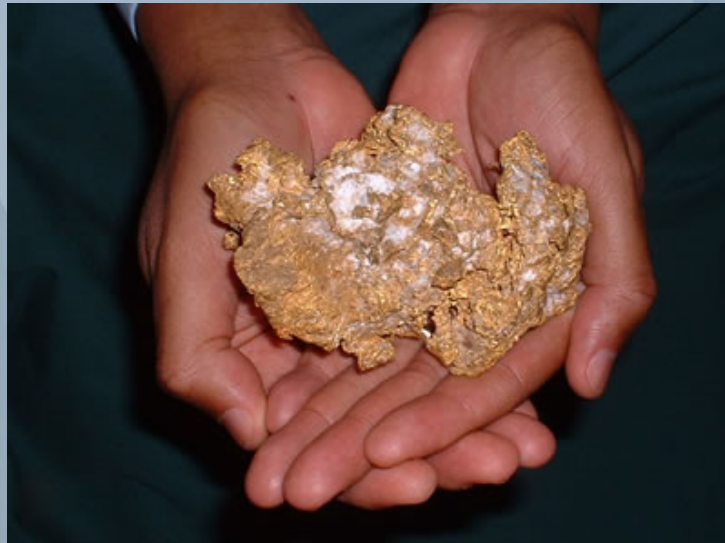


## Orezone Resources Inc.: Update on Essakane Gold Project Burkina Faso

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## 1 Summary

This Technical Report describes the Essakane Gold Project (the “Project”), a mineral exploration and development area located in the Oudalan Province of Burkina Faso. The Project is indirectly held by Essakane (BVI) Limited. Gold Fields Essakane (BVI) Limited has earned a 60% interest in Essakane (BVI) Limited and the remaining 40% interest is owned by Orezone Essakane (BVI) Limited.

Gold Fields Essakane (BVI) Limited is a wholly owned subsidiary of Gold Fields Orogen Holding (BVI) Limited. Orezone Essakane (BVI) is a wholly owned subsidiary of Orezone Resources Inc. (“Orezone”).

This report was prepared to allow the Directors of Gold Fields Essakane (BVI) Limited and Orezone Essakane (BVI) Limited to independently reach an informed decision regarding the economic viability of placing the Project into production. It reports the results of a positive Definitive Feasibility Study (“DFS”) which was completed by Gold Fields Essakane (BVI) Limited and delivered to the Directors of Essakane (BVI) Limited on September 11, 2007.

The report also details the updated Mineral Resources associated with the May 2007 resource model which Orezone announced on 19 September 2007.

The operating company in Burkina Faso is Gold Fields Burkina Faso SARL, which engaged GRD Minproc (Pty) Ltd (“GRD Minproc”) to undertake the DFS for the Project. GRD Minproc is the South African subsidiary of GRD Minproc, Australia.

Within this report, Gold Fields Burkina Faso SARL, Gold Fields Essakane (BVI) Limited, Orezone Essakane (BVI) Limited and Essakane (BVI) Limited will be jointly referred to as “Essakane”.

The Project is located in the northeast of Burkina Faso in proximity to the Gorouol River, between the settlements of Gorom-Gorom in the west and Falagountou in the east. A free milling, non-refractory gold deposit of 46.4 million tonnes has been identified and the intention of the Project is to mine and process this deposit at a rate of 5.4 million tonnes per year. The expected life of the mine is 8.6 years and during this time a total of 2.51 million ounces of gold will be produced.

The facilities for the project comprise a surface mining operation, an overburden storage facility, a gold processing plant and a tailing storage facility. The requisite infrastructure includes a village for the mine employees, the mining fleet and maintenance facilities, electrical power generation and water supplies. It will also be necessary to relocate 2 562 households that will be affected by the Project.

The estimated cost of the Project to an accuracy of +15% is US\$ 346.5 million. Operating and capital costs are based at July 2007 and no allowance has been made for escalation.

Construction of the mine and resettlement villages is planned to start in November 2007 and construction of the process plant in April 2008. Mining activities will commence in September 2009 and ore commissioning of the plant is planned for late October/November 2009. The process plant will be handed over in December of 2009 ready for full production in 2010.

This positive DFS contrasts with the pre-feasibility study (“PFS”) which was prepared by Grinaker-LTA for the Project and issued in October 2005. This PFS detailed the historical exploration and development of the Essakane Main Zone gold deposit (“EMZ”), which forms the basis of the Project.

The PFS base case utilised an annual processing rate of 5.4 million tonnes of ore at a head grade of 1.98 g/t over an 8.1 year life of mine through a gravity / CIL

process facility. The predicted overall capital cost of the Project was US\$ 311.1 million in August 2005, inclusive of allowances for estimating contingencies, owner's capital costs and pre-production cost. This capital cost estimate was to an accuracy of +/- 25% but did not include the cost of sustaining capital, reclamation and closure costs, or the cost of the DFS estimated at a combined additional cost of US\$ 32.2 million.

Operating costs were estimated to average US\$ 12.83 per tonne of ore processed over the life of the mine. The PFS developed a financial model based on a price of gold of US\$ 375 per fine ounce and concluded that the Project, at a capital cost of US\$ 311.1 million and operating costs of US\$ 12.83 per tonne of ore processed, was uneconomic as the internal rate of return (IRR) was 0.6%.

## 1.1 Summary of geology and mineralisation

The EMZ is currently the largest known gold deposit in Burkina Faso. Gold occurs in a quartz vein stockwork within an anticlinally folded succession of Birimian-age arenite and argillite. The geological model and assay data used in the DFS are the culmination of exploration work that started with BHP in 1995. Ranger Minerals and Orezone were also project operators before Essakane took over as operator in January 2006.

The so-called main arenite is the economically most important rock type within the EMZ. The arenite occurs within east limb, fold hinge and west limb lithostructural domains which have been recognized in surface trenches and drilling. The highest concentration of quartz veins and gold is found in the hinge zone of the anticline and in the upper part of the east limb main arenite. The top contact is a sharp grade contact which can be traced across the 2 500 m strike length of the modelled EMZ. The limits of mineralisation to the east and west are well defined by drilling through this grade boundary.

The top of the main arenite is 50 m below surface at the northern end of the geological model. The deposit is open to the north but economic mineralisation is progressively deeper. Quartz vein density decreases to the south and the opportunity to locate additional economic mineralisation appears poor. Drilling has not closed off gold mineralisation at depth below the geological model.

## 1.2 Summary of exploration concept

The EMZ is a coarse gold deposit with particles up to 5 mm in diameter. Analytical testing after the PFS showed that precision and accuracy of assaying could be improved by switching to LeachWELL rapid cyanide leach of 1 kg subsamples. Improved sample preparation and LeachWELL were introduced in January 2006. A re-assay program of historical pulp rejects was started at the same time, replacing 8 800 historical assay data by November 2006. This was increased to 28 640 re-assays by May 2007. Most of the new assays were completed at SGS Tarkwa in Ghana using the LWL69M method (a variety of LeachWELL) which is the same method used very successfully by Tarkwa Gold Mine in its grade control programs.

Approximately 25 000 m of core drilling was completed in 2006 to infill and expand the mineral inventory. Previous studies had relied mainly on logging of fine-grained chips from shallow reverse circulation drilling. The new cores added significantly to the confirmation of geological structure and modeling of gold mineralisation.

The orientation of drilling was changed from vertical to west-to-east dipped holes and drilling was extended at depth beyond the expected limits of surface mining. In-seam drilling confirmed the continuity of gold mineralisation to 300m within the main arenite layer.

Compared with the PFS, the January 2007 and subsequent May 2007 block models are based on a new and expanded geological model for the EMZ with new gold assay data.

Areas to be covered by the proposed plant infrastructure and overburden storage sites were sterilized by mapping, drilling and ground geophysical survey. No other significant gold bearing structures or zones have been located.

The exploration history includes work of BHP (1995-1996), Ranger Minerals (2000-2001), Orezone Resources Inc (2003-2005) and Essakane (2006-2007). Several sampling and analytical approaches have been applied in the past, including fire assay, BLEG (bulk leach extractable gold) cyanide leach and LeachWELL assisted cyanide leach. Visible gold is common in drill core and extensive Artisanal mining has taken place in the uppermost, heavily weathered part of the deposit. Testwork on sample repeatability has categorised the EMZ as a difficult sampling problem, with special requirements needed to adequately determine the gold grades.

The work of previous operators is characterised by poor records of analytical quality control. Essakane developed practical sampling and analytical protocols that were applied in the 2006 diamond drilling campaign. In addition, Essakane's analytical work was undertaken under cover of Certified Reference Materials, blanks and preparation duplicates in order to ensure reliable assay quality throughout the sampling campaign. Essakane also recovered reject sample materials from previous sampling campaigns that were stored at the mine site and treated these materials in the same way. Sample rejects from the Ranger Minerals' sampling programs had been stored in bio-degradable bags: none of this material could be recovered for re-assay. A twin hole drill campaign consisting of 27 holes was thus completed to compare Essakane assays with Ranger's results.

Detailed re-assay of 8 800 duplicate samples was completed by November 2006. These samples were recovered from the most densely drilled Panel F part of the deposit. The re-assay results were used to remediate the historical assays in the following way. For each analytical type, paired data sets were examined as quantile-quantile (QQ) plots and step-wise factors were developed that mapped the historical assay quantiles onto the re-assay quantiles. These factors were then applied to the remaining historical assays which had not been re-assayed. This process adjusts the global mean grade but is incapable of resolving the local imprecision present in the data. The January 2007 Mineral Resource estimate was based on the November 2006 data set which contained remediated data developed using 8 800 pairs of re-assay data.

Essakane continued the re-assay program into early 2007 and a total of 28 640 pairs of re-assay data were available by April 2007. These data were used to regenerate new remediation factors for the remaining historical data, and an updated Mineral Resource was estimated in May 2007. Data from the Ranger Minerals drilling were used without any adjustments for analytical bias.

### 1.3 Resource estimation

Each of the January and May 2007 mineral Resource estimates thus consists of: (i) LWL69M assays from Essakane's 2006 drilling; (ii) LWL69M re-assays of historical BHP and Orezone samples; (iii) Ranger Minerals assay data (as is); (iv) Remediated assays for BHP and Orezone samples which were not re-assayed by Essakane.

For the May 2007 estimate, five groups of historical BHP and Orezone assay data were remediated, amounting to 58 189 samples. Each group represents a specific assay laboratory and assay method (e.g., fire assay at Abilabs or ITS). Seventy three per cent of all remediated samples are BLEG assays from SGS Tarkwa and TransWorld laboratories in Ghana.

A summary of data statistics for May 2007 remediated pairs is shown in Table 1.1. The statistics of subsets above 1.0 g/t are also presented. Typically less than 7% of the unremediated BHP and Orezone data have grades above 1.0 g/t which illustrates the low grade of samples not re-assayed.

A count of 3 298 assays exceed 1 g/t within the unremediated dataset whereas 3 707 assays exceed 1 g/t in the corresponding remediated dataset. Remediation has thus changed 409 assays from less than 1 g/t to more than 1 g/t which is only 0.29% of the total May 2007 dataset of. On this basis the remediation process is not considered to be a significant geological risk.

Within the January 2007 estimate, the remediation factors changed 183 assays from <1 g/t to >1 g/t, representing 0.14% of the 133 255 assays used in the January model.

The main benefit of the additional re-assays is thus seen at lower gold cut-off grades, with a tonnage increase of 12% for both 0.5 and 0.8 g/t cut-off grades.

### 1.3.1 Uniform Conditioning (UC)

Essakane adopted a recoverable Resource estimation method instead of direct linear estimation of grades into SMU blocks. Testwork showed that edge-effects are present within the EMZ mineralisation and that a diffusion process model is a more appropriate random function model than a mosaic process to describe the mineralisation. UC was selected as the preferred estimation method, using a discrete Gaussian change of support model. This estimation process involves, firstly, estimation of the average grade of large blocks (panels) by Ordinary Kriging (OK), followed by prediction of the proportion of the panel that exceeds a defined cut-off grade and the average grade of that proportion.

### 1.3.2 Geological modeling

Geological modelling was undertaken in Datamine to build the main lithological units comprising Hangingwall Argillite, Main Arenite, Footwall Argillite and Footwall Arenite. The locations of these units in relation to the anticlinal fold axis determines the lithostructural domains used for Resource estimation.

The weathering profile consists of an upper Saprolite unit, a lower Saprolite (saprock) unit and the underlying Fresh domain. The boundaries of these units were modelled in Datamine and were developed from logs of all RC and DD drillholes supplemented by geotechnical drilling and logging. In some areas the study found large differences between early RC drillholes and adjacent 2006 drill cores. The reasons are inconsistencies between project geologists and project operators since 1995, and difficulties in picking gradational contacts from fine grained RC cuttings. The result, as was seen in the PFS, is irregular and geologically incoherent surfaces which could overstate the volumes of weathered rock. Greater weight has thus been allocated to the 2006 drillcores in modelling the weathering surfaces.

Gold occurs in quartz veins that are bedding parallel or steep and cross-cutting. Complex pressure-solution veining has also been identified in the fold axial zone. Veins vary from millimetres to tens of centimeters in thickness and the density of veining varies from isolated single veins to dense stockworks. Both north – south and east – west trending veins with visible gold are present. Statistics show that these vein directions have similar grade characteristics.

Table 1.1 Statistics of May 2007 remediated assays

Laboratory	Data	Count	Mean	Min	Max	Std Dev	CoV	Count >1g/t	Average >1g/t	Proportion of total assays
Abilabs fire Assay (Orezone)	Unremediated	8,767	0.53	0.005	206.82	4.03	7.62	570	6.31	6.5%
	Remediated	8,767	0.57	0.005	186.14	3.81	6.71	649	6.11	7.4%
ITS fire assay (BHP)	Unremediated	2,155	0.28	0.003	76.59	1.96	7.01	101	3.85	4.7%
	Remediated	2,155	0.26	0.003	58.97	1.53	5.81	110	3.22	5.1%
TransWorld LW (Orezone)	Unremediated	4,620	0.49	0.001	341.26	5.93	11.99	317	5.75	6.9%
	Remediated	4,620	0.68	0.001	546.02	9.47	13.89	333	8.06	7.2%
TransWorld BLEG (Orezone)	Unremediated	16,663	0.33	0.001	336.15	3.51	10.67	816	4.85	4.9%
	Remediated	16,663	0.43	0.001	537.84	5.58	12.97	889	6.40	5.3%
SGS Tarkwa BLEG (Orezone)	Unremediated	25,984	0.35	0.001	141.86	2.40	6.76	1,494	4.38	5.7%
	Remediated	25,984	0.43	0.001	148.95	2.75	6.43	1,726	4.88	6.6%

### 1.3.3 Resource estimation methodology

Resource estimation made extensive use of the geological model. Each main lithological unit was considered as a separate geostatistical domain. Visual checks of the domains were made to confirm that average grades and density of mineralization are different. Significant differences were also observed between the east and west limbs of the fold, and the limbs were thus included in the definition of geostatistical domains.

The influence of weathering types was also considered. In all cases, where a significant population of weathered samples exists, separate weathered and fresh lithological domains were defined.

Statistics of raw data showed high coefficients of variation (relative standard deviation) of gold grades. Samples were thus composited to 3 m lengths to reduce the variability of grades and capping was used to suppress the local impact of very high grades and also reduce the relative variance of the data. A total of ten domains were defined and domain boundaries are treated as hard.

Variography of raw assay data failed to define interpretable structures but non-linear transforms like pairwise relative and logarithmic transform show significantly clearer structures. All data were subjected to a Gaussian transform and directional variograms were developed on this transform. Models based on authorised structures were developed and then back-transformed to represent the spatial continuity of raw gold grades. These variogram models were used for the OK linear estimation of grades into large panels.

The drillhole spacing is a nominal 50 m(Y) x 25 m(X) with local areas drilled at 25 x 25 m spacing. Large panel dimensions were thus set at 25 m(X) x 50 m(Y) x 6 m(RL). The large panels were estimated using OK. Steps were taken to ensure that conditional biases were minimised within these estimates by monitoring the regression slope ( $Z|Z^*$ ).

Selective mining with SMU dimensions of 2.5 m(X) x 5 m(Y) x 3 m(Z) was considered appropriate. The UC approach assumes that the kriged large panel grade is the local mean grade of a 'distribution' of SMU grades within that panel, whose dispersion variance can be estimated from the relevant variogram model. In deriving the dispersion variance of the SMU, a modification to consider the information effect was also included. The information effect for each domain was estimated by simulating grade control drillholes and estimating SMU-blocks from these data.

### 1.3.4 Resource classification

Resource classification meets the requirements of SAMREC, JORC and the CIM guidelines adopted in NI 43-101. The classification was based on the geological and geostatistical quality indicators of the large panel estimates. Comparison between the distance to a sample, the theoretical regression slope  $Z|Z^*$  and the kriging efficiency showed a high correlation between the latter two parameters. A first pass classification was applied to define blocks with a kriging efficiency of 0.25 or greater as possible Indicated Resource blocks. This approach led to instances of isolated blocks with lower kriging efficiencies (i.e. Inferred blocks under this classification) surrounded by Indicated blocks as well as isolated Indicated blocks surrounded by Inferred blocks. To overcome this, a wireframe surface was developed from serial cross-sections that permitted rational exclusion or inclusion of isolated blocks.

No Measured Resources have been defined. The use of remediation factors and coarse gold sampling problems in the EMZ precludes classification of densely drilled parts of the resource as Measured.

Estimation of block confidence intervals was applied to the east limb main arenite using a non-linear kriging estimation technique. The non-linear estimates provided a mean grade estimate that is 6% lower than the linear estimate. Also, the analysis shows that high uncertainties can exist for the individual panel grades, meaning that grade control ahead of mining is important. In the context of full production, a 25 m x 50 m x 6 m panel of saprolite ore would be mined in one day and a fresh panel in two days.

### 1.3.5 Dry bulk density

Dry bulk densities were collected at the scale of core trays by weighing air-dried HQ diameter core and referencing the measured weight to the actual core length in the tray. Drill cores were dried in the sun before the trays were weighed. A core tray holds 3 m and, as such, errors arising from corrections for core loss are considered to be small.

Quality assurance was provided by measuring the densities of 10 – 20 cm lengths of sealed core using the standard immersion method. The immersion densities were found to be consistently higher for weathered rocks, caused by biased selection of intact core samples by technicians. The immersion density data for saprolite samples were thus not used.

### 1.3.6 Recoverable resource model

The UC estimate was converted to a Localised Uniform Conditioned (LUC) estimate to simplify the recoverable Resource model. The LUC block model is significantly simpler to work with. Comparison of the UC and LUC estimates confirms very close reproduction of the UC grade-tonnage results via the LUC process.

### 1.3.7 LWL69M leach efficiency model

All LWL69M assays represent gold-solution grades after leaching for 10 hours and were reported before fire assay of washed LWL69M tailing. Comparison of these gold-solution grades against historical BLEG data pairs showed that the LWL69M gold-solution values are still higher than total gold estimated by BLEG (g/t) + BLEG tailing (g/t). Historically, BLEG tailing grades were estimated by single fire assay if the BLEG solution grade was greater than 1 g/t. Single fire assay of 2kg BLEG tailings was clearly inadequate and the reason for this is an increase in the nugget effect caused by partial leaching, i.e., BLEG dissolved small particles leaving large gold particles in the tailings which were very difficult to detect in single 50g subsampling.

Essakane made use of routine screen fire assay of 1kg preparation duplicates to demonstrate efficient leaching of gold by LWL69M. Once this had been established the LWL69M protocol was changed to single 50 g fire assay of LWL69M tailing for 10 per cent of samples submitted to SGS Tarkwa. The result is an incomplete LWL69M tailing dataset.

Due to the incomplete dataset, the remediation process for historical data could only use LWL69M solution dataset, which was complete. Remediation of BLEG or other LW assays thus produced gold-solution equivalent values. In the same way, a fire assay sample that was not re-assayed by Essakane was remediated to a gold-solution equivalent value.

All variography and estimation in the January and May 2007 models has been completed using the LWL69M and remediated LWL69M gold-solution equivalent values. The gold-solution estimates were then converted to *in-situ* total gold values by applying LWL69M leach factors derived from the LWL69M values with matching tailing dataset. The data were used to estimate % leach depending on weathering type, lithology and grade. The recovery factors were applied to the

block estimates, converting solution and remediated solution grades to in-situ total gold grades.

### 1.3.8 Resource tabulation

Mineral Resources have been constrained within a US\$ 650/oz Whittle pit shell for reporting purposes. Pit shell optimisation was initially completed by Snowden on the January 2007 Mineral Resource model and reported to Essakane on 13 March 2007. The total and constrained Mineral Resource estimates are presented in Table 1.2. Orezone announced this resource estimate on 10 April 2007.

Snowden also completed pit optimization on the May 2007 model and the updated total and constrained Mineral Resource estimates are reported in Table 1.3 by gold cut-off grade and classification. Orezone announced this resource estimate on 19 September 2007 at gold cut-off grades 0.5 g/t and 1.0 g/t.

Table 1.2 January 2007 Mineral Resource Estimate released on 10 April 2007

		<b>Total Mineral Resource</b>			
<b>Jan-07</b>	<b>COG (g/t)</b>	<b>0.50</b>	<b>0.80</b>	<b>1.00</b>	<b>1.20</b>
Indicated	Tonnes (Mt)	68.7	46.0	36.5	29.3
	Grade (g/t Au)	1.56	2.01	2.31	2.60
	Au (Moz)	3.44	2.98	2.71	2.45
Inferred	Tonnes (Mt)	23.5	15.2	11.9	9.5
	Grade (g/t Au)	1.50	1.98	2.27	2.57
	Au (Moz)	1.13	0.96	0.87	0.79

		<b>Mineral Resource Constrained by US\$ 650/oz pit shell</b>			
	<b>COG (g/t)</b>	<b>0.50</b>	<b>0.80</b>	<b>1.00</b>	<b>1.20</b>
Indicated	Tonnes (Mt)	63.2	43.3	34.6	28.0
	Grade (g/t Au)	1.60	2.05	2.34	2.63
	Au (Moz)	3.26	2.85	2.60	2.37
Inferred	Tonnes (Mt)	14.7	10.4	8.4	6.9
	Grade (g/t Au)	1.69	2.13	2.41	2.70
	Au (Moz)	0.80	0.71	0.65	0.60

Note: Tonnes have been rounded to the nearest hundred thousand.

Table 1.3 May 2007 Mineral Resource Estimate released on 19 September 2007

<b>Total Mineral Resource</b>					
<b>May-07</b>	<b>COG (g/t)</b>	<b>0.50</b>	<b>0.80</b>	<b>1.00</b>	<b>1.20</b>
Indicated	Tonnes (Mt)	78.4	52.9	42.2	34.2
	Grade (g/t Au)	1.58	2.04	2.33	2.62
	Au (Moz)	3.99	3.47	3.16	2.88
Inferred	Tonnes (Mt)	27.4	17.6	13.7	10.8
	Grade (g/t Au)	1.44	1.89	2.17	2.46
	Au (Moz)	1.27	1.07	0.96	0.86

<b>Mineral Resource Constrained by US\$ 650/oz pit shell</b>					
	<b>COG (g/t)</b>	<b>0.50</b>	<b>0.80</b>	<b>1.00</b>	<b>1.20</b>
Indicated	Tonnes (Mt)	73.4	50.4	40.5	33.0
	Grade (g/t Au)	1.62	2.07	2.35	2.64
	Au (Moz)	3.82	3.35	3.06	2.80
Inferred	Tonnes (Mt)	16.1	11.6	9.5	7.8
	Grade (g/t Au)	1.66	2.06	2.31	2.58
	Au (Moz)	0.86	0.77	0.71	0.64

Note: Tonnes have been rounded to the nearest hundred thousand

Mineral Reserves have been estimated within a US\$ 500/oz mine design shell using only Indicated Mineral Resources and are presented in Table 1.4. The reporting cut-off gold grades in the Table relate to recovered mill grades, whereas the Diluted Grade column refers to diluted mill head grades.

Table 1.4 May 2007 Mineral Reserve estimate

<b>Category</b>	<b>Reporting cut-off (g/t Au)</b>	<b>Tonnage (Mt)</b>	<b>Diluted Grade (g/t Au)</b>	<b>Contained gold (Koz)</b>
Probable	Oxide 0.52	11.6	1.47	547
	Transition 0.58	10.1	1.71	555
	Fresh 0.64	24.8	1.94	1 547
<b>Total Probable</b>		<b>46.4</b>	<b>1.78</b>	<b>2 649</b>
Proven		-	-	-
<b>Total Proven</b>		<b>-</b>	<b>-</b>	<b>-</b>
<b>Total</b>		<b>46.4</b>	<b>1.78</b>	<b>2 649</b>

Note: Mineral Resources are inclusive of Mineral Reserves. Tonnes and ounces have been rounded and this may have resulted in minor discrepancies.

### 1.4 Summary of status of exploration, development and operations

Snowden provided external audit and inputs to QAQC, sampling protocols, geological modeling and estimation since January 2006 when Essakane undertook to complete a DFS by Q3 of 2007. Resource definition drilling for the DFS was completed by June 2006 and the assays were reported by November 2006. However, the re-assay of historical samples was extended to April 2007. Condemnation and geotechnical drilling was completed in the second half of 2006.

Essakane commissioned GRD Minproc as overall study manager for the DFS. The GRD Minproc scope of services was to conduct engineering for the Project and to estimate the capital and operating costs for the surface mine, metallurgical plant, tailing storage facility, infrastructure and resettlement villages to an accuracy of +15%.

GRD Minproc was responsible for the management and coordination of the activities of the following consultants employed by Essakane for the Project as detailed in Table 1.5.

**Table 1.5 Activities carried out by various parties and co-ordinated by GRD Minproc**

<b>Consultants</b>	<b>Description</b>
Essakane	Mineral resources and mine planning
Knight Piésold	Environmental study and report
	Tailing storage facility design
	Gorouol River water storage facility design
	Permitting
	Geotechnical report
	Hydrological study report
	Overburden characterisation
	Surface lake modelling
	Mine closure and reclamation report
	GCS
Overall water balance report	
Water supply for the resettlement villages	
Potable water supplies	
	Assessment of the Gorouol River alluvial aquifer
rePlan	Village relocation and settlement

The DFS addressed the activities listed in Table 1.6 in sufficient detail, identifying areas of potential cost savings, so as to ensure the technical, environmental, social and economic viability of the Project and to support a capital and operating cost estimate to an accuracy of +15%.

Table 1.6 List of activities documented within the DFS

Activity	Description
Environmental management and statutory requirements	Coordinate the work carried out by KP for completion of the EIA Report
Social, relocation and resettlement	Coordinate rePlan and Essakane activities on the resettlement villages and the required infrastructure. Preparation of the cost estimate for the resettlement village based upon rePlan designs.
Hydrological and hydrogeological investigations	Coordinate activities of KP and sub-consultants GCS into surface and underground water supplies and quality, and provide input into the preparation of the overall water balance.
Geotechnical investigation	Coordinate the geotechnical study by KP for the surface mine, roads, earthworks and civil works
Geology	Review the mineral resource block model and detailed mine plan provided by Essakane.
Overburden disposal	Review the detailed planning of the mine and overburden storage facility carried out by Essakane. Review the KP designs and proposals for the tailing storage facility and storage of overburden
Process engineering	Review testwork data generated to date and design the process plant
Tailing storage facility	Review and cost the tailing storage facility designed by KP.
Engineering	Prepare engineering designs in sufficient detail to specify and cost all capital equipment and services necessary for the Project.
Infrastructure and services	Identify and design the necessary infrastructure required to service the requirements of the Project.
Logistics and route survey	Prepare a logistics and route survey report that will ensure that materials, equipment and personnel are safely and reliably transported to site in a cost effective manner.
Capital cost estimate	Prepare a capital cost estimate for items of capital equipment and services with an accuracy range of +15%.
Operating cost estimate	Prepare an operating cost estimate for the Project with an accuracy range of $\pm 15\%$ .
Project implementation plan	Prepare a project implementation plan to include a milestone schedule showing critical activities for the completion of the Project.
Risk analysis	Carry out a Risk analysis with Essakane.
EPCM proposal	Prepare an EPCM proposal that will be incorporated into the DFS report

## 1.5 QP conclusions and recommendations

GRD Minproc has completed a positive DFS for surface mining and gravity / CIL processing of the EMZ gold deposit at a rate of 5.4 Mtpa for 8.6 years. The average annual gold production is 292 000 ounces and gold sales over the life of mine are estimated to be 2 507 000 ounces. The initial capital cost is estimated to be US\$346.5 million with an intended level of accuracy of + 15%.

The initial capital cost estimates have a base date of July 2007 and no allowance has been included for price escalation or currency fluctuations.

Based on a gold price assumption of US \$580/oz, and relative to an assumed life of mine oil price of US\$ 50/barrel, the pre-tax Project IRR (internal rate of return) is estimated to be 14.8% with a pre-tax NPV (net present value) of US\$ 173.3 million at a 5% discount rate.

Cash operating costs during the first 4 years of the project are estimated to be US\$294/oz. The DFS has demonstrated that the project is medium to low risk at a gold price assumption of US\$ 580/oz and oil price of US\$ 50/barrel.

The study shows that the Project's economics are most sensitive to gold and oil price. This is demonstrated in Table 1.7 which presents the summary of financial results for four separate sets of commodity price assumptions.

**Table 1.7 Summary of financial results for the Project**

	<b>Case 1</b>	<b>Case 2</b>	<b>Case 3</b>	<b>Case 4</b>
Gold price (US\$/oz)	580	460	650	720
Oil price (US\$/bbl)	50	40	60	80
Ounces recovered (000 oz)	2 507	2 507	2 507	2 507
Average annual production (000 oz)	292	292	292	292
Cash cost (US\$/oz)	298	269	321	356
Total cash cost (US\$/oz)	447	418	469	505
Total free carried cash cost (US\$/oz)	497	464	521	561
Pre-tax project IRR (%)	14.8%	5.8%	18.8%	21.5%
0% Pre-tax NPV (US\$ 000)	346 332	117 964	465 638	551 565
5% Pre-tax NPV (US\$ 000)	173 257	12 901	257 100	317 667
7.5% Pre-tax NPV (US\$ 000)	113 263	(22 610)	184 333	235 749

## 2 Introduction

This Technical Report has been prepared by Snowden for Orezone Resources Inc., in compliance with the disclosure requirements of the Canadian National Instrument 43-101 (NI 43-101). The triggers for the preparation of this report are the 11 and 19 September 2007 press releases of Orezone Resources Inc., disclosing the results of a definitive feasibility study (DFS) on the Essakane Gold Project and an increase in reported Mineral Resources, respectively.

Unless otherwise stated, information and data contained in this report or used in its preparation has been provided by Orezone Resources Inc.

The Qualified Persons for preparation of the report are Mr J Hawxby who visited the project site during October 2006 and September 2007, Mr I Glacken who visited site during August 2006, Mr M Harley who visited site during November 2006, Mr O. Varaud who visited site during October 2006 and Mr S Solomons who visited site in April 2007. Mr S Solomons is an employee of Gold Fields Australia Limited, which is an entity wholly independent of Orezone Resources Inc.

The responsibilities of each author are provided in Table 2.1.

**Table 2.1 Responsibilities of each QP**

<b>Author</b>	<b>Responsible for section/s</b>
Mr J Hawxby	4 – 6, 16, 18 (except 18.17) - 20
Mr I Glacken	1, overall compilation
Dr M Harley	6.3, 7 – 14, 17.1 – 17.2, 19 - 20
Mr O Veraud	17.3
Mr S Solomons	4 – 6, 16, 18 - 20

Unless otherwise stated, all currencies are expressed in US dollars (\$, US\$).

### 3 Reliance on other experts

There has been no reliance on experts who are not Qualified Persons in the preparation of this report.

## 4 Property description and location

### 4.1 Area

The EMZ deposit is located in the north central part of the Tassiri Permit, one of seven exploration permits (“*permis de recherche*”) comprising the Project in the Oudalan and Seno provinces of NE Burkina Faso. The northern end of a US\$650/oz Whittle pit shell developed on the EMZ crosses Tassiri’s northern boundary. The area of the Tassiri Permit is 175.5 km<sup>2</sup>. The areas of the other Permits are listed in Table 4.1.

Table 4.1 Tenement details: permit arrêté numbers and expiry dates

Permit Name	Arrêté	Date	Date	Surface area km <sup>2</sup>
		Granted	Expiry	
Tassiri	03/028/MCE/SG/DGMGC	10-Jul-00	10-Jul-09	175.5
Alkoma	03/030/MCE/SG/DGMGC	10-Jul-00	10-Jul-09	174.3
Dembam	03/026/MCE/SG/DGMGC	10-Jul-00	10-Jul-09	179.4
Gomo	03/029/MCE/SG/DGMGC	10-Jul-00	10-Jul-09	171.6
Gossey	03/027/MCE/SG/DGMGC	10-Jul-00	10-Jul-09	178.1
Lao Gountouré	03/031/MCE/SG/DGMGC	10-Jul-00	10-Jul-09	176.9
Korizéna	06/135/MCE/SG/DGMGC	21-Nov-06	21-Nov-15	192.2

### 4.2 Location

All the Permits are located on contiguous ground. The UTM co-ordinates of the corner points, as they appear in the respective arrêté, are listed in Table 4.2. The corner points of the Tassiri permit are marked in the field by four surveyed concrete posts.

### 4.3 Type of mineral tenure

Each exploration permit has been granted by the Minister of Mines, Quarries and Energy as an arrêté under Burkina Faso’s 2003 Mining Code (Code Minière, la loi n° 31 – 2003/AN du 08 mai 2003). The permits are presently in good standing and Essakane has been issued with Certificate # 1587/2007 (Issue date 04/10/2007) by Mr Seydou BALAMA at the Office Notarial in Ouagadougou.

The arrêté numbers and expiry dates are listed in Table 4.2. All permits except Korizéna have to be converted to mining licences (ML’s) by 10 July 2009 or be relinquished to the State with no residual interests. In terms of current Law each ML application requires a separate feasibility study but there are precedents in Burkina Faso for variations to this rule (e.g., Etruscan’s Youga project).

Table 4.2 Tenement details: boundary coordinates

Permit name	Point	Northing	Easting
Alkoma	A	1582851	177115
	B	1582633	194311
	C	1572484	194187
	D	1572699	177115
Dembam	A	1623457	177115
	B	1623226	194813
	C	1613078	194686
	D	1613305	177115
Gomo	A	1607850	194621
	B	1576161	205038
	C	1576161	194232
Gossey	A	1613305	177115
	B	1613078	194686
	C	1602929	194560
	D	1603154	177115
Korizéna	A	1603171	814519
	B	1603171	822612
	C	1599070	822612
	D	1599070	178012
	E	1593001	178012
	F	1593001	177115
	G	1572701	177115
	H	1572701	818352
	I	1584500	818352
	J	1584500	814519
Lao Gountouré	A	1603171	822612
	B	1602929	194560
	C	1592781	194435
	D	1592990	178012
	E	1599070	178012
	F	1599070	822612
Tassiri	A	1593001	177115
	B	1592781	194435
	C	1582633	194311
	D	1582851	177115
Projection	Ellipsoid:	Datum:	Clarke 1880

The Korizéna permit is valid until 21-Nov-2009 and would expire on 21-Nov-2015 after two renewals of 3 years each (the licence area is reduced by 25% upon the second renewal). The total entitlement of an exploration permit is nine years. Exploration permits are guaranteed by the Law and its associated decrees and arrêtés, providing the permit holder complies with annual exploration expenditures and reporting requirements.

The minimum annual exploration expenditure per permit is US\$ 100 000. Burkina Faso's 2003 Mining Code provides for an Exploration Permit to be superseded by an Exploitation Permit (synonymous with ML).

The requirements are:

- Exploration permit holders must have observed all prior obligations under the Mining Code.
- Applications for conversion to Exploitation Permit must be made at least three months before expiry of the Exploration Permit.
- The applications must be accompanied by a feasibility study and a deposit development and mining plan which must include an environmental impact study and rehabilitation plan.

#### 4.4 Issuer's interest

Orezone Resources Inc. owns 40% of the Project through its wholly owned subsidiary Orezone Essakane (BVI) Limited.

#### 4.5 Location of property boundaries

Figure 4.1 presents the location of the US\$ 650/oz EMZ pit shell (shown in red) in relation to all the Project permits and other gold prospects (borders of drill grids and soil anomalies are shown in blue hatching). Only the EMZ is the subject of the DFS and this Technical Report.

#### 4.6 Royalties, back-in rights, payments, agreements, encumbrances

The Government of Burkina Faso retains a 10 per cent free-carried interest in the Project. The effective interests in the Project are thus:

- Gold Fields Essakane (BVI) Limited 54%
- Orezone Essakane (BVI) Limited 36%
- Government of Burkina Faso 10%

#### 4.7 Environmental liabilities

Environmental liabilities as a result of mine development and closure are discussed in the subsequent text.

#### 4.8 Permits

Application for a Mining Convention is underway in Burkina Faso and is expected to be in place by the end of Calendar 2007. Four conditions have to be fulfilled prior to Project implementation:

- Acceptance of the ESIA (Environmental and Socio-Economic Impact Assessment) through a "positive notice" (Avis favorable) from the Burkina Faso Minister of Environment;
- Grant of a "Mining convention" by the Burkina Faso Government;
- Grant of a "Mining Permit" by the Burkina Faso Minister of Mines;
- Agreement with local populations on resettlement plans and process;

For acceptance of the ESIA, and in agreement with Burkina Faso's statutory requirements and International best practices, a completed Environmental and Socio-economic Impact Assessment was submitted to the Burkina Faso Minister of Environment on 8 August 2007.

The ongoing approval process is:

- Review of the document (up to 15 days);
- Public hearing (up to 60 days, including reporting);
- Review of the case by the Minister of Environment (up to 15 days);
- “Positive Notice” given by the Minister of Environment.

Approval of the ESIA is a critical step in the delivery of the Mining Permit and project cannot proceed without it. Approval should be obtained in early November 2007 provided the review process proceeds according to statute.

For granting of a Mining Convention, the Mining code proposes a ‘Standard Mining Convention’ which acts as a stability agreement. The Convention describes the Governmental commitments, operational tax regime and obligations of the company to Burkina Faso. Once executed, this Convention cannot be changed without the mutual agreement of both parties. If tax law changes are promulgated, the mining company can choose to adopt them (if deemed more advantageous) or stay with the current terms of the Convention.

The approval of the Convention requires the approval of the Cabinet. Typically approval of the standard Convention can be accomplished in a short period of time after the approval of the ESIA. However, certain points require further review and clarification in the instant case:

- Insure tax, legal and currency stability;
- Clarify terms of application;
- Avoid potential situations of double taxation;
- Include UEMOA mining code into the mining convention;

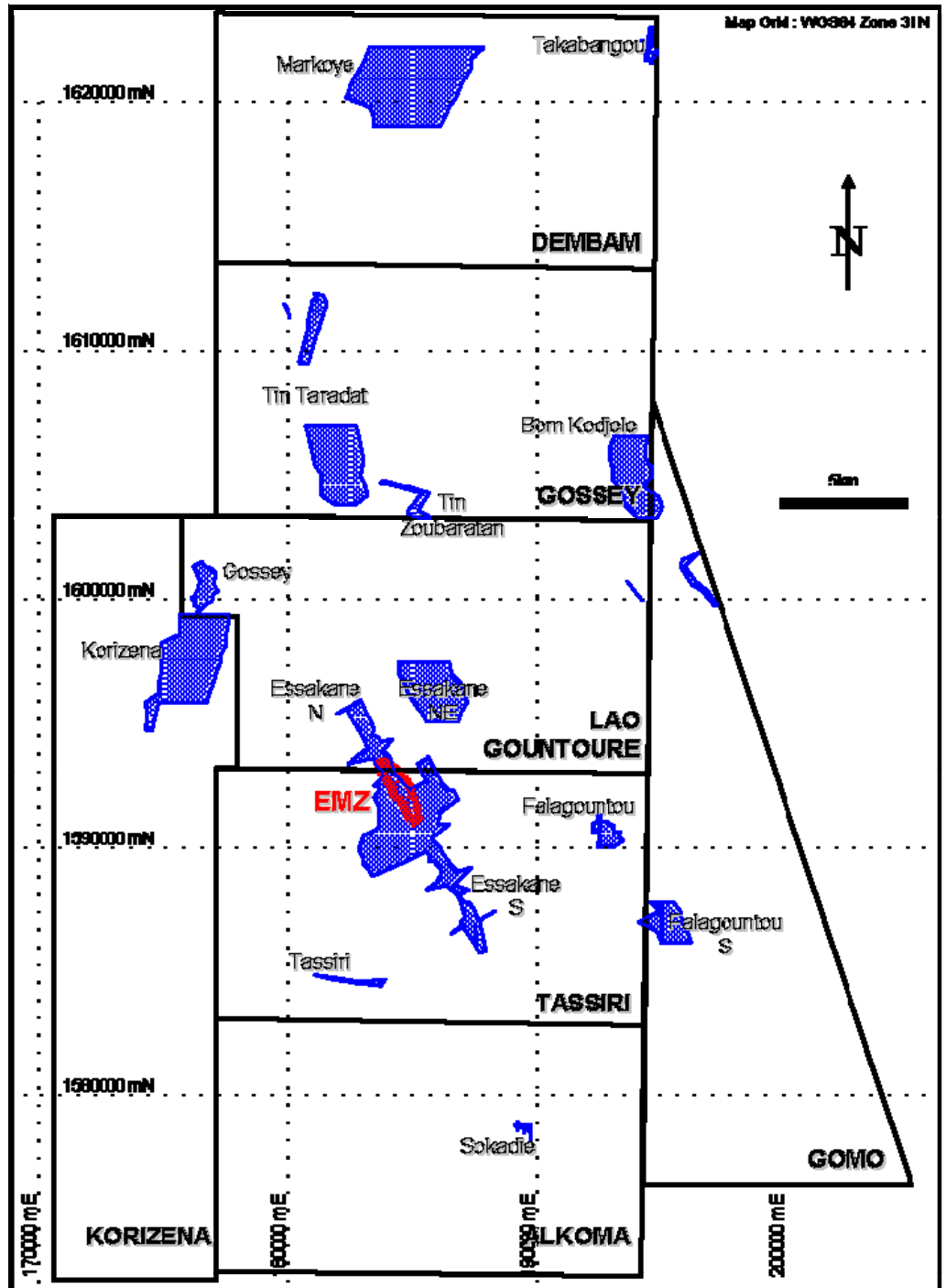
Discussions have started on these various aspects and an initial form of a Mining Convention is to be submitted to the Minister of Mines.

For granting of a Mining Permit, Essakane (BV) SARL currently holds seven exploration permits for the Essakane area. To start construction of the Project and mining, these exploration permits need to be converted into a Mining Permit. Burkina Faso requirements are:

- Realization and submittal of a Detailed Feasibility Study;
- Approval of an ESIA through a “positive notice” from the Minister of Environment;
- Agreement on a “Mining Convention” with the Government.

With the fulfilment of these conditions, a Mining Permit will be issued by the Minister of Mines. The Permit will be issued to a new legal entity, owned at 10% by the government of Burkina Faso.

Figure 4.1 Project permits and location of the EMZ Mineral Resources and other gold prospects



## 5 Accessibility, climate, local resources, infrastructure and physiography

### 5.1 Topography, elevation and vegetation

The Project area and specifically the area surrounding the EMZ are characterized by flat terrain. The EMZ forms a low ridge marked by extensive artisanal workings. Vegetation consists mostly of light scrub and seasonal grasses. Deforestation has been significant, particularly in the area surrounding the village of Essakane. The derelict heap leach pad and plant operated by CEMOB in the 1990's is located 1km east of the EMZ and contains approximately 1Mt of leached material.

### 5.2 Access

Access to and from the capital Ouagadougou is by paved road then by laterite road via the town of Dori, some 65km southwest of Essakane. Access via the town of Gorom-Gorom to the west is also possible. Within the permits access is via local tracks and paths which are suitable for two-wheel drive vehicles in the dry season but requires four-wheel drive vehicles and trucks in the wet season.

### 5.3 Proximity to population centre and transport

The Project straddles the boundary of the Oudalan and Seno provinces in the Sahel region of Burkina Faso and is approximately 330 km north-east of the capital, Ouagadougou. It is situated 42 km east of Gorom-Gorom which is the nearest largest town and the provincial capital of Oudalan. There are no major commercial activities in the project area and economic activity is confined to subsistence farming and Artisanal mining. There are no operating rail links and all transport is by road. There is a short air strip at Gorom-Gorom for chartered light aircraft.

### 5.4 Climate and length of operating season

The Essakane Site is located in the north east of Burkina Faso and the climate is typically Sahelian. Temperature ranges from 46°C to 10°C with evaporation rates of 3,000 mm/year. The mean annual rainfall is 450 mm with an estimated 100 year maxima of 143 mm in a 24 hour period.

A wet season occurs between late May and September, and the mean annual runoff in the Gorouol River is conservatively estimated to be 91 million m<sup>3</sup>/year. Rainfall is sporadic or absent during the rest of the year.

### 5.5 Surface rights

Permitting for surface rights is underway in Burkina Faso and all permits required for mine construction and associated infrastructure are expected by the end of Calendar 2007.

### 5.6 Infrastructure

The design infrastructure required for the proposed surface mine with capital cost estimates is described in Section 18. Existing infrastructure consists of an exploration camp with accommodation and canteen facilities for up to 150 persons.

#### 5.6.1 Power

Public infrastructure such as electrical power, maintained roads, scheme water and telecommunications does not exist in the Project area. The nearest grid-supplied power is at the town of Gorom-Gorom some 35km to the west-northwest. Electricity to the exploration site is provided by on-site diesel generators.

### **5.6.2 Water**

Water is pumped from wells (boreholes) in sufficient quantities for exploration drilling and the exploration camp, but a larger supply of water for mining operations would be engineered in the form of retaining dams to collect rainfall water during the wet season. This will include a small diversion weir on the Gorouol River west of the mine with gravity diversion into an Off-Channel Storage Facility (OCSF) with sufficient storage capacity to provide an assured water supply to the mine. Water to the exploration site is currently abstracted by pumping from water boreholes. The EMZ is located close to the Gorouol River and construction of water storage dams has been considered in the ESIA.

### **5.6.3 Mining personnel**

The Project may result in the displacement of 11,563 people living in 2,562 households and an initial draft Resettlement Action Plan (RAP) has been developed, in consultation with the community, to address the resettlement of these people. The RAP describes the policies, procedures, compensation rates, mitigation measures and schedule for resettlement.

The approach to involuntary resettlement is consistent with the International Finance Corporation's performance standards on Environmental and Social Sustainability and will adopt a collaborative approach involving the Government of Burkina Faso and the affected communities.

Essakane has initiated local training programs for artisans which include tuition in written and spoken French. It is envisaged that unskilled labour will be sourced locally with skilled labour drawn from Burkina Faso and the West African region.

### **5.6.4 Tailings and waste storage areas**

The EMZ is surrounded by ample flat and uninhabited land where tailings and surface dump storage will be possible and forms part of the ESIA. The tailings storage facility (TSF) is to be located southwest of the surface mine and the overburden storage facility (OSF) will be located east of the surface mine.

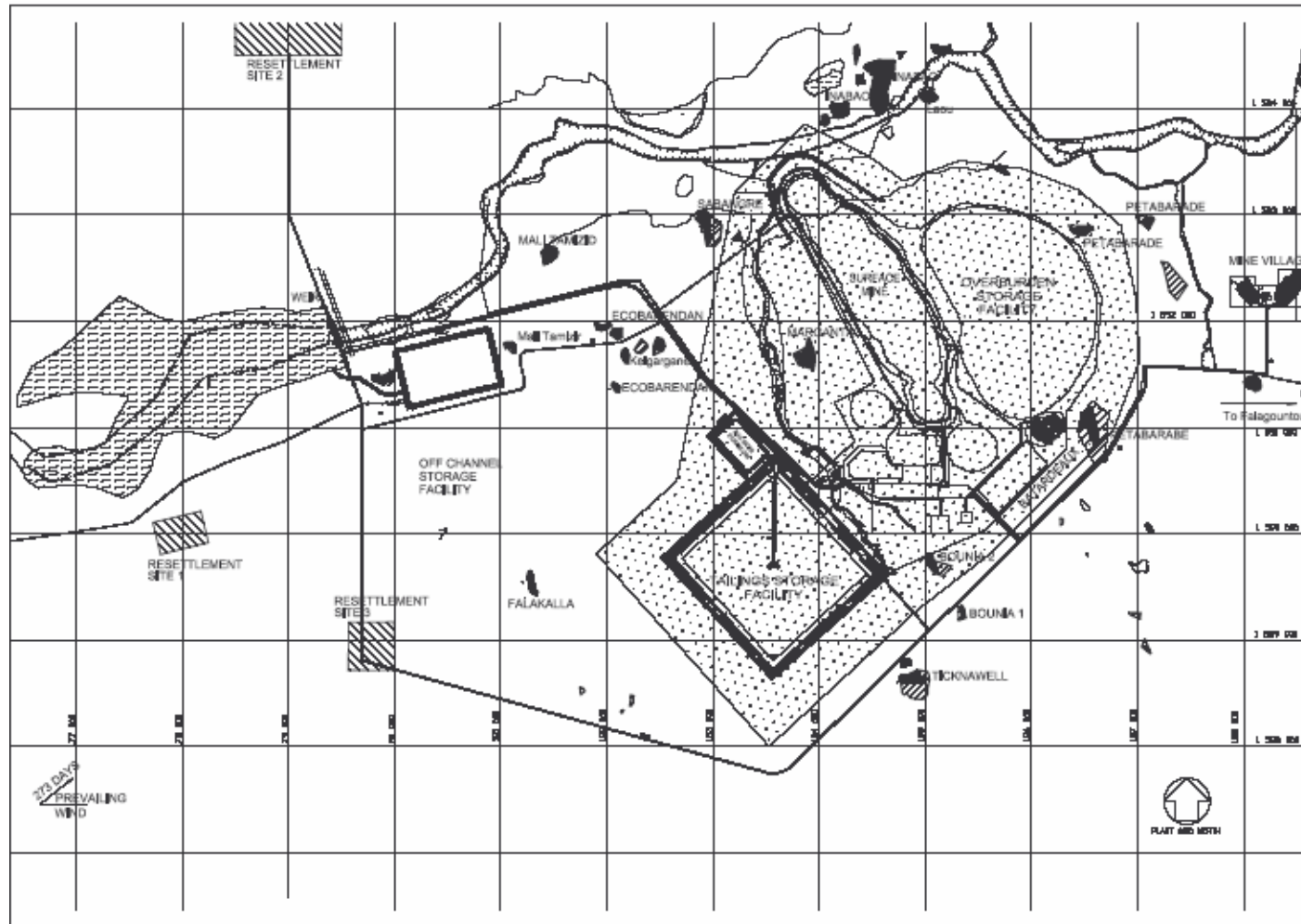
### **5.6.5 Heap leach pad areas**

Approximately 1 Mt of material on the existing CEMOB heap leach pad from previous mining operations may be processed as part of the EMZ mining operations. Currently there is no planning for heap leaching at the Project in the future.

### **5.6.6 Processing plant sites**

A description of the proposed plant site is given in Section 18. The overall site plan developed by GRD Minproc is presented in Figure 5.1.

Figure 5.1 Overall Project site plan



## 6 History

### 6.1 Prior ownership and ownership changes

The EMZ has been an active artisanal mining site (“orpailleur”) since 1985. Heap leach processing of gravity rejects from the artisanal winnowing and washings was carried out by CEMOB (Compagnie d’Exploitation des Mines d’Or du Burkina) in the period 1992 - 1999. From available records located in Burkina Faso by Orezone, CEMOB placed 1.01 Mt of material at an average grade of 1.9 g/t and achieved 73% recovery (Table 6.1). A sharp drop in reject grades after 1993 was caused by depletion of high grade laterite and the increased number of low grade workings around the initial discovery site. It is estimated that 250 000 oz of gold has been extracted from the local area since 1992.

Table 6.1 CEMOB gold production for period 1992 - 1999

Year	Tonnes to HL	Head grade (g/t)	Contained oz
1992	42 200	4.50	5 915
1993	115 751	5.10	18 388
1994	156 810	1.70	8 304
1995	148 165	1.50	6 923
1996	256 754	0.99	7 918
1997	165 125	0.84	4 321
1998	72 122	1.40	3 145
1999	50 072	2.00	3 151
Total	1 007 499	1.90	58 065

At its peak the deposit was worked by 25 000 miners. Cholera and major social problems prompted the Government to mobilize troops and administrators to organize the new mining community. A company named Société Filière Or (SFO) was formed in which the State held a 10% interest.

SFO controlled all mining and processing. Miners were granted small “claims” which were mined under SFO’s direction. During this period, the Bureau des Mines et de la Géologie du Burkina (BUMIGEB) undertook regional mapping and geochemical programs. BUMIGEB arranged and financed the program of heap leach test work between 1989 and 1991. The plant was constructed in 1992 and produced 18,000 oz in 1993 but averaged between 3 000 and 5 000 oz/year. Serious efforts were also made to leach saprolite from the EMZ but, based on verbal accounts, leaching failed because of high cement consumption and solution blinding in the heaps.

CEMOB was granted the Essakane Mining Research Permit in 1991. The permit covered most of the area which is now included within the Project (excluding the Gomo permit). BHP Minerals International Exploration Inc. (BHP) assisted CEMOB and explored the area from 1993 to 1996 under a proposed joint venture earn-in. This included an evaluation of the regional potential and other known gold prospects on the permit. BHP’s objective was to conclude an agreement with CEMOB after confirmation of potential. The company excavated and sampled 26

trenches (for 4 903 m) along the EMZ. Scout RC drilling was completed (including Falagountou and Gossey prospects), followed by RC drilling (7 949 m of vertical holes on a 100 x 50 m grid) and a few DD holes (1,510 m) in the main area of Artisanal mining on the EMZ. BHP estimated an in-house, Inferred saprolite (oxide) resource of 14.5 Mt @ 2.52 g/t for 1.2 Moz (at 1g/t cut-off grade) for the main arenite (based on due diligence documents provided to 3<sup>rd</sup> parties at the time BHP was withdrawing from further investment in West Africa).

Low gold prices and operational problems caused CEMOB to go into liquidation at the end of 1996. This complicated the CEMOB alliance and BHP decided to withdraw from the project around the time it withdrew from gold exploration and mining throughout West Africa. Gold Fields Ghana Limited reviewed BHP's data during due diligence periods in 1997 and 1998 but BHP was unable to demonstrate title to any share of Essakane. A number of other international mining companies (e.g., AngloGold, Ashanti Goldfields, Randgold, Placer Dome) also visited and evaluated the EMZ during and shortly after BHP's withdrawal. However, the Nigerian company Coronation International finally secured title. After the protracted liquidation of CEMOB, six new licenses were granted to Coronation International Mining Corporation (CIMC) by the Ministère de l'Énergie et des Mines in July 2000.

In September 2000, CIMC concluded an option agreement with Ranger Minerals (Ranger) whereby Ranger could earn a 40% interest by spending US\$8.0 million on exploration. Ranger undertook an aggressive exploration program, focusing on intensive RAB and RC drilling of an oxide resource between October 2000 and June 2001. Landsat imagery and aerial photographs were also acquired for regolith and detailed mapping. RAB drilling (12 867 m) was used to locate drill targets at Essakane North, Essakane South, Falagountou and Gossey. Follow up RC drilling at the EMZ amounting to 22 393 m was completed along with 1 070 m of DD twins and extensions. Ranger mapped and sampled veins in the BHP trenches and decided to drill towards local grid east at a dip of -60°.

Hellman & Schofield (for Ranger) estimated Measured plus Indicated oxide resources of 18.9 Mt @ 2.14 g/t for 1.3 Moz (at 1 g/t cut-off) and an Inferred resource of 5.2 Mt @ 1.76 g/t for 0.3 Moz, as shown in Table 6.2. Ranger also concluded a series of metallurgical tests on geological samples with Independent Metallurgical Laboratories Pty Ltd from Perth.

Ranger spent US\$ 1.7 million of the US\$ 8 million needed by June 2001 and withdrew because of unfavourable oxide project economics. After Ranger's departure CIMC was approached by Orezone with an offer to merge the companies. The merger was papered in March 2002 and Orezone became 90% owner of Essakane.

Table 6.2 BHP and Ranger estimates of EMZ oxide resources

Company	Metres drilled	Resource class	Tonnes (Mt)	Grade (g/t)	Gold ('000 oz)
BHP (Internal)	9 459	Inferred	14.5	2.5	1 200
Ranger (Hellman & Schofield)	36 330	M + I	24.1	2.1	1 600

Gold Fields Orogen Holding (BVI) Ltd (“Orogen”), formerly known as Orogen Holdings (BVI) Limited, a subsidiary of GFL Mining Services Limited, entered into an Option Agreement with Orezone Resources Inc. on 19 July 2002 (the “Option Agreement”).

The terms of the Option Agreement were: (a) Orogen, or a nominee, could earn a 50% interest in Orezone’s 90% share of the project by spending US\$ 8 million on exploration over 5 years; (b) Orogen, or a nominee, could earn a further 10% in the project by sole funding and completing a bankable feasibility study; (c) at Orezone’s election, Orogen, or a nominee, could earn a further 10% in the project by securing project finance for Orezone.

On the 1st April 2007, Orezone Resources Inc, Orezone Inc, Orezone Essakane (BVI) Limited, Gold Fields Essakane (BVI) Limited (“GF BVI”), Orogen and Essakane (BVI) Limited entered into a Members Agreement which gave effect to the terms of the Option Agreement mentioned above and also set out the terms and conditions on which the parties would joint venture. As GF BVI earned a 50% interest in Essakane (BVI) Ltd by spending the requisite US\$ 8 million on exploration, it now owns 60% in the Essakane Project as GF BVI earned a further 10% interest in Essakane (BVI) Limited having completed a bankable feasibility study on 11 September 2007.

Once an Exploitation Convention for the Essakane Project has been obtained from the national government in Burkino Faso (the “BF Government”), the BF Government shall be entitled to a 10% fully carried share interest in Essakane Burkino Faso SARL (“Essakane BF”) and accordingly will become a 10% shareholder in Essakane BF.

## 6.2 Previous exploration and development work

Previous exploration of the EMZ has been completed by CEMOB, BHP, Ranger and Orezone and can be summarised as follows:

- Trenching by CEMOB in the early 1990’s: A total of 5 trenches (705 m) were excavated.
- Trenching, RC and Diamond Core drilling (DD), airborne geophysics and mapping by BHP in 1995 and 1996: A total of 25 trenches (1 445 m), 117 vertical RC holes (5 732 m) and 9 DD holes (1 510 m) inclined at 60° to local grid west were completed.

- RAB, DD and RC drilling by Ranger between 2000 and 2001: A total of 21 RAB holes (541 m), 239 RC holes (19 777 m) and 15 DD holes (2 131 m) were completed. All holes were inclined at 60° to local grid east.
- DD and RC drilling, trenching, mapping and assaying by Orezone between 2003 and 2005: A total of 44 RAB holes (1 275 m), 658 RC holes (63 572 m), 211 DD tails (35 064 m) and 56 DD holes (7 245 m) were drilled at various angles but predominantly vertical.
- RC, DD and Aircore drilling (AC) and assaying by Essakane since January 2006. Total drilling amounts to 69 251 m as summarized in Table 6.3. Holes were inclined to local grid east or west depending on the collar position in relation to the EMZ fold axis. Generally the holes were inclined at 60° to grid east. The AC holes were vertical and inclined holes, drilled to bit refusal on condemnation programs at the Project site and on regional exploration programs on the surrounding permits. Geotechnical drilling comprised DD and RC drilling at the expected highwall positions of the EMZ design pit shell and were drilled on behalf of the geotechnical consultants.

Table 6.3 Drilling completed by Essakane

Drill type	Number of holes	Holes			Metres
		DD	RC	AC	TOTAL
AC	1 336			16 069	<b>16 069</b>
DD	126	20 145			<b>20 145</b>
RC	205		16 363		<b>16 363</b>
RCD <sup>1</sup>	73	11 574	5 101		<b>16 675</b>
<b>Total</b>	<b>1 740</b>	<b>31 719</b>	<b>21 464</b>	<b>16 069</b>	<b>69 252</b>

1. RCD represents RC collared drillholes with DD tails. RC drilling would stop at the water table and the rig would switch over to diamond drilling.

BHP drilled the EMZ to a notional spacing of 100 m along strike by 50 m across strike. Ranger’s infill drilling reduced the notional spacing to 50 m along strike and either 25 m or 50 m across strike on alternate sections. Exploration effort between July 2002 and January 2005 was initially focused on scoping the potential of the full Project area. Geochemical sampling and ground geophysical surveys were completed which culminated in 18 confirmed or newly defined targets. A number of follow up RAB, RC and DD programs were completed. Orezone started drilling the EMZ in February 2003 and began ramping up the number of rigs in late 2004 for vertical RC resource definition drilling. The nominal grid was 50 x 25 m (i.e., 50 m spaced lines with holes 25 m apart on section) with one high grade central area (Panel F) drilled to 25 x 25 m. The RC holes were drilled to the water table and sampled at 1.0 m intervals (producing 25 – 30kg per sample). RC drilling over DD was preferred to increase the sample size and thus offset the coarse gold problem. However, the large sample size created difficulties for splitting out representative 3 – 5kg subsamples. A few HQ diameter DD tails were drilled by Orezone in the early stages to test depth continuity on RC holes which had been stopped in the main arenite or in gold mineralization. Some of these tails returned significant gold assays in the footwall argillite. Systematic drilling of DD tails was introduced in May 2005 to evaluate the footwall units on a 100 x 50 m nominal grid.

Ranger and BHP's fire assaying demonstrated poor reproducibility of economic gold assays. Orezone thus introduced cyanide - saturated 2kg BLEG as the standard method to improve assay reproducibility. The concentration of NaCN was set at 5kg/tonne (10g per 2 litres) and the sealed bottles were rolled for 24 hours. Residues (tailings) for all BLEG solution grades greater than 1 g/t were fire assayed for gold. Although BLEG is a partial dissolution method, the results showed improved reproducibility (for duplicate 2kg splits from the same pulp) compared with BHP and Ranger's data. Systematic leach curves were not measured but the BLEG tailings consistently reported low fire assay gold values. Fire assay of the tailings showed an average BLEG leach of 97%. Umpire assaying of 10% of the BLEG samples was introduced in May 2005 but the two umpire laboratories (SGS Lakefield in Johannesburg and ABILABS in Bamako) were unable to reproduce assays on the pulp rejects, particularly at grades above 0.7 g/t. SGS Lakefield reported significantly higher values but ABILABS reported lower grades for all samples >0.7 g/t. From SGS it was subsequently learned that rolling for an additional 24 hours with fresh cyanide resulted in higher BLEG solution grades. Tests showed that gold was still being dissolved after 72 hours under the standard BLEG conditions. At ABILABS it was found that the bottles were rolled at very low speeds and reasons for under-reporting thus included poor mixing and oxygen starvation. Another reason for poor leach rates was poor grinds of analytical samples with grinds ranging from 50 – 85% passing 75 microns.

Essakane combined these findings with promising results from gravity and rapid cyanide leach tests, and in January 2006 it replaced Orezone's 2kg BLEG bottle roll process with LeachWELL rapid cyanide leach on 1kg subsamples (the "LWL69M" method). By this time Essakane had accepted the need to re-assay 40 000 pulp rejects by LeachWELL from the BHP, Ranger and Orezone programs because the 2kg BLEG and 50g fire assay data showed significant biases when compared with the 1kg LeachWELL assay pairs. The decision was also taken to only use SGS Tarkwa in Ghana since this laboratory had many years of experience with LeachWELL on Tarkwa gold mine samples.

The results from this LWL69M re-assay combined with the 2006 core drilling program form the basis of the January 2007 and May 2007 block models. All resource estimates prior to January 2007 used the original fire assay and BLEG + LW database.

### 6.3 Historical mineral resource and mineral reserve estimates

Orezone retained SRK (Cardiff) in August 2004 to complete a JORC classified Mineral Resource estimate and NI 43-101 report for the EMZ. This estimate was based mainly on historical data. SRK estimated an ordinary kriged Indicated resource of 30.5 Mt @ 2.0 g/t for 1.91 Moz and an Inferred resource of 4.4 Mt @ 2.0 g/t for 0.29 Moz (at 1.0 g/t cut-off grade) as shown in Table 6.4. It was subsequently found that incorrect relative densities had been used which overstated resource tonnages by 15%.

In May 2005 Orezone retained RSG Global (Perth) to audit an internal PFS block model and sign-off a JORC classified resource constrained within a Whittle pit shell at US\$ 375/oz gold price assumption. However, Orezone and RSG Global subsequently decided to re-estimate the resources and produce a small panel, recoverable resource estimate using Uniform Conditioning. This was completed in July 2005 and the RSG Global block model was used in the PFS with a conceptual depth extension to provide for drilling in progress. MIK was considered but RSG Global was unable to demonstrate grade continuity at economic grade thresholds.

Orezone presented a new grade domain and geological fold model to RSG Global in December 2005. The basis of this model was that (a) the main gold-bearing vein

sets are steep west dipping and parallel to the fold axis, and (b) strike continuity of sets of the main vein sets as seen in the trenches could be represented by wireframes with hard grade boundaries. Figure 6.1 provides an isometric view of this steep structure model looking northwest with the wireframes shown in red. The total classified January 2006 UC resource estimate at a US\$ 475/oz price assumption is given in Table 6.5. This resource estimate was announced by Orezone on 10 April 2006. Despite the large amount of additional drilling between the PFS and the January 2006 models, there were concerns about the reliability of the historical analytical data and the re-classification of the Mineral Resources by RSG Global did not yield a significant increase in Indicated resources.

The concept of estimating grades within steep grade wireframes with hard boundaries was dropped in August 2006 due to increased drill evidence for lithostructural and bedding parallel controls on mineralization. Other reasons were that the 2006 oriented core drilling by Essakane did not confirm vein sets within hard grade boundaries, and ongoing tests showed that grade estimation with hard boundaries was potentially inflating the grade of the mineral resource estimates.

**Table 6.4 EMZ Mineral Resource estimate completed by SRK in 2004**

<b>Classification</b>	<b>Tonnes (Mt)</b>	<b>Grade (g/t)</b>	<b>Gold ('000 oz)</b>
Measured	-	-	-
Indicated	30.5 <sup>1</sup>	2.0	1 910
Inferred	4.4	2.0	290

1. These tonnages were overstated by 15% due to incorrect allocation of densities to the weathering domains. SRK listed a number of technical caveats in its classification of Indicated resources based on uncertainty about quality of the historical assay data and poor QAQC documentation .

**Figure 6.1 December 2005 steep structure EMZ grade model**

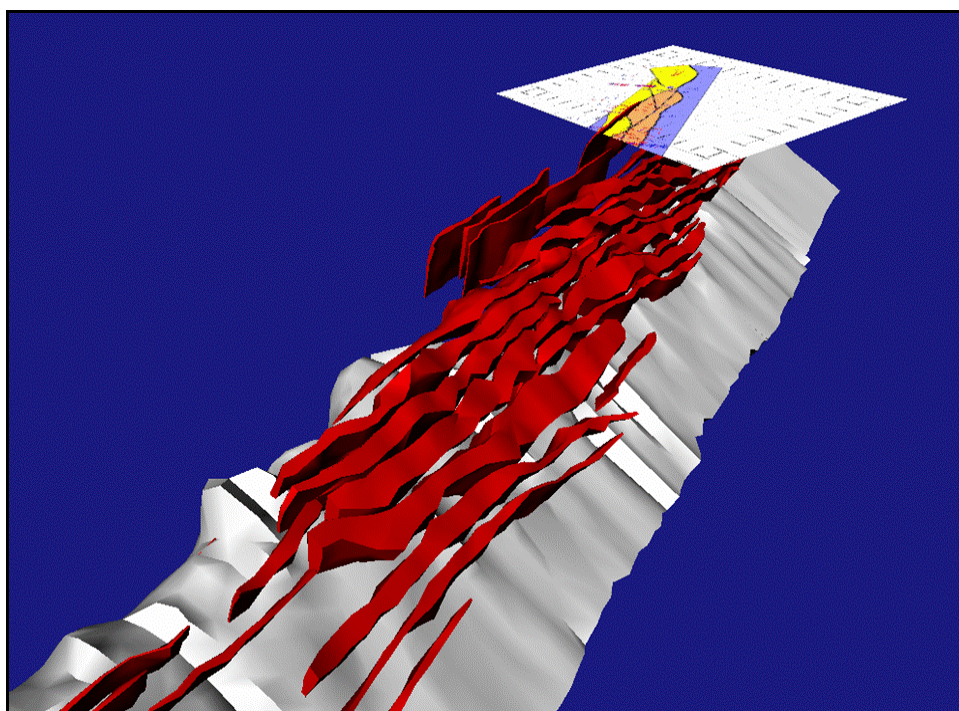


Table 6.5 January 2006 UC resource estimate – RSG Global

	Cut-off	0.5 g/t			1.0 g/t		
		Tonnes (Mt)	Grade (g/t)	Gold '000oz	Tonnes (Mt)	Grade (g/t)	Gold '000oz
Total Resource	Indicated	36.8	1.6	1 860	19.6	2.3	1 470
	Inferred	27.7	1.7	1 480	15.3	2.4	1 190
Resource reporting within US\$ 475/oz pit shell	Indicated	34.7	1.6	1 790	18.9	2.4	1 430
	Inferred	19.3	1.8	1 130	11.5	2.6	950

## 7 Geological setting

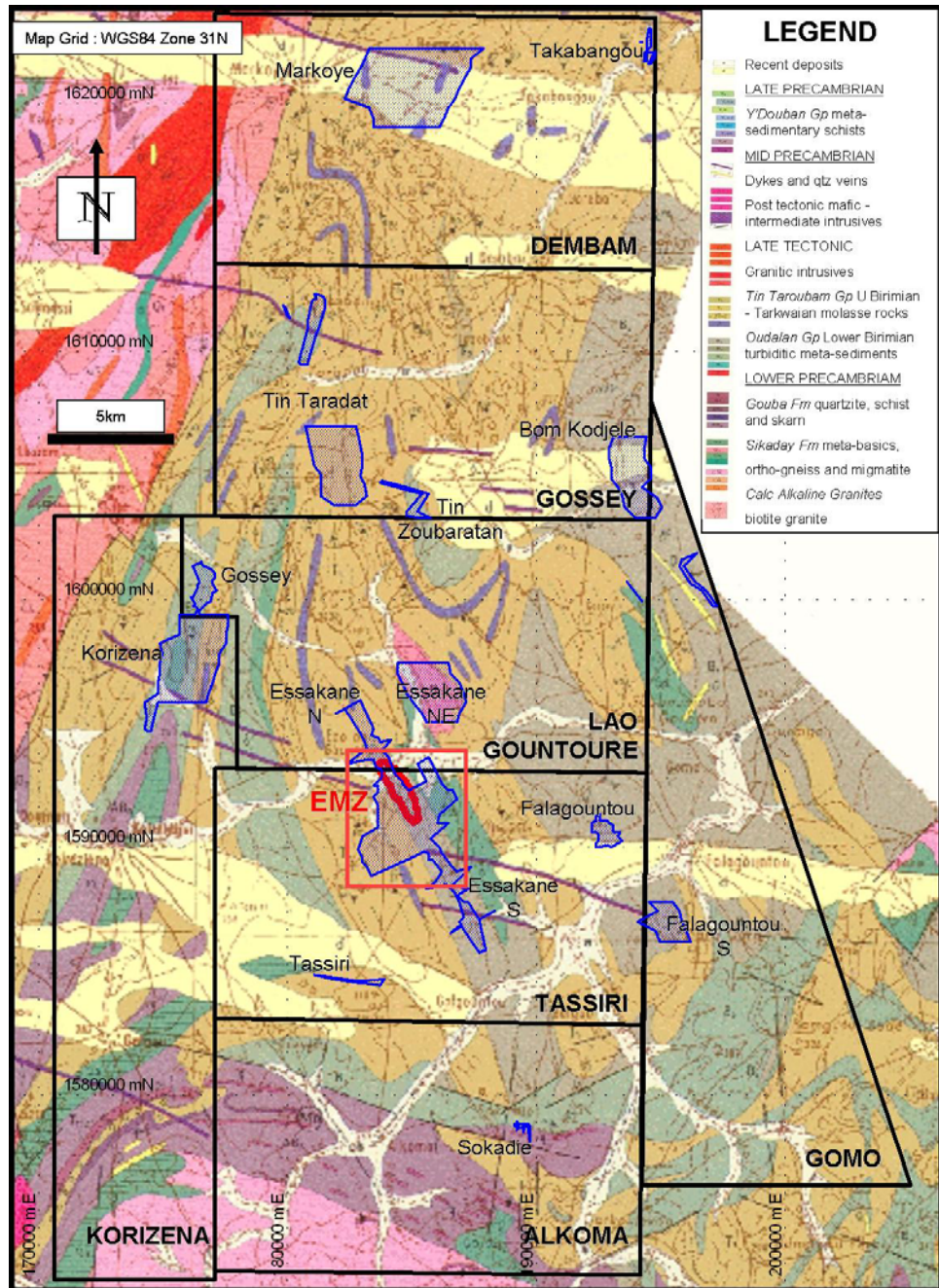
### 7.1 Regional geology

The Project occurs in an outlier of folded sedimentary Birimian rocks which are intruded in places by intermediate and mafic sills. The sediments in the district have been subdivided on the basis of lithology into deep-water turbidites (the Birimian) and coarse clastic basin margin sequences (the Tarkwaian). The Birimian rocks consist of wackes, arenites and mudrocks (argillites), pebbly arenites and minor tuffs which have been metamorphosed to lower greenschist facies. Arenite is the dominant lithology. Intermediate intrusives occurring as sills are common and appear to predate all gold mineralization in the district. The Tarkwaian rocks are typically sandstones with thin intercalated bands of matrix-supported, polymictic conglomerates but are unlike the type lithologies found in Ghana. In particular, the conglomerate matrices are not enriched in heavy minerals nor show the alteration mineral assemblages of Tarkwa and Iduapriem.

Figure 7.1 shows the boundaries of the permits comprising the Project and the EMZ feasibility study area (highlighted in red) in context with the regional geology. The bold red shape within the red perimeter is the crest line of a US\$ 650/oz surface mine shell on the EMZ. The map is reproduced from the BRGM's 1/200,000 map of the L'Oudalan district published in 1970. The Birimian and Tarkwaian are bounded to the west by the major NNE trending Markoye fault and to the south by the Dori batholith. The Markoye Fault is thought to be a left – lateral wrench fault that was an active basin margin fault at the time of deposition of the sediments. Other regional faults in the district appear to trend NE and WNW. Mesozoic age dolerite dykes are generally found in the latter. Fold axes within the Birimian trend NW and N except in the south where units are refolded adjacent to the batholith.

Gold prospects on the permits (shown as blue hashed areas in Figure 7.1) occur exclusively in Birimian rocks and are generally associated with quartz veining on the margins of mafic and intermediate sills. Exceptions are the EMZ and the Sokadie prospect (on the Alkoma permit). The EMZ is characterized by quartz veining in a folded Turbidite succession of arenite and argillite. At Sokadie the veins occur in a sheared volcanoclastic unit between undeformed andesite and metasediments. That is, as a general rule, gold occurs with quartz veining on the contacts of rock units with contrasted competency and as filling of brittle fractures in folded sediments.

Figure 7.1 Regional geological setting of the Essakane Gold Project



## 7.2 Local geology

The Project has been explored since 1995 by a variety of methods ranging from soil sampling and pitting to analysis of Aster and Landsat images. Outcrop is limited and there is an extensive cover sequence of residual soils and transported material. The distribution of transported cover is shown in Figure 7.2. As shown in the figure, the southern permits are characterized by a higher proportion of outcrop. The figure also shows the locations of soil sampling grids. Soil sampling has been successful in locating potential targets for follow up pitting and drilling. Samples have generally been assayed for gold and arsenic and an image of the processed data is presented in Figure 7.3. A total of 18 exploration targets were highlighted by this

method, some of which have been subsequently tested. However, the focus of exploration activity to date has been on the evaluation and development of the EMZ.

An interpretation of the structural geology of the Project area is shown in Figure 7.4. A number of fold axial traces can be observed and it is believed that a significant proportion of the gold occurrences on the permits are associated with this folding event.

The locations of all known gold prospects on the permits are shown in Figure 7.5 (highlighted in blue text). Perimeters around all drill grids and soil anomalies are also shown. Essakane expanded its programs during 2007 to include aircore drilling through overburden to explore for bedrock gold anomalies. These assay data are not available at the time of writing.

**Figure 7.2** Surface geology of the Project area showing transported cover

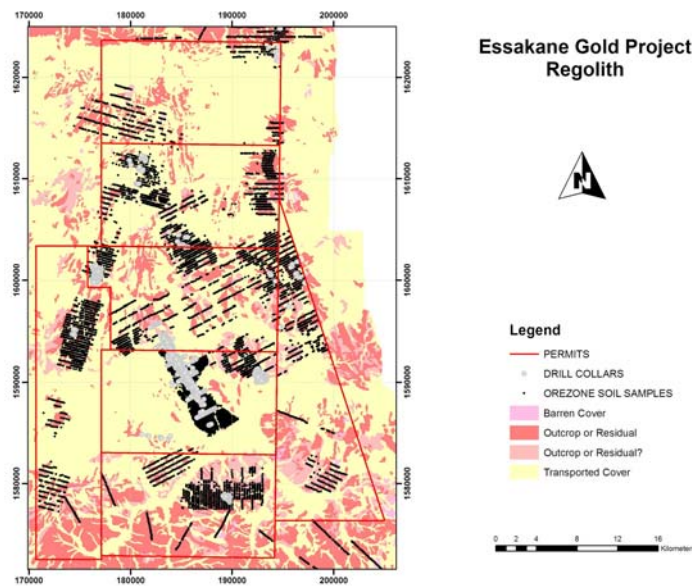


Figure 7.3 Au and As in soils for the Project area

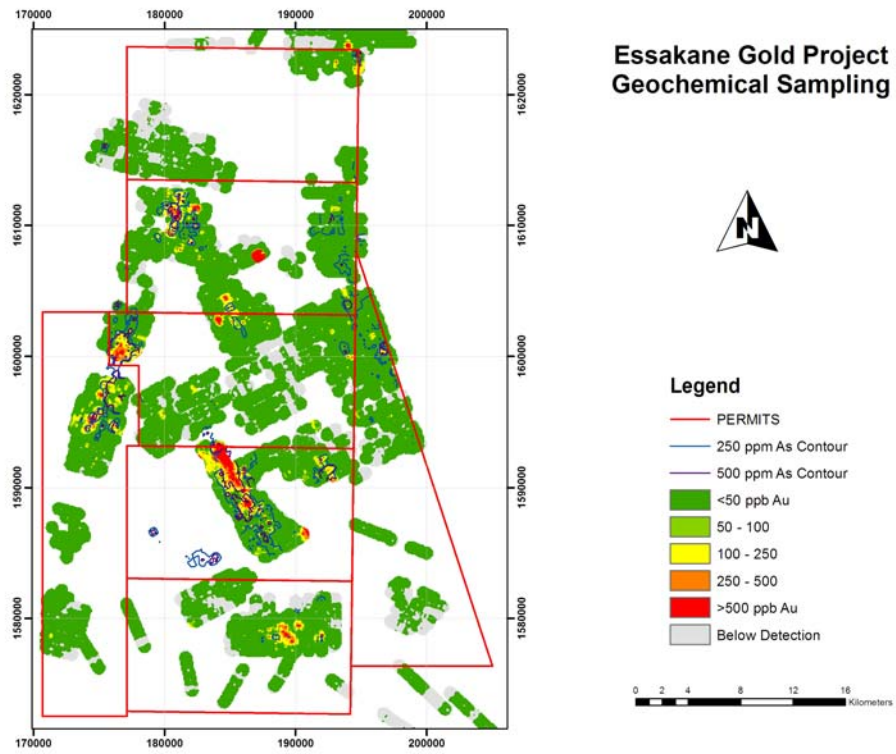


Figure 7.4 Fold model developed for the Project area

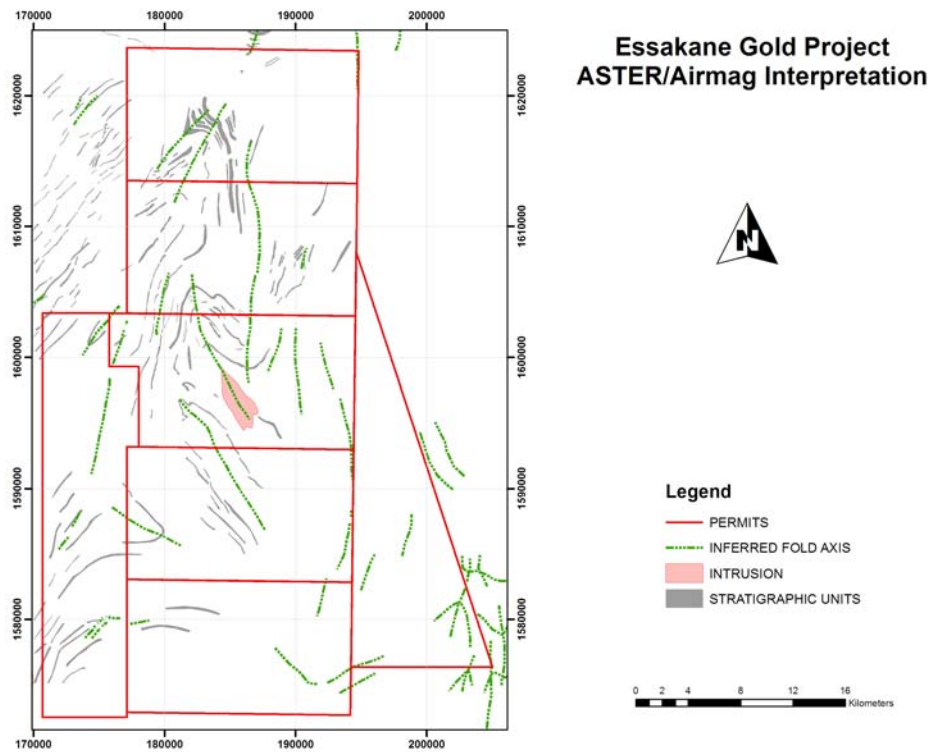
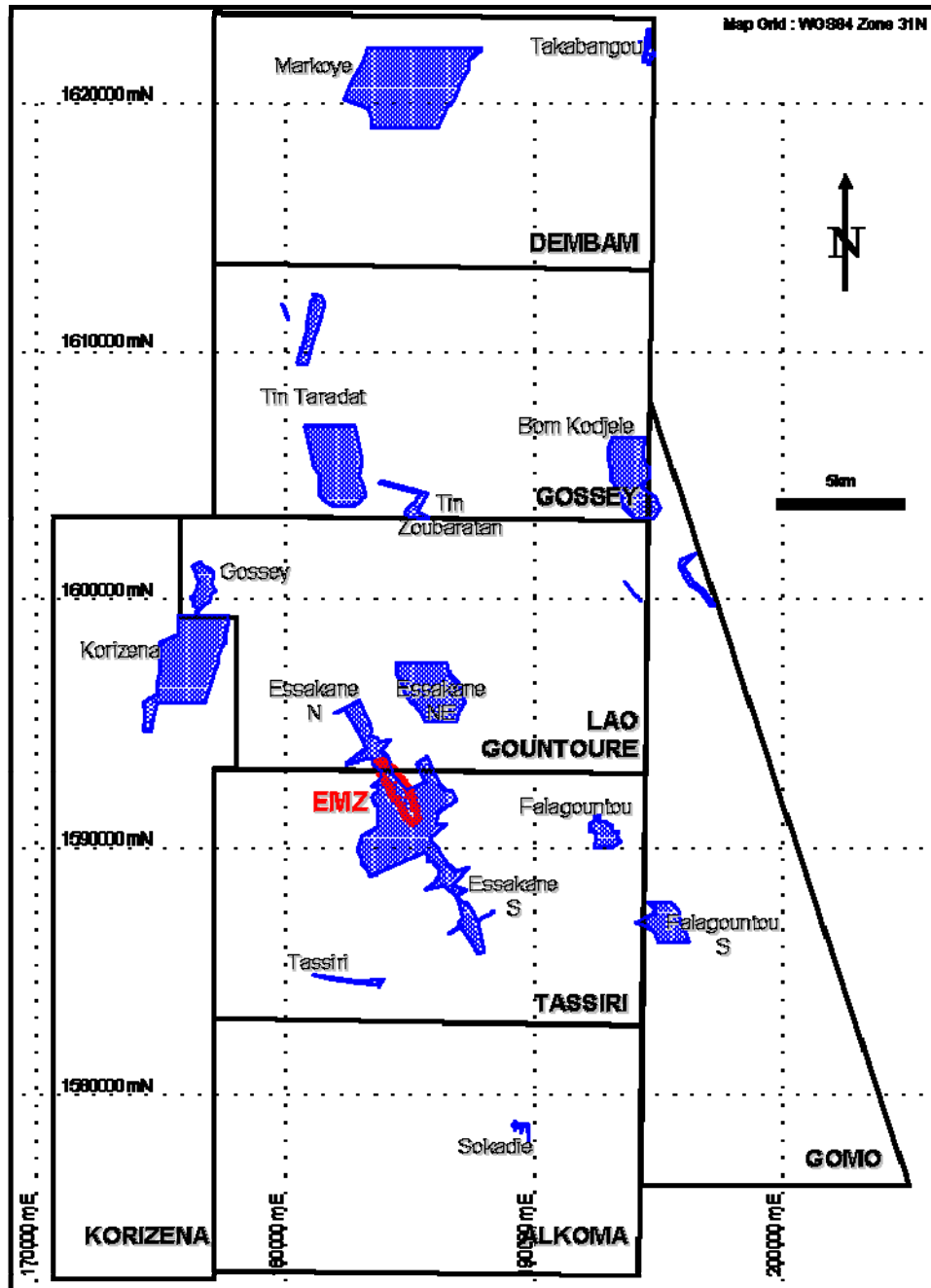


Figure 7.5 Location of all known mineralized zones in the Project area



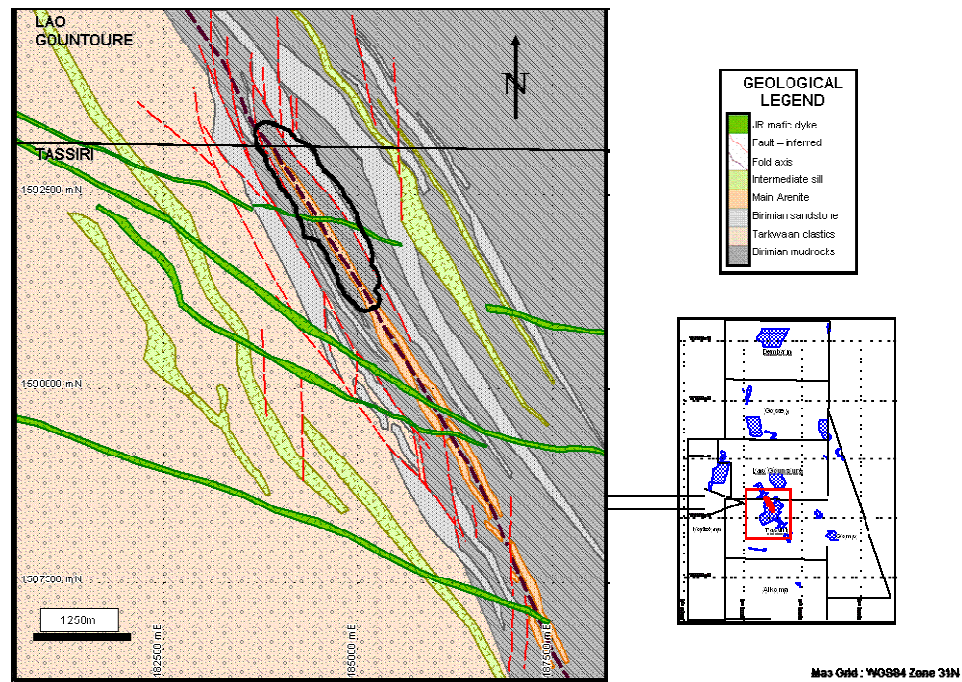
### 7.3 Property geology

The subsurface geology of the area surrounding the EMZ on the Tassiri permit is presented in Figure 7.6. This map was developed by Essakane and is based on pitting, drilling, surface mapping and interpretation of IP resistivity surveys carried out as part of the geohydrological study for the DFS. The economically important Main Arenite is shown in orange.

Tarkwaian clastic sediments occur to the west of the expected pit limits. These metasediments show only limited strain. The dominant alteration is pervasive silicification which produces outcrops of hard pebble conglomerate and quartzite. Birimian metasediments shown in grey consist of a deformed Turbidite succession of NW – SE striking argillite with interbedded wacke and arenite layers. Bouma cycles are preserved in the succession and occur within the EMZ stratigraphy. Conformable sills of intermediate composition have been emplaced into the Tarkwaian and Birimian stratigraphy. The margins of these sills are commonly the locus of quartz veining with associated sulphides and gold mineralization.

A series of late WNW – ESE dolerite dykes cross-cut all earlier rock units. The dolerite dykes have been intersected in drilling and generally outcrop as long trails of surface rubble. Residual caps of pisolitic and ferruginous laterite generally mark the presence of intermediate and mafic sills near surface.

Figure 7.6 Property geology



## 8 Deposit type

The EMZ is a greenstone – hosted orogenic gold deposit. Specifically, it is a quartz – carbonate stockwork vein deposit hosted by a folded turbidite succession of arenite and argillite. Gold occurs as free particles within the veins and also intergrown with arsenopyrite on vein margins or in the host rocks. Disseminated arsenopyrite in the host rock decreases away from the veins. The same relationship is seen away from lithological contacts, which generally show higher densities of bedding parallel veining. Oriented diamond core drilling by Essakane after the PFS showed that significant concentrations of gold with arsenopyrite can be found on the arenite – argillite lithological contacts in association with quartz veining or in veinlets of massive arsenopyrite. In weathered saprolite the gold particles occur without sulphides. The gold is free – milling in all associations.

BHP was the first international mining company to explore the EMZ and believed the stockwork was bounded by a series of west-verging thrust faults. This interpretation, developed by D. Pohl in 1995 for BHP, is shown in Figure 8.1 (copied from a Hellman & Schofield report for Ranger Minerals), was favoured by geologists up to late 2005 at which time Essakane changed the interpretation to an anticlinal fold (without thrust faults). This change came about from re-mapping the BHP surface trenches and drilling of oriented core drillholes. Previous operators had relied on reverse circulation (RC) chips without downhole structural measurements.

A cross-section through the PFS model is shown in Figure 8.2. The model was based on the BHP interpretation and the Mineral Resources were estimated by RSG Global (Perth). The shortcoming of this thrust model was that it assumed continuity of mineralization within grade domains without having a firm geological basis. That is, deep mineralization in arenite to the east could be correlated with shallow mineralization in argillite to the west if interpreted to be within the same grade domain. Each grade domain was thought to be separated by a thrust fault or thrust zone (without mineralization). The thrust domain model was thus abandoned (in August 2005) after further structural studies by Essakane. Subsequent core drilling by Orezone, oriented west – east and east – west, found no evidence of thrust faults. The PFS model was thus abandoned and all subsequent work has confirmed that the EMZ is an anticlinal fold with flexural slip between layers and brittle deformation within layers. The quartz veins fill brittle extension and shear deformation structures caused by the folding with at least two phases of quartz veining and gold mineralization.

Figure 8.3 depicts the late 2005 geological fold model which has been improved with further drilling and now forms the basis of the 2007 DFS model. The figure displays a cartoon overlay of expected vein orientations within the fold model. The fold is a NW – plunging anticline with a west - verging axial plane and near vertical west limb. The fold axis plunges 10° north. The east limb dips at 30 - 50° to the east.

Figure 8.1 Cross-section showing BHP's thrust model for the EMZ

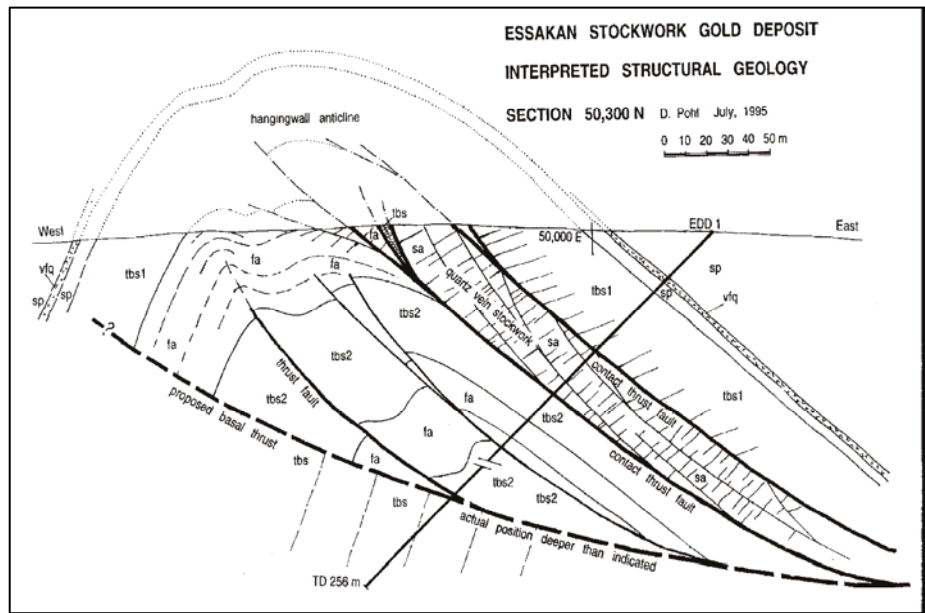


Figure 8.2 Cross-section showing the 2005 PFS thrust domain model

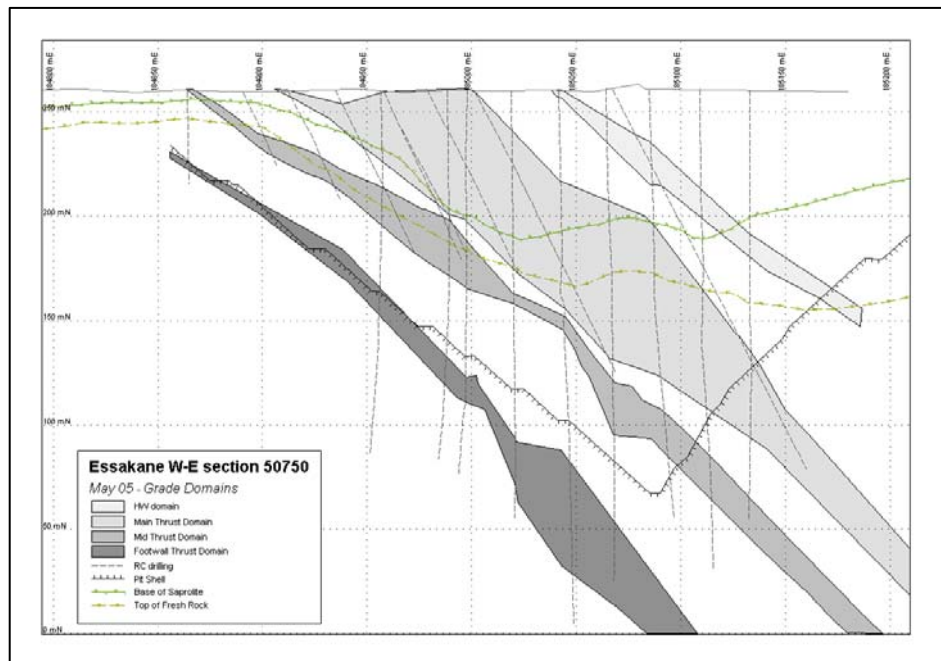
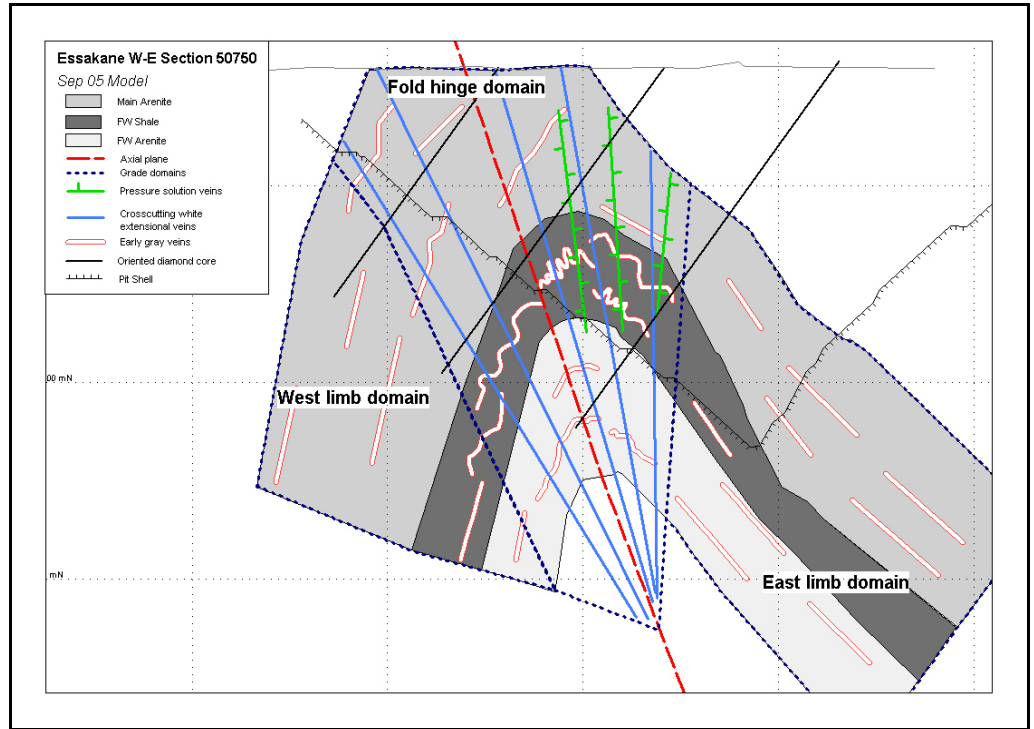


Figure 8.3 Cross-section showing the EMZ fold geological model



The vein arrays in the EMZ are complex and consist of: (i) Early bedding parallel laminated quartz veins caused by flexural slip and showing ptygmatic folding; (ii) Late, steep extensional quartz veins as vein filling in extension and shear joints formed by the folding (three major vein sets have been mapped on surface); (iii) Axial - planar pressure solution cleavage (with pressure solution seams normal and parallel to bedding).

The vein arrays occur in the east limb-, fold hinge- (or fold axis) and west limb-lithostructural domains. These domains form the basis of the DFS block model. The geology and economic potential of the EMZ is dominated by the persistent east limb main arenite. The top contact of the east limb domain is a sharp, sheared contact with no significant gold mineralization above it. The shearing appears to be bedding parallel but some loss of vertical succession has occurred. The main arenite below this contact is the lower coarse grained part of a Bouma cycle. The locus of bedding parallel deformation and alteration is within the east limb main arenite. Graphitic argillite occurs immediately above the contact. The deformation shifts into the hangingwall argillite unit to the north of the EMZ.

Mineralization has been confirmed to 270 m vertically below surface but the full depth extent in the fold hinge and east limb is not known. The geometry of the fold hinge zone is an anticlinal flexure that is easily recognized in the surface trenches and oriented drill cores. The fold closure is sharp and the transition from east limb to west limb takes place over a few metres. In arenite the position of the fold axis is generally marked by a breccia. In argillite it is marked by tight kink structures and sheath folds with rapid transitions from east dipping footwall rocks to near-vertical west limb beds below the fold axial plane.

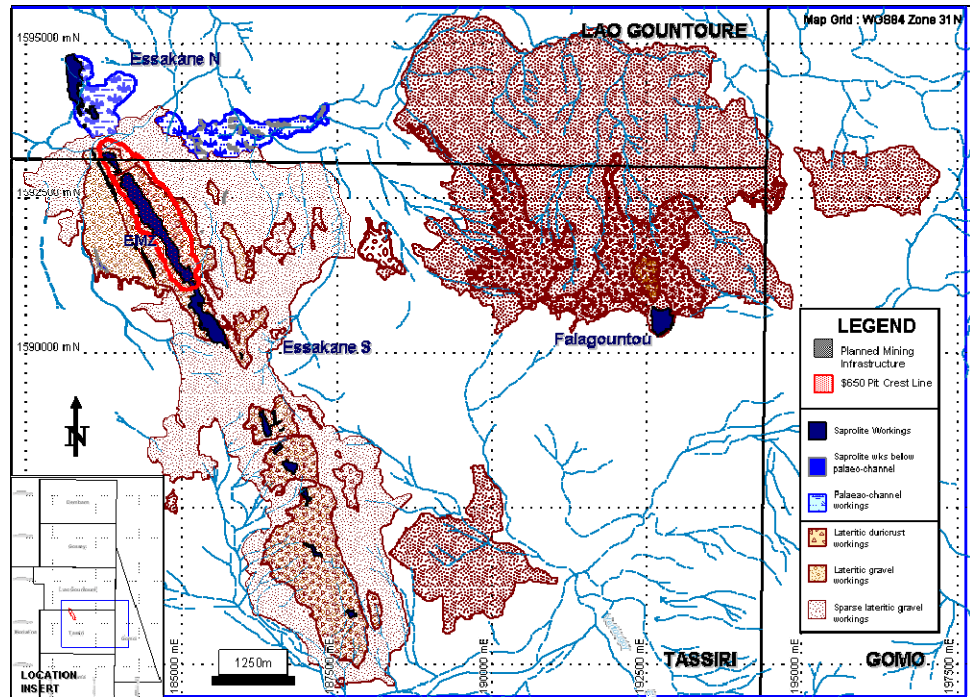
## 9 Mineralisation

### 9.1 General Description

The EMZ as modelled for the DFS has a strike length of 2 500 m and occurs at the northern end of the EMZ anticline. Weathered main arenite and quartz veins are fully exposed in surface trenches and Artisanal shafts along the low ridge marking the crest of the EMZ mineralization. Outcrop is otherwise poor and is obscured by 1 – 3 m of duricrust, alluvial silt and windblown sand. The laterite is a typical sub-Saharan ferruginous duricrust. The base of the regolith is clearly marked by a quartz pebble stoneline over large areas. Palaeochannels are also developed east of the EMZ. Gold is recovered from the stonelines and alluvial deposits by the local miners through tight clusters of vertical shafts, and by wind winnowing of scrapings of the surface over large areas.

The extent of these workings around the EMZ is shown in Figure 9.1. No evaluation of any alluvial deposits has been completed by Essakane. Workings range from clustered vertical shafts on the EMZ (shown in blue) to scattered shafts at Essakane South and wind winnowing of surface scrapings in the distal areas. The shafts are wide enough for one man to climb down. The colluvial and stoneline materials are worked through a system of shafts, pits and shallow sumps depending on the thickness of overburden. The overburden is generally 1 - 2 m thick except within the main drainage- and palaeo-channels where it reaches 5 - 8 m.

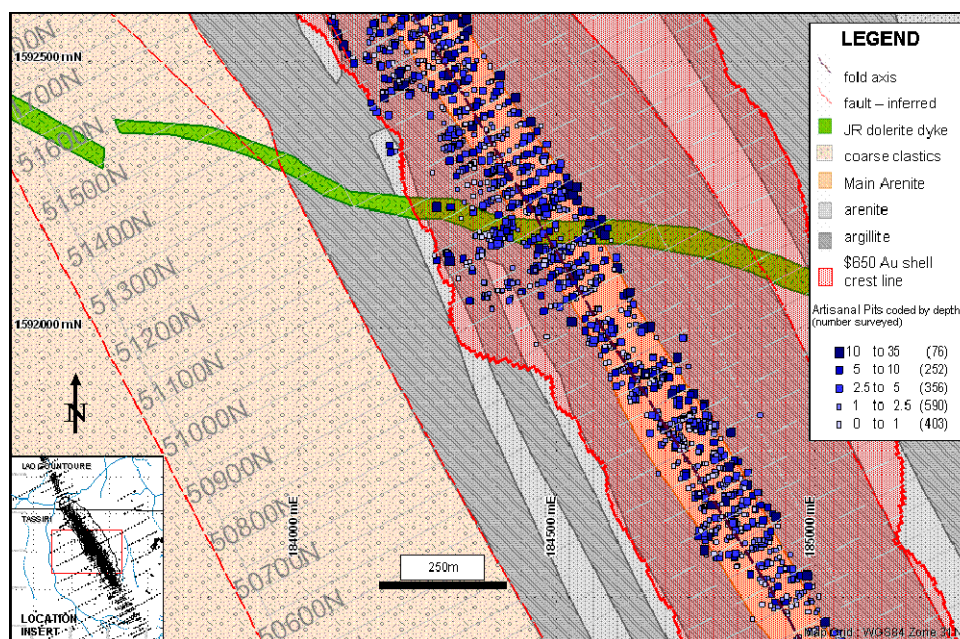
Figure 9.1 Artisanal workings surrounding the EMZ



The main area of pits and shafts on the EMZ measures 2.3 km NW – SE. Shafts generally reach depths of 35 m. According to verbal reports the vertical shafts were mined on steep veins to 15 m depth in mining blocks 25 m wide arranged from west to east. Below 15 m horizontal tunnels were mined with narrow connections for ventilation.

Three main areas of high grade gold were exploited: (a) fold hinge zone, (b) top contact of the east limb main arenite, and (c) along an argillite marker band which splits the upper and lower east limb main arenite. No maps or drawings of this mining have been located and were probably not maintained. However, Figure 9.2 shows the distribution of pits on the Panel F portion of the EMZ as mapped by Orezone in 2005 and also digitized off orthophotographs. Only 76 shafts deeper than 10 m were found to be open. Artisanal mining on the EMZ was halted by late 2005, primarily for safety reasons with increased drilling activity and movement of heavy vehicles.

Figure 9.2 Map showing artisanal pits and shafts on the EMZ



Collaring of RC and DD holes on the EMZ is made difficult by these shafts. Backfilled shafts and tunnels also created uncertainty about validity of RC samples in some cases. Essakane started weighing the 1 m RC samples at the drill rig in early 2006 to help identify disturbed ground. Loss of compressed air and poor sample return generally indicated proximity to voids. Mining voids have not been wireframed in any of the geological models although sample losses and voids are recorded in the database (as CNR or NR). The presence of voids is handled in the resource estimation, where CNR and NR intervals are included at nil grade in the calculation of 3 m assay composites. The result is lower block grades in overstated tonnes since the density of the 3m composite was not reduced. The lower block model grades near surface, compared with fresh rock below the water table, can be partly explained by selective mining of high grade quartz veins by the Artisanal miners.

Any modelled estimate of mined out ground would be understated since the drilling companies usually move the rig a few m from the planned position to avoid a shaft. This applies in particular to vertical drillholes. Pad preparation by bulldozer also backfilled shafts close to the collar position for safety reasons.

The northern limit of the DFS block model is at 52 300N. The fold hinge and east limb main arenite continue north of this point to at least 52 600N.

Drilling on grid line 52 800N to a vertical depth of 130 m remained in HW argillite but resource definition holes have successfully intersected the east limb main arenite up to 52 600N in recent drilling.

Table 9.1 presents the stratigraphic succession of rocks and a generalized description of lithologies within the January 2007 and May 2007 block models.

The main arenite is pale grey in colour and weathers to a white, clay-rich saprolite containing up to 43% kaolinite and 52% muscovite. Mineralized arenite with arsenopyrite (+ pyrite) weathers to a distinctive yellow saprolite. The clay minerals and typical modal abundance are listed in Table 9.2.

Clay contents were measured by XRD-PSD quantitative phase analysis. Weathered arenite when dry produces large amounts of dust and dust suppression during mining will be important. Vehicle access after rains is also very difficult.

Figure 9.3 shows an example of unweathered, fresh arenite which typically contains up to 35% feldspar. The core sample is from ERC0366D at 93 m and the width of the sample is 5 centimetres. Disseminated tourmaline and rutile occur in minor to trace amounts.

Figure 9.3 Photograph of fresh main arenite



Table 9.1 Description of rocks in the EMZ

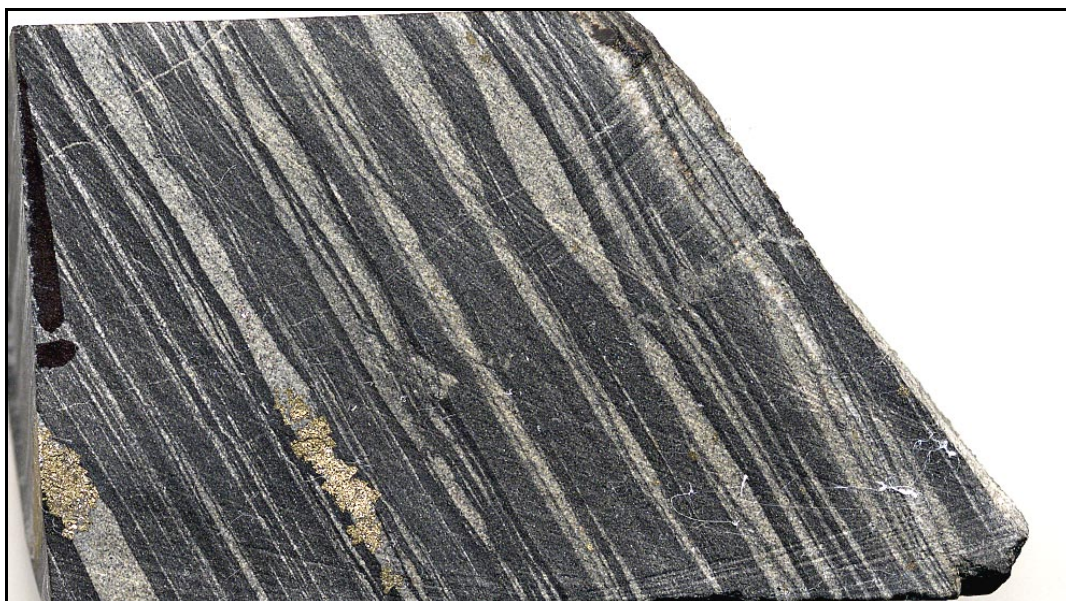
Stratigraphic Succession	Lithology	Comments
HW Argillite	Tourmalinized mudstone and siltstone with a basal 5m thick graphitic argillite. The base of the HW argillite is marked by a bedding parallel fault. Intrusive sills are common.	Largely barren on the east limb of the fold but the vertical west limb contains mineralized veins. Low grade gold occurs on the margins of intermediate HW sills.
Main Arenite	Equigranular arenite with subordinate lithic arenite and wacke. The east limb main arenite is the main host to quartz veins and gold mineralization. A thin argillite band commonly occurs in the middle of the main arenite and is used as a marker to separate an upper and lower main arenite. The upper main arenite has a higher vein density and is consistently mineralized compared with the lower main arenite. High Au values can be found on the lower main arenite – FW argillite lithological contact.	Main gold bearing unit in the EMZ with a sharp top grade contact. The east limb main arenite is the basal coarse grained unit of a beheaded Bouma cycle.  Bedding parallel shearing with associated veining, alteration and mineralization decreases to the north and south.
FW Argillite	Tourmalinized siltstone with thin arenite bands. It is mineralized in bedding parallel veins, steep quartz veins and pressure solution veins, mainly within the fold hinge domain.	Secondary gold bearing unit. High grade pressure solution structures can occur in the fold hinge with abundant and coarse visible gold.
FW Arenite	Arsenopyrite is common along the top and bottom contacts marked by flexural slip deformation.  Similar in appearance to the lower main arenite but tends to be massive with minor siltstone bands.	Gold bearing in fold hinge domain with some exceptionally high grade intercepts along contacts.

Table 9.2 Clay contents in weathered main arenite

Mineral	Average modal content (weight %)	Range (weight %)
Muscovite	35.7	25 - 52
Quartz	30.9	22 - 38
Kaolinite	32.9	25 - 43
Iron oxides	0.5	0.1 – 1.1

Figure 9.4 is a fresh FW argillite sample from EDD0368 at 105 m showing pyrite replacement in thin arenite bands. Argillite is fine-grained, dark grey and laminated, and contains high contents of granular tourmaline (up to 75%) which gives the rock a dark colour. In drilling the HW argillite is generally recognized by (i) frequency of sills of intermediate composition, (ii) graphitic nature of the lower beds above the sharp basal contact, (iii) absence of a fining-upward turbidite succession below the contact, and (iv) a low pyrite or arsenopyrite content. Graphite and rutile occur in minor to trace amounts.

Figure 9.4 Photograph of FW argillite with pyrite in thin arenite bands



Hydrothermal alteration and meteoric weathering are pervasive through the east limb main arenite. Hydrothermal alteration is generally associated with quartz veining and gold mineralization in deformed main arenite. The alteration assemblage is sericite > carbonate > silica  $\pm$  albite  $\pm$  arsenopyrite  $\pm$  pyrite. Disseminated tourmaline and rutile is found in accessory amounts. The main alteration minerals tend to occur in clearly defined veins and stringers.

Arsenopyrite and pyrite occurs within and adjacent to quartz veins as well as disseminated throughout areas of wallrock alteration. Traces of chalcopyrite, pyrrhotite, galena and hematite occur with arsenopyrite. Minor amounts of tourmaline with rutile are found in the main arenite and in interbedded arenite stringers in the footwall argillite. Remobilised graphite can accompany tourmaline.

The fine-grained argillites are strongly tourmalinized and have also been subjected to quartz, carbonate, sericite and quartz alteration. Fine needles of rutile generally accompany the tourmaline. Sulphide mineralisation preferentially occurs in the coarser arenaceous layers.

The deposit is characterised by multiple quartz and quartz – carbonate vein sets and stringers. Arsenopyrite and pyrite tend to be late and concentrated near the margins of the veins or in late cross-cutting stringers. The paragenetic sequence of veining is thought to be:

- Early quartz – carbonate – albite - (sericite) veins
- Quartz veins with tourmaline and pyrite containing gold
- Diffuse quartz – albite - carbonate veins with arsenopyrite
- Later tourmaline – rutile - arsenopyrite stringers with gold
- Late skeletal pyrite and carbonate - quartz - pyrite stringers

Weathering of arenite (database code S3) and argillite (database code S4) by meteoric processes has produced a consistent weathering profile which is described in Table 9.3. The ability of drillcore to absorb water and the rate of absorption was used from January 2006 to pick out the base of upper and lower saprolite. This provided a more consistent logging tool for geologists. Very little of the primary lithology can be recognized in the clay-rich saprolite near surface. The base of upper saprolite is easily recognized in drillcore, particularly after the core is allowed to dry in the sun and the clay fraction disaggregates. In general this is a fairly sharp contact and mining equipment would be able to dig to this without difficulty. The base of lower saprolite (or top of Fresh) is gradational and the contact is placed at the point that water is not absorbed by the rock. That is, the rock has no open pore space and Essakane took this to be the position of top of Fresh for geological and geotechnical modelling. However, oxidation of sulphides on vein margins and joints can extend into Fresh rocks for some m below this position.

Greater weight was given to the DD core logs when creating the Datamine weathering surfaces for the January 2007 geological model. The surfaces used in the DFS are the base of upper saprolite and the top of fresh. Knight Piésold's geotechnical RC and DD drillholes were included in the wireframing. Comparison of adjacent RC – RC and RC – DD logs on drill sections highlighted inconsistent logging of weathering by the various operators. This resulted in geologically incoherent surfaces both on - section and between drill sections. The final interpretation is smoothed and was completed with geological wireframes for lithology, fold axis, dykes and faults displayed on the computer screen.

The wireframes were developed using data in the Regolith field of the geological database. The codes in this field are summarized in Table 9.4.

Table 9.3 Description of weathering within the EMZ

Jan-07 regolith model	Thickness (m)	Comments
Laterite	1 - 3	Ferruginous zone grading from cemented, hard duricrust to pink, mottled zone (coded mz or WMZ in the Regolith field of the database). A cemented, pisolitic laterite is generally developed over the subcrop of intermediate and mafic intrusives. Artisanal mining has removed the laterite over the main arenite. Flagged as Oxide in the Oxidation field.
Upper Saprolite	30 - 50	Clay-rich, porous, friable and soft, highly weathered material with low strength. Absorbs water very rapidly. Quartz from quartz veins is the only relict material recovered in RC chips. The distinction between arenite and intermediate intrusive rocks is not always evident. Argillite is recognized by its grey colour. This material is expected to be free – dig and to slurry when mixed with water. Limited grinding required. Upper saprolite contains lenses of less weathered material and the proportion of these lenses increases with depth. Previous operators logged this material as sp (for saprolite) or ox (for oxide). Essakane introduced the code WSU for upper saprolite in early 2006. Flagged as Oxide by all operators in the Oxidation field.
Lower Saprolite	10 - 30	Weathered veins and fractures in porous to semi-porous host rock. The apparent porosity of this rock ranges from 1 – 5% and drillcores absorb water. The porosity decreases with depth and there is a corresponding increase in density. Grain size, bedding and texture are fully preserved in DD drillcores. Low – powder factor blasting would be required with grinding. Previous operators logged this as sr (for saprock) or Tr (for transitional). Essakane introduced the code WSL. Flagged as Oxide by Essakane but Tr by previous operators.
Fresh		Competent rock logged by previous operators as Fr in the Oxidation field but no code was applied in the Regolith field. Essakane has used FR in the Regolith field and FR in the Oxidation field. Core does not absorb water. The contact with the overlying weathered rock is gradational over a few m. Oxidation of vein margins and joints can extend a short distance into the fresh material but the host S3 and S4 lithologies are impermeable. Very low porosity. Blasting and grinding are required.

Essakane introduced weathering code WSR to describe partial weathering of sulphides on veins and fractures in fresh rock at the gradational base of the weathering profile. This classification was not used consistently, nor was it possible to add this level of detail to logs of previous RC drillholes. As a result WSR was not used in the January 2007 weathering model and top of Fresh was taken at the base of the lower saprolite in all subsequent modelling.

**Table 9.4 Comparison of logging codes to describe weathering**

Pre – 2006 logging		Essakane Project logging	
Code	Description	Code	Description
lt	laterite	WSM	mottled saprolite
sp	saprolite	WSU	upper saprolite
sr	saprock	WSL	lower saprolite
		WSR	Fresh rock with oxidation of veins and fractures
Fr	Fresh rock	FR	Fresh rock

Previous operators also logged the oxidation of the sulphides and this was adopted by Essakane for continuity of process and data (Table 9.5). Oxidation type is stored in the Oxidation field. Generally there is a logical link between Regolith and Oxidation but many historical holes do not have information or the logging does not match holes on either side of it.

**Table 9.5 Comparison of Regolith and Oxidation logging codes**

REGOLITH field		OXIDATION field
Pre Jan-06	Post Jan-06	
laterite	laterite	Oxide (OX)
sp	WSU	Oxide (OX)
sr	WSL	Oxide (OX) or Transitional (Tr)
sr	WSR	Transitional (Tr)
fr	FR	Fresh (FR)

**9.1.1 Gold deportment**

The EMZ is a coarse gold deposit. The rule-of-thumb definition for coarse gold is that particles are larger than 100 microns in diameter. This is reflected in Figure 9.5 which shows screen fire assay data for 96 x 1kg samples pulverized to 90% passing 75 microns. Significant amounts of gold report to the +106 microns oversize despite the fine grind. Fifty per cent of the gold fraction is coarser than 106 microns in samples assaying >5g/t with a strong maximum between 60 and 80% in high grade samples. In lower grade samples the proportion of gold coarser than 100 microns can vary from 5 – 80%. Strong heterogeneity would account for the sampling problems and imprecision in assaying EMZ samples. Excessive fragmentation of ore during blasting with loss of coarse gold particles should be avoided and free-dig on friable saprolite ores is strongly recommended.

Figure 9.6 presents Falcon SB40 gravity data for upper and lower saprolite samples and shows that most samples assaying >2g/t have more than 60% of the gold reporting to concentrate at a grind of 90% passing 425 microns. That is, two analytical methods indicate that gold in higher grade samples is generally coarse and is liberated at coarse grinds. Development of sampling and assay protocols for coarse gold has been an important component of the geological work completed by Essakane.

SGS Lakefield (Johannesburg) completed a gold deportment study on saprolite and fresh RC samples in low (0.7 g/t), medium (2.0 g/t) and high grade (>5.0 g/t) categories. The results of this work are summarized in Table 9.6. The study showed that significant amounts of coarse and fine-grained gold (<25 microns) are occluded in or attached to arsenopyrite in fresh samples. In low grade fresh samples 25% of the gold is <25 microns in diameter. Significant amounts of fine gold were also found in the upper saprolite sample.

**Figure 9.5 Screen fire assay results for 96 x 1kg pulverized samples**



Figure 9.6 Proportion of gold reporting to Falcon gravity concentrates

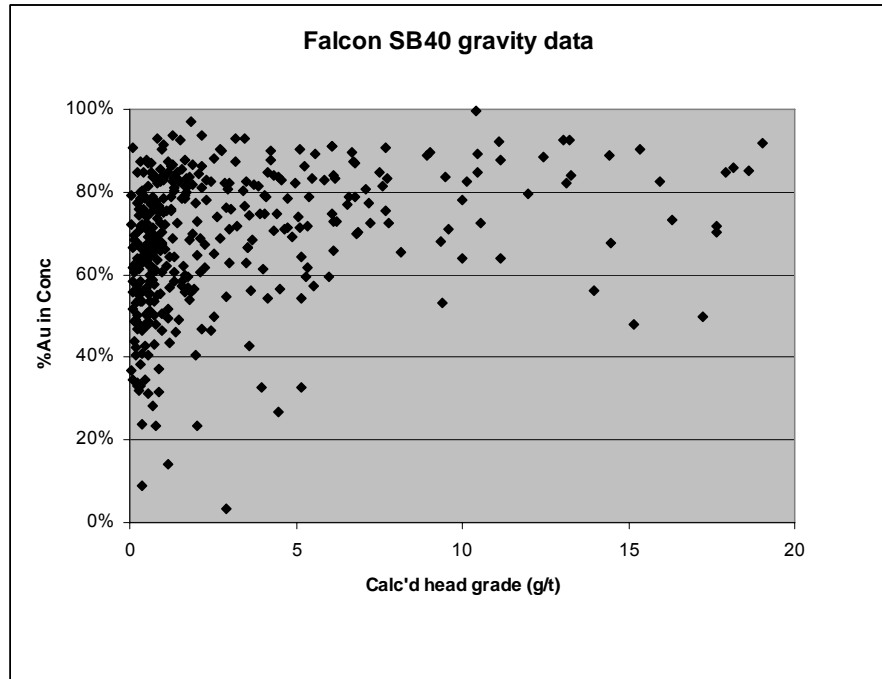


Table 9.6 Gold distribution from nine EMZ test samples

#	Gold distribution in the total sample (%)			Gold distribution in the sinks fraction as % of gold in total sample						
	%Au dist -25 µm	%Au dist floats	%Au dist sinks	Liberated	Attached to sulphides	Attached to Fe-Ox	Occluded in sulphides	Occluded in Fe-Ox	Total in sinks	
					Fresh					
HG	0.53	1.76	97.71	89.45	2.90	0.00	5.36	0.00	97.71	
MG	2.17	5.83	92.00	59.62	19.34	0.00	13.03	0.00	92.00	
LG	24.69	8.11	67.20	67.20	0.00	0.00	0.00	0.00	67.20	
					Lower Saprolite (Saprock)					
HG	2.25	4.06	93.69	47.41	36.75	0.00	9.53	0.00	93.69	
LG	17.31	6.56	76.13	58.11	17.75	0.00	0.19	0.08	76.13	
MG	13.35	7.68	78.96	66.06	0.00	0.00	12.90	0.00	78.96	
					Upper Saprolite					
MG	11.44	15.85	72.17	72.06	0.00	0.11	0.00	0.00	72.17	
MG	27.13	16.09	56.78	49.63	0.00	0.56	0.00	6.59	56.78	
HG	22.93	26.59	50.48	50.00	0.00	0.44	0.00	0.04	50.48	

The deportment study also measured diameters of gold particles (expressed as width x length) using an optical microscope and showed that, in general, high grade samples contain the highest proportion of large gold particles.

Visible gold particles have been recorded during core logging within and on the margins of quartz veins, intergrown with coarse arsenopyrite, and as isolated grains in the host rock. The usual associations are: (i) gold particles in white, extensional, quartz-carbonate veins, (ii) on fractures or peripheral to late carbonate which has developed along quartz grain boundaries, and (iii) associated with clusters of arsenopyrite grains. Mineralogical work shows that the gold occurs (iv) on sulphide grain boundaries, (v) as small filamental grains concentrated along fractures within the sulphide, or as coarse flakes >100 microns in size and wholly occluded by the sulphide, and (vi) interstitial to concentrations of tourmaline and arsenopyrite in the host rocks.

### 9.1.2 Structural controls on gold mineralization

Mapping of surface trenches and logging of oriented diamond core boreholes by Essakane has shown that there are distinct structural controls on gold mineralization in the EMZ. There are two basic controls: (i) gold associated with bedding parallel deformation within the main arenite, and (ii) gold associated with structures formed by the anticlinal folding event. The main structural features of the EMZ deposit are:

- The lithologies are folded into a west-verging anticline with a vertical west limb
- There are competency contrasts between arenite and argillite, and flexural slip along bedding planes is a pervasive deformation style in the deposit
- Early bedding-parallel, grey laminated quartz veins are related to flexural slip
- Late, steep extensional quartz veins with visible gold occur in the fold hinge and east limb domains
- Axial-planar pressure solution seams are developed in the fold hinge.

Oriented core drilling demonstrated that continuity of mineralization within the fold hinge domain is caused by a high frequency of steep extension N-S veins (commonly with visible gold) that strike parallel to the fold axis, and dissemination of mineralization along flexural slip and lithological contacts. Pressure solution veining appears to be more common in the footwall argillite and provides grade continuity down the fold axis. The lengths of individual veins are short and few veins longer than 10m are exposed in the surface trenches and workings. These tend to be the thicker veins. It is the density of veins which is the important factor. This pattern of mineralization extends into the east limb main arenite, with steep N-S veins supplemented by a lower frequency of E-W veins. Grade continuity is best developed along the following lithological contacts:

- Upper part of the east limb main arenite
- Marker argillite band within the east limb main arenite
- The arenite – argillite contact at the base of the main arenite
- The gradational contacts between the footwall argillite and footwall arenite units.

Continuity of mineralization in the steep west limb is poor and is caused by a low density of quartz veins. The tenor of mineralization is also low because the frequency of white, late-stage extensional quartz veins with visible gold is low, but

there are a few east – west extensional veins crosscutting the west limb which have been worked by the Artisanal miners. Dissemination of gold into wallrocks is rare and gold is largely confined to the early stage, bedding parallel grey veins.

## 10 Exploration

### 10.1 Project development

The 2006 project development exploration program on the EMZ was carried out by Essakane and focussed on quality of gold assay, quality of geological modelling and quality of mineral resource estimate. To this end Snowden was retained as lead consultant in 2006 to advise and vet data at important decision points. Essakane focussed on the following areas:

- Oriented HQ core drilling to extend resources
- Downhole surveying
- Twinning of Ranger RC boreholes
- Density measurements
- Sample preparation and assay protocols
- Re-assaying of BHP and Orezone pulp rejects
- QAQC best practice
- Analysis of preparation (field) duplicates
- Geological modelling and resource estimation
- Condemnation drilling
- Exploration potential and assessment of blue sky

Essakane contracted Boart Longyear and West African Drilling Services for the RC and DD drilling in 2006. Core orientation was carried out using a downhole spear with wireline attachment. Drill cores were placed in angle iron racks at the drill site and oriented by an Essakane geologist. A continuous top node line was drawn along the length of the core in black indelible ink. The start and end depths of the drilled interval were written on the core along with the metre marks. The cores were then packed into metal core trays at the drill site and transported to a dedicated logging area within the secure camp area. Wooden blocks with depths were also used to mark the start and end of drill runs. The borehole number, tray number and from - to depths of the interval were also written on the core tray.

The core was allowed to dry in the direct sun for at least two days. Each sun-dried core tray was then weighed and the bulk density of the core was calculated. This process is described in the next Section. The core was then logged by Essakane geologists with information recorded onto standard log sheets. After logging each core tray was photographed. The core was cut on the metre marks by diamond saw and each one metre sample was placed in a plastic sample bag and carried to the adjacent on-site sample preparation laboratory managed by SGS Essakane. For RC drilling, the entire sample was collected at the drill rig and transported to a dedicated RC sample preparation area within the secure camp site. Samples were allowed to dry in the sun before any sub-splitting took place.

Downhole surveying was carried out by the drilling contractors using Eastman downhole cameras. Survey results were calculated by Essakane geotechnicians. On average, camera shots were taken at downhole depths of 6, 31, 56, 81, 106, 131, 156 and 181 m. Drillhole collar positions were initially positioned by handheld GPS on local grid lines and dipped by the Essakane geotechnicians. After drilling the collar

position was picked up by a contract surveyor using a total station theodilite. Essakane completed a full re-survey by total station theodilite of all historical borehole collar positions. The collar positions are preserved by plastic standpipe and/or cement blocks with written hole IDs.

Orezone was project operator from July 2002 up to December 2005. Staff and senior project managers were employed by Orezone and reported directly to Orezone. Essakane took over as operator of the project in January 2006 with new senior project managers appointed by Essakane. Reporting to Orezone was by monthly progress reports and inter-office meetings in Ouagadougou. Periodic site visits by senior Orezone managers also took place.

## 10.2 Exploration potential

Exploration on the other targets and permits has highlighted potential to expand the Project's mineral resource inventory. A review of previous drilling and updated geological modeling and resource estimation by Essakane has resulted in a potential inventory of 1.9 Moz of gold contained in 38 Mt at an average grade of 1.6 g/t. Resources are all Inferred and include 220 Koz of geological potential at the northern extension of the EMZ.

The mineral inventory which is based on varying levels of drilling, but which is not part of the DFS, is presented in Table 10.1.

Table 10.1 Exploration potential

Classification		Comments	COG	Tonnes Mt	Au g/t	Au Koz
EMZ extensions						
EMZ depth extensions_1	JORC Inferred	May-07 kriged resources outside \$650/oz pit shell	1.0	5.9	1.7	320
CEMOB HL	JORC Inferred	Kriged resources on both leach pads	0.0	0.9	0.9	25
EMZ North	Target	200 m extension of block model	1.0	3.0	2.3	220
Essakane North	JORC Inferred	Kriged resources to 140 m depth	0.6	6.4	1.3	267
Subtotal				16.2	1.6	832
Other Prospects						
Falagountou	JORC Inferred	Kriged resources	0.6	7.1	2.0	462
Sokadie	JORC Inferred	Kriged resources to 200 m depth	0.6	5.1	1.1	180
Gossey	JORC Inferred	Kriged resources to 140 m depth	0.6	9.9	1.4	440
Subtotal				22.1	1.5	1 082
Total (100%)				38.3	1.6	1 914

## 11 Drilling

### 11.1 Introduction

Essakane drilled 20 364 m of oriented HQ diameter core between September 2005 and June 2006 for the EMZ project development and feasibility study program. The main objectives were (i) infill drilling to upgrade Inferred Resources within a US\$ 475/oz pit shell, (ii) expansion of the resource inventory to an upside shell at a gold price assumption of US\$ 650/oz, (iii) better understanding of the geology and controls on mineralization (e.g., vein orientations) to advance the geological modelling, and (iv), also improved the quality of assay samples. A summary of drilled metres by operator is listed in Table 11.1.

Table 11.1 Drill programs by operator for the Project to date

Operator	Hole type	Holes	Metres	RC precollar	DD tail	Orientation
BHP	DD	9	1 511			E - W and W - E
BHP	RC	88	5 732			Vertical
Ranger	DD	2	182			W to E
Ranger	RC	214	19 776			W to E
Ranger	RCD <sup>1</sup>	13	1 950	1 061	888	W to E
ORZ	DD	79	12 242			Mostly Vertical
ORZ	RC	520	51 288			Mostly Vertical
ORZ	RCD	233	38 986	24 612	14 374	Mostly Vertical
Essakane	DD	58	8 843			W - E and N - S
Essakane	RC	44	3 911			W - E and N - S
Essakane	RCD	72	16 449	4 929	11 520	W - E and N - S
TOTAL	All	1,332	160 869	30 602	26 783	

1. Hole type RCD means the holes were pre-collared with RC then completed by DD

Essakane changed the drill orientation from mostly vertical to west-to-east inclined holes at minus 60° degrees. This decision was based on the 2005 trench re-mapping and analyses of data by Essakane, but also to confirm the concept of steep grade zones used in the December 2005 geological model. Depth of drilling was guided by US\$ 475/oz and US\$ 650/oz Whittle shells developed for the model. The Whittle inputs were the same as used in the PFS. In-seam core holes dipped -50°E were also drilled to establish grade continuity down dip within the east limb main arenite and the footwall argillite. The longest in-seam hole was drilled to 300m (without excessive deviation) and demonstrated continuity of lithology, alteration and mineralization to end-of-hole. North – south holes collared on the EMZ were also drilled (4 186m in 22 holes) to assess frequency and grade of E – W quartz veins. Another objective was to measure continuity of mineralization between the East – West oriented drill sections. E – W veins with visible gold were intersected but the outcome seemed inconclusive at the time and the N – S drilling program was stopped.

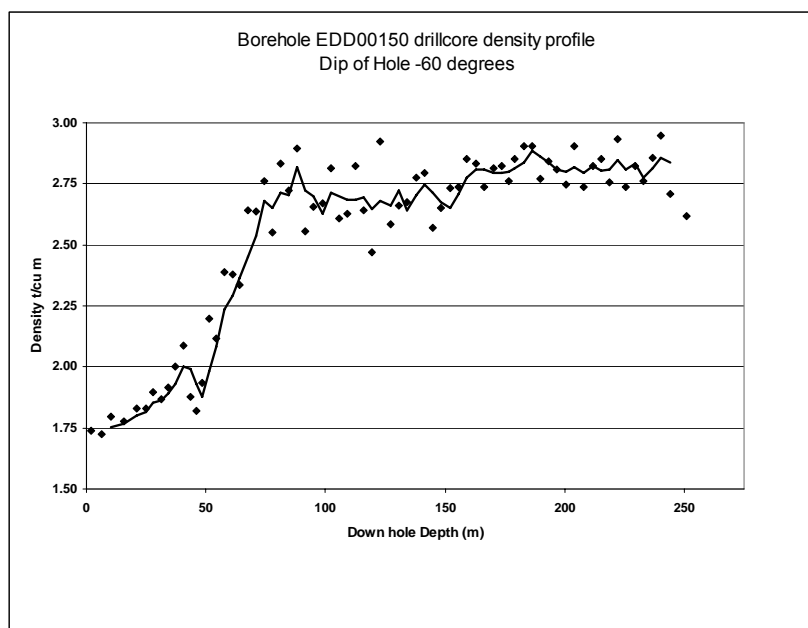
Essakane experienced serious delays in assaying the drill core samples and, coupled with quality assurance problems, Essakane and Snowden failed the on-site SGS

assay laboratory in August 2006. This meant that reliable assays were not available to guide further drilling until well after the EMZ program was completed. The drilling program ended in June 2006 but final assays were only reported between 6 August and 30 November 2006 after all 2006 drill samples had been re-sampled and assayed at SGS Tarkwa in Ghana.

## 11.2 Measurement of relative density (RD)

Immersion methods could not be used to measure the RD of upper saprolite due to the high clay content and friable nature of this material. All earlier resource estimates thus used average densities for Oxide (saprolite), Transitional and Fresh material types calculated from (i) immersion method data for Fresh and Transitional with the latter coated in paraffin wax or wrapped in plastic cling-wrap, and (ii) excavated 1 m<sup>3</sup> surface pits for Oxide. Essakane solved this problem by weighing the core trays after air drying the core in the direct sun for at least two days. The calculation of RD makes provision for core loss and the weight of the tray. Check samples 10 cm in length were taken from each tray as soon as competent core appeared in the trays and measured by immersion method. These data provided the first opportunity to model changes in relative density of saprolite with increasing depth. Figure 11.1 shows an example of a downhole density profile using the core tray method. The solid line is a moving average. In this example the weathering types are (a) upper saprolite 0 - 47 m, (b) lower saprolite 47 - 80 m, and (c) Fresh below 80 m. Additional detail is provided in Section 17.

Figure 11.1 Example of a downhole density profile



### 11.3 Twinned Ranger drilling

Ranger's field staff (in line with company policy) stored all sample rejects in bio-degradable plastic bags. The bags deteriorated rapidly in the high ambient temperatures and Orezone could not salvage any samples. Essakane thus twinned twenty seven of Ranger's RC holes in January 2006 with the new collar located 2 m from the existing hole (to a maximum of 5 m depending on ground conditions and proximity to Artisanal shafts). The twins were the first priority holes drilled in January 2006 to decide if Ranger's holes could be used in the DFS resource modelling. Comparison of the drillhole assay profiles and QQ analysis showed that re-drilling was not required and that Ranger assays could be used without factoring (for bias when compared with LWL69m gold-solution values). Additional detail is provided in Sections 14 and 17.

## 12 Sampling method and approach

### 12.1 Sample preparation and assay protocols

The sample preparation protocol used by Essakane for DD and RC samples was developed in conjunction with Snowden. The basis of the protocol is to reduce the GSE and FSE sampling errors in a coarse gold environment. Snowden also undertook an EMZ gold heterogeneity test.

The sampling protocols for DD samples are shown in Figure 12.1. Most of the 2006 drillholes were sampled as 1 m lengths of full core. The first 1kg assay subsample was split out only after the sample had been crushed to 80% passing 2mm. The entire 1kg subsample aliquot was pulverized to 90% passing 75 microns and assayed without further sub-sampling.

Sampling protocols for RC samples are shown in Figure 12.2. RC drilling during 2006 was mainly used as a pre-collar to DD holes. Drilling changed to DD as soon as wet samples were returned (generally at a depth of 45 – 50 m) or the weight of the 1 m RC sample (measured at the rig) was consistently below the expected weight over an interval of 5 m.

Efforts to diamond core drill from surface through the upper saprolite failed in most cases due to loss of drilling fluid and caving of holes. All holes on the EMZ are cased with hard PVC plastic tubing to 40 m which will have to be pulled prior to mining. Steel casings if used were removed.

All operators sampled the 5.25 – 5.50 inch RC holes at 1 m intervals at the drill rig. The entire sample was vented directly into a large sample bag after passing through the rig cyclone and delivered to the on-site sample preparation facility. BHP, Ranger and Orezone reduced the large 20 – 40kg RC rig sample down to 3 – 5 kg with an 8 : 1 riffle splitter.

Essakane in 2006 changed this to a single 1 : 1 stainless steel riffle splitter (unless the split was still larger than 15 kg). The 10 – 15 kg split was dried and pulverized to 90% passing 425 microns in a vertical spindle Keegor mill. ESSA and Eriez rotary splitters were then used to split out a 1kg sample which was pulverized to 90% passing 75 µm and assayed by LeachWELL rapid cyanide leach.

Figure 12.1 2006 sampling protocols for DD samples

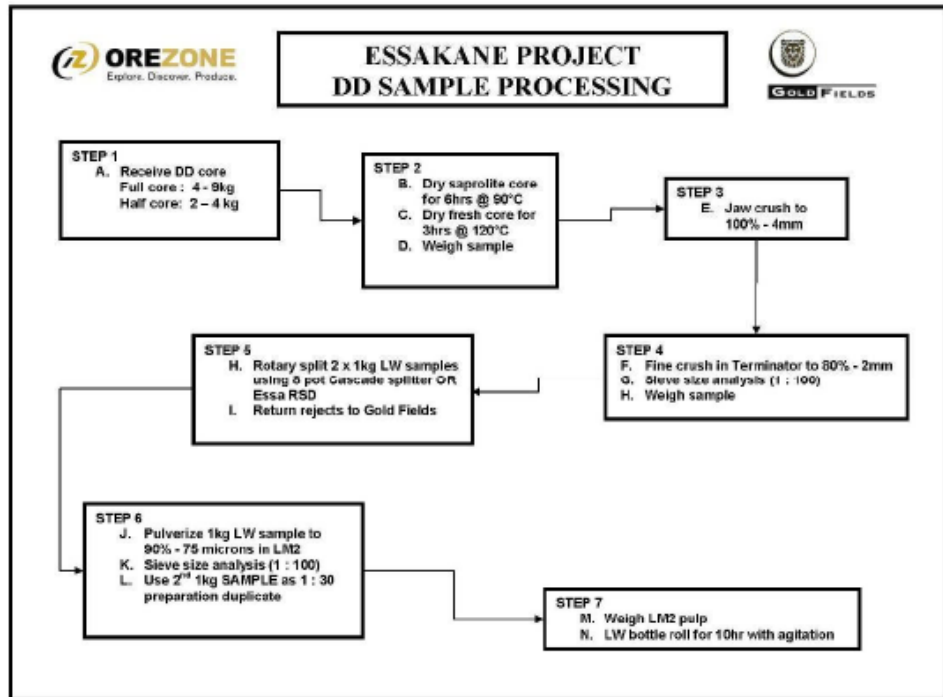
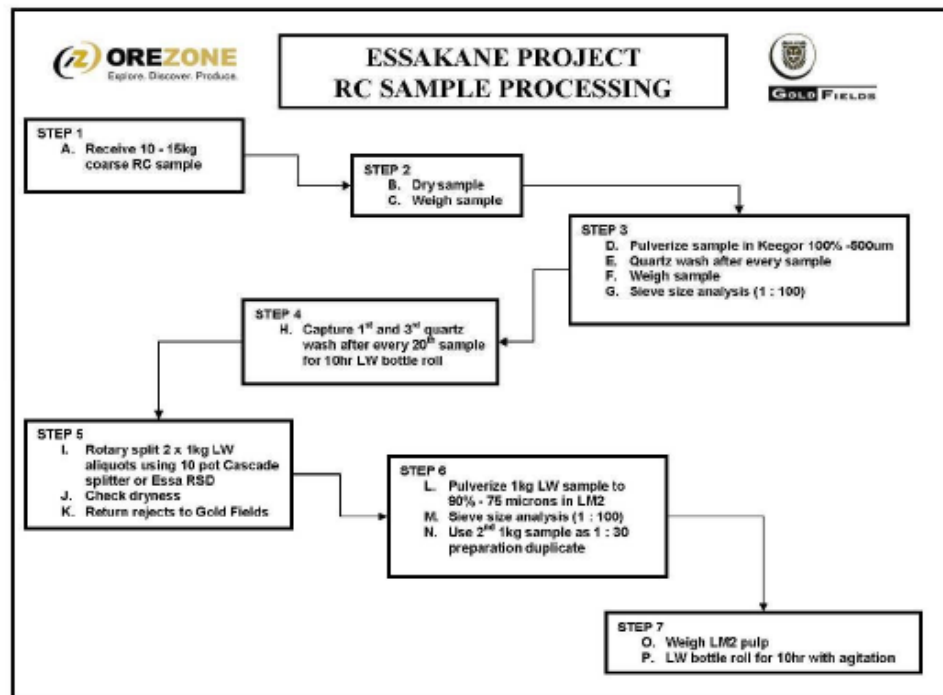


Figure 12.2 2006 sampling protocols for RC samples



The 1kg splits were pulverized and bagged at SGS Essakane which was under fulltime SGS management and transported by road to SGS Tarkwa in Ghana in sealed bags. The bags were sealed with metal clips and placed in large calico grain bags which were tied off. SGS Tarkwa collected the sample bags every 1 – 2 weeks using its vehicle and drivers. Essakane supplied a fulltime geologist to SGS Tarkwa to receive the samples at Tarkwa and manage the unloading and sample preparation

process. A small number of assays were completed by SGS Burkina Faso in Ouagadougou in 2007.

The standard LWL69M assay method used by SGS Tarkwa during 2006/07 was as follows: (i) Weight of sample = 1kg; (ii) Liquid : solid ratio = 2:1; (iii) 1 LeachWELL tablet added; (iv) Bottle roll duration = 10hrs, (v) Agitation by adding glass beads; (vi) Gold analysis by AAS, (vii) Fire assay of 1:10 tails.

## 12.2 Re-assay program

Essakane completed a range of bottle roll leach and gravity concentration tests in late 2005 and demonstrated that, on average, previous BLEG and LeachWELL bottle roll assays were biased low because of anomalously poor dissolution of coarse gold. Re-assay of BHP and Orezone pulp rejects started in May 2006. The pulps were recovered from storage and new 1kg splits were re-assayed using exactly the same LWL69M method at SGS Tarkwa. Eriez rotary splitters were used to split out the 1kg subsamples after drying the pulps.

Selection of drillholes and pulps for re-assay was made from geological cross-sections in Datamine showing gold grades, weathering type, stratigraphic unit, lithostructural domains and original assay laboratory. Pulps were generally selected in continuous intervals starting at 5 m above the HW argillite contact to the end-of-hole. Every effort was made to re-assay every borehole on every section within the west and east limbs of the fold. Mafic and intermediate intrusives, or long intervals assaying below detection limit, were not re-assayed to reduce sample pressure on SGS Tarkwa. A cut-off of 0.3 g/t was used as a guideline for selecting the limits of mineralized economic intervals for re-assay. Below a grade of 0.3 g/t, check assays accumulated by Essakane had showed that the differences between the original BLEG grades and the new LWL69M pairs are generally small.

Samples were re-assayed from 67 BHP boreholes (out of 100 drilled by BHP) and 465 Orezone boreholes (out of 785) in two phases:

- Panel F of the EMZ located between grid lines 50 700N and 51 000N
- All other available pulps located north and south of the Panel F block.

The pulp rejects were dried for four hours at approximately 110°C. Lumps were broken down with a pestle or passed through a Keegor mill at a grind of 500 microns if compacted. A 1kg sample was split out using Eriez rotary splitters and pulverized to 90% passing 75 microns in an LM2 mill for 3 minutes. Milling times were reduced from 5 minutes to control over-grinding and gold losses by smearing. In comparison, sieve tests on the original pulps generally showed poor grinds in the range 50 – 85% passing 75 microns.

The pulp re-assays have been used in the January 2007 and May 2007 models in the following way:

- If a new LWL69M assays exists: Replace the previous fire assay, BLEG or LeachWELL assay irrespective of source laboratory
- If no LWL69M assay exists: Apply the calculated remediation factors to the original fire assay, BLEG or LeachWELL assay according to original laboratory, assay method and g/t intervals

However, Ranger assays were used unadjusted. The calculation of remediation factors is discussed in Section 14.

## 13 Sample preparation, analyses, and security

### 13.1 Sample splitting

Essakane has been involved with the preparation of two sets of samples from the EMZ: (i) locating and re-preparation of sample rejects left in storage on site by previous operators, and (ii) preparation of samples generated by Essakane’s own drilling operations in 2006. These materials have all been analysed using LWL69M rapid cyanide leach. The majority of these assays were completed at the SGS Tarkwa laboratory in Ghana; a lesser number were completed at the SGS Burkina Faso Laboratory in Ouagadougou. Details of the comparisons of historic assay results and the re-assay results are presented in Section 14.

Sample preparation for pulp rejects involved the steps set out in Table 13.1.

Table 13.1 Sample preparation and assay procedures for LWL69M re-assay samples

SGS scheme code	Method	Description
<b>PRP86</b>	Dry and Split pulp sample	<ol style="list-style-type: none"> <li>1. Empty “As received” pulp reject into drying pan</li> <li>2. Turn the old sample bag inside out and make sure all sample falls into the pan</li> <li>3. Only use the new Sample ID tags</li> <li>4. Dry the sample at 110°C</li> <li>5. Split out 1000 g using a Cascade rotary splitter. Do not add or remove sample material with a spatula if the sample weight is not exactly 1000 g. Process the whole split.</li> <li>6. The same rules apply if a 50 : 50 Jones riffle splitter is used. Split the sample once and process the whole split.</li> <li>7. Process the entire “as received” pulp if the as received weight is &lt;1500 g.</li> <li>8. <b>DO NOT</b> mat roll under any conditions.</li> <li>9. Store any rejects for 60 days then return to Essakane</li> </ol>
<b>SCR32 1 : 100</b>	Wet screen 100g “as received” sample at 106µm	<ol style="list-style-type: none"> <li>1. Wet screen 1 : 100 “as received” samples to evaluate the historical grind by Keegor</li> <li>2. Report weight of coarse and fine fractions</li> </ol>
<b>PUL47</b>	LM2 pulverize the whole split which will be between 700 and 1500g.	<ol style="list-style-type: none"> <li>1. Pulverize the whole split in an LM2 mill to 80% passing 75 microns</li> <li>2. Care must be taken to avoid over - grinding and loss of gold by smearing</li> </ol>
<b>SCR34 1 : 100</b>	Wet screen 100g of the LM2 pulp at 75µm	<ol style="list-style-type: none"> <li>1. Wet screen 1 : 100 of the LM2 pulps to evaluate the LM2 grind performance</li> <li>2. Report weight of coarse and fine</li> </ol>

SGS scheme code	Method	Description
		fractions
<b>LWL69M</b>	<b>LeachWELL bottle roll the whole split (this will be approximately 1000 gs). DO NOT tamper with the split sample by adding or removing material to get precisely 1000 g.</b>	<ol style="list-style-type: none"> <li>1. NaCN Leach period = 10 hours</li> <li>2. Liquid : Solid ratio = 2 : 1</li> <li>3. No. of LW tablets = 1</li> <li>4. Add 4 glass balls to assist agitation</li> <li>5. Allow to settle and decant the pregnant solution</li> <li>6. Analyze by solvent extraction AAS finish</li> <li>7. Report as LWL69M_ppm</li> </ol>
<b>FAS31K 1 : 10</b>	<b>For 1 : 10 Preparation Duplicates by <u>screen fire assay</u></b>	<ol style="list-style-type: none"> <li>1. Randomly select the 2<sup>nd</sup> half of a split as per PRP86</li> <li>2. Weigh the whole split (TOTWT) in grams</li> <li>3. Pulverize the whole split to 90% passing 75 microns in an LM2 mill</li> <li>4. Dry screen the whole pulp through a 106 micron sieve cloth</li> <li>5. Weigh CORS fraction</li> <li>6. Fuse the CORS fraction and the sieve cloth in a Pb collection fire assay</li> <li>7. Duplicate fire assay the FINE fraction as FA50g by Pb collection</li> <li>8. Report TOTWT, CORSWT, CORS_AU, AU (x2), CALC_ppm</li> </ol>
<b>GFL Standards / Blanks</b>	<b>LWL69M</b>	<b>Report results as LWL69M_ppm</b>

The following protocols were in place for the 2006 drill samples:

- Essakane developed specific sample splitting procedures for RC and DD samples based on estimates of sampling errors by Dr S Dominy of Snowden.
- The large RC samples measuring 20 – 40kg in weight were split in a 1 : 1 stainless steel riffle splitter to reduce the sample weight to approximately 10kg.
- The 10kg RC samples were then split into 10 x 1kg subsamples using 8- or 10-pot rotary splitters. The subsequent sample preparation procedures are presented in Figure 12.2.
- Essakane had 4-, 6-, 8- and 10-pot rotary splitters available on site that it could use to split out 1kg subsamples depending on the initial weight of the sample.
- Full core sample of HQ diameter core was generally carried out. The sample preparation procedures for drill core are described in Figure 12.1.

### 13.2 Certified Reference Materials and blanks

Essakane introduced a comprehensive QAQC system involving insertion of Certified Reference Materials (CRMs) supplied by Rocklabs. A list of reference materials used in the assay and re-assay program is provided in Table 13.2. The Count column lists the number of times each CRM was used. The CRMs were selected on the basis of a range of gold grades and Oxide or Sulphide oxidation type. Oxide CRMs were inserted with upper and lower saprolite samples. Sulphide CRMs were inserted with Fresh arenite and argillite samples.

The procedures for CRMs were:

- Insertion rate 1 in 20
- Range of gold grades 0.8 – 8.3 g/t
- Oxide and sulphide samples
- Sample weight 200 g
- Analysis by LWL69M rapid cyanide leach
- Check fire assays on CRM tails
- Acceptance range for LWL69M solution values = 95 – 105% of Expected Value.

Table 13.2 List of certified Rocklabs reference materials

Selected CRMs supplied by Rocklabs Ltd.			
Type	CRM	EV (Au g/t)	Count
Oxide	OXF53	0.810	435
	OXG46	1.037	695
	OXH52	1.290	39
	OXI40	1.857	316
	OXI54	1.868	439
	OXJ47	2.384	582
	OXK48	3.557	432
	OXL34	5.758	34
	OXL51	5.850	1086
	OXN49	7.635	62
Sulphide	SE19	0.583	23
	SH13	1.315	42
	SH24	1.326	1039
	SJ22	2.604	579
	SJ32	2.645	292
	SK21	4.084	555
	SN26	8.543	317

Results for every batch of CRMs reported by the assay laboratory were assessed by Essakane prior to upload of any assay data into the SQL database. The average of the CRM results for every batch was reported to the laboratory manager in a qualitative way by e-mail (e.g., trends showing over- or under-estimation; evidence for poor instrumental drift corrections; differences occurring at AAS operator shift changes; decay of gold standard solutions). Records of these assessments are stored in the Essakane database.

Coarse quartz blanks were inserted at a rate of 1 : 15 and generally were preparation blanks as shown by the Count column below. 1kg bags of quartz blank material (provided in bulk at 4 mm crush size) were inserted into the sample stream and prepared in the same way as any other RC or DD sample. Additional quartz blank samples were generally inserted after samples containing visible gold. Aliquots of 200g were used if a Rocklabs blank was used.

Analytical blanks used in 2006/07 programs	EV (Au g/t)	Count
Blank pulps supplied by Rocklabs Ltd		
AUBLANK5	0.005	30
AUBLANK7	0.005	70
AUBLANK8	0.005	107
Preparation quartz blanks from local sources		
BLB001	0.005	239
FALQ001	0.005	1959
FALQ002	0.005	6174

### 13.3 Preparation duplicates

Essakane prepared duplicate assay samples in the following way:

- Rate = 1 : 10 in 2006 reduced to 1 : 20 in 2007
- Taken as a second 1kg split at the rotary splitting of -2mm crushed material for DD and RC samples. In the case of re-assays the preparation duplicate was a second 1kg split of sample pulp.
- Identity not known to the laboratory and samples sometimes sent in different batches.
- Initially analysed by total Au screen fire assay (SFA) to demonstrate that LWL69M dissolution was achieving >95% leach 90% of the time.
- Changed to LWL69M with fire assay of tailings because SGS Tarkwa was slow in reporting SFA results and after the leach efficiency of LWL69M had been proven by the SFA data.
- Splitting and assay of selected samples to extinction to assess intra-sample variability (caused by coarse gold).

The precision of these preparation duplicates is discussed in Section 14. A summary of % leach for LWL69M based on the fire assay of tailings is presented in Tables 17.6 and 17.7.

### 13.4 Security

Essakane started transporting 1kg pulps in sealed sample bags by road to SGS Tarkwa in Ghana in May 2006. This continued through to completion of the re-assay program in April 2007.

Essakane has represented that there are no known issues relating to tampering with samples. A fulltime Essakane employee, reporting directly to the laboratory Manager, was seconded to SGS Tarkwa to act as receiver and manage the movement of samples within the laboratory. Sample bags were sealed with metal clips before transport and Essakane represents that no instances of sample spillage or leaks were reported by the employee assigned to SGS Tarkwa.

At no time was any employee, officer or agent of Orezone Resources Inc involved in preparation or transport of assay samples.

Essakane elected to undertake sample preparation at a custom built SGS facility on site in order to monitor quality control during sample preparation. The facility was managed and serviced at arms length by SGS Essakane.

### 13.5 Assay laboratory

Almost all of Essakane's samples were analysed by SGS Tarkwa in Ghana. A small number of samples were analyzed by SGS Burkina in Ouagadougou during 2007 to assist with a backlog of samples. SGS Tarkwa was selected as the primary assay laboratory because of its experience in LWL69M assay as a provider of this method to Tarkwa Gold Mine for all its grade control assaying. The Essakane employee mentioned above was also responsible for ensuring that the analytical protocols agreed with the Laboratory were adhered to.

Two independent audits of SGS Tarkwa were completed during the course of the assay and re-assay programs in 2005 and 2006.

### 13.6 Quality control measures

Essakane tested the performance of LeachWELL versus BLEG cyanide leach performance as a function of grind in 2005 and determined that 90% passing 75 microns was important in terms of reproducibility of assay result. SGS maintained a fulltime laboratory manager at SGS Essakane to ensure that the Terminator crushers and LM2 pulverizers were operating according to the prescribed sample specifications. Daily C - Class sieve tests were reported by SGS directly to Essakane.

LM2 pulverizers were used in the preparation of all samples. After milling, the sample was emptied directly from the LM2 bowl through a steel cone jig into a plastic sample bag to ensure that the entire 1kg pulp was recovered. Scooping direct from the bowl or mat rolling the pulp was not permitted at any time. However, according to available records, mat rolling was used in some instances by previous operators. A 2 x quartz wash was used after every grind and the material was discarded into bins placed at the LM2. Essakane sampled and assayed these quartz wash bins on a regular basis to assess gold losses caused by smearing in the LM2 bowls. It was noticeable that gold values reported to the quartz wash after high grade samples, indicating that gold losses from gold particles can occur.

Regular laboratory checks were carried out by Essakane. Periodic technical reviews were also carried by SGS auditors on request from Essakane.

### 13.7 Check assay methods

By agreement with Snowden, Essakane introduced an umpire check assay process by analyzing 1:10 preparation duplicate samples by 1kg screen fire assay at SGS Tarkwa. The main objective was to establish early on if LWL69M was achieving

better than 95% leach on all samples within 10hrs of leaching irrespective of gold grade and rock type. The results for the first 384 pairs showed that this was achieved; the results are discussed in Section 14. The procedure for preparation duplicates was thus changed to assaying 10% of the LWL69M tailing by 50g fire assay. This step allowed estimation of leach efficiency as a % recovery factor.

Essakane also introduced analysis by gravity using on-site SB40 Falcon concentrators. Remaining sample rejects up to 30kg in weight were pulverized to 90% passing 425 microns in vertical spindle Keegor mills and passed through the concentrators. The gravity concentrates and 1kg splits of the dried tailing were airfreighted to SGS Lakefield in Johannesburg for fire assay. Samples were selected on the basis of (i) samples with visible gold, (ii) samples with high arsenopyrite contents, (iii) whole borehole check assays of reject samples. The results confirmed that 1kg LWL69M is an effective measure of gold grade in the EMZ samples.

### 13.8 Adequacy of sampling

Within the technical difficulties of sampling a severe coarse gold deposit such as the EMZ, Essakane has maintained acceptable levels of quality control and quality assurance during sample preparation and assaying. It has demonstrated by check assays (using total gold analytical methods such as screen fire assaying and gravity analysis) that 1kg LWL69M is an appropriate analytical method for EMZ sampling. On this basis the LWL69M re-assay and associated remediation programs to normalize historical assays to LWL69M gold solution assays are justified.

## 14 Data verification

### 14.1 Introduction

A significant proportion of the assay data for the Project has been generated by previous operators. However, much of this historical data have been generated either with inadequate QAQC measures in place, or uncertified reference materials were used, and thus made the quality control measures equivocal. RSG Global completed a review of the recorded quality control data for the PFS. The key findings of this review are summarized below:

- The use of uncertified (unaccredited) standards should be discontinued; use of 250 gram standards for BLEG and LeachWELL cyanide leach assays should be examined;
- Orezone standards appear to have been mixed up during laboratory submission; much of Orezone standard data appears unusable;
- The Abilabs QAQC results appear to be substandard;
- A bias between BHP FA (ITS FAA) and BLEG assays may be result of incomplete dissolution during BLEG process;
- Indications exist that unaccounted Au is present within samples that report BLEG results less than 1.0 g/t; this suggests that tails samples should be taken for all BLEG samples > 0.5 g/t;
- Heterogeneity testwork should be undertaken to optimise the subsampling protocol.

### 14.2 Essakane comparative analysis of assays

#### 14.2.1 LWL69M rapid Cyanide Leach at SGS Tarkwa

The LWL69M rapid cyanide leach procedure provided by SGS Tarkwa is used by Tarkwa Gold Mine for its grade control assaying. SGS Tarkwa has thus acquired considerable expertise with this method. The method is described in this section. Supporting work that compares this analytical technique with 1kg SFA is also described together with work on replicate LWL69M assays. This analytical process has been accepted by Essakane as suitable for EMZ samples following tests during 2005.

The first set of preparation duplicates was analysed at SGS Tarkwa using LWL69M cyanide leach for the original sample and a conventional 1kg SFA on the duplicate. The 1kg preparation duplicates for SFA were collected from the rotary splitters, then pulverized separately in an LM2 mill to 90% passing 75 microns. That is, assay variance between original and duplicate in these preparation duplicate pairs also contains intra-sample inhomogeneity at a coarser grind.

The historical samples were originally pulverized in most cases by Keegor vertical spindle mills. Checks by Essakane during the pulp re-assay program found that the historical grinds had varied from 50 – 95% passing 75 microns.

The results of the duplicate pairs at a grind of 90% passing 75 microns are described below. SFA at Tarkwa used a 106 micron cloth screen to select the oversize fraction. The oversize was fired as a single charge, with the cloth screens included and fired in the crucible.

The undersize was analysed as a duplicate 50 g fire assay. The undersize tailing was not assayed to extinction.

Three hundred and forty eight pairs were analysed using 1kg LWL69M cyanide leach and 1kg SFA with both assays completed by SGS Tarkwa. Comparison of the results showed one anomalous SFA sample value (Sample ID=219730, SGS Tarkwa LWL69M=5.95 g/t and SGS Tarkwa SFA=211.27 g/t), which reported a gold value thirty five times greater than the LWL69M assay results. This sample pair was excluded from further analysis. Comparative statistics for the remaining 347 sample pairs are presented in Table 14.1.

Table 14.1 LWL69M compared with screen fire assays

Statistic	LWL69M	Screen fire assay
Count	347	347
Minimum	0.005	0.01
Maximum	27.1	25.25
Average	0.98	0.92
Standard Deviation	2.78	2.40
Variation	7.72	5.74
CoV	2.83	2.61
Correlation	0.91	

The paired results are also presented within a scatter plot in Figure 14.1. The data scatter is relatively high despite the correlation coefficient of 0.91.

These data can also be compared meaningfully within a quantile-quantile plot (QQ plot) that compares the grades for equivalent quantiles within the sample distribution. A QQ plot for the SFA (y-axis) and LWL69M assay data (x-axis) is presented in Figure 14.2. At low grades (<2.5 g/t) the SFA data typically exceed the LWL69M results (see also Figure 14.3). This effect can be partly explained by the fact that the LWL69M leach does not account for all the gold present: a small amount remains entrained within the leach tailing. However, at higher grades (above 5 g/t) LWL69M assays tend to report higher grades than SFA.

SFA is considered to be one of the preferred analytical approaches for analysis of samples containing coarse gold. One major disadvantage of this technique is the length of time required to complete one assay: laboratories undertaking this analysis need a large number of vibratory screens to complete large numbers of assays. The LWL69M assay takes a similar length of time to complete but large numbers of assays can be completed concurrently relative to SFA.

Slow delivery of the selected mesh to SGS Tarkwa also resulted in SFA duplicate results being reported many weeks (and sometimes months) after completion of the original LWL69M result. SFA as the primary assay method for 50 000 Essakane samples was thus considered to be impractical.

Figure 14.1 Scatterplot for LWL69M vs screen fire assay

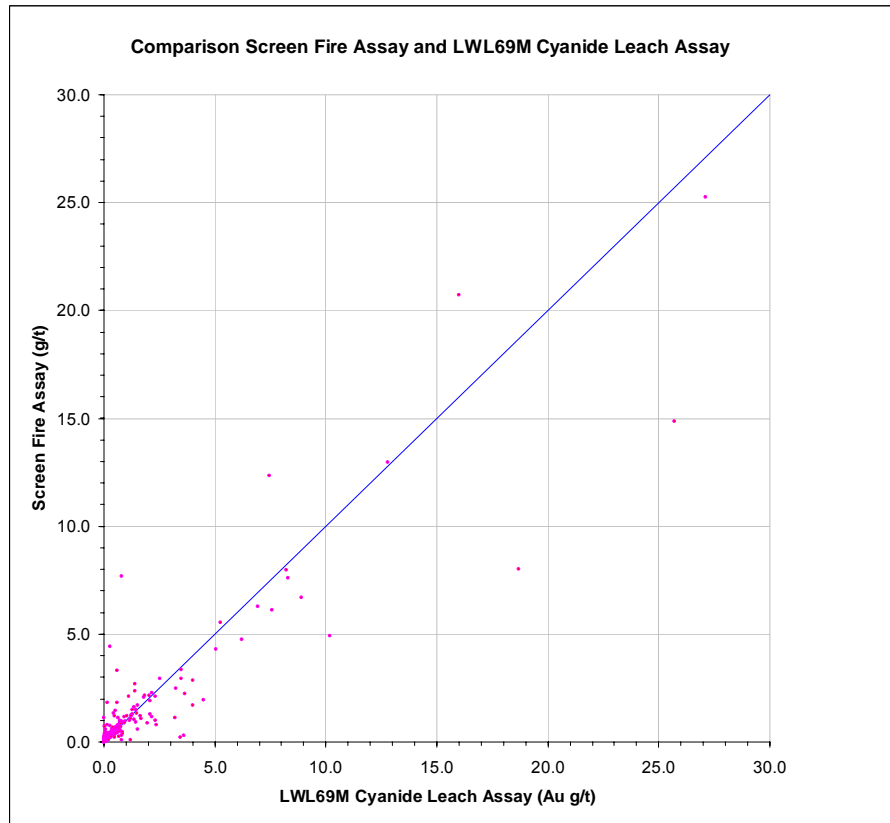


Figure 14.2 Preparation duplicates: QQ plot of LWL69M vs SFA results

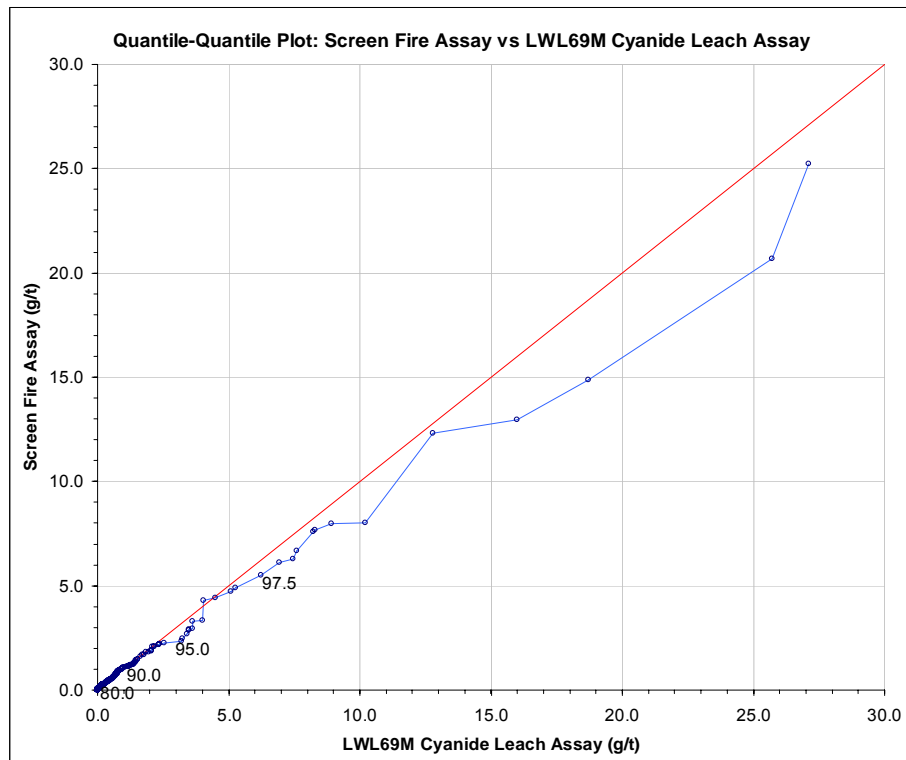
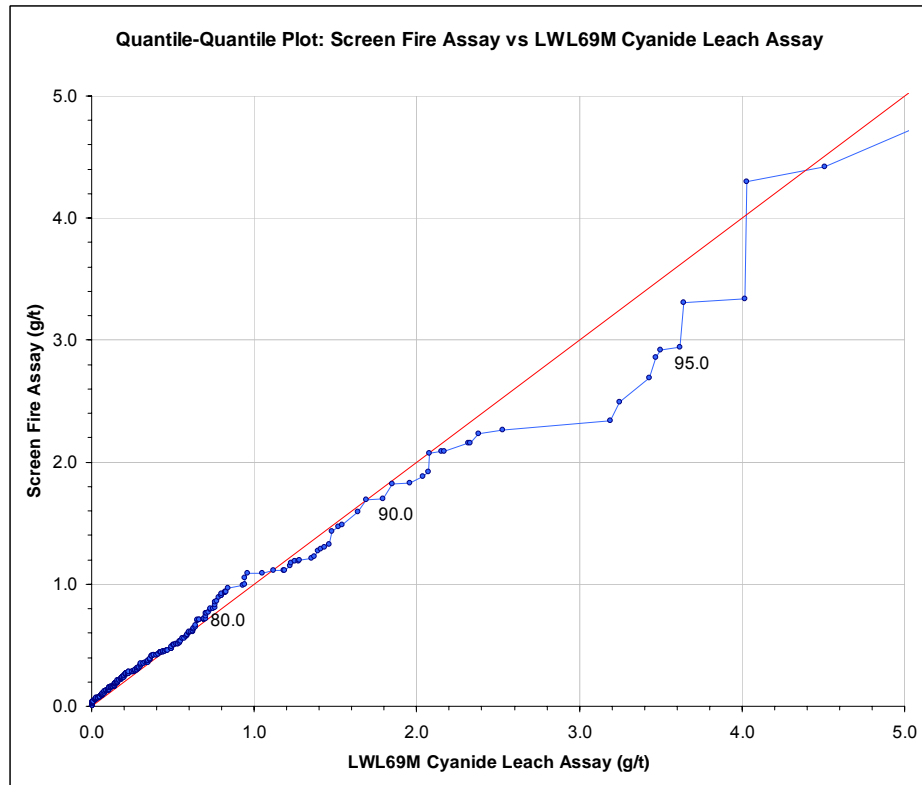


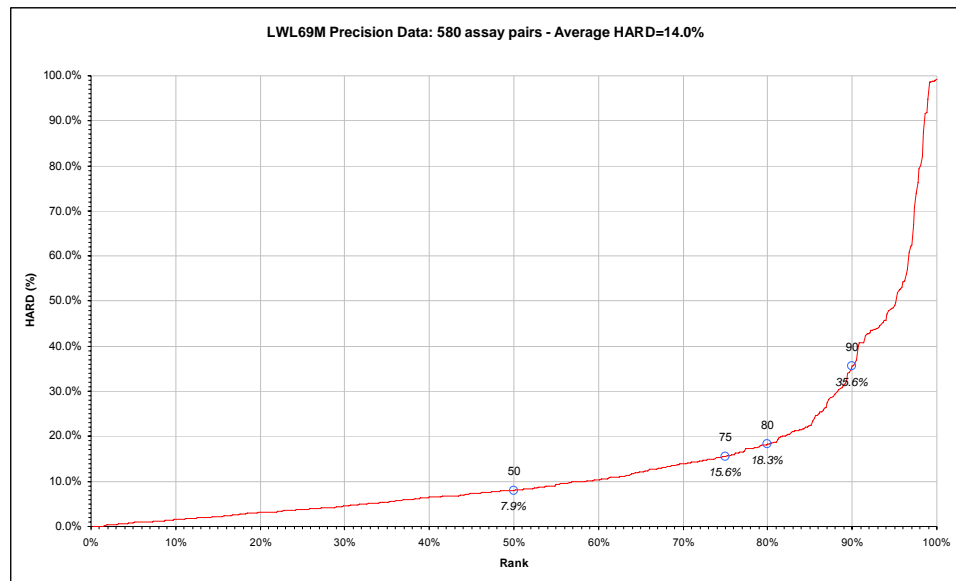
Figure 14.3 Close-up of Figure 14.2: QQ plot of LWL69M versus SFA results



Comparison of the two sets of assays reveals a 6% difference in the mean values, with LWL69M reporting higher (on average) values relative to SFA. The linear correlation of the two sets of data is reasonable, although it is influenced by the higher grade samples to a certain extent. Whilst the oversize fraction (+106 microns) is generally considered to contain the majority of gold, particles of 40 to 90 microns contained within the undersize fraction may account for significant gold grades within this fraction as well. It is notable that the undersize fraction, which may have a mass of several hundreds of grams, has been assayed in duplicate using a 50 g aliquot fire assay. It is considered likely that the grade deficit observed between LWL69M and SFAs may be attributed to undersampling of the SFA undersize fraction.

One hundred and four samples were also prepared and rotary split to yield multiple 1kg samples. The large RC drill samples were crushed and then pulverized within an open circuit vertical spindle mill to P<sub>100</sub> less than 425 microns. Assay aliquots of 1kg were split out from the pulverized sample using a Cascade rotary splitter. Each 1kg aliquot was then pulverized separately within a closed circuit LM2 mill to P<sub>90</sub> passing 75 microns. A total of 580 sample pairs were developed in this manner and were all subjected to LWL69M at SGS Tarkwa. The paired assay data were used to construct a ranked Half Absolute Relative Deviation (HARD) plot, which provides a measure of the analytical and sampling precision. The resultant HARD plot developed from these data is presented in Figure 14.4.

Figure 14.4 Ranked HARD Plot for 580 pairs of LWL69M assays



The Ranked HARD plot shows that 90% of the sample pairs have a HARD value of 35% or less. Long (1998) recommends that for coarse rejects, agreement of  $\pm 20\%$  on 90% of pairs is desirable, whilst for pulp duplicates agreement of  $\pm 10\%$  on 90% of the pairs is required. Long (1998) uses a Ranked Absolute Relative Deviation (ARD) plot to monitor precision which gives two times the value obtained from an HARD plot. Accordingly, when using a HARD plot, Long's  $\pm 10\%$  agreement on 90% of pulp duplicate pairs is equivalent to  $\pm 20\%$  agreement on the HARD plot. In the same way, Long's  $\pm 20\%$  agreement for 90% of coarse reject pairs is equivalent to  $\pm 40\%$  HARD. The sampling protocol for these preparation duplicates split the aliquots at a nominal grain size of P<sub>100</sub> -425 microns. This is coarse for normal pulps but fine for normal coarse rejects, implying that neither of Long's recommended values of  $\pm 20\%$  and  $\pm 40\%$  for 90% HARD are applicable to this material.

To summarize, LWL69M assays are comparable to SFA and have a precision that approaches that recommended by Long (1998) for coarse rejects, but falls below that recommended for pulp duplicates.

The 2006/07 sample preparation protocols were reviewed by Dr S. Dominy for Snowden. For RC samples the entire field sample ( $\pm 30\text{kg}$ ) was passed through a 1:1 riffle splitter to yield  $\pm 15\text{kg}$ . This 15kg sample was dried and pulverized within a vertical spindle mill to P<sub>100</sub> -500 microns. Rotary splitters separated the sample into equal 1kg subsamples. In the case of a 10-pot splitter, Pot 1 was always taken as the LWL69M assay sample and Pot 6 was the preparation duplicate sample.

For diamond core samples, the full core was crushed to P<sub>100</sub> -4 cm in a Bruno crusher and then passed through a Terminator crusher to achieve P<sub>80</sub> -2 mm. Rodding was a problem from time to time with damp samples: when this occurred the sample (which failed P<sub>80</sub> -2 mm) was dried and pulverized in a Keegor mill to P<sub>100</sub> -500 microns before the first splitting stage. All the pots on the rotary splitters were numbered and the original and duplicate samples were always opposite pots (e.g., 1 and 3 for a 4-pot splitter or 1 and 4 for a 6-pot). Selected 1kg samples were pulverized to P<sub>90</sub> -75 microns in an LM2 mill and the entire 1kg pulp was then assayed with no further splitting. That is, every effort was made to limit the number of sample handling steps.

In Snowden's opinion the EMZ must be regarded as having a severe sampling problem, characterized by extreme variations and a very high sampling constant ( $K=14,100\text{g/cm}$  based on an estimated gold liberation diameter of 600 microns).

The LWL69M assay proceeds with the following steps:

- Pulps with masses of nominally 1000 g are combined with 2000 ml of water within large polyethylene jars.
- Four glass marbles and 1 LeachWELL tablet are added to the slurry.
- The jar is closed after dissolution of the LeachWELL tablet and placed on a roller table.
- Leaching proceeds for 10 hours with continuous rolling of the sample jars assisted by the glass marbles within the leach vessel.
- The jars are then unloaded from the roller table and allowed to stand for approximately 90 minutes to permit the pregnant solution and sample sediment to separate.
- Approximately 200 ml of pregnant cyanide leach solution is abstracted by pipette and stored in open polystyrene cups.
- 30 ml of this solution are decanted into a graduated test tube into which four millilitres of Di-isobutyl ketone (DIBK with 1% aliquat 336) is added using an automatic dispenser.
- The glass vessel is closed with a screw-top plastic lid and shaken vigorously for approximately one minute. During this process the gold-cyanide complex preferentially partitions into the organic phase by solvent extraction.
- The test tubes are placed in racks and sent to Atomic Absorption for Au analysis.

The AA spectrometer is a double beam instrument and in its passive state all blank solution aspirated into the instrument is DIBK. In addition, all standards that are read during instrument calibration are hosted in DIBK-media such that the fuel-air mixture is consistent and does not change when standards and samples are aspirated into the instrument. The flame is maintained using an air-acetylene mixture; no nitrous-oxide or oxygen is employed. Calibration of the instrument makes use of standard solutions derived from SpectroSol certified Au-bearing solutions in an ionic acid media. Solutions with variable Au concentrations are prepared by dilutions from the standard 1000 ppm Au reference solution. The standard diluted solutions are absorbed into DIBK. SGS Tarkwa used DIBK Au concentrations of 1 ppm, 2 ppm, 5 ppm, 10 ppm and 25 ppm as the AA standards for calibration of the AA at the start of each shift and for instrument drift corrections during the shift.

### 14.3 Essakane validation and remediation

Essakane implemented re-assay programs designed to upgrade the quality of the historical analytical database and add a significant number of new assays. A large number of sample rejects are stored at site, representing sample rejects stored by BHP, Ranger Minerals and Orezone. Of these, the Ranger Minerals rejects were stored within bio-degradable plastic bags that did not permit viable recovery of sample rejects. Validation and re-assay of Ranger materials was thus not possible through assay of reject materials; a set of twin drillholes were developed to validate the Ranger drillhole data.

### 14.3.1 Ranger Minerals twin hole validation program

Ranger fire assayed 21 844 samples at TransWorld Laboratories in Ghana. Sample rejects were stored on site in bio-degradable plastic bags which unfortunately perished rapidly and prevented successful recovery of sample rejects. A series of holes were thus drilled in January 2006 to twin selected Ranger holes. Twenty three holes yielded sample data that could be directly compared with the corresponding lengths of Ranger drillholes, consisting of 1 555 pairs of assay data. In this case direct comparison of assay values is difficult because the samples are not the same materials.

However, the QQ plot for the corresponding LWL69M assays and the TransWorld LWL69M assays shows a similar grade distribution within the two sample sets. The average grades and standard deviations of the two data sets are also similar (LWL69M average = 0.98 g/t and std dev = 2.82, TWLFAA average = 0.95 g/t and std dev = 2.86).

Statistics of the twinned hole assay data are presented in Table 14.2. There is a poor correlation between the two sets of data because, although the holes are twinned, the incidence of high grades within steeply dipping veins implies that a direct comparison of grades in twinned holes *by depth* is not a valid sample-sample comparison. Despite no metre-by-metre linear correlation, the mean grades, standard deviations and QQ plots are very comparable. The scatterplot between LWL69M assays and the TransWorld fire assays is presented in Figure 14.5. The QQ plot comparing the SGS Tarkwa and TransWorld Fire Assays is presented in Figure 14.6.

Table 14.2 Comparison of twinned Ranger and Essakane drillholes

Statistic	LWL69M Assay	TWL FA Assay
Count	1 555	1 555
Average	0.98	0.95
Minimum	0.01	0.01
Maximum	49.10	51.73
Variance	7.97	8.16
standard Deviation	2.82	2.86
CoV	2.89	3.01
Linear Correlation	0.01	
RMA Slope	1.01	
RMA Intercept	-0.04	

Despite the poor correlation within the scatterplot, the comparative statistics of the Ranger data and the QQ plot show that the Ranger FA data (assayed by TransWorld) are comparable to LWL69M cyanide leach assays developed at SGS Tarkwa in 2006.

Figure 14.5 Twinned holes - scatterplot of Ranger FA vs SGS LWL69M

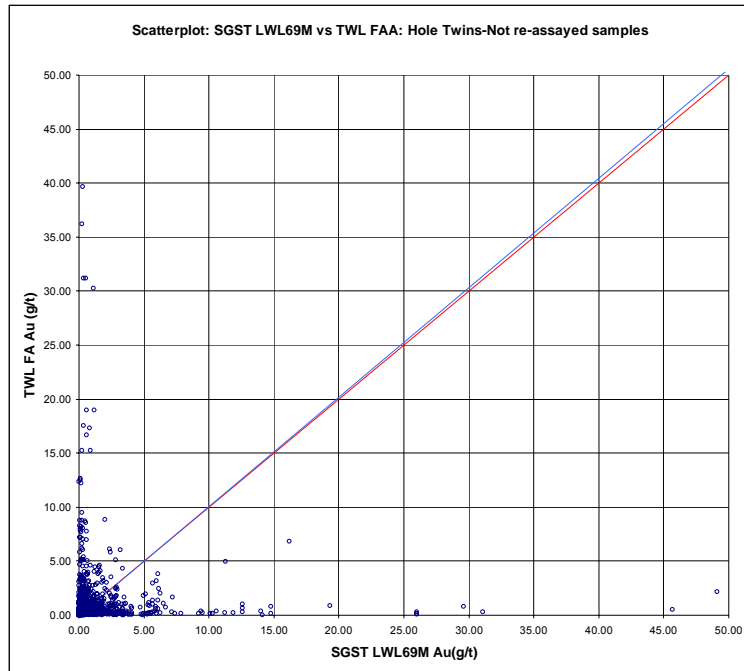
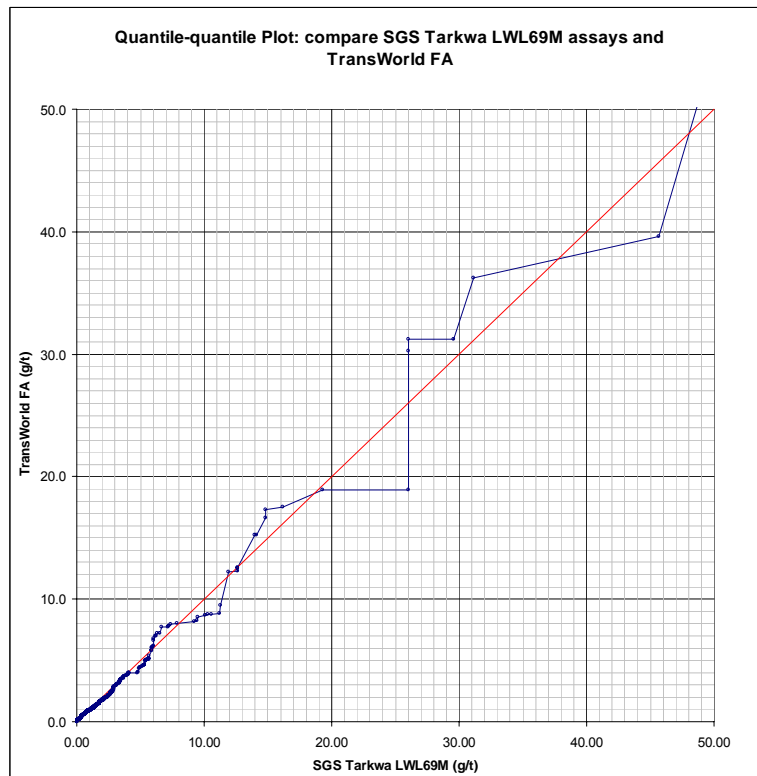


Figure 14.6 QQ plot of Ranger FA vs SGS LWL69M re-assays



The re-assay data acquired up to May 2007 have been systematically compared with the original assay data, with the various assays grouped by laboratory and assay method. Data statistics were compiled and analysed and scatterplots and QQ plots were developed for each laboratory and method group. Comparison of these

results has revealed systematic biases between historical assay data and the LWL69M re-assay data. A program of correction or remediation has been devised in the following manner:

- Acquire the paired sample data for each laboratory and method from the master database;
- Generate statistics, scatterplots and QQ plots for each paired dataset;
- Within each QQ plot, observe the areas of greatest divergence and classify these in terms of grade ranges;
- Iteratively develop a set of stepwise factors and corresponding grade ranges within which the factors are applicable, such that application of these factors results in a closer correspondence between the quantiles of the modified assay data and the original LWL69M assay results.
- Check the statistics of the modified assay results and, if acceptable, apply these factors to the remaining unpaired assay data.

Inherent in this procedure are a set of assumptions, the major ones being listed below:

- This process seeks to remove global biases and equilibrate the historical assay data with the later LWL69M assays, but no procedure is capable of removing or reducing the local analytical imprecision. This imprecision is handled within the Resource Classification approach and, as a result, no EMZ material can be classified as a Measured Mineral Resource.
- It is considered preferable to retain imprecise (but globally unbiased) measurements within the estimate because they help to improve the quality of the estimated results (Emery *et al.*, 2005). Hence, data subjected to remediation was combined with demonstrably correct LWL69M assay data and can thus participate in the Mineral Resource estimate on an equal basis.

For the May 2007 exercise, a total of 28 640 re-assay samples were available. In the following sections, the statistics of the LWL69M re-assays and their paired data is presented for each of the major laboratory - method groups. QQ plots comparing the assays are also presented.

### 14.3.2 Abilabs fire assay

Table 14.3 Statistics of Abilabs FA and paired LWL69M re-assays

Abilabs FAA	LWL69M	ABLFAA	Remediated ABL FAA paired
Count	7 105	7 105	7 105
Average	0.98	0.89	0.98
Minimum	0.005	0.005	0.005
Maximum	168.00	248.53	223.68
Standard Deviation	4.47	5.08	4.77
CoV	4.58	5.69	4.88

The QQ plot in Figure 14.7 supports a bias between the Abilabs FA and the LWL69M re-assay data. The raw data QQ plot in Figure 14.7 is shown as the blue line; the orange line represents the QQ plot after application of the remediation factors. In broad terms, the nature of the bias is similar to that observed within the first comparison in January 2007, viz., LWL69M reports higher grades for the majority of the data distribution but LWL69M reports lower grades for the highest grade samples (+5 g/t in the January 2007 dataset). In the complete dataset up to May 2007, the LWL69M reports significantly higher grades relative to Abilabs FA up to approximately 50 g/t, at which point Abilabs FA report higher grades for the highest value samples.

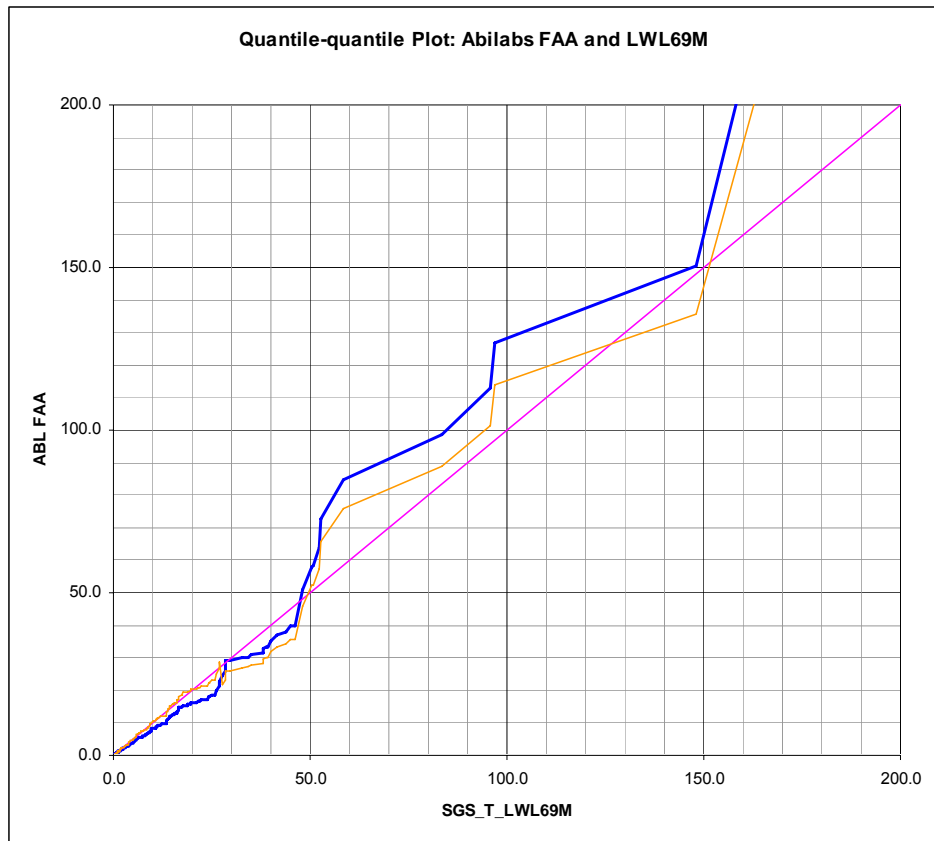
**14.3.3 ITS fire assay**

Table 14.4 details a comparison of fire assays from ITS and LWL69M assays from SGS.

Table 14.4 Statistics of ITS FA vs SGS LWL69M re-assays

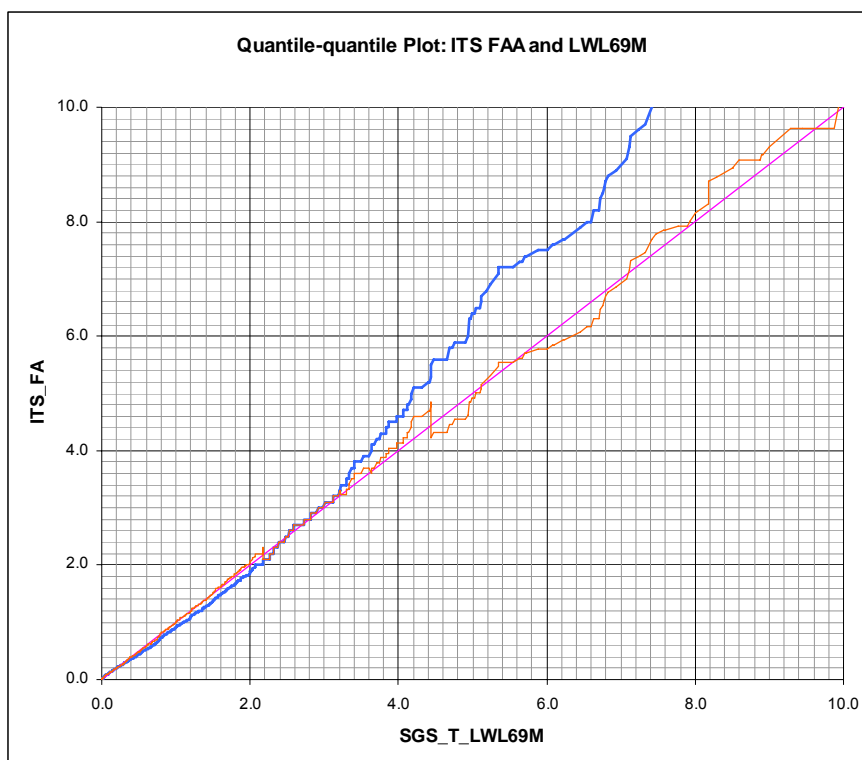
ITSFAA	LWL69M	ITSFAA	Remediated ITS FAA paired
Count	2 428	2 428	2 428
Average	1.50	1.71	1.51
Minimum	0.005	0.002	0.002
Maximum	200.00	255.00	196.35
Standard Deviation	5.90	7.47	5.76
CoV	3.92	4.36	3.81

Figure 14.7 QQ plot of Abilabs FA vs SGS LWL69M re-assays



The QQ plot in Figure 14.8 shows a complex bias pattern between ITS FA and SGS LWL69M. At low grades (>2.2 g/t) the LWL69M reports higher values than the ITS FA assays; above this value ITS FA systematically reports higher grades than the LWL69M assays.

Figure 14.8 QQ plot of ITS FA vs SGS LWL69M re-assays



**14.3.4 SGS Tarkwa BLG**

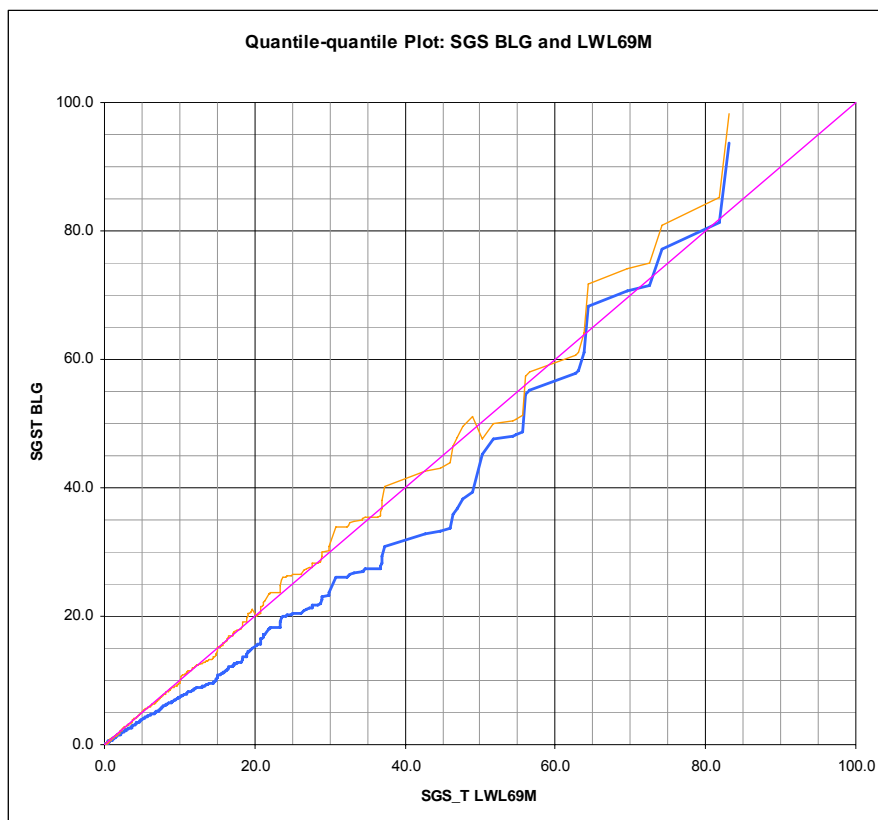
Table 14.5 shows a comparison of bulk leachable gold (BLG) assays from SGS Tarkwa and the LWL69M assays from the site laboratory.

Table 14.5 Statistics of SGS BLG assays vs SGS LWL69M re-assays

SGS Tarkwa BLG	LWL69M	SGS Tarkwa BLG	Remediated SGST BLG paired
Count	12 267	12 267	12 267
Average	1.11	0.90	1.08
Minmum	0.00	0.00	0.001
Maximum	430.00	188.80	198.24
Standard deviation	5.64	3.71	4.23
CoV	5.08	4.14	3.90

The QQ plot in Figure 14.9 supports a systematic bias between the 2006 LWL69M assays and the historical SGS BLG assays. LWL69M reports higher grades for all but the highest grade samples.

Figure 14.9 QQ plot of SGS BLG vs SGS LWL69M re-assays



**14.3.5 TransWorld BLG**

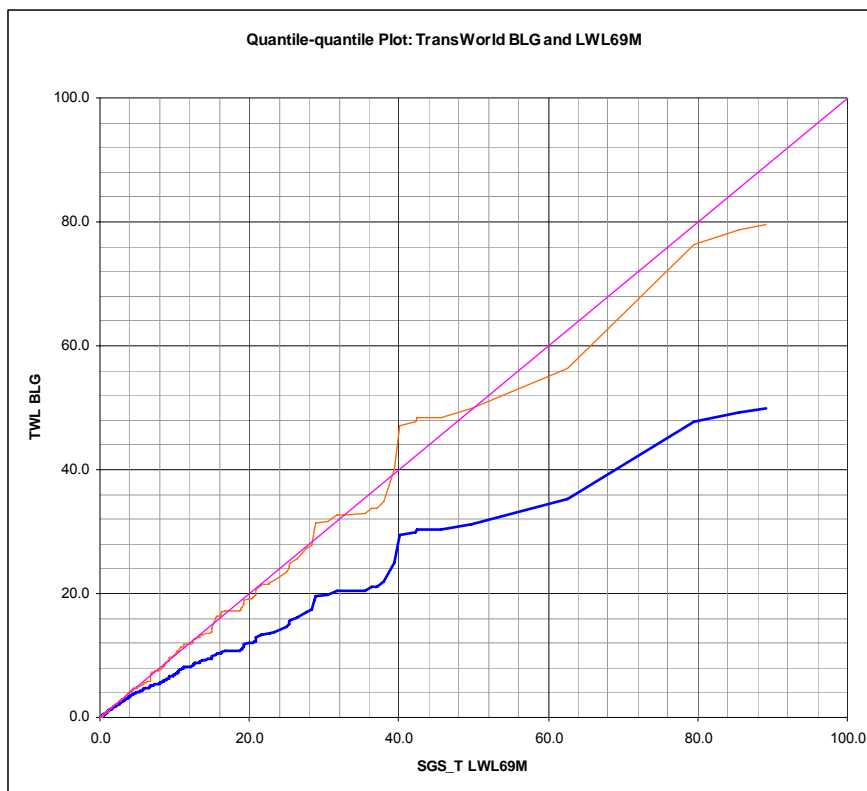
Table 14.6 shows a comparison of BLG assays from TransWorld in Ghana and the SGS LWL69M assays.

Table 14.6 Statistics of TransWorld BLG vs SGS LWL69M re-assays

TransWorld BLG	LWL69M	TWLBLG	Remediated TWL BLG paired
Count	5 436	5 436	5 436
Average	1.10	0.88	1.09
Minimum	0.00	0.00	0.001
Maximum	89.10	49.79	79.66
Standard Deviation	3.63	2.28	3.53
CoV	3.29	2.60	3.24

The QQ plot in Figure 14.10 supports a consistent and systematic bias between LWL69M assays and the TransWorld BLG assays, with BLG showing a negative bias compared to the LWL69M assays.

Figure 14.10 QQ plot of TransWorld BLG vs SGS LWL69M re-assays



### 14.3.6 TransWorld LeachWELL

Table 14.7 details a comparison of TransWorld LeachWELL (LW) assays with the SGS LWL69M assays.

Table 14.7 Statistics of TransWorld LW vs SGS LWL69M re-assays

TransWorld LW	LWL69M	TWL LeachWELL	Remediated TWL BLG paired
Count	1 404	1 404	1 404
Average	1.45	1.14	1.39
Minimum	0.00	0.00	0.001
Maximum	63.70	26.45	42.32
Standard Deviation	3.57	2.05	3.13
CoV	2.47	1.81	2.25

The QQ plot of Figure 14.11 supports a systematic bias with SGS LWL69M reporting systematically higher grades than the corresponding TransWorld LW assays.

Figure 14.11 QQ plot of TansWorld LW vs SGS LWL69M re-assays



The remediation steps described above are summarized in Table 14.8. These factors are only applied to assays that do not have a corresponding SGS LWL69M re-assay. The re-assay LWL69M data are used in preference of all other assays because these results have been generated under cover of a comprehensive QAQC program.

The current analytical database consists of samples from a variety of sources. In assembling the assay database for mineral resource estimation, the LWL69M re-assay data are preferentially retained over other data. The break-down of the assay database by data source is presented as Table 14.9. Approximately 42% of all assays have been subjected to remediation. The TWL FAA data are the Ranger Minerals data that have been accepted as valid on the basis of the twinned drillhole program.

To assess the relevance and impact of the remediation program, the statistics of the assays that have been subjected to remediation are compared in Table 14.10 with the statistics of the raw or un-remediated assays.

The average grades of the remediated groups are lower than the average grade of the total data set and are also lower than the average grades of the LWL69M and the TransWorld fire assay data. This is a direct result of the sample selection procedures: remediation has mainly affected the lower grade material. In addition, the statistics of samples with grades greater than 1g/t are compared in the Table before and after remediation. Remediation results in a comparatively small change in the proportion of samples exceeding 1 g/t.

Table 14.8 Factors applied to historical assay data

Laboratory	Grade ranges					
<b>ITS FA</b>	>0.3g/t and <2.1g/t <b>1.1</b>	>2.1g/t and <3.4g/t <b>1.0</b>	>3.4g/t and <4.0g/t <b>1.0</b>	>4.0g/t and <5.5g/t <b>0.9</b>	>5.5g/t <b>0.8</b>	
<b>Abilabs FA</b>	>0.33g/t and <0.8g/t <b>1.1</b>	>0.8g/t and <3.3g/t <b>1.15</b>	>3.3g/t and <24.0g/t <b>1.25</b>	>24.0g/t <b>0.9</b>		
<b>SGS Tarkwa BLG</b>	>0.0g/t and <0.75g/t <b>1.1</b>	>0.75g/t and <1.6g/t <b>1.15</b>	>1.6g/t and <2.9g/t <b>1.25</b>	>2.9g/t and <7.5g/t <b>1.3</b>	>7.5g/t and <15.0g/t <b>1.4</b>	>15.0g/t <b>1.3</b>
<b>TransWorld LW</b>	>0.2g/t and <1.2g/t <b>1.05</b>	>1.2g/t and <1.5g/t <b>1.1</b>	>1.5g/t and <3.5g/t <b>1.15</b>	>3.5g/t and <4.5g/t <b>1.2</b>	>4.5g/t and <7.0g/t <b>1.35</b>	>7.0g/t <b>1.6</b>
<b>TransWorld BLG</b>	>0.5g/t and <0.7g/t <b>1.05</b>	>0.7g/t and <1.3g/t <b>1.075</b>	>1.3g/t and <1.8g/t <b>1.1</b>	>1.8g/t and <3.0g/t <b>1.15</b>	>3.0g/t and <5.0g/t <b>1.2</b>	>5.0g/t <b>1.4</b>

Notes: For assay results relative to the grade categories, multiply the laboratory assay data by the factor. Application of these factors is intended to reduce the observed biases (mapped using QQ Plots) between SGS Tarkwa LWL69M assays and historical results reported by other techniques.

Table 14.9 Sources of remediated assays as a proportion of the entire database

Data Source	Number	Average	Std dev	Variance	Minimum	Maximum	CoV	% of database
Analabs Damang LW AAS	2	0.36	0.25	0.06	0.110	0.60	0.69	0.001%
TWL FAA	21 844	0.85	3.46	11.99	0.005	125.78	4.08	15.7%
Abilabs FAA [remediated]	8 767	0.57	3.81	14.52	0.005	186.14	6.71	6.3%
ITS FAA [remediated]	2 155	0.26	1.53	2.35	0.003	58.97	5.79	1.6%
TWL LW [remediated]	4 620	0.68	9.47	89.63	0.001	546.02	13.97	3.3%
TWL BLG [remediated]	16 663	0.43	5.58	31.17	0.001	537.84	12.97	12.0%
SGS Tarkwa BLG [remediated]	25 984	0.43	2.74	7.53	0.001	148.95	6.44	18.7%
SGS Burkina LWL69M	5 602	0.86	4.38	19.20	0.005	220.00	5.08	4.0%
SGS Tarkwa LWL69M	53 218	1.02	5.19	26.89	0.005	430.00	5.07	38.3%
<b>Total</b>	<b>138 855</b>	<b>0.75</b>						<b>100.0%</b>

Table 14.10 Statistics of remediated data compared with the original assay

Laboratory	Data	Count	Average grade (g/t)	Minimum (g/t)	Maximum (g/t)	Std dev	CoV	Count>1g/t	Average>1g/t (g/t)	Proportion of total assays
Abilabs Fire Assay	Unremediated Assays	8 767	0.53	0.005	206.82	4.03	7.62	570	6.31	6.5%
	Remediated Assays	8 767	0.57	0.005	186.14	3.81	6.71	649	6.11	7.4%
ITS Fire Assay	Unremediated Assays	2 155	0.28	0.003	76.59	1.96	7.01	101	3.85	4.7%
	Remediated Assays	2 155	0.26	0.003	58.97	1.53	5.81	110	3.22	5.1%
TransWorld LW	Unremediated Assays	4 620	0.49	0.001	342.26	5.93	11.99	317	5.75	6.9%
	Remediated Assays	4 620	0.68	0.001	546.02	9.47	13.89	333	8.06	7.2%
TransWorld BLG	Unremediated Assays	16 663	0.33	0.001	336.15	3.51	10.67	816	4.85	4.9%
	Remediated Assays	16 663	0.43	0.001	537.84	5.58	12.97	889	6.40	5.3%
SGS Tarkwa BLG	Unremediated Assays	25 984	0.35	0.001	141.86	2.40	6.76	1,494	4.38	5.7%
	Remediated Assays	25 984	0.43	0.001	148.95	2.75	6.43	1,726	4.88	6.6%

## 15 Adjacent properties

There is no information from adjacent properties applicable to the Essakane Gold Project for disclosure in this report.

## 16 Mineral processing and metallurgical testing

### 16.1 Overview

At an early stage of the PFS in 2005 it was determined that a conventional crushing, milling, CIL gold plant would be required. Heap leaching would not be economically viable due to poor recoveries with Fresh ore and uneconomic quantities of cement required for agglomeration of the clay-rich Saprolite.

#### 16.1.1 Testwork programs

The following testwork programs have been carried out on the major EMZ ore types since 1990:

- McClelland Laboratories (1990) on behalf of the PNUD. Tests on orpailleur (Artisanal miner) tailing largely devoted to heap leach amenability.
- Independent Metallurgical Laboratories Pty Ltd (2000/2001) on behalf of Ranger Minerals. Cyanidation, gravity concentration and heap leach testing carried out on four Oxide and two Fresh (primary) ore samples.
- SGS/Lakefield South Africa (2004) on behalf of Essakane. Gravity concentration and cyanidation/CIL tests on an Oxide (EMZ-1) and a Sulfide (EMZ-2) sample.
- Gold Fields Ghana Damang Mine (2005) on behalf of Essakane. Cyanidation tests on four EMZ-4 Oxide samples.
- Gold Fields Ghana Tarkwa Mine (2005) on behalf of Essakane. Cyanidation and column leach tests on an EMZ-3 sample.
- SGS/Lakefield Johannesburg (2005) on behalf of Essakane. Gravity concentration and cyanidation/CIL tests on two Fresh ore samples (EMZ-8 Argillite, EMZ-9 Arenite) and one Oxide ore sample (EMZ 10), along with comminution testing of the samples.
- Kappes Cassidy and Associates (2005/6) on behalf of Essakane. Heap leach amenability testing, and gravity concentration, cyanidation and CIL testing on two Fresh ore composites (EMZ-12 Arenite and EMZ-13 Argillite) and two Oxide ore composites (EMZ-16 Arenite Saprolite and EMZ-18 Arenite Saprolite).
- McClelland Laboratories (2006) on behalf of Essakane. Gravity concentration, cyanidation and CIL testing on two Fresh ore composites (EMZ-14 Arenite and EMZ-15 Argillite) and two Oxide ore composites (EMZ-17 Arenite Saprolite and EMZ-19 Arenite Saprolite).
- Phillips Enterprises, LLC (2006) for McClelland Laboratories Inc. Comminution testing of samples provided by McClelland Laboratories.
- SGS-Lakefield (Canada) 2006/7 for Essakane. Comminution testing and mill circuit modelling on samples provided by McClelland Laboratories. (Appendix 8)
- McClelland Laboratories (2007) on behalf of Essakane. Gravity concentration, intensive cyanidation and standard cyanidation testing on the following variability samples: two Fresh Argillite composites (ERC 1630D and ERC 1626D), an Oxide Arenite composite (ERC 1629D), two Fresh Arenite samples (both EDD0084), a Transition/Fresh Arenite

composite (ERC1612D), an Oxide/Transition Arenite/Argillite composite (ERC 1648D), and a Fresh Argillite composite (ERC1648D).

- SGS Johannesburg (2007) on behalf of Essakane. Cyanidation and CIL tests on whole ore samples to investigate the effects of percent solids, cyanide concentration, residence time and dissolved oxygen, along with oxygen uptake testing and carbon adsorption kinetic testing for design purposes. In addition, tails samples will be prepared for further geotechnical and geochemical evaluation by Knight Piésold and Patterson & Cooke. Samples being used represent the various mining phases as follows: (i) WSU Upper Saprolite Oxide composite (various EDD intervals); (ii) An Arenite blend of 57% WSU and 43% Lower Saprolite WSL; (iii) An Arenite blend of 80% Fresh ore and 20% WSL; (iv) Fresh Arenite; (v) Fresh Argillite.

In addition, test work on tails samples were carried out by Golder & Associates, Knight Piésold, and Patterson & Cooke to characterize the geochemical and geophysical properties of CIL tailings.

### 16.1.2 Ore types and samples

Metallurgical test samples were selected by Essakane during 2006 from a large number of RC and DD drillholes which intersected oxide, transitional and Fresh oxidation types and the main gold-bearing lithologies within US\$ 500/oz and US\$ 650/oz surface mine shells. Bulk surface samples of the saprolite ore types were not taken. Samples for the DFS were prepared by Essakane and shipped to the various laboratories in sealed plastic drums. The sampling programs have been extensive and due care was taken in selecting and compositing representative samples based on:

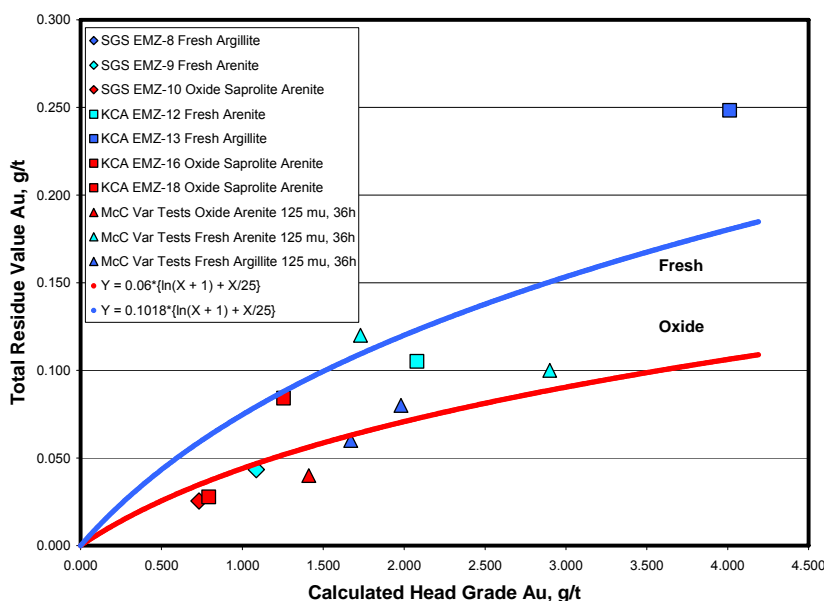
- Weathering type
- Oxidation type
- Gold grade
- Lithology
- Arsenopyrite contents
- Position within the expected limits of the pit shell.

### 16.1.3 Testwork results

Comminution test results on hard Fresh ore dictated the design of the grinding circuit. They were also used to establish an economic optimum grind  $P_{80}$  of 125 microns and a CIL leach residence time of 36 hours for Fresh ore. The gold in milled ore concentrates readily by gravity. Removing a gravity concentrate was also found to enhance the cyanide leach kinetics of the gravity tails during CIL, as did the presence of activated carbon. This endorsed the use of a CIL circuit as opposed to a leach-CIP circuit.

Metallurgical recoveries, as characterized by the final residue values for given ore types and head grades, are summarized in Figure 16.1.

Figure 16.1 Residues vs head grades for relevant testwork programs



The data in Figure 16.1 were characterized by a global head grade model to generate the recovery curves. These curves allow for a 0.01 g/t tailings solution loss and a 0.01 g/t tailing contingency loss at a head grade of 2.0 g/t and therefore should represent practical extractions achieved in the plant. Translated into gold recovery terms these model relationships yield:

**Oxide Ore (Saprolite): Gold Recovery % = 99.76 – 6\*{ln[Au + 1]}/[Au]**

**Fresh Arenite, Argillite: Gold Recovery % = 99.59 – 10.18\*{ln[Au + 1]}/[Au]**

The recoveries calculated from these relationships will vary with the head grade. At a 2.0 g/t head grade, for example, the Oxide ore recovery projected from the Oxide model is 96.5%, while that for Fresh ore is 94.0%. Saprock for the DFS was considered a 50:50 blend of Oxide and Fresh ore. These recovery models were used for ore Mineral Reserve estimation and for all financial evaluations. The LOM gold recovery for the Project’s Mineral Reserves at a 1.78 g/t average grade is estimated to be 94.6%.

**16.1.4 Design criteria**

The design criteria for the gold plant, in part determined from the testwork, can be summarised as follows:

- Plant throughput 5.4 million tonnes per year
- Project design life 8.6 years
- Plant operating schedule 365 days per year
- Plant availability 95%
- Plant feed rate 650 tonnes per hour
- Plant ROM feed size 90% minus 800 mm
- SAG Mill feed size 90% minus 200 mm
- Leach feed size 80% minus 125 microns
- Leach time for oxide ore 26 hours with a feed density of 40% solids

- Leach time for fresh ore 36 hours with a feed density of 50% solids

The process flow diagram is shown in Figure 16.2. The layout of the process plant is shown in Figure 16.3.

Figure 16.2 Process flow diagram

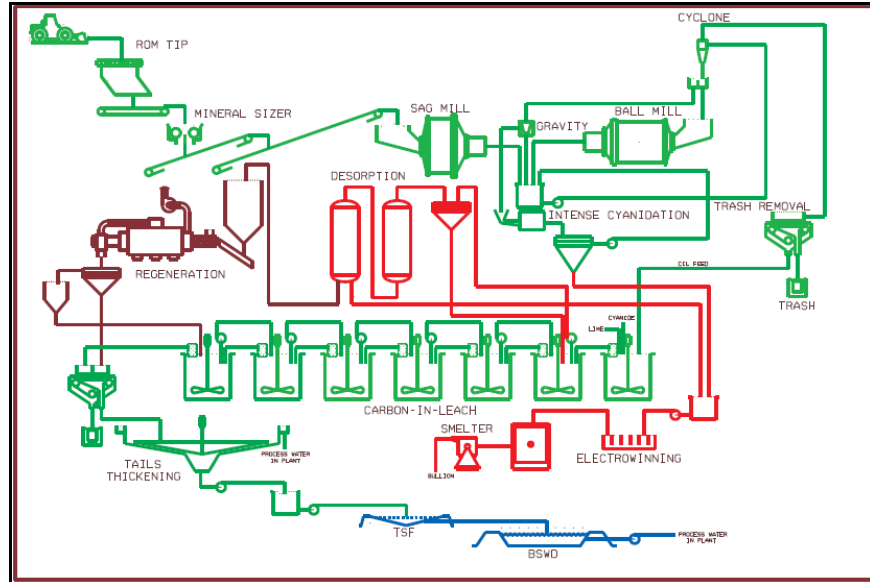
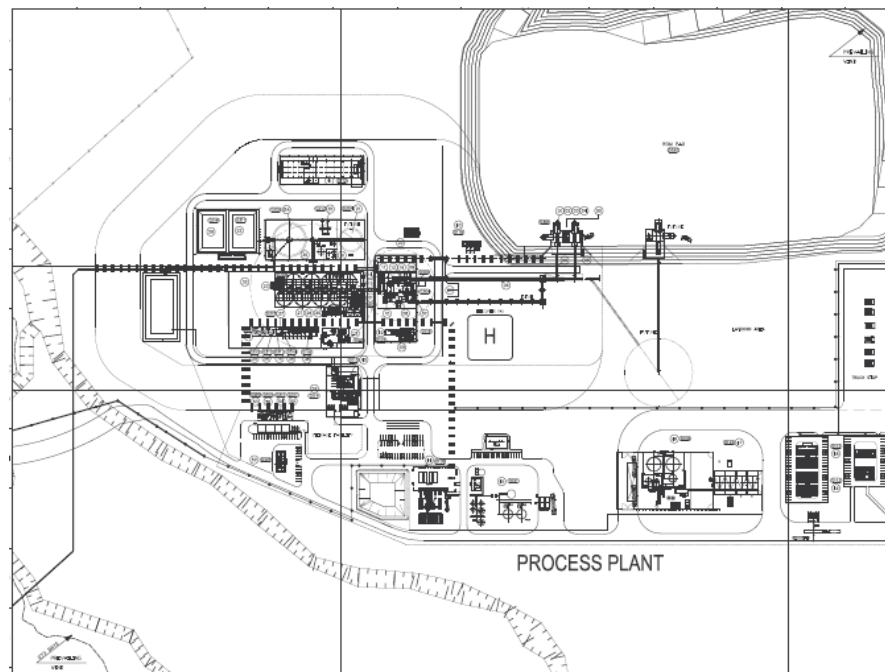


Figure 16.3 Process plant layout



## 16.2 Process flow-sheet development

### 16.2.1 Design philosophy

The GRD Minproc process design accommodates the sequential processing of ores with widely differing physical characteristics whilst keeping the plant as simple as possible. To achieve this it was considered wise to provide sufficient milling capacity from the outset and make provision for the installation of primary jaw crushing and pebble crushing at a later stage should it prove necessary. Similarly, provision was made for the installation of a pre-leach thickener at a later stage to accommodate the differing leach characteristics of the ores.

By adopting this philosophy towards the development of the flow-sheet it was possible to design a plant that will achieve a dependable, safe start-up with the saprolite ore whilst providing the flexibility to do whatever is necessary to process the fresh ore when the time comes.

### 16.2.2 Crushing

The plant will use two mineral sizers in parallel. Each mineral sizer will be capable of the full ROM feed rate and will feed the SAG mill feed conveyor via a sacrificial conveyor equipped with an over belt magnet for trash removal. Feed to the mineral sizers will be controlled by apron feeders equipped with variable speed drives.

The plant layout provides for a primary jaw crusher and a crushed ore stockpile should it be required in the future.

### 16.2.3 Milling

The milling plant will be a standard SAB configuration with a 30 ft diameter by 14 ft EGL primary SAG mill rated at 7 MW feeding a 20 ft diameter by 33.5 ft EGL Ball Mill also rated at 7 MW. The SAG mill will be equipped with a variable speed drive whilst the ball mill will run at a fixed speed. The plant layout provides for the installation of a pebble crusher should it become necessary in the future.

### 16.2.4 Gravity concentration

There is a high percentage of free gold in the ore and this will be removed by gravity concentration and intensive cyanidation. The removal of the free gold prior to leach reduces the possibility of gold lock-up and improves the leach characteristics of the ore. Centrifugal gravity concentrators will be used.

### 16.2.5 Carbon in leach (CIL)

Feed to the CIL will not be thickened for the first three years of operation, but provision has been made for the installation of a pre-leach thickener for the Fresh ore.

The residence time in the CIL circuit is 26 hours for the saprolite ore and 36 hours for the fresh ore. The CIL circuit will have eight tanks, one leach tank and seven CIL tanks, each with a capacity of 4 000 m<sup>3</sup>. The required residence times are achieved by the different feed densities for the different types of ore.

### 16.2.6 Elution, electro-winning and regeneration

Elution will use a split AARL process with conventional electro-winning cells. The elution system will be heated by diesel fired heaters and regeneration of the carbon will be in a diesel heated re-generation kiln.

### 16.2.7 Tailing thickening and pumping

The tailing thickener has been sized on the saprolitic material as it represents the worst case. Tailings will be thickened to 59% solids for pumping to the TSF

through a dual HDPE tailing pipeline installed on sleepers above ground within a bunded area to contain any loss of slurry.

### 16.2.8 Gold room and smelt

The monthly gold production will be 848 kg and it is anticipated that a gold smelt will be undertaken every second day. Gold will be smelted in a 115 litre diesel fired smelt furnace. A six tray, electrically heated oven will treat the calcine on a daily basis.

Gravity gold from the centrifugal concentrators will be leached in an intense cyanidation process and the resulting pregnant solution will report to the electro-winning cells thus joining the CIL production.

### 16.2.9 Reagents

The reagents that will be used within the process plant are:

- Flocculants for the tailings thickener
- Lime for pH control
- Cyanide for CIL and, if required, for elution
- Caustic soda for elution
- Hydrochloric acid for washing the carbon
- Oxygen for CIL sparging

Aside from the oxygen, reagents will be delivered to site by road transport and will be transferred to the mixing tanks outside the plant. Once prepared, the reagents will be pumped to the day tanks within the plant fence.

## 16.3 Process plant design criteria

The process plant design criteria forms the basis upon which the plant is designed. The process plant design criteria encompass:

- Process description
- Process flow sheets
- Design criteria

### 16.3.1 Material balances

The mass and water balances for the plant have been developed by GRD Minproc. The total water demand of the plant is 460 m<sup>3</sup>/hr and this is made up from 96 m<sup>3</sup>/hr of clean water from the boreholes and 364 m<sup>3</sup>/hr of dirty water which will come from the off channel storage facility, the tailing storage facility and the surface mine sump.

### 16.3.2 Process equipment selection

During the DFS several vendor specific types of equipment were identified:

- Feed preparation: MMD mineral sizers
- Milling: FLSmith SAG and Ball mills
- Classification: Multotec cyclone clusters
- Gravity concentration: Falcon concentrators and a Gekko intense cyanidation reactor
- CIL circuit: Kemix agitators and carbon screens

- Thickening: Outokumpu tailing thickener
- Oxygen: Linde pressure-swing-adsorption unit

### 16.3.3 Process control philosophy

The plant is designed to incorporate a moderate level of automation. A SCADA system will control the process using PLCs, instrumentation and control valves. Condition monitoring of equipment for maintenance purposes will be recorded on SCADA.

Some vendor supplied equipment such as the mill lubrication systems and the thickener will be supplied with PLC control that will report to the SCADA.

Motor control and protection will be by Simocode and the three modes of operation will be initiated through the SCADA. All safety circuits will be hard wired and cannot be defeated by SCADA.

## 17 Mineral Resource and Mineral Reserve estimates

Mineral Resource and Mineral Reserve estimates are currently reported for the planned mining operations at the Essakane Main Zone (Table 17.1 and Table 17.2)

Table 17.1 Constrained May 2007 Mineral Resources reported at 0.5 g/t Au cut-off grade.

Mineral Resource Constrained by US\$ 650/oz pit shell				
Category	Tonnage (Mt)	Grade (g/t Au)		Contained gold (Moz)
Indicated	73.4	1.62		3.82
<b>Total Indicated</b>	<b>73.4</b>	<b>1.62</b>		<b>3.82</b>
Inferred	16.1	1.66		0.86
<b>Total Inferred</b>	<b>16.1</b>	<b>1.66</b>		<b>0.86</b>

Table 17.2 May 2007 Mineral Reserve estimate

Category	Reporting cut-off (g/t Au)	Tonnage (Mt)	Grade (g/t Au)	Contained gold (Moz)
	Oxide 0.52	11.57	1.47	547
Probable	Transition 0.58	10.07	1.71	555
	Fresh 0.64	24.77	1.94	1 547
<b>Total Probable</b>		<b>46.41</b>	<b>1.78</b>	<b>2 649</b>
Proven		-	-	-
<b>Total Proven</b>		<b>-</b>	<b>-</b>	<b>-</b>
<b>Total</b>		<b>46.41</b>	<b>1.78</b>	<b>2 649</b>

Note: Mineral Resources are inclusive of Mineral Reserves. Tonnes have been rounded and this may cause minor discrepancies.

### 17.1 Disclosure

Mineral Resources reported in Section 17 were prepared by Dr M. Harley, Mineral Resources Manager – International Projects, a full time employee of Gold Fields International Services Limited, and reviewed by Mr I.M. Glacken, Group General Manager – Resources for Snowden.

Mineral Reserves reported in Section 17 were based on surface mine optimization and mine planning studies undertaken by Mr O. Varaud, Chief Mining Engineer – Essakane Feasibility Study, a full time employee of Gold Fields Burkina Faso Sarl, and reviewed by Mr J F. Hawxby, Senior Project Manager for GRD Minproc (Pty) Ltd.

All persons named above are Qualified Persons as defined in NI 43-101. Snowden, Gold Fields International Services Limited, and GRD Minproc (Pty) Ltd., are independent of Orezone Resources Inc. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

### 17.1.1 Known issues that materially affect mineral resources and mineral reserves

Snowden is unaware of any issues that materially affect the Mineral Resources and Mineral Reserves in a detrimental sense. These conclusions are based on the following:

- Essakane has approved “*permis de recherche*” operating licences and Essakane rehabilitates drill sites and drill access roads on an ongoing basis.
- Essakane has represented that there are no outstanding legal issues; no legal actions or injunctions pending against the Project
- Essakane has represented that the mineral and surface rights have secure title
- There are no known marketing, political, or taxation issues
- Essakane has represented that the Project has strong local community and national support
- There are no known infrastructure issues.

## 17.2 Assumptions, methods and parameters – Mineral Resource estimates

The basis of the Mineral Resource estimate for the EMZ deposit is discussed in this section.

The estimates were prepared with the following steps:

- Data validation: this was undertaken by Mr Samuel Tianhoun and Mrs Michelle Chard of Essakane and reviewed by Snowden.
- Data preparation: this and subsequent steps are discussed below.
- Geological interpretation and modelling was undertaken by Mr M Briggs, Mr P Davies and Mr I Kolga of Essakane. Wireframes describing the major lithological contacts and intrusive bodies (dykes and sills) were developed.
- Block models describing the geology of the EMZ were developed by Dr M Harley of Gold Fields International Services Limited, using the constraints of the geological wireframes.
- Drillhole compositing was undertaken at a 3 m length, following testwork on variography using 1 m and 3 m sample lengths. Intervals without sample values (missing intervals) were not included within the compositing process, but were retained as missing values without modification of the unsampled lengths.
- Statistical analysis of sample sets was undertaken to establish a suitable domaining strategy. Mineralisation is controlled by a combination of lithological and structural controls and post-mineralisation modification of sample grades has taken place within the upper parts of the deposit, where a complex weathering pattern exists. A combination of lithological, structural and weathering characteristics were used to select a suite of ten separate domains within which the deposit has been segregated.
- Analysis of top cuts (caps) was undertaken by examining the cumulative mean grade and cumulative standard deviation of the data sets; in all cases attempts were made to have the capped data show significant reduction in the coefficient of variation.

- Variography of raw composite grades within each domain has taken place. Typically data show high skewness and high coefficients of variation (uncut data may have CoV values in excess of 5); this behaviour required variography of transformed sample data. Pairwise relative variography was undertaken but was not used for estimation. Variography of Gaussian-transforms (normal scores) of the gold grades was undertaken and variograms developed using this method were back-transformed to raw sample space using a hermite-polynomial transform based method. These back-transformed variograms were used in the subsequent grade modelling, derivation of kriging plan and boundary conditions.
- Grade interpolation into 25 mE x 50 mN x 6 mRL panel blocks was undertaken using Ordinary Kriging. The search parameters were designed to minimise conditional bias and to develop the best estimate of the local mean grade within the deposit. The Uniform Conditioning (UC) non-linear recoverable resource estimation technique was then applied to the estimated panel grades. Each panel is subdivided into Selective Mining Units (SMU) measuring 2.5 mE x 5 mN x 3 mRL. UC provides (i) an estimate of the proportion of each panel that exceeds a cut-off gold grade, and (ii) the average grade of that proportion above each cut-off grade.
- The Panel grade estimates were validated in two ways. The declustered data statistics for the composites in each domain were compared with the volume-weighted panel estimates. These results were compared and in all cases the observed differences were less than 5%. A series of swath plots comparing the average sample grades with easting, northing and RL coordinates were also developed and the block grades were placed on these charts. While there is inevitable smoothing of the block grades, the plots show that the major trends present within the composite data are reproduced in the estimated block grades.
- The Panel grade estimates were defined as the basis for resource classification. The kriging efficiency was employed as a first-order tool and all blocks with kriging efficiency greater than 0.25 were flagged. Wireframes were then developed on serial cross-sections around the higher-efficiency block estimates. During this process drill samples were posted on the sections and wireframes were developed taking cognisance of the local sample density as well as the local estimation quality. Isolated blocks outside these envelopes that have kriging efficiencies greater than 0.25 were disregarded. All blocks within the envelope were then classified as Indicated Mineral Resources. No material was considered to qualify as Measured Mineral Resources in recognition of the high skewness of the original sample data and because remediated assays participate in the estimation. Mineral resource classification has followed the JORC guidelines.
- The classification categories of Probable and Proved Ore Reserve under the JORC Code are equivalent to the CIM categories of Probable and Proven Mineral Reserve (CIM, 2005).

### 17.2.1 Drillhole locations

Details of the drillhole survey procedures are presented in Section 10.

A local coordinate grid was developed by BHP during its exploration of the EMZ. The axes of this grid are orientated oblique to the National Grid axes such that the EMZ strikes parallel to the northing axis in the local grid. The use of the local grid was followed by subsequent operators. Drillhole locations plotted in the National and Local Grid coordinate systems are shown in Figure 17.1.

### 17.2.2 Database

A detailed SQL Server database containing all information pertaining to drill holes is maintained by Essakane on site. Back-up copies are stored off-site. Access to this database is restricted to authorized site personnel.

### 17.2.3 Geological interpretation and modeling

The EMZ is a quartz – carbonate stockwork vein deposit hosted by a folded turbidite succession of Birimian arenite and argillite. Gold occurs as free particles within the veins and also intergrown with arsenopyrite either on vein margins or in the host rocks. Disseminated arsenopyrite and occluded gold contents decrease away from the veins. The same relationship is seen away from major lithological contacts, where the frequency of veining is also higher. In weathered saprolite the gold particles occur without sulphides. The gold is free – milling in all associations.

The main structural features of the EMZ deposit are:

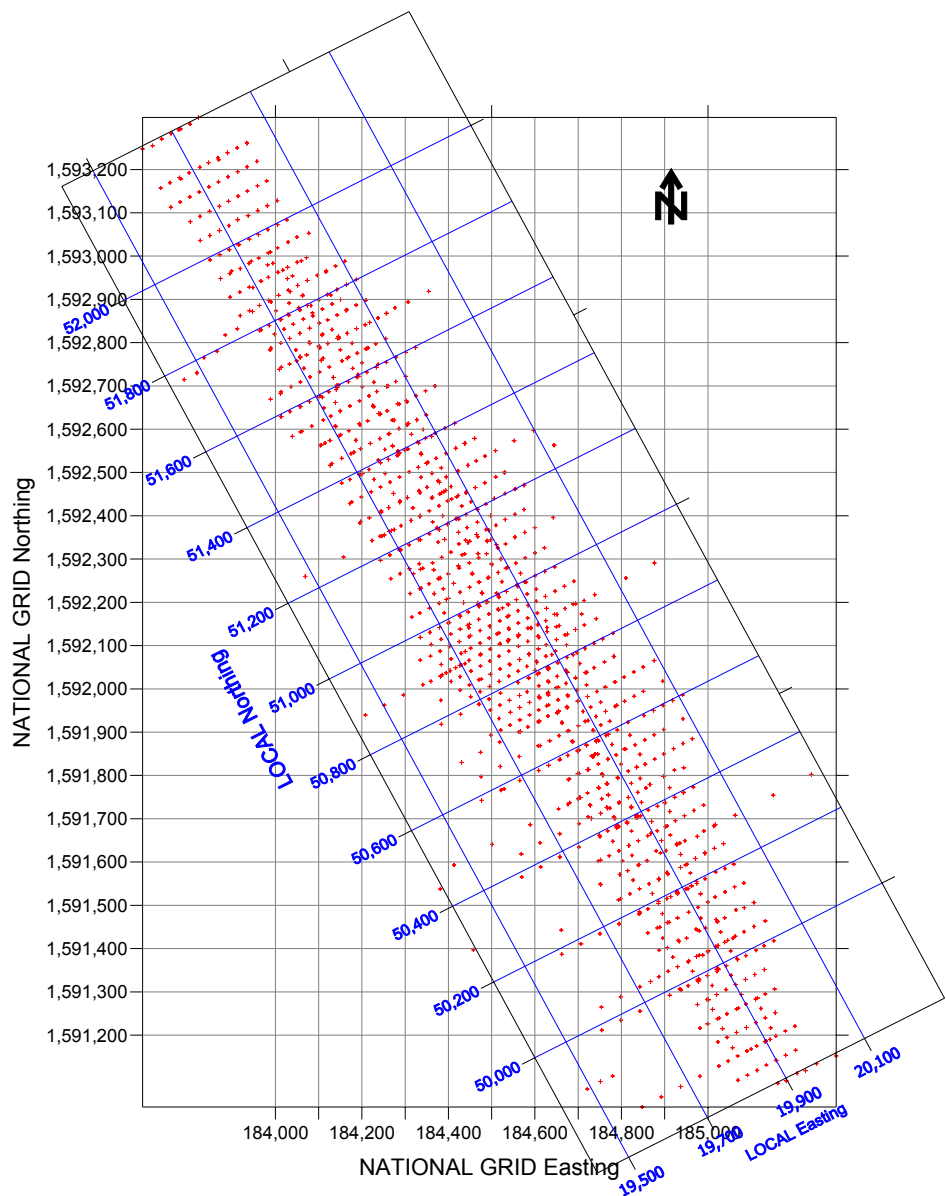
- Lithologies are folded into a west-verging anticline with a vertical west limb.
- There is a marked competency contrast between arenite and argillite. Flexural slip along bedding surfaces is a pervasive deformation style.
- Early bedding-parallel, grey laminated quartz veins are related to flexural slip.
- Late, steep extensional quartz veins with visible gold occur in the fold hinge and east limb domains.
- Axial-planar pressure solution seams are developed in the fold hinge.

The EMZ has been modelled for the DFS with a strike length of 2 500 m. The EMZ occurs at the northern end of the EMZ anticline. Open wireframes for the top and bottom surfaces of the main arenite were developed on site working in Datamine. Open wireframes describing the footwall surfaces of the FW argillite and FW arenite were also developed. Other relevant surface models include (i) the topographic surface, (ii) base of the upper saprolite, and (iii) top of Fresh weathering domain.

Closed form wireframes describing basic and intermediate dykes intrusive into the EMZ were also modeled.

The HW argillite above the main arenite represents the volume located between the HW surface of the main arenite and the topographic surface. Block models of the main arenite unit (Rock Code 200), the FW argillite (Rock Code 300) and the FW arenite (Rock Code 400) were generated for mineral resource estimation.

Figure 17.1 Location of EMZ drillholes on the National and local grids



Two models were generated for each of the principal rock units: one at the scale of the proposed SMU (2.5 mE x 5 mN x 3 mRL) and one at the scale of the large panels (25 mE x 50 mE x 6 mRL). Features such as dykes in the large panel model have been modeled as sub-cells at the resolution of the SMUs. Details relevant to the selection of the large panel and SMU dimensions are presented in the subsequent text.

The SMU model defines the block model resolution. SMU blocks do not contain any subcells and all geological boundaries are described as stepped surfaces at the resolution of the SMU cell. The panel model has been regularised from the SMU model so the individual volume estimates of each lithology in each panel are expressed by the contained SMU volume in that panel.

An additional waste model, representing all materials not described by the main arenite, FW argillite and FW arenite block models, was created at the panel scale. This waste model contains the HW argillite (Rock Code 100) and unmineralised

dyke material (Rock Code 600) and has no grade estimates. Relative density information is the only physical attribute that has been modeled in the waste block model, taking cognizance of the rocktype and weathering characteristics. Details of the modeling of relative density are presented in the subsequent text.

#### 17.2.4 Data analysis

There are three major lithological divisions within the EMZ (main arenite, FW argillite, FW arenite), three weathering zones (Saprolite, saprock, Fresh) and two structural domains (east limb and west limb with a sub-domain defined by the fold axial zone). Drill hole sample data were segregated according to these divisions and the statistics of these subsets examined. Typically the data display extreme skewness and high coefficients of variation ( $CoV = \text{standard deviation}/\text{mean}$ ). Where considered reasonable, or where too few data were present in a single set, data subsets were combined. An example where too few data exist within a subset is that of the Footwall Arenite. Because this unit is confined to depth, small volumes of this unit are developed above the base of the weathered zone; too few data exist within the Footwall Arenite-Saprock and Footwall Arenite-Saprolite datasets to allow these units to be estimated or evaluated individually. Whilst the volumes of material present may be classified as saprock, the estimation of this material includes the adjacent samples from the Fresh rock as well. This process resulted in the definition of ten distinct domains defined in terms of structure, lithology and weathering type, within which inference of statistical properties, variography and ultimately grade estimation was constrained.

Histograms of capped 3 m composites for domains within the East Limb of the EMZ are presented in Figure 17.2. Corresponding data from the West Limb of the EMZ are presented in Figure 17.2.

#### 17.2.5 Declustering

As part of the routine data analysis, declustering of drillhole data was undertaken using a grid-defined, cell-based declustering process. In addition, a sample-centred, cell-based declustering process was also used to confirm the results of the grid-based approach. In the majority of cases consistent behaviour was observed with the EMZ drillhole data displaying a significant clustering effect, with the tendency for the declustered mean grade to be less than the naïve average grade. This implies a tendency to oversample within the higher grade zones of the deposit, relative to the areas identified as lower grade. Estimates of the declustered average and declustered sample variances were prepared for comparison with the kriged estimates.

#### 17.2.6 Compositing of assay intervals

Assay intervals within the drillholes were composited to 1 m and 3 m downhole intervals using a downhole composite process in Datamine. The compositing mode used is one that does not produce short composites at the end of sampling runs. Instead, this process seeks to produce composites that are as close to the defined interval as possible, but may vary in accordance with the length of the composite run. The maximum composite length is constrained to be 1.5 times the defined interval.

Variography tests were completed on one of the main domains using 3m and 1m composites and the statistics of these datasets were also compared. The longer composite shows a significant reduction in coefficient of variation (CoV) relative to the shorter composite length, which was the main reason why the longer composite length was selected.

Figure 17.2 EMZ East Limb - Histograms & data statistics for capped gold grades (3m composites)

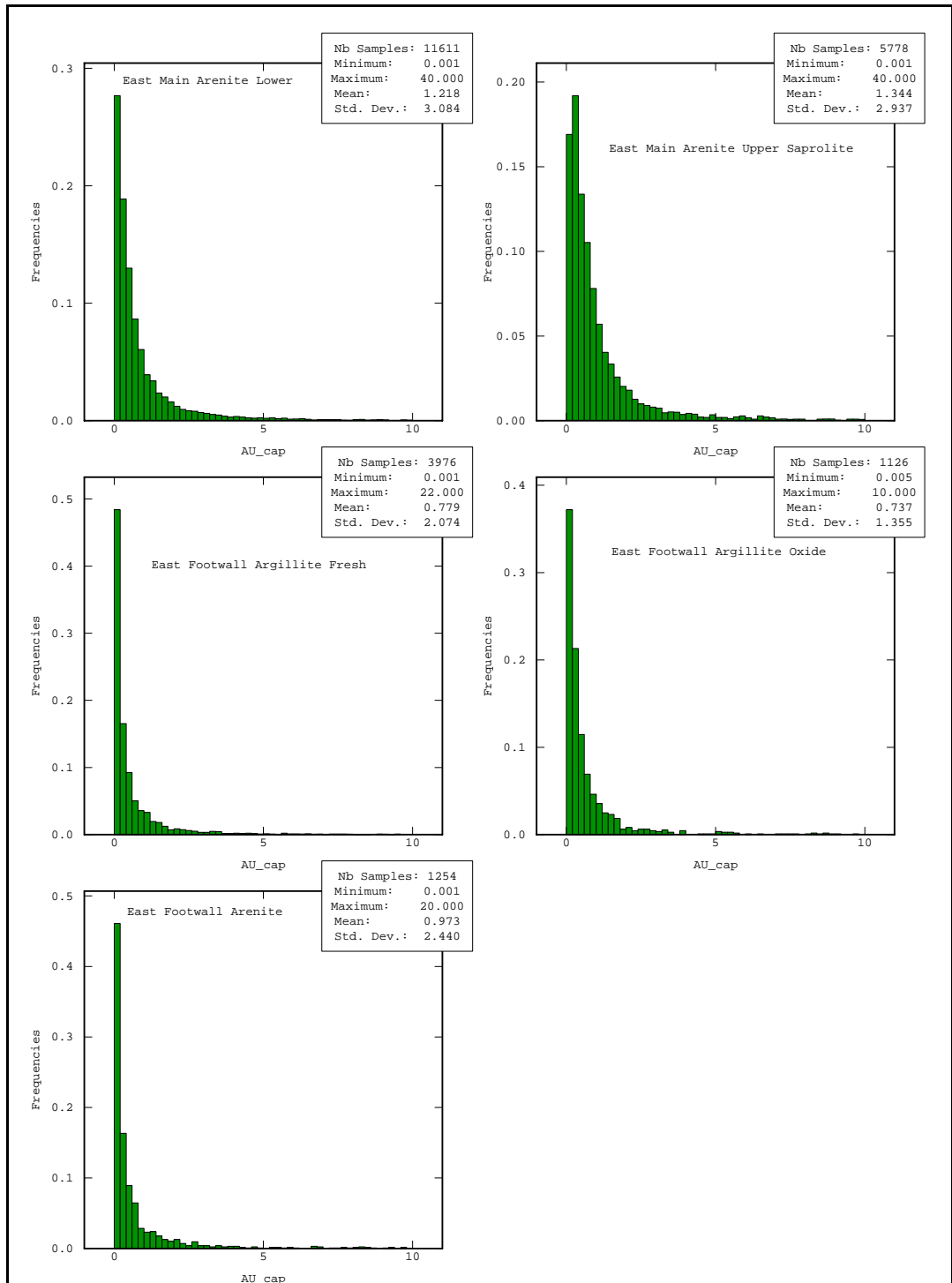
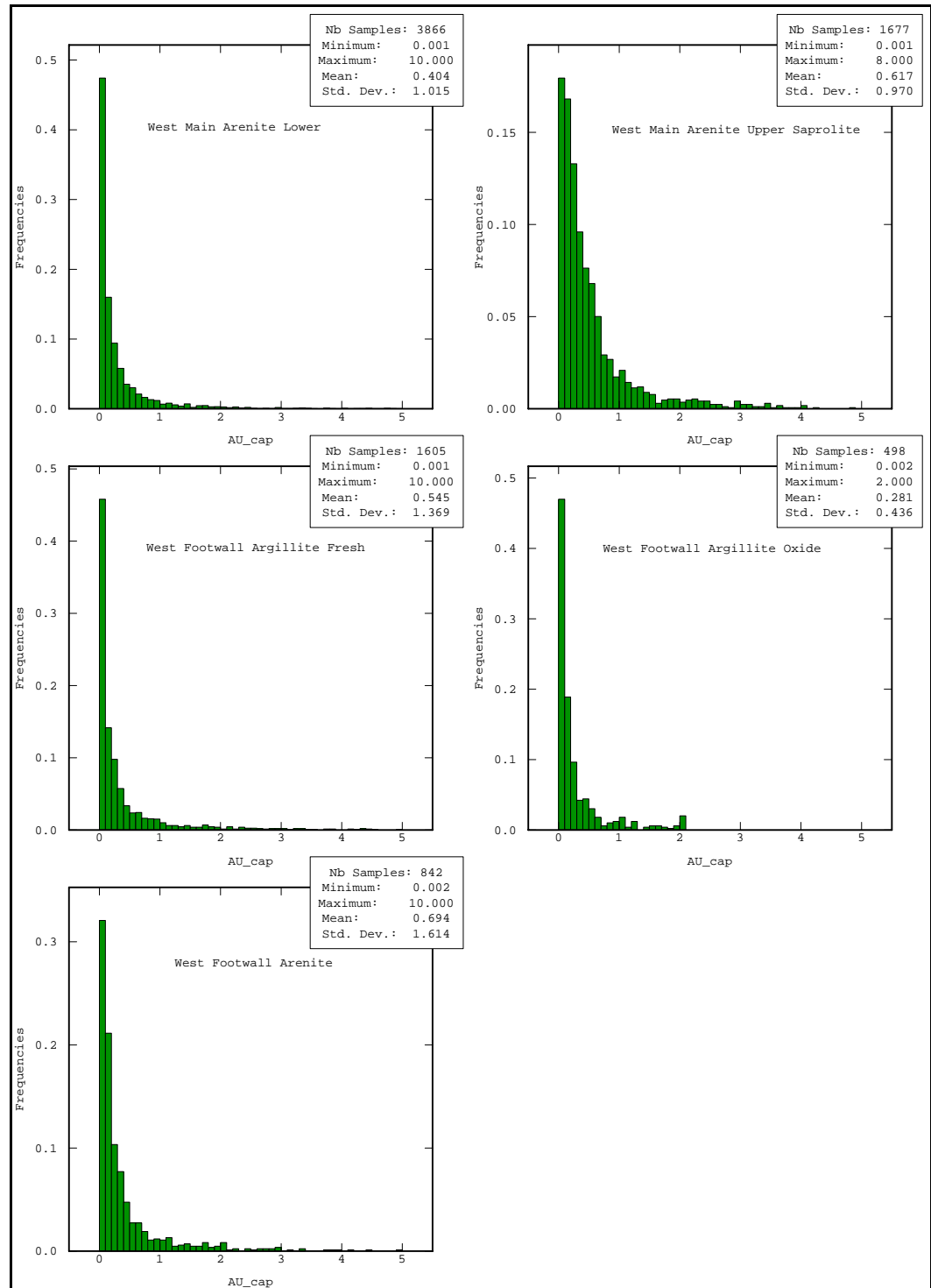


Figure 17.3 EMZ West Limb - Histograms & data statistics for capped gold grades (3m composites)



### 17.2.7 Top cuts (data caps)

Data capping levels were examined using the cumulative mean and cumulative standard deviation of the domain specific data sets. Typically the data sets display extreme skewness: the histograms show a high grade tail that is very long, very thin and is incomplete with data values separated by uninformed grade ranges. Data capping, or systematic modification of the highest grade samples, is frequently practiced to restrict or limit the influence of these highest data values. In most long-tailed distributions it is common for the variance of the data set to be disproportionately influenced by the few highest grade samples.

The capping approach that has been followed in the January and May 2007 models was to examine the cumulative mean value as well as the cumulative standard deviation of the data set as a function of the sample grade. Capping levels for each domain have been selected using the following criteria:

- Where the introduction of higher-grade samples creates a significant step increase in either the cumulative mean or the cumulative standard deviation.
- Where large gaps exist within the range of sample values (i.e., there is a break in the histogram leading to plateaux within the cumulative histogram).

The statistics of the domain specific data before and after capping are presented in Table 17.3.

### 17.2.8 Variogram analysis

Variography of raw composite grades failed to generate any interpretable variogram structures, mainly because the data (even after capping) retain high degrees of skewness. Capping and cutting of the highest grade samples failed to significantly improve the quality of experimental variograms. Variograms of Gaussian-transforms (normal scores) of the declustered gold grades were developed and this approach yielded well-structured experimental variograms that were modelled. These modeled variograms were then back-transformed to raw sample space using a hermite-polynomial based procedure (the mathematical background of this approach is provided in Rivoirard, 1994). Pairwise relative variograms and variograms of the logarithms of gold grades were also developed to check the 'normal score' variograms.

The pairwise relative variograms and log variograms show similar shapes and similar ranges to the normal score variograms. It is fairly common practice to directly model pairwise relative variograms and use these within estimation (e.g., Srivastava and Parker, 1989) despite this variogram representing a non-linear transformation of the data, for which there is no analytical back-transform method. In this case the approach that has been followed has some distinct advantages:

- The sills of the back-transformed variograms honour the declustered variance of the raw data;
- The method is analytically complete (the relationship between the variables and the transform can be fully described);
- The resultant variograms describe the untransformed data, not a non-linear transform of the data.

Table 17.3 Statistics of uncapped and capped gold grades by domain

Domain	Variable	Count	Mean	Std. dev.	CoV	Mean reduction	Variance reduction	Cap value	Data affected
East Main Arenite	Au	11,611	1.27	4.23	3.33			40	
Lower	Au-capped	11,611	1.22	3.08	2.53	4.3%	46.9%	99.82%	20
East main Arenite	Au	5,778	1.39	3.95	2.83			40	
Upper Saprolite	Au-capped	5,778	1.34	2.94	2.19	3.5%	44.6%	99.81%	10
East FW Argillite	Au	3,976	0.83	3.33	3.99			22	
Fresh	Au-capped	3,976	0.78	2.07	2.66	6.7%	61.2%	99.70%	11
East FW Argillite	Au	1,126	0.77	1.67	2.17			10	
Oxides	Au-capped	1,126	0.74	1.35	1.84	4.1%	34.3%	99.56%	4
East FW Arenite	Au	1,254	1.06	3.44	3.26			20	
Fresh	Au-capped	1,254	0.97	2.44	2.51	7.9%	49.8%	99.52%	6
West MainArenite	Au	3,866	0.44	1.47	3.37			10	
Lower	Au-capped	3,866	0.40	1.01	2.51	7.2%	52.3%	99.64%	13
West Main Arenite	Au	1,677	0.66	1.46	2.22			8	
Upper Saprolite	Au-capped	1,677	0.62	0.97	1.57	6.5%	56.0%	99.46%	9
West FW Argillite	Au	1,605	0.62	2.07	3.35			10	
Fresh	Au-capped	1,605	0.55	1.37	2.51	11.9%	56.3%	99.07%	14
West FW Argillite	Au	498	0.33	0.85	2.57			2	
Oxides	Au-capped	498	0.28	0.44	1.55	15.6%	74.0%	97.99%	10
West FW Arenite	Au	842	0.77	2.19	2.86			10	
Fresh	Au-capped	842	0.69	1.61	2.33	9.6%	45.8%	98.57%	12

Directional variograms were developed taking cognizance of the local geology. Within the East Limb (i) the longest ranges are typically subparallel to the strike of the local geology, and (ii) the shortest ranges are typically perpendicular to the stratigraphic layering, implying a strong influence from the layer parallel veins in the development of variographic structures. Relative nugget effects were confirmed using downhole variograms. Variogram parameters for the back-transformed variograms are detailed in Table 17.5.

### 17.2.9 Block model set up

The principal block model used for the EMZ mineral resource estimation consists of large panels with dimensions of 25 m (easting) x 50 m (northing) x 6 m (RL). These panels are truncated where necessary against the major geological boundaries and the topographic surface. Internally, the panels are subcelled to a dimension of 2.5 mE x 5 mN x 3 mRL, which represents the SMU dimension.

The block model parameters for the panel model are described in Table 17.4. The block model has been developed within the local coordinate system rather than the UTM coordinate system. Local coordinates are rotated relative to the UTM coordinates such that the EMZ strikes parallel to the rotated northing axis.

The choice of panel dimension was strongly influenced by the drilling grids that have been developed on the EMZ. Drillhole collars locally approximate a 25 m x 25 m grid (as in the Panel F block) but a 50 mN x 25 mE grid is approximated over most of the deposit. The SMU was selected deliberately small for the reason that selective mining to a small SMU is required for the EMZ to achieve maximum head grade on this deposit.

Table 17.4 EMZ block model parameters (local coordinates)

Direction	Minimum	Maximum	Increment
Easting	19600.0	20250.0	25m
Northing	49525.0	52475.0	50m
Elevation	-49.0	275.0	6m

Table 17.5 Variogram parameters

Backtransformed Models	Rotation				Structure 1			Structure 2			Structure 3					
Domain	X	Y	Z	C <sub>0</sub>	C <sub>1</sub>	Range X	Range Y	Range Z	C <sub>2</sub>	Range X	Range Y	Range Z	C <sub>3</sub>	Range X	Range Y	Range Z
East Main Arenite LOWER	0	40	-5	5.443	2.060	23.0	25.0	9.5	0.844	163.6	520.0	35.0				
East Main Arenite UpSap	0	40	-5	4.113	1.595	6.3	6.3	7.0	0.652	30.0	30.0	26.0	0.787	199.0	400.0	55.0
West Main Arenite LOWER	0	5	0	0.575	0.325	19.8	29.0	10.0	0.197	132.0	180.0	55.0				
West Main Arenite UpSap	0	5	0	0.508	0.348	19.0	27.0	12.0	0.095	88.2	88.2	30.0				
East FW Argillite Fresh	0	50	-5	1.578	1.146	8.0	13.9	7.2	0.813	38.0	42.0	38.2	0.439	110.0	395.6	56.5
East FW Argillite Oxide	0	50	-5	0.640	0.620	7.3	7.3	6.0	0.402	152.4	183.4	35.7				
West FW Argillite Fresh	90	0	110	0.811	0.520	35.0	45.0	22.0	0.175	173.0	100.0	35.0				
West FW Argillite Oxide	90	0	110	0.084	0.035	26.9	14.0	18.0	0.026	190.0	45.0	28.0				
East FW Arenite	0	45	-6	1.609	1.850	12.0	15.0	5.5	0.340	100.0	350.0	20.0	1.310	17.0	28.6	40.0
West FW Arenite	0	-80	5	0.731	0.503	6.8	6.8	6.0	0.348	27.3	43.9	17.0	0.295	30.4	145.0	29.0

**17.2.10 Grade interpolation and boundary conditions**

It is anticipated that the EMZ will be mined selectively by a truck and shovel surface mining operation. Small block estimates developed using linear estimation approaches (e.g., kriging or inverse distance) frequently fail to predict grade tonnage relationships correctly, leading to what is known as the “vanishing tonnes problem” described by David (1977). Estimating into larger blocks may more correctly predict the grade tonnage relationships but do not reflect the expected mining practice. Accordingly, more sophisticated non-linear estimation techniques have been developed for the EMZ to address these issues. Examples of non-linear techniques include multiple-indicator kriging (MIK), disjunctive kriging, multigaussian kriging and uniform conditioning (UC). Information about the practice of uniform conditioning is available in Rivoirard (1994) and Zaupa-Remacre (1984).

The EMZ mineral resource estimate has been developed using UC which provides:

- For each panel an estimate of the proportion of that panel that exceeds an applied cut-off gold grade.
- An estimate of the average grade of that proportion.

The average grade of each large panel has been estimated using Ordinary Kriging. The domain boundaries are considered to represent hard domain boundaries in this process. The search parameters applied in the development of the kriged panel estimates have been designed to minimize conditional bias of the panel estimates following the approach of Vann et al. (2003).

Panel estimates have been validated by comparing the declustered sample data mean with the volume-weighted kriged estimate for each domain. Partial statistics are shown below for illustration.

**Mean of declustered drillhole data vs OK large panel estimates**

Domain	3m drillhole composites		Volume weighted OK large panel estimates	
	Count	Mean	Count	Volume Weighted Mean
East Main Arenite Lower	11,495	1.04	6,313	0.98
East Main Arenite Upper Saprolite	5,888	1.17	1,442	1.20
East FW Argillite Fresh	3,976	0.74	2,657	0.73
East FW Argillite Oxides	1,127	0.64	450	0.66
East FW Arenite Fresh	1,254	0.87	1,451	0.85
West Main Arenite Lower	3,865	0.37	2,420	0.36
West Main Arenite Upper Saprolite	1,700	0.62	524	0.62
West FW Argillite Fresh	1,605	0.46	1,418	0.45
West FW Argillite Oxides	498	0.23	231	0.22
West FW Arenite Fresh	842	0.60	907	0.59

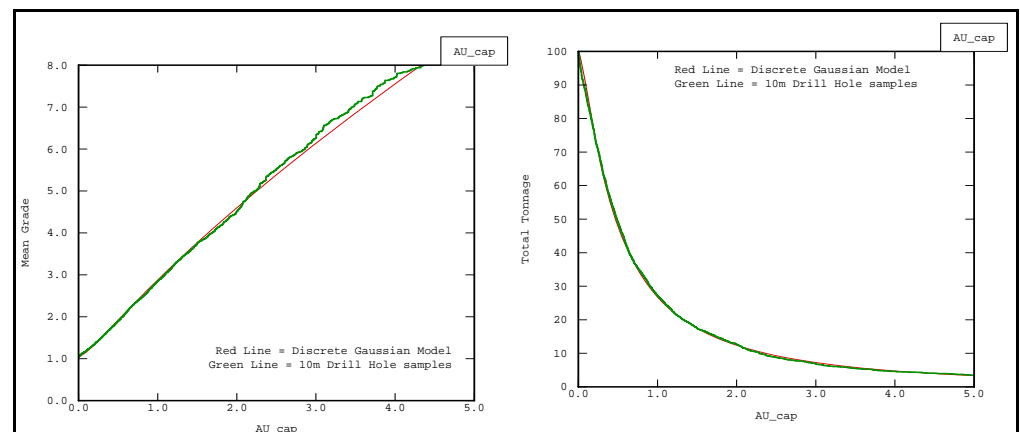
The naïve averages of the 3m composite grades may be significantly different from the declustered averages. The volume-weighted block grades in contrast differ by 5% or less. A series of swath plots comparing the declustered average sample grade with coordinates were also developed. Block grades (volume weighted) were placed on these charts and confirm that the estimates honour local grade trends and variations that are present within the sample data.

The change of support modeling within the UC process was validated in the following way:

- In the East Main Arenite lower domain the samples were composited to 10m downhole lengths.
- The histogram of 10m downhole sample grades was then estimated using the same change-of-support model employed in the uniform conditioning.
- The results of the experimental composite grades and the estimated composite grade distribution were contrasted in grade-tonnage curve.

The comparative results are presented in Figure 17.4 and show an excellent fit which supports the change-of-support model. The red line represents the estimated composite grades (derived using a discrete Gaussian change of support model) and the green line is the experimental data derived from downhole compositing.

Figure 17.4 East Limb Main Arenite G – T results for 10 m downhole composites



For each panel the kriged grade is an estimate of the local mean grade. This value is also the estimate of the average grade of all the SMUs contained within each panel. UC provides estimates of the proportion of a panel occupied by SMU with grades greater than a given cut-off. An estimated grade-tonnage curve for each panel can then be derived. For each panel, these grade tonnage curves have been used in the January and May 2007 models to estimate discrete SMU grades that satisfy this grade tonnage curve and these SMU estimates have been localised by methods described by Abzalov (2006).

All grade modeling has used LWL69M solution grades combined with the remediated and Ranger's raw data. However, LWL69M does not measure the total gold because some Au is retained in the tailing.

The LWL69M assays were thus converted to *in situ* gold values in both the January and May 2007 models so that metallurgical recovery factors could be applied. This was done by using the 50 gram Fire Assay of LWL69M tailings to calculate % Leach in the following way:

$$\% \text{ Leach} = \text{LWL69M-solution grade} / (\text{LWL69M-solution grade} + \text{Tailing g/t}) * 100$$

The leach efficiency was modeled within each of the major RCODE lithology domains. These data are presented in Table 17.6 by domain and weathering type. There are sufficient data to derive factors for all units except for FW argillite upper saprolite which daylight at the southern end of the design pit shell.

Table 17.6 Summary statistics for % Leach of LWL69M assays

RCODE	Data	Upper Saprolite	Lower Saprolite	Fresh
Main Arenite	Count	411	265	423
	Average	97.6%	95/5%	95.5%
FW argillite	Count	2	29	141
	Average	95.36%	95.55%	94.46%
FW arenite	Count			92
	Average			94.63%

Factors were modelled for rocktype and weathering domains as a set of step-wise conditional means. The final models are presented in Table 17.7 and shown graphically in Figure 17.5. Data points used in the analysis are shown as blue symbols. Anomalous data points excluded from the analysis are shown as red symbols.

Table 17.7 %LWL69M leach by rocktype and weathering domain

Domain	From	To	From	To	From	To	From	To	From	To
Main Arenite Fresh	0.00	0.20	0.20	0.60	0.60	1.00	1.00	3.00	3.00	100.00
	<b>90.02%</b>		<b>97.63%</b>		<b>98.12%</b>		<b>98.31%</b>		<b>98.96%</b>	
FW Argillite Fresh	0.00	0.70	0.70	2.00	2.00	10.00	10.00	100.00		
	<b>90.68%</b>		<b>94.91%</b>		<b>97.20%</b>		<b>98.76%</b>			
FW Arenite Fresh	0.00	1.00	1.00	3.50	3.50	100.00				
	<b>93.09%</b>		<b>95.66%</b>		<b>98.36%</b>					
Main Arenite Upper Saprolite	0.00	0.20	0.20	0.40	0.40	1.30	1.30	2.00	2.00	100.00
	<b>92.32%</b>		<b>97.66%</b>		<b>98.12%</b>		<b>98.40%</b>		<b>99.05%</b>	
Main Arenite Lower Saprolite	0.00	0.20	0.20	0.70	0.70	1.50	1.50	5.00	5.00	100.00
	<b>84.74%</b>		<b>97.88%</b>		<b>98.25%</b>		<b>99.11%</b>		<b>99.26%</b>	
FW Argillite Upper Saprolite	Use Main Arenite Upper Saprolite Curve									
FW Argillite Lower Saprolite	Use Main Arenite Lower Saprolite Curve									

The Localised Uniform Conditioning (LUC) estimates that have been used in all subsequent DFS mine planning studies contain this conversion of Au-solution grades to *in situ* total Au grades. The % Leach factors are applied to the SMU grades after Au-solution grades have been estimated into the blocks.

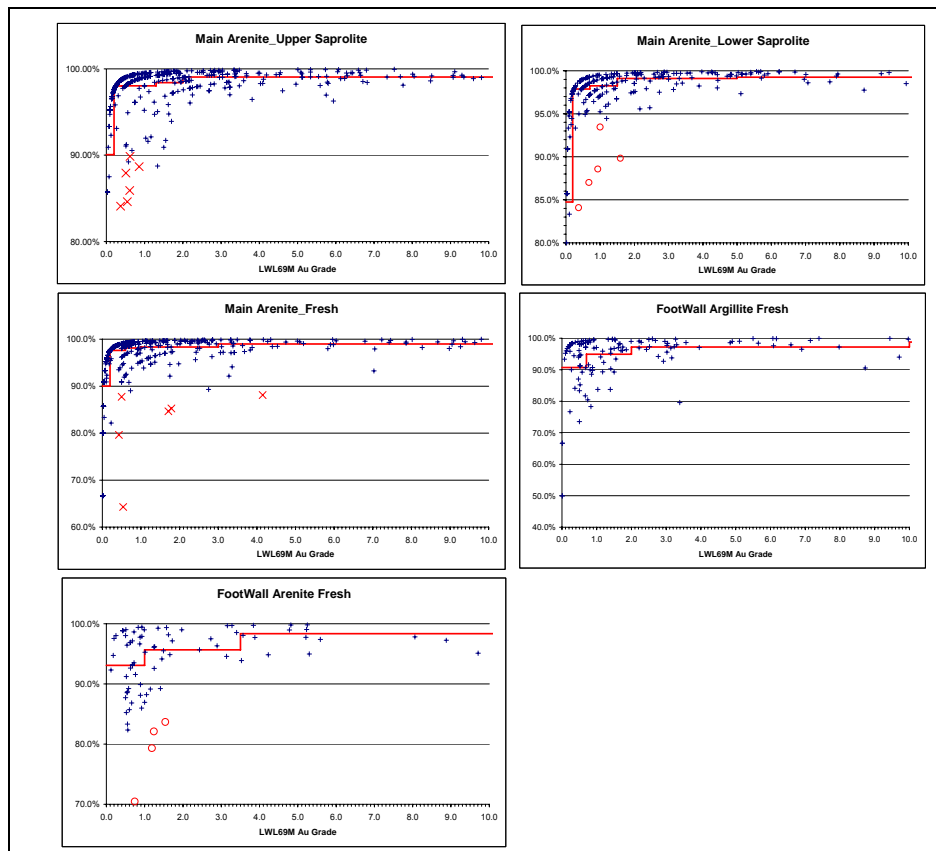
**17.2.11 Density**

Essakane measured density by immersion method on approximately 6 000 saprock and Fresh arenite and argillite core samples. Each sample was 10 – 25 cm in length and was sealed with paraffin wax. Dry bulk densities were also measured by weighing air-dried drill core while in the core trays. For each tray the total core length was measured and the mass of the core minus the mass of the metal tray was also measured. Based on the nominal diameter for HQ core, the core volume was estimated and divided by the mass which gave a bulk density estimate. This method was particularly useful for the friable upper saprolite intervals.

Comparison of immersion and core tray measurements shows similar values for Fresh intervals but saprock immersion data are biased high. Essakane believes this bias is caused by selection of more competent samples for immersion, and wetting of porous samples.

The two density data sets have been merged within one spatial database but with the immersion data for saprock excluded. Density values show a well defined trend which increases with increasing depth until a constant Fresh rock density is reached. Density is less sensitive to rock type and a density model was created using IDW<sup>2</sup> interpolation. The search neighbourhood was deliberately restricted in the vertical to ensure the density profile is honoured in the estimated block values

**Figure 17.5 LWL69M % Leach by grade, rocktype and weathering**



### 17.2.12 Model validation

Snowden validated the EMZ model using the following techniques:

- Comparison of top cut input grades with tonnage weighted output grades
- Inspection of the model against the input composites
- Comparison of moving window input and output statistics

### 17.2.13 Mineral Resource classification

Snowden and Essakane concluded that classification of any EMZ mineral resources as Measured was not possible on the basis of coarse gold sampling problems and application of remediation factors to 42% of the assay data. Classification of the remaining materials into Indicated and Inferred Mineral Resources made use of drilling density as well as estimation quality given by theoretical slope of regression  $Z|Z^*$  and kriging efficiency of the large panel grade estimates.

Essakane determined that the kriging efficiency and the theoretical regression slope  $Z|Z^*$  were almost linear and thus defined threshold kriging efficiencies for a threshold regression slope. Blocks with kriging efficiency greater than 0.25 were flagged in the model and a set of perimeters enclosing the blocks were defined on successive cross-sections. Some isolated blocks with kriging efficiencies less than 0.25 were included in the process, in the same way that isolated blocks outside the perimeters were excluded. Drillholes on each section were also posted on-screen and were used to make local decisions about placing the perimeters. The blocks contained within this volume were classified as Indicated Mineral Resources and all remaining material was defined as Inferred Mineral Resources. Most of the Inferred material is located below the US\$ 500/oz design shell used in this DFS.

### 17.2.14 Mineral Resource reporting

Snowden reported Mineral Resources for the January and May 2007 models constrained by US\$ 650/oz Whittle pit shells. The Whittle input parameters were the same as used in the DFS US\$ 500/oz mine design.

Table 17.8 reports the total May 2007 Mineral Resource estimate and the Mineral Resource estimate constrained within a US\$ 650/oz pit shell by gold cut-off grade and classification.

The classification categories of Inferred, Indicated and Measured under the JORC Code are equivalent to the CIM categories of the same name (CIM, 2005).

The Mineral Resource estimates have been developed by UC post processing of 25 x 50 x 6 m panels estimated by ordinary kriging. A discrete gaussian change-of-support model has been applied to derive estimates for 2.5 x 5 x 3 m SMU's.

The May 2007 estimates include an Information Effect to account for future in-pit grade control drilling and sampling at 5 x 5 m hole spacing.

The May 2007 estimate is based on an analytical database containing 42% LWL69M assays, 42% historical BHP and Orezone assays which have been remediated using LWL69M factors, and 16% Ranger assays which were accepted without requiring modification.

The May 2007 Mineral Resource estimates presented in Table 17.8 represent *in situ* gold grades and gold ounces.

Table 17.8 May 2007 Mineral Resource estimate by gold cut-off grade and classification

<b>Total Mineral Resource</b>					
<b>Classification</b>	<b>Au Cut-off grade (g/t)</b>	<b>0.5</b>	<b>0.8</b>	<b>1.0</b>	<b>1.2</b>
Indicated	Tonnes (Mt)	78.4	52.9	42.2	34.2
	Grade (Au g/t)	1.58	2.04	2.33	2.62
	Ounces (Moz)	3.99	3.47	3.16	2.88
Inferred	Tonnes (Mt)	27.4	17.6	13.7	10.8
	Grade (Au g/t)	1.44	1.89	2.17	2.46
	Ounces (Moz)	1.27	1.07	0.96	0.86

<b>Mineral Resource constrained by US \$650/oz pit shell</b>					
<b>Classification</b>	<b>Au Cut-off grade (g/t)</b>	<b>0.5</b>	<b>0.8</b>	<b>1.0</b>	<b>1.2</b>
Indicated	Tonnes (Mt)	73.4	50.4	40.5	33.0
	Grade (Au g/t)	1.62	2.07	2.35	2.64
	Ounces (Moz)	3.82	3.35	3.06	2.80
Inferred	Tonnes (Mt)	16.1	11.6	9.5	7.8
	Grade (Au g/t)	1.66	2.06	2.31	2.58
	Ounces (Moz)	0.86	0.77	0.71	0.65

### 17.3 Assumptions, methods and parameters – Project reserve estimates

In June 2007, Essakane and GRD Minproc estimated Mineral Reserves based on the January 2007 and updated May 2007 Mineral Resources for the EMZ.

#### 17.3.1 Pit optimization

Essakane and GRD Minproc were provided with a geological block model in January 2007. Mine planning was carried out to the optimisation stage and a final surface mine design was produced for a US\$ 500/oz gold price assumption. Only Indicated Mineral Resources were considered in this design process.

An updated block model was provided in May 2007. Because of time constraints the US\$ 500/oz mine design on the January 2007 was used to calculate Mineral Reserves on the new model.

Essakane adopted the following procedures during the design process:

- The January 2007 model was imported into MineSight software
- Dilution was estimated at 14% with a 0.31 g/t gold grade. Dilution parameters were estimated by a bench-by-bench assessment of mineable ore.
- Mining shapes were digitized around groups of resource blocks with dimensions greater than or equal to 5m as this is the minimum mining width for the 7 m<sup>3</sup> bucket of the smaller selected backhoe excavator.

The 14% dilution of the mining shapes includes:

- Internal waste that will have to be mined with the ore.
- 0.5 m dilution skin around the mining shapes.

Metallurgical recovery factors were provided by Mr M. Brittan, Head Consulting Metallurgist Gold Fields - International Projects, for each ore weathering type and were used to calculate the recovered gold grade in each block. The equations which were used are:

- Saprolite Au rec (%) =  $99.7 - 6 \times \{\ln(\text{Au}+1)\}/(\text{Au})$
- Saprock Au rec (%) =  $(\text{Saprolite Au rec \%} + \text{Fresh Au rec \%})/2$
- Fresh Au rec (%) =  $99.5 - 10.18 \times \{\ln(\text{Au}+1)\}/(\text{Au})$

The MineSight model was imported into Datamine to create the Whittle model. This model included pit slope codes, mining costs for ore and waste and processing costs. Only the Indicated blocks with centroids located inside the mining shapes were exported as ore blocks for the optimization. The ore blocks with centroids falling outside the mining shapes represent an approximate 10% ore loss.

Resource blocks were re-blocked into 5mE x 10mN x 3mRL blocks to reduce the computation time for each Whittle run. Total dilution was also reduced from 14% to 8.4% for the optimizations to compensate for dilution caused by the re-blocking.

The process cut-off grade method was used for ore selection. The optimizations were run on a floating cut-off grade (variable per block). The cut-off grades were applied to the recovered gold grade. Optimizations were run with a 0% discount factor and excluded the capital cost of mining equipment.

The main input parameters used in the optimization are summarized in Table 17.9. The fuel price used in the design optimization was US\$ 0.61/litre or US\$ 45/bbl. A total cost of US\$ 10 million was estimated for Mining Fixed Costs which includes:

- Mine supervision
- Maintenance supervision
- MARC contract fee
- Tyre management
- Contracted blast services
- Equipment damage
- Surface Mine dewatering
- Ancillary mining equipment.

Variable mining costs for load, haul and dump were calculated using cycles times and fuel consumptions figures that were generated using the Caterpillar FPC software. Haul profiles were measured from the centroid of the surface mine bench to the centroid of the corresponding lift on the overburden storage facility, or to the centroid of the ROM pad. The 3 m flitches were combined into 12 m benches for this exercise. Haul profiles were digitized with a maximum 10% gradient.

Operating costs used in the Whittle optimizations are based on the following hourly rates:

- Loading: US\$ 319.45 and US\$ 151.87 for the Liebherr 994B (9350) and 984C excavators
- Hauling: US\$ 77.43 for a Caterpillar 785C not including the variable fuel consumption (59.3 l/h average)
- Drilling: US\$ 143.36 for a Sandvik/Tamrock Pantera 1500.

Other costs included in the optimization were:

- Blasting: US\$ 0.30/bcm for Saprolite (Powder Factor = 0.20kg/bcm) and US\$ 0.74 per bcm for Saprock (Powder Factor = 0.55 kg/bcm)
- Ore-rehandling: US\$ 0.06/tonne assigned to the processing cost
- Pre-splitting: US\$ 0.11/bcm
- Labour: Estimated at US\$ 6.17/h
- Grade Control: Estimated at US\$ 0.54/bcm

The undiscounted shell corresponding to the US\$ 500/oz gold price was selected for the surface mine design.

Table 17.9 Summary of Whittle input parameters – January 2007 resource model

Optimisation parameter	Unit	Factor		
Conversion	g/oz	31.1034768		
Gold price	US\$/oz	<b>500.00</b>		
Royalty		3.00%		
	US\$/oz	15.00		
Refining charge	US\$/oz	1.08		
Net gold price US\$/oz	US\$/oz	483.92		
Net gold price US\$/g	US\$/g	15.56		
		<b>Overheads</b>		
GFL administration	US\$/y	0		
Essakane administration	US\$/y	305 020		
JV administration	US\$/y	7 267 862		
Crusher feed (50 % of FEL costs)	US\$/y	322 738		
Process capacity	t/y	5 400 000		
		<b>Saprolite</b>	<b>Saprock</b>	<b>Fresh Rock</b>
Recovery (metallurgical)*	%	100.0	100.0	100.0
Processing cost	US\$/t	6.84	7.76	8.67
Ore premium	US\$/t	0.00	0.00	0.00
Tailing storage	US\$/t	0.00	0.00	0.00
Sundry	US\$/t	0.00	0.00	0.00
GFL administration	US\$/t	0.00	0.00	0.00
Essakane administration	US\$/t	0.06	0.00	0.00
JV administration	US\$/t	1.35	1.35	1.35
Crusher feed	US\$/t	0.06	0.06	0.06
<b>Whittle 'processing cost'</b>	<b>US\$/t</b>	<b>8.30</b>	<b>9.22</b>	<b>10.13</b>
<b>Mining cost – fixed</b>	<b>US\$/t</b>	<b>0.41</b>		
<b>Mining cost - variable</b>				
Load & haul - average	US\$/t	0.64		
Drill & blast	US\$/t	0	0.17	0.27
Grade control	US\$/bcm	0.54		
Pre-splitting (average)	US\$/bcm	0	0	0.11
Mining dilution	%	8.40	8.40	8.40
Mining loss	%	0.00	0.00	0.00
Overall surface mine slope (incl ramp system)	degrees	27-21	39-25	58-49
Mining rate	Mt/y	24.5		

**17.3.2 Pit design**

Essakane and GRD Minproc based the mine design criteria on a conventional surface mine operation using Liebherr 994B (9350) for the waste and 984C for the ore hydraulic excavators and Caterpillar 785C off-highway trucks. The geotechnical and slope stability recommendations were provided by Knight Piésold and are described in their report Essakane DFS SM Report 5094-42-01 dated March, 2007.

These recommendations are summarized in Table 17.10. The haul road was designed for Caterpillar 785C haul trucks. For double lane traffic the minimum width was 26 m and includes a 1 m wide drainage ditch and a 2 m high safety berm. For single lane traffic a minimum width of 16 m was used. The maximum road gradient was 10%.

The final surface mine design is 2.6 km long and approximately 500 m wide. Its highest elevation is 65 mRL which gives a maximum depth of 204 m.

Knight Piésold was provided with the final surface mine design and issued their sign-off in the document 5094-L002 Slope Angles.doc dated 13 June 2007. The final surface mine design is shown in Figure 17.6.

**Table 17.10 Geotechnical configuration for the US\$ 500/oz surface mine design**

Surface mine design parameter	West wall			East wall		
	Saprolite	Saprock	Fresh rock	Saprolite	Saprock	Fresh rock
Bench height (m)	6	6	12	6	6	12
Batter angles (degrees)	43	72	85	43	65	80
Bench stack height (m)	36	48	60	36	36	60
Berm width (m)	4	4	5	4	4	5
Geotechnical berm width (m)	13	13	0	13	13	16
Ramp width (m)				26	26	26
Bench stack angle (degrees)	27	37	59	20	25	48
Overall Slope Angle (degrees)		44			37	

**17.3.3 Other assumptions and parameters**

The cut-off grade calculations for the surface mine reserves were calculated in the following way:

- The final mine design reserves are reported above the Whittle processing cost that includes processing, administration and ore premium for each weathering type.
- The cut-off grades are applied to the recovered gold grade.
- The process recovery is stored in each block of the model.
- Mining losses were accounted for by excluding ore blocks that could not grouped together to allow for a 5 m minimum mining width.
- The processing cut-off calculation is shown in Table 17.11.

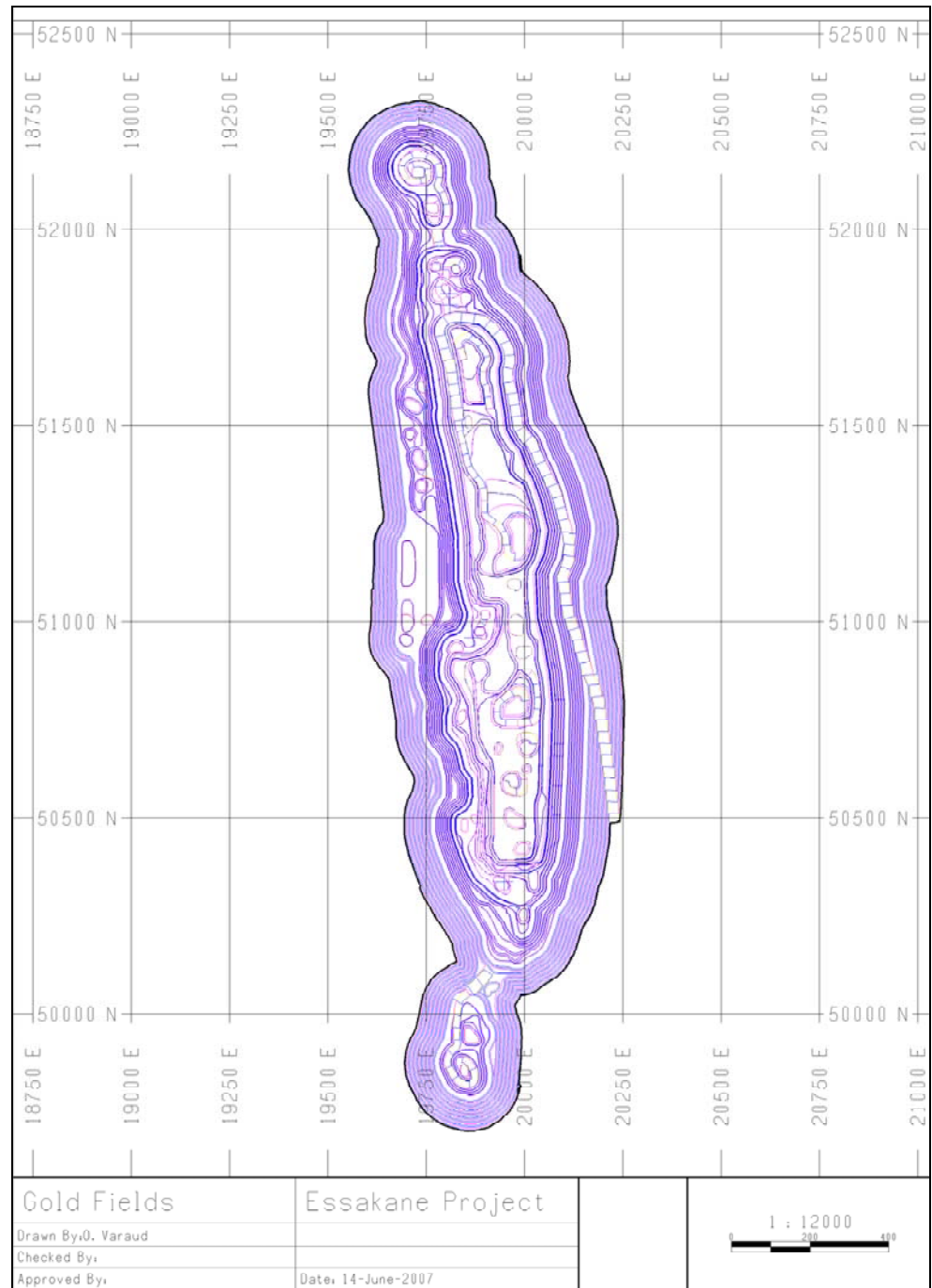
Table 17.11 Cut-off grade calculation for surface mine reserves

Cut-off grade parameter	Unit	Factor		
Conversion	g/oz	31.1034768		
Gold price	US\$/oz	<b>500.00</b>		
Royalty		3.00%		
Refining charge	US\$/oz	15.00		
Net gold price US\$/oz	US\$/oz	1.08		
Net gold price US\$/g	US\$/g	483.92		
		<b>Overheads</b>		
Essakane administration	US\$/y	305 020		
JV administration	US\$/y	7 267 862		
Crusher feed	US\$/y	322 738		
Process capacity	t/y	5 400 000		
Cut-off calculations		<b>Saprolite</b>	<b>Saprock</b>	<b>Fresh Rock</b>
Recovery (metallurgical)	%	100.0%	100.0%	100.0%
Processing cost	US\$/t	6.84	7.76	8.67
Ore premium	US\$/t	-0.14	-0.16	-0.19
Surface haulage / Rehandle	US\$/t	0.00	0.00	0.00
Tailing storage	US\$/t	0.00	0.00	0.00
Essakane administration	US\$/t	0.06	0.06	0.06
JV admin	US\$/t	1.35	1.35	1.35
Crusher feed	US\$/t	0.06	0.06	0.06
<b>Process cut-off cost</b>	<b>US\$/t</b>	<b>8.16</b>	<b>9.06</b>	<b>9.94</b>
<b>Process cut-off grade</b>	<b>g/t Au</b>	<b>0.52</b>	<b>0.58</b>	<b>0.64</b>

### 17.3.4 Mineral Reserve classification

The classification categories of Probable and Proved Ore Reserve under the JORC Code are equivalent to the CIM categories of Probable and Proven Mineral Reserve (CIM, 2005).

Figure 17.6 Plan of the US\$ 500/oz surface mine design (local grid)



### 17.3.5 Mineral Reserve reporting

Probable Mineral Reserves are given in Table 17.11 and in Table 17.12 which gives the surface mine design inventory based on the May 2007 resource model.

Mineral Reserves for the Project at US\$ 500/oz and using the May 2007 model are 46 413 340 tonnes at 1.78 g/t. Non-reserve material listed in Table 17.11 represents Inferred material contained within the US\$ 500/oz design shell but not used in the optimization process.

Table 17.12 LOM Mine Mineral Reserve inventory – May 2007 resource model

<b>EMZ DFS - LOM SCHEDULE INVENTORY - MAY-07 RESOURCE MODEL</b>									
<b>Diluted tonnage and grades (14% dilution at 0.31 g/t Au) by Weathering Type</b>									
Weathering type	Cut-off grade	g/t Au rec	Tonnage	Au (g/t)			Total Au (koz)		
				Rec	Dil	In-situ	Rec	Dil	In-situ
	Saprolite	0.52	11 573 506	1.42	1.47	1.63	528	547	608
<b>Reserve</b>	Saprock	0.58	10 071 913	1.63	1.71	1.91	529	555	618
<b>Material</b>	Fresh Rock	0.64	24 767 921	1.84	1.94	2.17	1 461	1 547	1 729
	<b>Total</b>		<b>46 413 340</b>	<b>1.69</b>	<b>1.78</b>	<b>1.98</b>	<b>2 518</b>	<b>2 649</b>	<b>2 955</b>
<b>Non-</b>	Saprolite	0.52	112 018	0.95	0.99	1.09	3	4	4
<b>Reserve</b>	Saprock	0.58	10 614	1.35	1.42	1.57	0	0	1
<b>Material</b>	Fresh Rock	0.64	459 068	1.99	2.10	2.35	29	31	35
	<b>Total</b>		<b>581 700</b>	<b>1.78</b>	<b>1.88</b>	<b>2.09</b>	<b>33</b>	<b>35</b>	<b>39</b>
<b>GRAND TOTAL</b>			<b>46 995 040</b>	<b>1.69</b>	<b>1.78</b>	<b>1.98</b>	<b>2 551</b>	<b>2 684</b>	<b>2 995</b>
OVERBURDEN			141 050 466						
Strip Ratio*			3.05						

## 18 Other relevant data and information

### 18.1 Mining operations

#### 18.1.1 Introduction

RSG Global undertook the PFS mining component for the Project in October 2005. This study was based on a 2005 EMZ resource model which is described in Section 8 and Figure 8.2 of this Technical Report. The 2005 geological model was revised in late 2005 due to changed interpretations of data and Essakane recognized potential upside in the EMZ that had not been fully tested. In January 2006 Essakane started a program of oriented diamond core drilling to infill and expand the EMZ resource inventory, and also re-assayed 28 640 historical samples in two phases using a LWL69M rapid cyanide leach process. This project development work resulted in a wholly new DFS mineral resource model dated January 2007. Re-assaying continued through to April 2007 and an updated resource model was issued in May 2007.

Revised geotechnical recommendations were provided by Knight Piésold (KP) in April 2007 and form the basis of this DFS.

A mill production throughput of 5.4 million tonnes per year (Mt/y) has been adopted for the DFS. In the PFS a series of mine optimizations was run for six production throughput cases ranging from 3.0Mt/y to 6.6Mt/y. The exercise indicated that the best value would be obtained at the maximum throughput level considered. However, security of water supply and the necessity of balancing a single mining fleet with the life of mine (LOM) dictated a lower mill throughput level and 5.4Mt/y was in turn selected and adopted.

Detailed haul profiles and cycle times have been used in the mining cost calculations. Processing and mining costs were updated.

A detailed mine design including sequential mining phases was produced from the January 2007 Resource model and signed off by KP. This was subsequently updated for the May 2007 Resource model using the US\$500MI mine design shell, and adopted as the basis for this DFS

#### 18.1.2 Operation

Mining of the EMZ will involve a conventional surface mine, selective ore mining and an owner-operator approach with blasting, fleet maintenance and tyre management (MARC) contracts. Annual ore production is targeted at 5.4 Mt. This concept and annual ore production rate are consistent with the PFS.

There are two main rock types in the design shell (arenite and argillite) with three weathering zones. With increasing depth these are upper saprolite, saprock (also called lower saprolite) and Fresh rock. The weathering zones will be sequentially mined over the life of the operation starting with the friable upper saprolite. The upper saprolite (fully weathered) material is assumed to be 'free dig' while the saprock (semi-weathered) material and Fresh rock are assumed to require drilling and blasting.

Mining will be undertaken on 6 m benches with the material excavated in two discrete 3m high flitches. The final bench height recommended by KP 6 m in saprolite and saprock and 12 m in Fresh rock.

Grade control will be done by 5 ¼ inch RC drilling with assays taken every 1 m intervals that will be averaged into 3 m composites to reflect the operating flitch height and interpreted by mine geologists. An 8 m square drill pattern is assumed

with drilling costs being applied only to that material below the main arenite hanging wall contact on both limbs of the fold. The RC drilling will drill and sample a minimum of six 3 m benches at one time. All holes will be drilled at a 60° dip and at 45° to the strike of the orebody to maximise the sampling efficiency for both northerly and easterly striking vein sets.

The mining equipment for the project comprises hydraulic excavators (in backhoe configuration) and off-highway haul trucks. Drilling will be done by track mounted top hammer hydraulic drills with a 6.5 m high mast to allow for single pass drilling.

The primary fleet will ultimately comprise:

- 3 excavators, one Terex RH 70 (120 t with 7m<sup>3</sup> bucket) and two Terex 120-E (290 t with 15 m<sup>3</sup> bucket)
- 13 Caterpillar 785C trucks (nominally 140 t)
- 6 Atlas Copco L7-40 drill rigs with COP 40-50 drifters (89-127 mm diameter) using T51 drill consumables.

Ancillary equipment will consist of

- 2 front end loaders (Caterpillar 992K) for crushing loading/stockpile reclaim
- 2 tracked dozers (Caterpillar D10T) for maintenance and roadway construction
- 1 wheel dozer (Caterpillar 824H) for operating bench overburden stockpile
- 2 graders (Caterpillar 16M) for roadway maintenance
- 2 water trucks (Caterpillar 773F) for dust suppression around the site
- 1 tool carrier for stores, tyre handling and general purposes
- 1 mobile fuel truck and 1 mobile lubrication truck for servicing equipment in the field
- 1 low loader for long distance equipment transportation within the mine area (mine to workshop).

## 18.2 Mine production

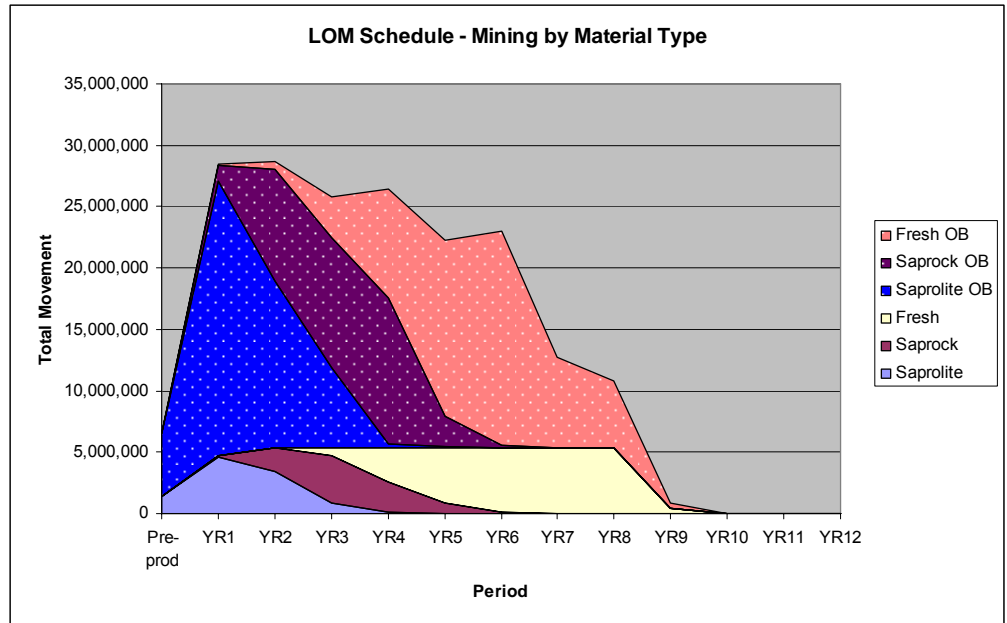
The US\$ 500MI mine design provides for 46.4 Mt of ore and 141.6 Mt of overburden will be mined over an 8.6 year mine life. The strip ratio (on a tonnes basis) is estimated at 3.05 : 1.

The diluted ore grade (mill feed) is estimated at 1.78 g/t Au based on average dilution of 14% at 0.3g/t.

A three month pre-Production period is required when 1.4 Mt of ore and 5.1 Mt of overburden will be mined. The ore will be stored in two temporary stockpiles which will be reclaimed over the life of the mine. Annual total material mining rates commence at 29 Mt and reduce over time to 10 Mt in the penultimate year.

Ore to the plant will be a blend of the three weathering types commencing predominantly with saprolite (no fresh rock) for the first year, saprolite and saprock (minimal fresh rock) for years 2 and 3, saprock and fresh rock for years 4 and 5 and primarily fresh rock for years 6, 7 and 8 and reclaimed low grade saprolite ore in the final year. This production profile by material type is shown graphically in Figure 18.1 with details presented in Table 18.1.

Figure 18.1 LOM schedule by material type (May 2007 model)



### 18.2.1 Surface mine layout and infrastructure

The US\$ 500MI surface mine design shell is approximately 2 600 m long, 500 m wide and 200 m deep. The mine is long enough to avoid any ramp switchbacks. The mine ramp commences on the hanging wall side before turning anti-clockwise to the footwall side. The ramp changes from double lane traffic (26 m) to single lane traffic (16 m) part way along the footwall.

Overburden will be stored in a single site (overburden storage facility) east of the surface mine with access opposite the mine ramp exit. This site will have a total area of 227 hectares and its average height at the end of the mine life will be around 50 m.

Other mining infrastructure involves a mine office complex (mine offices, change house and canteen), equipment workshop with overhead cranes and external wash down bays, blasting and explosives compound including magazines, diesel storage and dispensing facility and a drill core storage facility.

As the northern part of the surface mine extends towards the Gorouol River, a flood protection wall will be built around the northern end to ensure that no flood waters enters the surface mine during the wet season. This bund will be constructed by the mining crew in Year 2 when sufficient fresh rock becomes available.

Table 18.1 Essakane US\$ 500/oz LOM detailed mill feed schedule

Essakane US\$ 500/oz LOM Schedule – Detailed mill feed schedule by source and weathering type																				
Period	Saprolite			Saprock			Fresh Rock			Stockpile Reclaim			TOTAL							
	kt	Rec Au	Dil Au	SG	kt	Rec Au	Dil Au	SG	kt	Rec Au	Dil Au	SG	kt	Rec Au	Dil Au	SG	kt	Rec Au	Dil Au	SG
		g/t	g/t			g/t	g/t			g/t	g/t			g/t	g/t			g/t	g/t	
Pre-Prod	0	0.00	0.00	0.00	0	0.00	0.00	0.00	0	0.00	0.00	0.00	0	0.00	0.00	0.00	0	0.00	0.00	0.00
1	4 614	1.45	1.50	1.75	138	1.50	1.58	1.92	1	0.85	0.92	1.85	647	1.87	1.93	1.70	5 400	1.50	1.56	1.75
2	3 462	1.69	1.75	1.86	1 862	1.64	1.72	2.12	75	1.13	1.21	2.50					5 400	1.67	1.73	1.96
3	852	1.61	1.67	2.00	3 804	1.91	2.00	2.34	744	1.67	1.78	2.63					5 400	1.83	1.92	2.33
4	62	1.16	1.21	2.29	2 461	1.55	1.62	2.48	2 877	1.63	1.73	2.71					5 400	1.58	1.67	2.60
5	3	1.56	1.61	2.39	839	1.68	1.76	2.53	4 558	1.77	1.87	2.75					5 400	1.75	1.85	2.71
6					114	1.46	1.53	2.63	5 286	1.94	2.05	2.74					5 400	1.93	2.04	2.74
7	0	0.00	0.00	0.00	0	0.00	0.00	0.00	5 400	1.91	2.02	2.77					5 400	1.91	2.02	2.77
8	0	0.00	0.00	0.00	0	0.00	0.00	0.00	5 400	1.86	1.97	2.78					5 400	1.86	1.97	2.78
9	0	0.00	0.00	0.00	0	0.00	0.00	0.00	426	1.83	1.94	2.79	2 786	0.62	0.66	1.92	3 212	0.78	0.83	2.03
Total	8 993	1.56	1.61	1.82	9 219	1.73	1.81	2.35	24 766	1.84	1.94	2.75	3 433	0.86	0.90	1.88	411	1.69	1.78	2.43

The water table is approximately 20 to 80 m below surface in the vicinity of the surface mine, with the deeper end at the Southern end away from the river. A water aquifer exists in the saprolite and saprock zones and dewatering of the proposed surface mine is required prior to start-up. Numerous bores will be drilled around the surface mine perimeter and the groundwater extracted using submersible pumps. All bores will be connected to a common pipeline around the surface mine perimeter that will deliver water to the plant clean raw water pond for use in the process. A branch line from the perimeter main will also provide water to a water tank (turkey's nest) for use by water trucks for dust suppression when water is not available from the in-mine pumping system. Groundwater flow is expected to reduce over the life of the mine owing to extraction being greater than recharge.

In addition, an in-mine pumping system will be installed at the bottom of the surface mine to extract any seepage and wet season excess water from the mine. Additional slurry pumps will be added over time to stage pump the water owing to an increasing depth of mining. Water will be pumped to the plant dirty water pond for use in the process with a branch line to the turkey's nest for use by water trucks for dust suppression.

### 18.2.2 Mine establishment and work conditions

The mining operation has been designed around three operation crews working two 12 hour shifts per day (one crew off) on a six days on / three days off roster system for 362 days/year (three days discounted for weather affected production stoppage time). It is assumed that there will be 10 effective hours achieved from a 12 hour shift with equipment operating between 5 000 and 6 500 hours/year.

The mining structure will comprise both Expatriate and National employees. The majority of Expatriates will be in supervisory and technical roles and will be required to fully train National staff before relinquishing their positions. Mining establishment numbers vary over the life of the operation depending on equipment numbers but at a maximum would be in the order of 215 people made up of:

- 109 operations personnel
- 46 technical services personnel
- 60 contracted blasting, fleet maintenance and tyre management personnel.

To assist in the training of local employees for operating mining equipment and to reduce capital costs, the mining crew will undertake the excavation of the off-channel storage facility that involves 4 million m<sup>3</sup> of material. Two excavators (one small and one large), a small fleet of trucks and some ancillary equipment (grader, tracked dozer, water truck) will be used. Experienced contract operators for the large excavator and some ancillary equipment will be hired to ensure that the work can be done in the time allowed. Prospective local mine employees will be trained on the small excavator, trucks and some ancillary equipment. This work will be done prior to the pre-production mining period.

### 18.2.3 Further work

Essakane and GRD Minproc recommend investigation of the following items during the detailed design phase to try to reduce operating costs:

- Use of a large excavator for ore mining
- Contract drilling which would include grade control drilling
- Optimum production blasthole size
- Refine the surface mine optimisation and production scheduling.

## 18.3 Tailing storage facility and return water

The Tailing Storage Facility (TSF) and the Bulk Water Storage Dam (BWSD) were designed by KP and the pumps and pipelines for both systems by Patterson and Cooke under instruction from KP.

### 18.3.1 Hydrology

The TSF has been designed to cater for the 1 in 100 year flood which is 143 mm of rainfall in a 24 hour period. This flood run-off will be collected in the BWSD which is a 300 000 m<sup>3</sup> fully lined storage dam. The BWSD has a deeper section, with a capacity of 30 000 m<sup>3</sup>, which will collect the dry weather process water decant from the TSF. This has been done to reduce the losses due to evaporation. The pumping system on the BWSD will deliver 460 m<sup>3</sup>/hr to the process water pond within the plant area.

### 18.3.2 Evaporative drying tests

The settling and drying tests show acceptable results and coupled with the high evaporative rates good consolidation will be achieved within the TSF with final densities of 1.3 t/m<sup>3</sup> to 1.4 t/m<sup>3</sup>.

### 18.3.3 Tailing deposition method

Tailing will be deposited from a ring main around the TSF using a conventional spigot system managed on a rotational cycle to ensure drying and consolidation of the tailing beaches. A penstock will decant the liquor and deliver it to the BWSD. The penstock will be raised as the TSF increases in height.

The starter walls of the TSF range from 5 to 14 m in height and will be constructed from suitable borrow material until the raises can be achieved using the deposited tailing. The outer slopes of the TSF will be clad with selected overburden rock placed concurrently with the tailing deposition. A storm water diversion canal will divert surface water flows around the TSF.

### 18.3.4 TSF location and model

The site for the TSF was chosen taking into account the suitability of the ground conditions and the effect that the TSF will have on operations, the local population and the environment.

The TSF has been designed for a final capacity of 60 Mt and will occupy an area of 1.38 km by 1.47 km with a maximum rate of rise of 2.6 m/yr in the final years of the mine. The minimum and maximum heights of the TSF are at the south and north corners respectively and will be 24 m and 34 m.

### 18.3.5 TSF seepage analysis

Seepage analysis has been conducted using finite element software and the primary seepage is in the upper 1.5 m in a direction parallel to the ground surface. This seepage will be collected in a filter drain and directed into the BWSD. The low permeability of the material in the early stages of deposition will inhibit seepage of tailing water into the groundwater.

### 18.3.6 Tailing pumping

The tailing will be transported from the plant to the TSF using two pairs of pumps, one running and one on standby. The second pump of each pair will have a variable speed drive to manage the flow rates. The slurry design flow rate is nominally 685 m<sup>3</sup>/hr at a solids density of 59%.

There will be a dual delivery pipeline to the TSF and this is a 400 mm OD grade 100 HDPE line run on sleepers within an unlined earth bund to collect spillage. The TSF ring main will be provided with flushing facilities.

### 18.3.7 TSF return water and off-channel storage pumping facility

The off-Channel Storage Facility (OCSF) pumps are designed to provide the plant process water demand of 460 m<sup>3</sup>/hr and the 355 mm OD grade PE100 HDPE pipeline from these pumps will feed into the BWSO at the TSF. This pipeline will be continuously welded and buried to protect it from fire and damage.

The pumps at the BWSO will have the same capacity as those at the OCSF and will use an identical pipeline to transfer water from the BWSO to the plant process water pond.

### 18.3.8 Water supply and management

Water for the operations will come from:

- A diversion weir will be built on the Gorouol River and water will gravitate during the rainy season to an off-channel storage facility that will supply the plant process water
- A series of 25 boreholes will be drilled around the perimeter of the surface mine to de-water the mine and this water will be used in the plant for clean water applications
- A well field along the Gorouol River
- Run-off from the tailing storage facility will be used to augment the above two supplies
- Potable water will be provided by suitably sited boreholes

### 18.3.9 Mine water balance

Return water and rainfall run-off from the TSF will gravitate to the BWSO for storage. Water from the OCSF will be pumped to the BWSO and from there to the process water pond within the plant area.

The total plant water demand is 460 m<sup>3</sup> per hour comprising raw water 96 m<sup>3</sup> per hour and a process water 364 m<sup>3</sup> per hour. The raw water demand will be met by groundwater sources and the process water demand from the BWSO.

The feed to the BWSO will be from the OCSF that has a safe yield of 2 million m<sup>3</sup>/year and this is augmented by the return water from the TSF that is expected to yield an additional 1.37 million m<sup>3</sup>/year inclusive of storm runoff and this will meet the plant process water requirement of 3.2 million m<sup>3</sup>/year. The plant raw water supply will be met by the surface mine de-watering boreholes for the first four years of the mine and then from the well field along the Gorouol River. Domestic water for the mine and resettlement villages and plant will be by boreholes.

## 18.4 Electrical, control and instrument systems

### 18.4.1 Bulk power supply

A power station will be constructed on site as there is no National grid supply to the Project. This power station will utilize a combination of diesel engines fuelled by heavy fuel oil (HFO) and by diesel fuel oil. There will be three generating sets, each with a rating of 8 MW running on heavy fuel oil and five sets each with a rating of 1.6 MW running on diesel fuel. The design load is 19.9 MW and the three HFO sets will supply this base load. The diesel sets will be used to supplement the system, to start the 7 MW mills and as stand-by.

The quoted cost of electrical power from the selected power station operator is US\$ 0.155 per kWh and this is the largest contributor to the mine operating costs. The power station will be owned by the Project and operated by the supplier. The HFO and diesel fuel cost has been incorporated in the cost of electrical power. However, the supply of fuel will be the responsibility of the Project.

#### 18.4.2 Fuel oil supply and storage

The requirement for HFO is 97 000 litres per day and this will be trucked to the Project by one of the international fuel companies which will be under contract to the Project. Receipt of HFO and on-site storage will be the responsibility of the power station supply and operating company. The HFO storage tank farm will comprise two 1 000 m<sup>3</sup> storage tanks complete with all the necessary ancillaries for the receipt and distribution of the HFO and this will be sufficient for 15 days of operations. For diesel fuel there will be one 200 m<sup>3</sup> storage tank.

#### 18.4.3 Transformers and mini-sub

Transformer sizes have been rationalised to 1 MVA and 2 MVA to reduce the number of spares required and to keep fault levels within economic levels. Boreholes in the well-fields will be supplied by pole mounted transformers and mini-sub will be used to supply office, workshop and domestic loads. The secondary voltage is 415V, 3 phase and 240V single phase.

Selected loads have variable speed drives and those above 1 MW will have a dedicated converter transformer with a 690V secondary. The SAG mill motor is rated at 7 MW and, as it has a variable speed drive, it will have a custom made transformer to supply the motor at 6.6 kV. To rationalise on the mill motors, the ball and SAG mills will both have 7 MW motors at 6.6 kV and, to accommodate the ball mill motor, a 10 MVA, 11kV/6.6 kV auto transformer will be installed.

#### 18.4.4 Mill major drives

The mill motors will both be 7 MW motors at 6.6 kV and they will be wound rotor motors with continuously rated slip rings. The ball mill will have a LRS starter whilst the SAG mill motor will run on a variable speed drive and the rotor will be shorted out. The mill motors will be selected to meet the insulation requirements of variable speed drive.

The mill discharge pump motors are each rated at 1025 kW and these will be supplied at 690V by variable speed drives.

#### 18.4.5 Emergency generators

The village and identified critical loads will have dedicated emergency, diesel fuelled generators.

#### 18.4.6 Electrical enquiries and tenders

A full set of tender documents was prepared for each of the electrical requirements of the project including site installation. Multiple tenders were received against these enquiries and, after adjudication, the recommended prices included in the Capital Estimate of this DFS.

#### 18.4.7 Control system

The plant will be controlled by a SCADA system that has two processor units running and processing data simultaneously to ensure reliable operation. Should one unit fail the other will maintain uninterrupted plant control.

### 18.4.8 Communications

Communication between the Control Room and the MCCs will be by an Ethernet, fibre optic cable network. All vendor package equipment incorporating PLC control modules will be connected to this network.

### 18.4.9 PLC panels

Each sub-station will have one PLC controller with sufficient memory and I/O capacity for the instrumentation, control valves and motor and variable speed drive controls. Interface between the PLCs and the motor starters and VSDs will be by Profibus-DP and Simocode motor control and protection modules.

## 18.5 Engineering

### 18.5.1 Engineering design methodology

The standards and codes of practice used in the DFS establish the minimum requirements for design, fabrication, supply and installation of equipment and works for the Project. These standards and codes have been selected in an effort to use the latest proven technology wherever possible. In the designs particular attention has been paid to:

- Seamless progression from the study to detailed design for implementation
- Use of proven equipment
- Selection of equipment taking into account the site logistics
- Conformity with equipment being used in West Africa
- Prudent use of standby plant
- Maintenance and crane access.

Engineering designs were produced in 2D in Auto CAD and progressed to allow the following activities to be undertaken:

- Completion of all PFDs and PCDs
- Preparation of area Gas
- Completion of the plot plan and the plant layout
- Building plans
- Selection of major contractors and equipment vendors
- Preparation of the bills of quantities for all of the site activities other than piping which was taken to 80% with the remaining 20% being factored.

### 18.5.2 Specifications

All equipment and fabrication has to meet the latest statutory requirements and safety requirements required by the South African Mineral Act and Regulations, the South African Occupational Health and Safety Act and the requirements of the South African National Standards.

### 18.5.3 Contractors and vendors scope of works

All vendor equipment in the study was grouped into Engineering disciplines and 110 vendor packages covering over 2 000 items were identified. Similarly site works

identified 9 contract packages that are needed for the implementation of the Project.

## 18.6 Infrastructure, services and ancillary facilities

### 18.6.1 Administration and general areas

All architectural buildings will be prefabricated structures on concrete raft foundations.

### 18.6.2 Messing and catering

Messing and canteen facilities will be provided at the mine village and the service will be provided by a management company.

### 18.6.3 Training and induction

All site personnel will have to attend site induction. Contractors will have to cover the costs for their labour to attend the induction but the Project will cover the induction costs.

### 18.6.4 Electrical power

Electrical power for the site will be provided by a 32 MW hybrid HFO and diesel fuel power station with smaller diesel generators as back-up in strategic areas of the plant and at the mine village.

### 18.6.5 Mining fleet and plant vehicles

The mining fleet and plant vehicles will be owned and operated by the Project and maintenance will be contracted out. The vehicles required by the EPCM contractor have been included within the disbursement costs of the EPCM estimate.

### 18.6.6 Fuel storage

The HFO and diesel fuel tank farm and facilities for the power station will be owned by the Project and operated by the power station contractor whilst the diesel fuel tank farm for the mining fleet will be supplied and operated by the fuel company. All fuel will be trucked to site by the fuel company.

### 18.6.7 Reagent storage

Dry reagents will be stored within a steel clad warehouse.

### 18.6.8 Plant area

Steel clad buildings within the plant area will use natural ventilation as much as possible. The plant workshops will be designed to support a gantry crane. All structural steel work within the plant area is designed for purpose and will be supplied fabricated and painted and site erected.

Concrete foundations, bases and structures have been designed to meet the geotechnical conditions on site and propriety linings will be used in areas subjected to abrasion or chemical attack.

Plant offices, laboratories change houses and security buildings will be prefabricated structures on concrete raft foundations and the MCC buildings will be concrete frame with concrete block brick infill as these buildings will have cable basements. The MCC buildings will have fabricated steel roof trusses and IBR or similar roofing.

### 18.6.9 Mine area

The mine fleet workshop will be a steel clad building on a concrete base. This building will have two gantry cranes. Two sites have been identified for the magazines and these structures are to the supplier's specifications.

### **18.6.10 Potable water and sewage treatment plants**

Potable water will be supplied from boreholes that have been tested and are suitable for human consumption. Water treatment will comprise filtration and chlorination.

In total three sewage plants will be installed, two for the plant and mining area and one for the mine village.

### **18.6.11 Roads and drainage**

The project area covers the existing regional road and provision has been made to divert this road around the mine site. The bypass road has been designed for a life of 10 years to accommodate light traffic. Suitable material for the roads has been identified within the Project area. Roads within the plant will be 4 m wide and of gravel construction. The haul road will be 20 m wide and will be constructed by Mining and will have to support the 140 t mine haul trucks.

All stormwater run-off within the plant and from the fuel storage depots is considered to be polluted and will be channelled to separate pollution control ponds. Spillage within these areas will be contained within bunded areas and returned to source by dedicated spillage pumping systems, as a policy of zero discharge has been adopted.

### **18.6.12 Housing and community buildings**

The mine village will be built from prefabricated structures and this village will initially be used as the construction camp. Construction housing can be augmented by the existing CEMOB housing if necessary, although the intention is to demolish these buildings once operations commence. Other structures within the mine village will be the recreational and dining facilities which will also be prefabricated.

### **18.6.13 Communications**

The site will be provided with a satellite communications system and full wi-fi coverage for computer, internet and Skype communications and networking. There is also cell phone coverage in the area and construction crews will provide their own radio communications for use within the construction area.

### **18.6.14 Fire protection**

The Project is classified as a low risk industrial facility and will be more completely defined in the implementation phase. Budget allowance has been made for the provision of a fire water system and a fire tender in addition to the statutory placement of fire extinguishers.

### **18.6.15 Medical facilities**

Provision has been made for one clinic within the plant area and another outside. These clinics will be equipped and staffed to be able to stabilise patients for medical evacuation by road or helicopter to Dori or Ouagadougou where there are better equipped facilities.

### **18.6.16 Human resources**

Human resource management during construction will be managed by the individual contractors with overall supervision by the Project.

## **18.7 Logistics and route survey**

### **18.7.1 Route survey**

The preferred route to site will be by sea to either Tema or Takoradi in Ghana and then by road to site via Kumasi in Ghana to Ouagadougou in Burkina Faso and to site via Dori and Falagountou. The roads through Ghana are congested but the

authorities are familiar with the movement of abnormal or oversized loads and will provide the necessary escorts.

The roads in Burkina Faso present little in the way of problems up to Dori. From Dori to site the roads to site are gravel. In several places the approaches to river crossings are very steep and will have to be filled to accommodate the larger low-beds. During the rains there are occasions when the roads in this section are impassable. However, this situation only lasts for a few days each year and can be managed.

### 18.7.2 South African ports

Two ports will be used for the export of goods from South Africa and these are Durban and Richards Bay. Both ports are fully equipped to handle containers but Richards Bay is best placed for out-of-gauge loads.

### 18.7.3 West African ports

The most convenient ports in West Africa are at Tema and Takoradi in Ghana. Tema is the more congested of the two but it is more regularly served by the international shippers and will be the port of choice for containers and flat racks. Takoradi is not a deep water port but is ideal for break-bulk shipments on smaller charter vessels. Both ports have adequate cranes and most shipping lines have sufficient stock on board to cope with the project loads.

### 18.7.4 Burkina Faso clearance procedures

The necessary procedures for the import of goods into Burkina Faso have been investigated. These procedures are in place but can be lengthy and a period of 14 days should be allowed from arrival at Ghanaian Port to eventual de-stuffing on site.

### 18.7.5 Freight forwarding agents

Bids were received from several international freight forwarders and UTi have been selected as the preferred freight forwarders for the Project as they were the lowest bid. UTi is represented by All Ships in Tema and All Ships have its own transport fleet, cranes and container handling equipment.

## 18.8 Project implementation

GRD Minproc has been instructed to commence the detailed design for the Project for a period of four months up to the end of October 2007. At this stage it is assumed that the project will be completed on an EPCM basis although there is a possibility that it may be awarded as EPC. The key elements of the implementation philosophy are

- The project execution strategy and form of contract
- The initial detailed design period
- Organisations and resources
- Safety
- The project management plan.

### 18.8.1 Project objectives

The objectives of the Project are to establish a safety culture of zero HSEC incidents to enable the Project to be completed on schedule and within budget so that the ore reserves at the EMZ can be mined and processed economically and with minimum disruption to the local population and the environment.

These objectives will be achieved by using the best HSE practices, proven project management systems and design and construction methods that have been developed through extensive experience gained from similar projects.

### **18.8.2 EPCM model**

The EPCM model relies upon close cooperation between the Owner and the Contractor and allows the Owner to have considerable input into the design and construction of the project as well as dictating the level of risk.

### **18.8.3 Project implementation scope and strategy**

The EPCM Contractor will be appointed to undertake the engineering, procurement, construction and management of the Project and will be aided by the Owner's team. All procurement will be managed by the EPCM Contractor for and on behalf of the Owner. The owner will be responsible for the overall direction of the project, all social and environmental aspects, all financial dealings and compliance with local rules, regulations, taxes and duty requirements.

### **18.8.4 Contracting strategy**

Several items of plant and services have been identified as having to be secured early in the project implementation in order to meet the scheduled dates and these are highlighted within this report. During the preparation of the DFS several contractors and vendors were identified as Preferred and these company's will be involved with the detailed design phase of the project.

### **18.8.5 Project organisation**

Detailed design and procurement will be carried out in the offices of the EPCM contractor with the support of an owner, representative or team. As the project implementation progresses selected personnel will be transferred to site for construction and eventual commissioning of the facilities.

### **18.8.6 Roles and responsibilities**

Responsibility for the successful implementation of the Project will be managed and directed by the following:

- The Project sponsors
- The Project steering committee
- The Owner's team
- The EPCM contractor

### **18.8.7 Health and Safety, Environmental and Community**

A comprehensive and site specific HSCE Management System will be put in place to ensure that the project is implemented in a safe and responsible manner, recognising the impact that it may have on the community and the environment. All site contractors will be required to subscribe to this system.

### **18.8.8 Project controls**

The EPCM Contractor will use proven systems to schedule and track the progress of the project through all of its stages. The EPCM Contractor will control and report on all costs and commitments as well as preparing forecasts that will highlight any deviations from the budget.

**18.8.9 Procurement and contracting**

The EPCM Contractor will call for tenders for goods and services and adjudicate all bids received. Orders and contracts will be placed after consultation with the Owner. Procurement and contracting encompasses the expediting and transport of goods to site.

**18.8.10 Construction**

The EPCM Contractor will establish a site based management team representing safety, site management and all the relevant engineering disciplines for the direction and management of all construction activities.

**18.8.11 Commissioning and handover**

Commissioning will be carried out by the Owner’s team assisted by a commissioning team set up by the EPCM Contractor.

**18.8.12 Schedule information**

Selected project milestones are shown in Table 18.2. Figure 18.2 provides the project milestone schedule.

Table 18.2 Selected Project milestones

<b>Selected Project milestones</b>	
<b>Task</b>	<b>Date</b>
Grinding mills contract award	31 May 2007
Commence detailed design	02 July 2007
Award the mine village and resettlement housing contracts	14 September 2007
Award the contract for the powerstation	14 September 2007
Award the mining fleet purchase order	28 September 2007
Commence building the mine village and resettlement houses	08 November 2007
Establish the contractor’s lay-down area	14 January 2008
Commence the plant earthworks	03 April 2008
Complete the mine village houses	27 June 2008
Commence excavation of the off channel storage facility	31 October 2008
Complete the relocation village houses	16 January 2009
Commission the powerstation	21 April 2009
Commence mining activities	02 September 2009
Commence ore commissioning	27 October 2009

**18.9 EPCM proposal**

The EPCM estimate prepared for the DFS has been based upon the GRD Minproc rates for the third quarter of 2007. The scope of services covers the provision of all of the necessary engineering, procurement, construction and management resources and systems required to bring the Project into production on time and within budget. The EPCM budget is shown in Table 18.3.

Table 18.3 Essakane Gold Project – EPCM budget

Essakane Gold Project – EPCM Budget			
Discipline	Manhours	Cost (ZAR)	Cost(US\$)
Johannesburg office	114 058	70 918 970	9 849 857
Site Office	98 880	34 005 684	4 723 012
Disbursements		7 568 718	1 051 211
<b>Total Budget</b>	<b>212 938</b>	<b>R 112 493 372</b>	<b>\$ 15 624 079</b>

## 18.10 Environmental and social impact assessment

The Environmental and Social Impact Assessment (ESIA) has been prepared by Knight Piesold (KP) and rePlan and includes baseline data of the relevant environmental impacts associated with the Project and the mitigation measures required to minimise the impact of the Project upon this baseline.

The ESIA has been prepared in accordance with Essakane's commitment to corporate responsibility and meets the requirements of the Burkina Faso Government, the World Bank and the Equator Principles by which the project is defined as being Category A.

### 18.10.1 Legal review

The ESIA has been compiled in accordance with the laws and regulations of Burkina Faso of which there are over 400 relating to the environment. The most pertinent of these include the creation of an Ecological Debt for the use and exploitation of natural resources for monetary gain. In addition there are establishment taxes, annual fees and a variable tax related to pollution emissions.

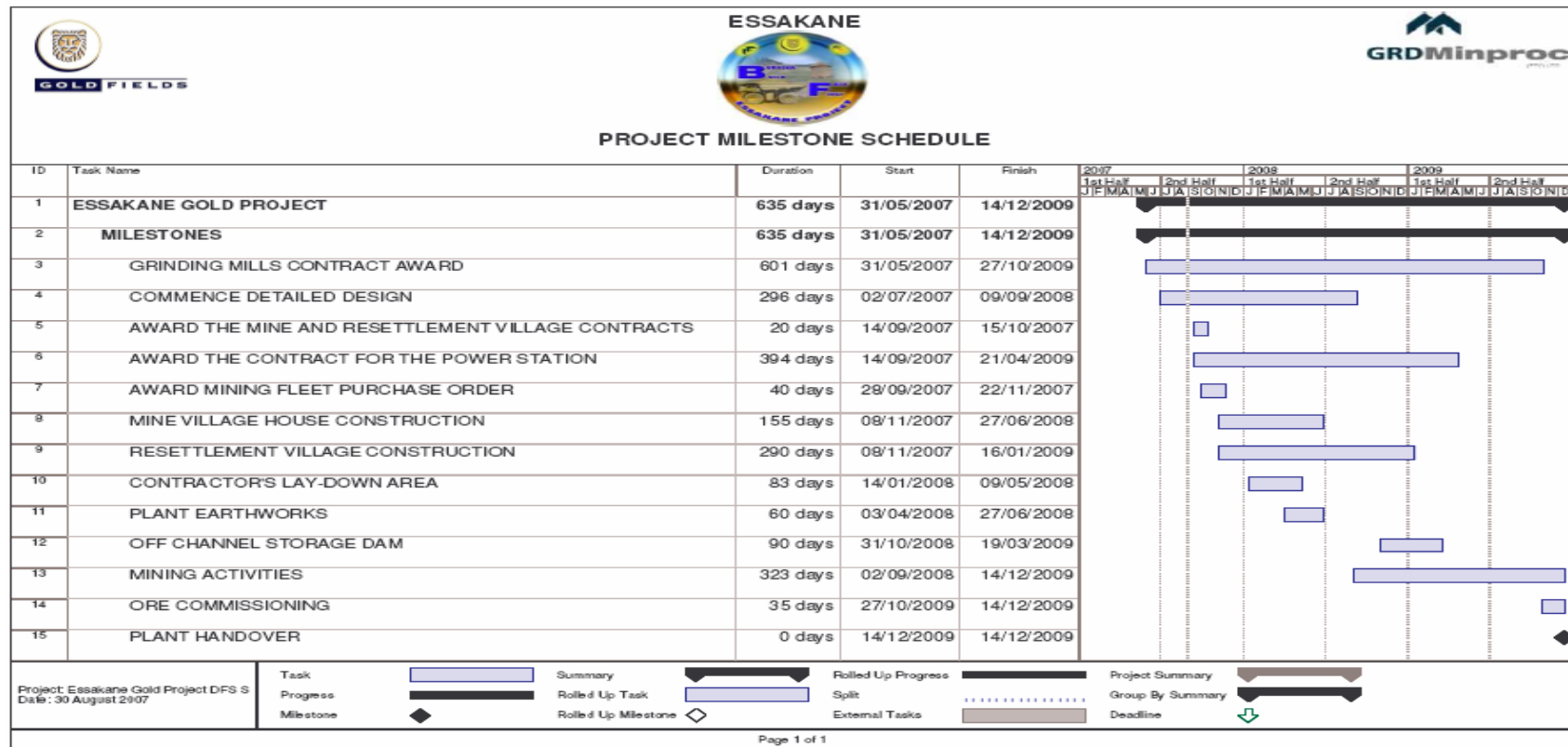
The development of the Project will have important beneficial impacts upon the region and in recognition of this the Mining Act aims to promote investment in the mining sector whilst ensuring that the environment is protected. To this end draft legislation makes provision for the establishment of a Mine Site Reclamation Fund, statutory reporting requirements and certain public health and safety regulations.

Relevant Burkina Faso legislation includes:

- The Mining Act
- The Forestry Act
- The Agrarian and Land Re-organisation Act
- The Water Management Act
- The Pastoral Act.

Burkina Faso is a democratic country with an Executive represented by the President and National Assembly. The country has ten regions and the footprint of the Project straddles the border between the Oudalan and Seno Provinces. The Ministry of the Environment and Living Standards is the leading environmental management ministry, but the Ministry of Mines and Quarries also deals with various aspects of environmental management.

Figure 18.2 Project milestone schedule



The World Bank Group has developed environmental and social safeguards that include:

- An environmental assessment to ensure that the project is environmentally sound
- Protection, maintenance and rehabilitation of the natural habitat
- Reduction of deforestation
- Safety of dams and tailing storage facilities that includes pre-qualification and inspection of contractors and works
- Protection of international waterways against pollution and effects upon existing riparian rights
- Planned resettlement of people affected by the project and protection of indigenous people and cultural property
- Prohibition of child labour
- The IFC has adopted the following eight Performance Standards that must be met in its project financing:
  - Social and environmental assessment and management system
  - Labour and working conditions
  - Pollution prevention and abatement
  - Community health, safety and security
  - Land acquisition and involuntary resettlement
  - Biodiversity conservation and sustainable natural resource management
  - Indigenous people
  - Cultural heritage

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- Social and environmental assessment and management system
- Labour and working conditions
- Pollution prevention and abatement
- Community health, safety and security
- Land acquisition and involuntary resettlement
- Biodiversity conservation and sustainable natural resource management
- Indigenous people
- Cultural heritage

The Equator Principles are voluntary social and environmental guidelines to which 45 banks and financial institutions adhere. These guidelines set limits on emissions, set standards for erosion control and pollution as well as guidelines for safety and

training. Burkina Faso is party to several international accords and treaties that, amongst others govern the movement, use and disposal of hazardous materials and these have to be observed.

### 18.10.2 Environmental baseline information

The environmental study for the DFS covered an area of 50 km<sup>2</sup> and included the surface mine, its infrastructures, access roads and the resettlement villages. The river course of the seasonal Gorouol River is located within this area.

The site is within the Sahelian climate zone and has a tropical type of climate that is subject to a short wet season of less than four months. During this period the entire annual rainfall of about 400 mm is experienced. The main wind is the dry Harmattan for most of the year and this reverses direction during the wet season. Temperatures range from 46 °C to 10 °C with humidity from 98.5% in the wet to 0.5% in the dry with extremely high evaporation rates.

Surface water and stream flow is confined to the wet season from June to September. Siltation of any river-based water storage facilities is extremely rapid due to the torrential nature of the rainfall.

Field hydrological drilling results indicate that the majority of the groundwater occurs within the first 60 m below surface and that there is a minor aquifer system underlying the site. Groundwater is generally of good potable quality but elevated levels of arsenic have been recorded within boreholes that intersect the EMZ ore body. A total of 25 dewatering boreholes around the surface mine will yield 6m<sup>3</sup> / hr per borehole and will provide 100 000 m<sup>3</sup> per month.

The overburden is considered to be non acid generating with tests indicating pH above 7.5. Run-off from the upper and lower saprolite is likely to contain arsenic at levels above those recommended by WHO for drinking water.

There is very little in the way of vegetation at the site as a result of the activities of the Artisanal miners. In general seven vegetation classes can be encountered in the region. Similarly there is little in the way of wild life mammals and reptiles although there are several species of birds in the area.

Five types of soil were identified within two classes. They all present limiting factors for agricultural activity of which the most important are rooting conditions, oxygen and erodibility. The soil types will have an important role to play in the selection of the resettlement sites.

Agriculture is very important in the Sahel but yields are low to mediocre. All land resources belong to the whole village under the control of the Chiefs. Animal husbandry is practiced by many households but the area is open to herds from Mali and Niger which complicates the livestock numbers. The majority of the population resident in the village of Essakane give Artisanal mining as their primary source of income and there are both permanent and seasonal miners in the area. Mining, as currently practiced, is not considered to be the livelihood of choice as the work is dangerous and has associated health problems.

The majority of the population of the Sahel is Muslim although there are Christians and Animists.

### 18.10.3 Potential environmental and social impacts

Baseline information has been gathered during the 2005 season and the potential environmental and social impacts have been investigated for each phase of the Project: (i) Construction, (ii) Operation, and (iii) Closure.

During the construction phase the Gorouol River weir and off channel storage will impact on a riverbank forest although careful planning is expected to reduce the high significance to moderate. There will also be loss of revenue from Artisanal mining and a certain loss of land. The positive impact of the Project will be an increase in employment in the area, increased opportunity for economic activities and an expanded skills database.

During the operational phase the majority of impacts are considered to have moderate significance and can be mitigated to low significance with the introduction of management actions. The positive socio-economic impacts will continue through the operational phase.

Impacts with closure involve the activities that will be required to rehabilitate the land to a stable post-mining status. It is envisaged that the overall impact on the environment will be positive in the long term once rehabilitation is completed. Work has been done on final rehabilitation of the surface mine and the current decision is towards the creation of a surface mine lake by allowing the Gorouol River to flood the surface mine on closure, but further study is recommended.

The tailing storage facility will have a final height of 22 m and a capacity of 60 Mt. The outer slopes will be clad with selected rocks during operations and the beach surfaces will be allowed to consolidate. Upon closure the TSF penstock will continue to decant water from the TSF and this water will be directed into the surface lake. Secondary recoverable water will be collected in a lined catchment pond and allowed to evaporate.

The overburden storage facility will be about 50 m high. This facility will be stabilised during the operation phase and on closure it will be re-contoured to a stable landform. The bulk water storage facility will be rehabilitated after closure whilst the Gorouol River weir and the off channel storage will remain for use by the local community. The plant and ancillary facilities will be demolished and the entire area ripped and graded to blend in with the surrounding topography and vegetation.

### 18.10.4 Environmental management program

An Environmental Management Program is required in terms of the laws of Burkina Faso and has to address the following:

- Social management and adjustment program
- Emergency response and contingency plan
- Resettlement action plan
- Environmental monitoring
- Closure plan.

### 18.10.5 Public consultation

The social impact assessment for the DFS was undertaken by rePlan, a Canadian registered independent company. Extensive opportunities were provided for all stakeholders to express concerns and expectations regarding the Project.

Essakane has committed to engaging the public throughout the lifetime of the mine and has drawn up and implements a Public Consultation and Disclosure Action Plan to cover:

- Goals and objectives
- Background
- Project stakeholders
- Engagement activities
- Results of engagement activities

## 18.11 Social, relocation and resettlement

The Project may result in the displacement of 11,563 people living in 2,562 households and an initial draft Resettlement Action Plan (RAP) has been developed in consultation with the community to address the resettlement of these people. The RAP describes the policies, procedures, compensation rates, mitigation measures and schedule for resettlement.

The approach to involuntary resettlement is consistent with the IFC's performance standards on Environmental and Social Sustainability and will adopt a collaborative approach involving the Government of Burkina Faso and the affected communities.

### 18.11.1 Legal and institutional framework

Burkina Faso is a democratic unitary state and the national government is divided into three main authorities: (i) The legislative authority; (ii) The executive authority; (iii) The judicial authority. Government policy is administered by regional and municipal authorities throughout the country.

There is a well developed legal framework concerning the social and environmental impact of mining activities and the Project has committed to adhere to these and to best international practices.

### 18.11.2 Baseline conditions

The baseline conditions have been established through reviewing existing information and from information collected by the social assessment team working within the Project footprint area. During this period a good understanding of the baseline social conditions, as well as a comprehensive database of people and dwellings has been established.

The Project footprint covers eight villages that will have to be resettled and involves 2 562 households and 11 563 people. There are two schools in the area offering limited education facilities and only about 5% of the community possess an employable skill.

Health facilities are very basic and the area is served by a small clinic and a maternity home with the nearest hospital in Dori which is 70 km away.

### 18.11.3 Project impacts

The people who have been physically displaced are entitled to either resettlement or compensation. Whilst Essakane has tried to minimise the impact of the Project upon agricultural land there is a total of 337 hectares that will be affected as it will be within the fenced area for the Project.

In addition to the loss of agricultural land the Project will also affect about 270 fixed small businesses. However, it is assumed that once they are re-established they will benefit from the economic boom that the Project will bring to the area.

#### **18.11.4 Public engagement**

Essakane has embarked on a comprehensive public consultation and disclosure plan to improve decision making through dialogue with legitimate stakeholders. This process has highlighted the concerns of the local people that range from expectations of employment to the fear that they will lose their livelihoods due to the lack of appropriate skills.

People and groups who will be resettled and those who will be affected by the resettlement program have been identified and will be kept informed of developments by public meetings, small group feedback sessions and handouts. This policy will be implemented by a Resettlement Consultations Committee made up of representatives of all stakeholders.

#### **18.11.5 Compensation strategy**

Compensation policies and procedures will be defined in an open and transparent manner with the intention of restoring and improving the livelihoods of the affected people with reward for self reliance and self help. Only those whom have been identified as having a legitimate interest in respect of immovable assets such as structures, crops and land use will be eligible for compensation.

#### **18.11.6 Resettlement package**

Householders identified for resettlement will select their own resettlement house in accordance with pre-defined guidelines. The underlying principle is that there will be a minimum size of house that is made available. Above this there are standard buildings of various sizes and households will be able to choose the size that corresponds to the size of the buildings and facilities that they presently own. In cases where people elect to “down size” there will also be cash compensation for the balance.

Traditionally buildings in the area have been constructed from sun baked mud bricks and these will now be upgraded to sandcrete block on raft foundations and they will have galvanised corrugated sheet steel roofs. Walls will be cement “bagged” to render them waterproof. Resettlement houses will not have plumbing or electrical wiring, but will be provided with toilets and septic tanks on a communal, shared and in a few cases, on an individual basis. Water supplies will be from boreholes and hand operated pumps located conveniently within the village complex.

The resettlement villages will be serviced by a clinic and two schools with provision for staff houses. There is also provision for various religious buildings.

#### **18.11.7 Relocation package**

Relocation compensation is where eligible owners are paid cash for their assets and this will be determined on a case by case negotiated basis although indications are that about 30% of the affected households will opt for the cash payout.

#### **18.11.8 Entitlement processing**

Studies have identified, recorded and valued all assets eligible for replacement or compensation. If a person is eligible for resettlement or relocation compensation there will be a sign off procedure that will be followed. This procedure will ensure that the person meets the requirements and understands the level of compensation that will be offered.

Once owners have vacated their houses and relocated to the new houses they will be given a period during which they can recover materials from the vacated buildings. After this the old buildings will be demolished.

Those receiving cash compensation can choose how they would like the payment made and financial management advice will be provided.

#### **18.11.9 Livelihood restoration and community development program**

The main objective of the program is to enhance food production for the affected population. The program will endeavour to address some of the traditional constraints by the introduction of adult learning centres and market gardening amongst others. There will also be economic opportunities presented by the establishment of the Project in the way of employment and an increase in the spending power of the populace.

#### **18.11.10 Management of grievance and disputes**

As the majority of the affected people are not literate, grievance hearings will be held daily on a face to face basis by a committee established for this purpose. There will be an arbitration mechanism that will rule upon any appeals.

#### **18.11.11 Organisational framework**

The RAP will be implemented by a team of professionals comprising senior members from Essakane and Government Ministries.

#### **18.11.12 Monitoring and evaluation**

The RAP provides for the monitoring of the resettlement program to assist and improve the living conditions of the affected people. This will verify that all commitments within the RAP are met.

### **18.12 Permitting**

Four conditions have to be fulfilled prior to Project implementation:

- Acceptance of the ESIA (Environmental and Socio-Economic Impact Assessment) through a “positive notice” (Avis favorable) from the Burkina Faso Minister of Environment;
- Grant of a “Mining convention” by the Burkina Faso Government
- Grant of a “Mining Permit” by the Burkina Faso Minister of Mines
- Agreement with local populations on resettlement plans and process

#### **18.12.1 Acceptance of the ESIA**

In agreement with Burkina Faso’s statutory requirements and International best practices, a completed Environmental and Socio-economic Impact Assessment was submitted to the Burkina Faso Minister of Environment on 8 August 2007. The ongoing approval process is:

- Review of the document (up to 15 days)
- Public hearing (up to 60 days, including reporting)
- Review of the case by the Minister of Environment (up to 15 days)
- “Positive Notice” given by the Minister of Environment

Approval of the ESIA is a critical step in the delivery of the Mining Permit and project cannot proceed without it. Essakane anticipates that approval should be obtained in early November 2007 provided the review process proceeds according to statute.

### 18.12.2 Granting of a Mining Convention

The mining code proposes a ‘Standard Mining Convention’ which acts as a stability agreement. The Convention describes the Governmental commitments, operational tax regime and obligations of the company to Burkina Faso. Once executed, this Convention cannot be changed without the mutual agreement of both parties. If tax law changes are promulgated, the mining company can choose to adopt them (if deemed more advantageous) or stay with the current terms of the Convention.

The approval of the Convention requires the approval of the Cabinet. Typically approval of the standard Convention can be accomplished in a short period of time after the approval of the ESIA. However, certain points require further review and clarification in the instant case:

- Insure tax, legal and currency stability
- Clarify terms of application
- Avoid potential situations of double taxation
- Include UEMOA mining code into the mining convention

Essakane has started discussions on these various aspects and an initial form of a Mining Convention will be submitted to the Minister of Mines shortly.

### 18.12.3 Granting of a Mining Permit

Essakane currently holds seven exploration permits in the Project area. To start construction of the Project and mining, these exploration permits need to be converted into a Mining Permit. Burkina Faso requirements are:

- Realization and submittal of a Detailed Feasibility Study
- Approval of a ESIA through a “positive notice” from the Minister of Environment
- Agreement on a “Mining Convention” with the Government

With the fulfilment of these conditions a Mining Permit will be issued by the Minister of Mines. The Permit will be issued to a new legal entity which would be owned 10% by the Burkinabe government.

### 18.12.4 Consultation with affected Burkinabe citizens

While consultation and agreement with the affected citizens is not a legal requirement to start work, it has been the policy of Essakane to operate in a transparent and supportive manner, enabling free and informed consent regarding Project activities by all stakeholders. Essakane has continued to build on the good relationships previously established by Orezone. In that time, numerous economic, educational, and health initiatives have been emplaced, with good reception by local citizens and the Burkina Faso government as well.

Essakane engaged formal consultations with the local population and Regional and National authorities in July 2007. Overall, Essakane represents that the Project dynamic is good and a strong support is evidenced at all levels within Burkina Faso for the Project.

## 18.13 Geotechnical and hydrogeological

GRD Minproc contracted Knight Piesold and GCS (a South Africa based geohydrology company) for the geotechnical design and water balance aspects of

the DFS. Investigations covered site geology, seismicity, foundation conditions and borrow areas.

**18.13.1 Geotechnical**

The studies show that subsurface conditions vary throughout the site, but in general the site is underlain by a few metres of primarily stiff to dense fine grained soils or completely weathered saprolite. The depth to fresh bedrock is generally more than 30 m below ground level and the water level is at about 25 m below surface.

Table 18.4 presents the geotechnical recommendations for the surface mine design. These data were based on RC and oriented diamond core drilling within the surface mine footprint.

Table 18.4 Geotechnical parameters for DFS mine design

<b>Knight Piésold Geotechnical Recommendations for Surface Mine Design</b>						
<b>Surface mine design parameter</b>	<b>West wall</b>			<b>East wall</b>		
	<b>Saprolite</b>	<b>Saprock</b>	<b>Fresh rock</b>	<b>Saprolite</b>	<b>Saprock</b>	<b>Fresh rock</b>
Bench height (m)	6	6	10	6	6	10
Batter angles (degrees.)	43	72	87	43	66	82
Bench stack height (m)	42	36	60	42	36	60
Slope height (m)	42	36	172	42	36	172
Geotechnical berm width (m)	13	13	16	13	13	16
Overall slope angle (degrees)	30	45	58	30	45	58

Foundation conditions were assessed at each of the following sites:

- Tailing storage facility (TSF)
- Overburden storage facility (OSF)
- Process plant area
- Bulk water storage dam (BWSD)
- The Gorouol River weir
- The off channel storage facility (OCSF)
- The surface mine flood protection wall and Gorouol River diversion
- The mine village site
- Access roads, borrow areas and erosion control facilities

Foundation conditions are adequate for conventional foundations but further tests are required for the heavily loaded foundations to set the design basis. The Gorouol River weir will require a compacted clay cut-off wall but is not expected to require grouting of fractured bedrock. The allowable bearing pressures for foundations are typically 150 kPa to 200 kPa at a depth of 0.5 m to 1.5 m.

Adequate borrow sources have been identified for the soil, rock and aggregate requirements for the Project. Planned activities such as the OCSF would provide the material for the surface mine flood protection wall and the TSF starter walls.

Clean sands have not been identified for construction and these will have to be processed from the identified aggregate sources.

Erosion protection of walls and embankments will be provided by rock rip-rap on the slopes adjacent to potential water flows. All rip-rap will be underlain by heavy, non-woven geotextile or a sand-gravel bedding. Long term closure covers will consist of laterite and ferricrete gravels and rocks.

The soils have low levels of sulphates and a Type I or II cement can be used in the concrete foundations.

### **18.13.2 Additional geotechnical investigations**

The DFS has made provisions in the capital cost estimate for additional geotechnical testwork in the following areas:

- The mine village site
- Access roads
- Selected sites for process plant
- The Gorouol river diversion
- The off channel storage facility, BWSD and TSF

### **18.13.3 Regional hydrogeology**

Geophysical surveys were conducted to identify site drilling targets and various boreholes were drilled across the project area. These indicated that the majority of the groundwater occurs in the first 60 m below ground level. The drilling logs indicate that the site is underlain by a minor aquifer system with discrete areas of enhancement. The ground water located up-gradient of the ore-body is of good potable quality.

### **18.13.4 Mine hydrogeology**

The ground water modelling indicates that aquifers within the surface mine can be dewatered by out-of-mine dewatering pumps and that a total of 25 boreholes, with an average yield of 2 litres/s will be required. The dewatering volume is predicted to be 100 000 m<sup>3</sup> per month but will decrease over time.

Upon mine closure the rate of evaporation will exceed the direct rainfall and groundwater seepage and KP recommends that the surface mine lake is artificially recharged by diverting the Gorouol river into the surface mine as this will ensure that the surface mine lake is of good quality.

### **18.13.5 Preliminary overburden characterisation**

Overburden leach tests are ongoing but there is little or no likelihood of acid rock drainage occurring on site although there may be arsenic enriched leachate associated with the early mine overburden.

### **18.13.6 River hydrology and water resources**

The Project is located in the north east of Burkina Faso and the temperature ranges from 10 – 46 °C with evaporation rates of 3 000 mm per year. The mean annual rainfall is 450 mm with an estimated 100 year maxima of 143 mm in a 24 hour period.

The mean runoff in the Gorouol River is conservatively estimated to be 91 million m<sup>3</sup>/year and is concentrated in the three months July to September.

### 18.13.7 Gorouol River water supply

The water requirement of the mine from the Gorouol River is estimated at 2 million m<sup>3</sup>/year and that for the local population at 0.5 million m<sup>3</sup>/year. Several options were considered in the DFS before deciding to build a diversion weir on the Gorouol River that will divert water, during the wet season, into an off channel storage facility. The weir will have a storage capacity of 1.37 million m<sup>3</sup> but will decrease in time by siltation and annually by evaporation. This water will be available for use by the local population.

The off channel storage facility will have a storage capacity of 3.3 million m<sup>3</sup> and this will ensure the 2 million m<sup>3</sup>/year supply to the mine. By using off channel storage the effects of evaporation are reduced.

## 18.14 Capital cost estimate

The capital cost estimate for the Definitive Feasibility Study has been prepared to an overall accuracy of +15% as at the third quarter 2007.

### 18.14.1 Plant, housing and infrastructure

To meet the accuracy required by the capital cost estimate the following designs and documents were produced for the plant, housing and infrastructure:

- Process block diagram
- Process flow diagrams and mass balance
- Process control diagrams
- Equipment list
- Electrical load list
- Electrical SLD and cable block diagrams
- Major pipeline list
- Plant general arrangement drawings
- Plant building drawings
- Plant layout
- Plant area bulk earthworks layout
- Mine village building drawings
- Resettlement village drawings
- Site plot plan
- Site bulk earthworks sketches

From these drawings and designs it was possible to generate specifications and bills of quantities and call for quotations for equipment and services from vendors and contractors. In the majority of cases multiple bids were received, however, for some important services only one bid was tendered despite going to several sources. Volume of work was generally cited as the reason for not tendering.

### 18.14.2 Owner's costs

Owner's Costs for the Project have been estimated for the period July 2007 to December 2009 (inclusive), after which time the project will have been commissioned and in operation. The estimate of the Owner's costs comprises the following items:

- Capital equipment including
  - mining equipment (all mobile equipment required for mining including that required after plant operation has commenced)
  - mining infrastructure (ROM pad, haul roads, explosives compound)
  - vehicle fleet (site light vehicle fleet and other vehicles such as ambulance, fire and emergency services trailers, busses and fork lift)
  - mine village furnishings (bedroom and lounge room furniture as well as club house and public area furniture)
  - Ouagadougou and mine offices furniture and IT equipment (office furniture and IT equipment for new employees at the Ouagadougou and mine site offices including photocopiers, printers and plotters and training room facilities)
  - operating systems (SAP cost including implementation as well as specific geological and mining software)
  - first fill (power station HFO, process plant grinding media and reagents and first month plant consumption)
  - insurance spares (spares for key process plant equipment identified by GRD Minproc)
  - operating spares (spares for plant and infrastructure equipment; spares for mining equipment already included in the mine equipment cost)
- Preproduction - three month period prior to the commencement of plant operation in which 6.5 million tonnes will be mined. This period will be used as the final step for in the training of local employees on mine equipment
- Social-economic and resettlement compensation – includes costs associated with the resettlement negotiations, various compensation packages, consultants, land acquisition, local community donations, programs and ventures (e.g., nursery), ISO 14001 implementation and project information
- Custom duties, fees and taxes – custom duties on initial project equipment, related fees and taxes on services provided by foreign organisations
- Insurances and legal services – marine, public liability, professional indemnity and other project insurances and legal fees
- Working capital – process and administration costs associated with 17 production days (plant handover to end of year). Mining costs for this period already included in preproduction cost.
- Other costs including
  - employee salaries and wages and Ouagadougou office
  - consultants and contractors
  - regional exploration and geology
  - Essakane camp
  - training, health, safety and security

Table 18.4 presents a summary of the Owner's costs and excludes any contingency.

Table 18.4 Summary of Owner's costs

Description	Cost US\$ X 10 <sup>6</sup>	Total cost US\$ X 10 <sup>6</sup>
- mining equipment	49.499	
- mining infrastructure	3.543	
- site vehicles	1.415	
- mine village furnishings	0.790	
- mine office furniture & IT	0.279	
- operating systems	1.614	
- first fill	5.808	
- insurance spares	3.594	
- operating spares	1.480	68.022
Preproduction		5.057
Socio-economic & resettlement		7.557
Custom duties, fees & taxes		4.434
Insurance & legal		3.809
Working capital		1.805
Employee salaries & wages		10.410
Ouagadougou office		1.290
Consultants & contractors		1.998
Regional exploration & geology		1.300
Training		1.205
Essakane camp		0.804
Health, safety & security		0.482
Total		108.173

### 18.14.3 Cost estimate summary

Table 18.5 presents a comparison between the capital cost of the PFS and the current DFS estimates. Table 18.6 shows the breakdown of the estimate for the Project by disciplines. Owners cost includes US\$ 6.7 million that will have been committed for mining commitment and is scheduled for delivery after project handover.

Table 18.5 Comparison of capital cost estimates for PFS and DFS

Description	PFS Estimate	DFS Estimate
	US\$ X 10 <sup>6</sup> (October 2005)	US\$ X 10 <sup>6</sup> (July 2007)
Water storage and infrastructure	15.3	14.3
Mine fleet	38.1	49.0
Mining other (Pre-prod & Infrastructure)	4.6	11.8
Process plant and ancillaries	79.3	111.1
Infrastructure, accommodation and roads	33.8	18.1
EPCM	26.5	15.6
Relocation costs	13.7	18.5
Power supply and infrastructure	36.9	36.7
Working capital	12.9	17.8
Overburden and tailing storage facilities	17.2	17.4
Owner's costs	13.3	21.9
Contingency	19.5	14.3
Total	311.1	346.5

### 18.15 Sustaining and ongoing capital

Sustaining capital has been assumed for replacement or upgrading of some capital items, such as light vehicles, IT hardware, survey equipment, in-mine dewatering pumps. In addition, an allowance has been made for Phase 1 processing capital, required in years 2 and 3 for the change in ore type to the plant. At this stage, the mine production schedule indicates a mine life of around nine years and, as such, it is not anticipated that any of the main mine fleet will require replacement.

Although a detailed replacement or upgrading schedule has not been developed, an annual allowance of US\$ 200 000, US\$ 300 000 and US\$ 100 000 for mining, processing and administration respectively has been assumed. In addition, part of the light vehicle fleet has been assumed to be replaced every three years and as such an allowance (US\$ 240 000) has been included in the administration figure. The allowances have reduced in the penultimate and final years.

The Phase 1 processing capital, totalling US\$ 0.6M, includes primary crushing modifications, a pre-leach thickener and additions to the gravity circuit in years 2 and 3 for the harder ore.

Table 18.6 Capital cost estimate by discipline

Discipline	Estimate US\$
Earthworks	40 313 277
Civilworks	13 454 637
Structural steelworks	5 778 048
Platework	13 643 084
Equipment	45 216 064
Piping	8 061 618
Electrical	45 038 367
Instruments	1 074 279
Buildings	13 320 189
Miscellaneous	5 392 811
Accuracy Provision at an average of 6.86%	13 124 719
Total cost	204 417 093
Owner's costs	108 173 000
EPCM fee	15 624 079
Contingency at 7% of total cost	14 309 197
Consultants	3 995 963
<b>TOTAL</b>	<b>346 519 332</b>

## 18.16 Operating cost estimate

GRD Minproc has developed operating costs for the mine, the process plant and administration and used the third quarter of 2007 as a base date. All costs are given in United States Dollars (US\$).

### 18.16.1 Mining operating costs

The operating costs for the mine have been based upon supplying 5.4 Mt of ore per year to the process plant over the life of the mine. These costs take into account the ore types that have to be mined and the overburden that has to be moved. Table 18.7 and Table 18.8 provide a breakdown of the unit mining operating costs by expense type for tonnes moved and ore mined, respectively, over the life of mine.

### 18.16.2 Process plant operating costs

The process plant operating costs have been based on a throughput of 5.4 Mt of ore per year and take into account the three predominant weathering types that have to be processed over the life of the mine.

The breakdown of these costs is shown in Table 18.9.

Table 18.7 Total tonnes mined unit cost - US\$ 500/oz design

US\$ 500MI (May 2007 Resource Model) TONNES MOVED UNIT COST OVER MINE LIFE											
Costs by Expense type	PreProd	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	TOTAL
	US\$/t	US\$/t	US\$/t	US\$/t	US\$/t	US\$/t	US\$/t	US\$/t	US\$/t	US\$/t	US\$/t
Lubrication	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.03	0.03	0.03	0.02
GET/Undercarriage/drill consumables	0.02	0.02	0.03	0.04	0.06	0.09	0.09	0.10	0.10	0.07	0.06
Fuel	0.21	0.24	0.27	0.31	0.34	0.40	0.41	0.47	0.51	0.56	0.34
Tyres	0.06	0.07	0.08	0.09	0.09	0.11	0.11	0.12	0.13	0.12	0.09
MARC variable	0.20	0.23	0.25	0.29	0.31	0.38	0.38	0.41	0.44	0.49	0.32
Grade control	0.12	0.11	0.11	0.10	0.11	0.11	0.09	0.16	0.19	0.05	0.11
Explosives & blasting accessories	0.00	0.01	0.05	0.11	0.18	0.23	0.25	0.25	0.25	0.06	0.14
Pre-split	0.00	0.00	0.00	0.00	0.01	0.02	0.02	0.03	0.03	0.00	0.01
MARC fixed	0.10	0.09	0.09	0.10	0.10	0.12	0.11	0.21	0.24	0.42	0.12
Fixed others	0.05	0.04	0.04	0.05	0.05	0.05	0.05	0.09	0.11	0.19	0.06
Labour	0.10	0.09	0.10	0.10	0.09	0.10	0.10	0.15	0.16	0.23	0.11
<b>TOTAL</b>	<b>0.88</b>	<b>0.92</b>	<b>1.05</b>	<b>1.21</b>	<b>1.36</b>	<b>1.62</b>	<b>1.64</b>	<b>2.01</b>	<b>2.20</b>	<b>2.21</b>	<b>1.38</b>

Table 18.8 Ore tonnes mined unit cost – US \$500/oz design

US\$ 500MI (May 2007 Resource Model) ORE MINED UNIT COST OVER MINE LIFE											
Costs by Expense type	PreProd	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	TOTAL
	US\$/t	US\$/t	US\$/t	US\$/t	US\$/t	US\$/t	US\$/t	US\$/t	US\$/t	US\$/t	US\$/t
Lubrication	0.08	0.10	0.09	0.09	0.10	0.10	0.10	0.06	0.05	0.27	0.09
GET/Undercarriage/drill consumables	0.10	0.12	0.14	0.19	0.31	0.35	0.40	0.23	0.21	0.61	0.24
Fuel	1.00	1.33	1.31	1.37	1.65	1.64	1.75	1.11	1.05	4.88	1.42
Tyres	0.29	0.40	0.40	0.41	0.45	0.45	0.45	0.28	0.27	1.01	0.39
MARC variable	0.96	1.24	1.22	1.28	1.53	1.55	1.61	0.97	0.91	4.21	1.31
Grade control	0.56	0.61	0.52	0.44	0.52	0.44	0.37	0.38	0.40	0.42	0.46
Explosives & blasting accessories	0.00	0.03	0.25	0.47	0.87	0.96	1.06	0.60	0.52	0.54	0.57
Pre-split	0.00	0.00	0.01	0.00	0.05	0.07	0.09	0.07	0.07	0.00	0.04
MARC fixed	0.49	0.48	0.43	0.44	0.49	0.49	0.49	0.49	0.49	3.62	0.50
Fixed others	0.22	0.22	0.20	0.20	0.22	0.22	0.22	0.22	0.22	1.66	0.23
Labour	0.49	0.51	0.47	0.42	0.46	0.43	0.43	0.35	0.33	2.01	0.44
<b>TOTAL</b>	<b>4.19</b>	<b>5.04</b>	<b>5.03</b>	<b>5.31</b>	<b>6.66</b>	<b>6.70</b>	<b>6.97</b>	<b>4.76</b>	<b>4.52</b>	<b>19.20</b>	<b>5.69</b>

Table 18.9 Process plant operating costs

	Oxide – Saprolitic ore US\$/tonne milled	Fresh – Arenite/Argillite ore US\$/tonne milled
Power	2.44	4.04
Reagents	2.22	3.57
Maintenance	0.58	0.78
Assay incl labour	0.06	0.06
Plant labour	0.24	0.24
Engineering	0.20	0.20
Tailing storage facility	0.00	0.00
Water	0.00	0.00
Other	0.06	0.06
<b>Total</b>	<b>5.82</b>	<b>8.96</b>

### 18.16.3 Overall operating costs

Table 18.10 shows the overall operating costs for the mine based upon a feed of 5.4 Mt/y of ore to the processing plant. No allowance has been made for escalation over the life of the project.

Table 18.10 Overall annual operating costs

Area	Overall annual operating costs (US\$x10 <sup>6</sup> )								
	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9
Mining	27.6	30.8	31.9	35.9	36.2	37.7	25.7	24.4	8.2
Process plant	33.2	36.3	41.8	46.7	49.6	50.8	51.0	51.0	22.5
General and administration	7.6	7.6	7.4	7.4	7.3	6.9	6.8	5.6	4.0
<b>Overall opex</b>	<b>68.4</b>	<b>74.7</b>	<b>81.1</b>	<b>90.0</b>	<b>93.1</b>	<b>95.4</b>	<b>83.5</b>	<b>81.0</b>	<b>34.7</b>
Ore tonnes milled x10 <sup>6</sup>	5.4	5.4	5.4	5.4	5.4	5.4	5.4	5.4	3.2
Operating cost (US\$/tonne milled)	12.7	13.8	15.0	16.7	17.2	17.7	15.5	15.0	10.8

## 18.17 Financial analysis

The financial analysis was completed by Essakane. All Project financial analyses used the US\$ 500/oz design shell reported on the May 2007 block model and were conducted on a pre-tax basis. Table 18.11 presents four gold and crude oil price scenarios that were considered in the analysis.

Table 18.11 Commodity price scenarios

Case	Gold price (US\$ per oz)	Oil price (US\$ per barrel)
1	580	50
2	460	40
3	650	60
4	720	80

LOM gold production is 2.507 million ounces over 8.6 years and averages 292 000 ounces per annum with peak production of 333 000 ounces in Year 6. The gold production profile is presented in Table 18.12. Based on Case 3 price parameters, cash costs average US\$ 321/oz over the life of mine and range from US\$ 275 to US\$ 373/oz.

Table 18.12 Gold production profile over life of mine

Year	Annual gold production									Total
	1	2	3	4	5	6	7	8	9	
Gold (koz)	260	288	316	274	303	333	330	321	81	2 507

Average LOM operating costs for Case 3 price parameters are as follows:

- Mining: US\$ 1.43 per tonne of material mined
- Processing: US\$ 9.12 per tonne ore processed
- G&A: US\$ 1.34 per tonne ore processed

The following initial project capital, on-going capital, working capital and reclamation cost estimates were used:

- Initial project capital - US\$ 340.0 million disbursed 7%, 47%, 46%, and <1% between Year -3, Year -2, Year -1 and Year 1, respectively.
- On-going project capital - US\$ 17.7 million 85% disbursed over the period from Year 1 to Year 4.
- Working capital includes US\$ 8 million of gold-in-process inventory which is recovered in Year 9 which is the final year of operation.

- Reclamation and closure costs of US\$ 12 million and US\$ 3 million, respectively, are evenly disbursed following the end of production in Years 10 to 12.

Calculations for total cash costs include all operating, capital, reclamation and closure costs. The government of Burkina Faso has a 10% free carried interest in the Project and the free carried cost per ounce is calculated on that basis.

A summary of financial results are presented in Table 18.13. Discount rates of 0%, 5% and 7.5% were utilized to develop pre-tax Net Present Values (NPV) and pre-tax Internal Rate of Returns for each Case.

**Table 18.13 Summary of financial results for four commodity price scenarios**

	Case 1	Case 2	Case 3	Case 4
Gold price (US\$/oz)	580	460	650	720
Oil price (US\$/bbl)	50	40	60	80
Ounces recovered (000 oz)	2,507	2,507	2,507	2,507
Average annual production (000 oz)	292	292	292	292
Cash cost (US\$/oz)	298	269	321	356
Total cash cost (US\$/oz)	447	418	469	505
Total Free carried cash cost (US\$/oz)	497	464	521	561
Pre-tax project IRR (%)	14.8%	5.8%	18.8%	21.5%
0% Pre-tax NPV (US\$ 000)	346 332	117 964	465 638	551 565
5% Pre-tax NPV (US\$ 000)	173 257	12 901	257 100	317 667
7.5% Pre-tax NPV (US\$ 000)	113 263	(22 610)	184 333	235 749

Essakane completed a +/-10% variance analysis on Case 3 to assess Project sensitivity to capital and operating costs. The results are shown in Table 18.14 and demonstrate that the Project is highly leveraged to operating costs (primarily cost of power) and less so towards capital. A US\$ 1 per barrel movement equates to approximately US\$ 2 per ounce change in cash operating costs for the Project. Figure 18.3 graphically shows the 0% pre-tax NPV sensitivities for Case 3.

Table 18.14 Financial analysis - Case 3 sensitivities

<b>Case 3 Sensitivities</b>						
<b>0% Pre-Tax NPV (US\$ 000)</b>						
		Initial CAPEX				
		90%	95%	100%	105%	110%
OPEX	90%	575 110	558 107	541 104	524 100	507 097
	95%	537 378	520 374	503 371	486 367	469 364
	100%	499 645	482 641	465 638	448 634	431 631
	105%	461 912	444 908	427 905	410 902	393 898
	110%	424 179	407 176	390 172	373 169	356 165

<b>5% Pre-Tax NPV (US\$ 000)</b>						
		Initial CAPEX				
		90%	95%	100%	105%	110%
OPEX	90%	340 979	325 462	309 945	294 428	278 910
	95%	314 557	299 039	283 522	268 005	252 488
	100%	288 134	272 617	257 100	241 583	226 066
	105%	261 712	246 195	230 678	215 160	199 643
	110%	235 289	219 772	204 255	188 738	173 221

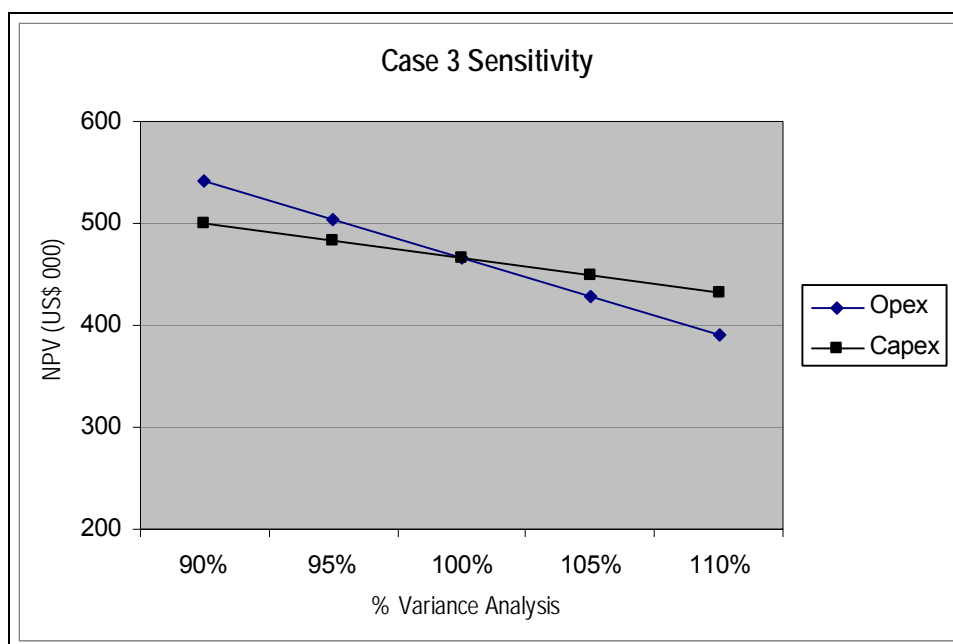
  

<b>7.5% Pre-Tax NPV (US\$ 000)</b>						
		Initial CAPEX				
		90%	95%	100%	105%	110%
OPEX	90%	258 751	243 898	229 046	214 193	199 341
	95%	236 394	221 542	206 690	191 837	176 985
	100%	214 038	199 186	184 333	169 481	154 629
	105%	191 682	176 830	161 977	147 125	132 272
	110%	169 326	154 474	139 621	124 769	109 916

<b>Pre-Tax IRR</b>						
		Initial CAPEX				
		90%	95%	100%	105%	110%
OPEX	90%	24.1	22.6	21.2	19.8	18.6
	95%	22.9	21.4	20.0	18.7	17.5
	100%	21.6	20.2	18.8	17.6	16.4
	105%	20.4	18.9	17.6	16.4	15.2
	110%	19.1	17.7	16.4	15.2	14.0

Figure 18.3 NPV sensitivities for Case 3 (0% pre-Tax)



### 18.18 Payback and mine life

The intention of the Project is to mine and process the EMZ at a rate of 5.4 Mt/yr. The expected life of the mine is 8.6 years and during this time a total of 2.507 Moz of gold will be produced.

The Payback period for the initial capital of the project on a pre-tax basis is estimated to be:

- Case 1: 4.2 years
- Case 2: 6.1 years
- Case 3: 3.7 years
- Case 4: 3.1 years

## 19 Interpretation and conclusions

### 19.1 Geology and Mineral Resources

Starting in January 2006, Essakane undertook a two phase LWL69M rapid cyanide leach assay program comprising:

- Re-assay of 28,640 historical drillhole samples
- Assay of step-out and infill drilling on the EMZ which was largely oriented HQ diameter diamond core drilling

The basis for the re-assay program was to validate the historical samples and to establish which of the previous assay methods had under- or over-reported gold assays. With some exceptions (e.g., high grades reported by BHP fire assay) it was found that most of the historical assaying had under-estimated the gold grades, and re-assaying increased the average grade of the 28 640 samples by 16.5% from 0.97 to 1.13 g/t. The proportion of LWL69M assays by May 2007 ultimately made up 42% of the total sample database. The 28 640 assay pairs were used to create remediation factors which were applied to 58 189 samples (42% of the sample database) which had not been re-assayed. The average grade of this low grade population increased by 18.4% from 0.38 to 0.45 g/t, which is below the economic cut-off grades and attests to the relatively low risk of the remediation process in the estimation of the EMZ Mineral Resources.

Essakane was the first company to focus on diamond core drilling of the EMZ and to introduce oriented core drilling. The information gained in this way provided significant improvements in the quality of the geological modeling. The improvements in geological modeling and gold assay combined to significantly increase the ability to estimate and classify the EMZ mineral resources despite the coarse gold sampling problem.

In Snowden's opinion, Essakane's drilling, sampling of RC and DD drillholes, sample preparation and analytical procedures, and quality control procedures, meet industry standards. The procedures used by Essakane for remediation of historical data has a number of precedents and, in Snowden's opinion, there was sufficient check assaying of new and old samples to confirm that the combined historical and new Essakane database is acceptable for the May 2007 Mineral Resource estimation.

The EMZ Mineral Resources were estimated by large panel Ordinary Kriging with post processing by Uniform Conditioning into SMUs with dimensions of 2.5 mE x 5.0 mN x 3 mRL. This is a small SMU dimension but is consistent with the general geology and gold distribution within the deposit. Snowden participated in discussions regarding SMU size and corresponding grade control grids, and concurs that the 8 x 8 m RC drill grid used and costed in this DFS is likely to be acceptable if supported by in-pit geological ore spotting. Snowden notes that a 5 x 5 m grade control grid is considered to be the optimum spacing for maximizing gold grade through selective mining.

### 19.2 Mineral Reserve estimates and mining

In accordance with the CIM (2000) definitions in NI 43-101, Essakane and GRD Minproc derived Probable Mineral Reserves from Indicated Mineral Resources. No Inferred Mineral Resources were used in the estimates or in the mine planning for the US\$500/oz mine design shell. The total Probable Mineral Reserves for the Project are 2 649 000 ounces of gold contained in 46.41 Mt at an average head grade of 1.78 g/t. These reserves yield a positive cashflow based on project economic

assumptions and detailed technical evaluations of mine production and ore processing.

Essakane and GRD Minproc used the Mineral Resource block model to develop detailed mining shapes. Appropriate mining dilution and ore loss was then applied.

Mine equipment has been sized according to the nature of the mineralization and the production schedule required to mine and process 5.4 Mtpa of ore for a mine life of 8.6 years. Mining will be carried out using three backhoe excavators and thirteen 140 t haul trucks. Mining of the EMZ is sequential, starting with the upper saprolite layers in the early years and progressing through to 100% fresh ore from Year 6. Overburden and tailings storage facilities will be rehabilitated with rock cladding and re-vegetated where possible. On closure the surface mine may become a surface lake that will be recharged from the Gorouol River annually during the wet season, but this closure concept requires further investigation.

### 19.3 Project risk assessment

GRD Minproc has completed a matrix based risk assessment (RA) to identify and evaluate project implementation risks associated with the Project as presented in the DFS. The RA does not, however, address issues such as political risk, financial risk (including taxation and revenues) and changes in government legislation. The RA is based upon the GRD Minproc Project Risk Management Plan.

GRD Minproc's RA by severity is presented in Table 19.1. Ratings less than 10 are classified as high risk, between 11 and 15 as medium risk and greater than 15 low risk. The assessment classifies the Project as medium to low risk with the cost of heavy fuel oil regarded as the highest risk because of impact on the operating costs.

Other potential high risk items identified by GRD Minproc are:

- training of Nationals as mining plant operators and the subsequent effect that this could have on mine productivity
- the security of water supply for the process plant.

A total of 17 medium risk items have been recognized, of which the most significant are:

- granting of the mining convention
- the supply of aggregate for civil works
- logistics
- crainage
- Contractor personnel resources.

Table 19.1 Project implementation risks

Ranking	Section	Rating	Comment
1	Operating costs	9	High risk: cost of HFO and reagents
2	Construction	14.1	Medium risk: logistics, contractor resources and earthworks
3	Schedule	14.5	Medium risk: project go-ahead and equipment deliveries
4	Mining	16.8	Low risk: permit, training and productivity
5	Detailed design	17.7	Low risk:
6	Engineering design	18.2	Low risk: final geotechnical results
7	Budget	21.0	Low risk:
Overall	Total project	15.9	Medium to low risk driven by the operating costs

## 20 Recommendations

### 20.1 Mineral Resource evaluation

The following program is recommended by Snowden as important to the EMZ project development and planning process prior to handover of the Plant to the Owner on 14 December 2009 to extend the Life of Mine beyond 8.6 years:

- Infill diamond core drilling to upgrade Inferred Resources located below the US\$ 500/oz mine design shell and nominally within the US\$ 650/oz pit shell developed by Snowden, followed by updated mineral resource and reserve estimates for a range of higher gold price assumptions and oil prices.
- Infill drilling north of the May 2007 block model (described as EMZ North) to extend mineral resources that may convert to reserves at gold price assumptions of US\$ 650/oz or higher.
- Exploration of resource extensions south of the May 2007 block model.
- Mapping of structural details within the surface trenches and training of geologists and geotechnicians in grade control standards and ore spotting.
- Scout drilling of depth extensions to evaluate potential for reserve growth at higher gold prices and/or selective underground stoping below the HW contact of the main arenite from declines developed out of pit bottom.
- Ongoing evaluation of gold prospects on the adjacent permits and development of a business plan for the District.

An estimate of costs for these initiatives is provided in Table 20.1.

Table 20.1 Recommended exploration program and Budget 2008/10

Location	# Holes	Metres	Cost US\$ ('000)	Purpose
EMZ US\$ 650/oz	88	17 600	2 800	Upgrade resources + reserves inventory
EMZ North	25	5 000	800	Expand resources + reserves inventory for area 200 m north of May 2007 block model
EMZ South	20	3 000	480	Expand resources + reserves inventory for area 200 m south of May 2007 block model
Trench Mapping			25	Training of mining geologists + Ore spotters
Depth Extensions	10	3 000	480	Scout potential for EMZ underground mining in main arenite in 6 – 10 m cut below HW contact in interval >200m BS
Other permits	45	6 750	1 080	Upgrade of mineral resources + reserves inventory: Essakane North, Falagountou, Gossey
<b>Total</b>	<b>188</b>	<b>35 350</b>	<b>5 665</b>	

## 20.2 Mining

GRD Minproc recommends that further mining studies be concentrated in the following areas to reduce operating costs:

- use of a large excavator for ore mining
- switch to a single contractor blasthole and grade control drilling
- optimize production blasthole size
- refine the surface mine optimisation and production scheduling

GRD Minproc also recommends that further geotechnical and hydrological work is completed in the pre-Production to Year 2 mining area to provide greater confidence in ground conditions.

## 20.3 Processing

GRD Minproc and Essakane recommend that further metallurgical studies investigate gravity/intensive cyanidation work on Fresh ore sulphide samples, and that more comminution characterization work on representative Fresh ore samples are completed. No impact on metallurgical design is anticipated but operating cost benefits may result with improved plant optimization.

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## 22 Dates and signatures

**Name of Report: Update on Essakane Gold Project,  
Burkina Faso**

**Date: October 2007**

**Issued by: Orezone Resources Inc.**

signed [ ]

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Ian M Glacken Date

signed [ ]

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John F Hawxby Date

signed [ ]

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Michael Harley Date

signed [ ]

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Olivier Varaud Date

signed [ ]

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Simon Solomons Date

## 23 Certificates

**CERTIFICATE of QUALIFIED PERSON**

(a) I, Ian Martin Glacken, Group General Manager – Resources, of Snowden Mining Industry Consultants Pty Ltd., 87 Colin St., West Perth, Western Australia, do hereby certify that:

(b) I am the co-author of the technical report titled 'Update on Essakane gold project, Burkina Faso' and dated October 2007 (the 'Technical Report') prepared for Orezone Resources Inc.

(c) I graduated with the following degrees:

- BSc (Honours) Geology, Durham University (UK), 1979
- MSc Mineral Exploration and Mining Geology, Royal School of Mines (Imperial College), London (UK), 1981
- MSc Geostatistics, Stanford University (USA), 1996
- Grad. Dip. Computing, Deakin University (Australia), 1997.

I am have the following professional qualifications:

- Fellow of the Australasian Institute of Mining and Metallurgy
- Chartered Professional Geologist (AusIMM)
- Member of the Institute of Materials, Mining and Metallurgy, UK
- Chartered Engineer (Europe).

I have worked as a Geologist continuously for a total of 26 years since my graduation from university and have worked in exploration and mining geology in addition to resource evaluation and geostatistics. I have worked in a wide range of commodities and geological environments including gold, copper, nickel, lead/zinc, uranium and phosphate.

I have read the definition of 'qualified person' set out in National Instrument 43-101 ('the Instrument') and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements of a 'qualified person' for the purposes of the Instrument. I have been involved in a Mineral industry (resource evaluation) consulting practice for 9 years, including gold deposit evaluation for all of that time.

(d) I have made a current visit to the Essakane property in July 2006 for 6 days.

(e) I am responsible for the preparation of the sections of the Technical Report as detailed in Table 2.1. and for the overall compilation of the Technical Report.

(f) I am independent of the issuer as defined in section 1.4 of the Instrument.

(g) I have not had prior involvement with the property that is the subject of the Technical Report before the site visit.

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

(i) As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Perth WA this 9<sup>th</sup> day of October, 2007



Signed **IAN M GLACKEN**  
GEOLOGY

Ian M Glacken, FAusIMM(CP), CEng



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## **Essakane DEFINITIVE FEASIBILITY STUDY CERTIFICATE of QUALIFIED PERSON**

(a) I, John Francis Hawxby, Senior Project Manager of GRD Minproc (Pty) Ltd, Highbury House, Hampton Office Park North, 20 Georgian Crescent, Bryanston, South Africa 2021 do hereby certify that:

(b) I have reviewed the relevant portions of the technical report titled 'Update on Essakane gold project, Burkina Faso' and dated October 2007 (the 'Technical Report') prepared for Orezone Resources Inc.

(c) I graduated with a BSc in Electrical Engineering from the University of Natal, South Africa, in 1970.

I am registered as a Professional Engineer with the Engineering Council of South Africa (Reg. No. 20040131) and I am a Member of the Institution of Engineering and Technology in the United Kingdom (Membership No. 15020256).

I have worked as an Engineer continuously for a total of 36 years since my graduation from university, initially in the electrical supply industry, followed by 23 years in the mining industry and for the last 9 years I have been a Project Manager for the design and construction management of process plants for the mining industry throughout Africa.

I have read the definition of 'qualified person' set out in National Instrument 43-101 ('the Instrument') and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements of a 'qualified person' for the purposes of the Instrument. I have been involved in the preparation of definitive feasibility studies and the design, construction and commissioning of process plants, for the mining industry, for 9 years. This has included work for CVRD in Gabon, Gold Fields and Anglo Ashanti in South Africa and Golden Star Resources in Ghana.

(d) I have made two current visits to the Essakane site in Burkina Faso. The first was in October 2006, at the start of the DFS and the second in September 2007.

(e) I am responsible for the preparation of the sections of the Technical Report as detailed in Table 2.1.

(f) I am independent of the issuer as defined in section 1.4 of the Instrument.

(g) Other than the preparation of the definitive feasibility study I have not had prior involvement with the property that is the subject of the Technical Report.

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

(i) As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Bryanston South Africa this 9th day of October 2007

Signed

John F. Hawxby (Pr.Eng, MIET, MSAIEE, BSc. Eng.)

Senior Project Manager

GRD Minproc (Pty) Ltd

**CERTIFICATE of QUALIFIED PERSON**

(a) I, Michael Harley, Mineral Resource Manager - International Projects of Gold Fields Mining Services Limited (Corporate Office), 24 St Andrews Road, Parktown, Johannesburg, South Africa, do hereby certify that:

(b) I am a co-author of the technical report titled Orezone Resource Inc. Update on Essakane Gold Project, Burkina Faso and dated October 2007 (the 'Technical Report') prepared for Orezone Resources Inc.

(c) I graduated with a PhD in Geology and CFSG Diplome in Geostatistics.

I am a member of the South African Institute of Mining and Metallurgy and a member of the Australian Institute of Mining and Metallurgy.

I have worked as a Geologist continuously for a total of 16 years since my graduation from university.

I have read the definition of 'qualified person' set out in National Instrument 43-101 ('the Instrument') and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfill the requirements of a 'qualified person' for the purposes of the Instrument. I have been involved in Mineral Resource evaluation consulting practice for 14 years, including hydrothermal gold mineralisation for at least 5 years. This has included work for Kalahari Goldridge Mining in South Africa, Golden Star Resources in Ghana and Anglo Gold Ashanti in South Africa, Namibia and Ghana.

(d) I have made one visit to the Essakane Gold Project on 1 November 2006 to 6 November 2006.

(e) I am responsible for the preparation of the sections of the Technical Report as detailed in Table 2.1.

(f) I am independent of the issuer as defined in section 1.4 of the Instrument.

(g) I have not had prior involvement with the property that is the subject of the Technical Report.

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

(i) As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Johannesburg this 9<sup>th</sup> day of October 2007

Signed

Michael Harley BSc (Hons), PhD, CFSG, MSAIMM, MAusIMM  
Mineral Resource Manager - International Projects.



### CERTIFICATE of QUALIFIED PERSON

(a) I, Olivier Varaud, Chief Mining Engineer of Gold Fields Burkina Faso Sarl, 146, Rue 13.49 Quartier Zogona, Ouagadougou, Burkina Faso, do hereby certify that:

(b) I am the co-author of the technical report titled "Update on Essakane Gold Project, Burkina Faso" and dated October 2007 (the "Technical Report") prepared for OreZone Resources, Inc.

(c) I graduated with a Bachelor of Science in Mining Engineering.

I am a member of AusIMM. My membership number is 226311. I have worked as a MINING ENGINEER continuously for a total of 12 years since my graduation from university as a mining consultant for Mintec Inc., as Senior Mine Planning Engineer for CBG in Guinea, as Chief Mining Engineer for Abooso Gold Fields at the Damang Mine, Ghana, and as Chief Mining Engineer for Gold Fields Burkina Faso on the Essakane Gold Project. I have read the definition of 'qualified person' set out in National Instrument 43-101 ("the Instrument") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements of a 'qualified person' for the purposes of the Instrument. I have been involved in MINING operational practice for 12 years, including pit optimization, surface mine design, mineral reserve calculation and long term scheduling for the Damang Expansion Project, and Resource and Reserve reporting, CBG Resource and Reserve reporting and LoM scheduling, and several pre-feasibility and feasibility studies for mines in Central and South America for at least 5 years.

(d) I have made two visits to the Essakane property on October 2006 and September 2007.

(e) I am responsible for the preparation of the sections of the Technical Report as detailed in Table 2.1.

(f) I am independent of the issuer as defined in section 1.4 of the Instrument.

(g) I have not had prior involvement with the property that is the subject of the Technical Report.

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

(i) As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Ouagadougou, Burkina Faso this 9<sup>th</sup> day of October 2007.

Signed

Olivier C. Varaud BSc (Mining), MAusIMM



### CERTIFICATE of QUALIFIED PERSON

(a) I, Simon Solomons, Project Manager-Essakane Feasibility Study of Gold Fields Australia Limited, do hereby certify that:

(b) I am the co-author of the technical report titled Update on Essakane Gold Project, Burkina Faso and dated October 2007 (the 'Technical Report') prepared for Orezone Resources Inc.

(c) I graduated with a Bachelor of Applied Science (Mining Engineering) Honours Class 1 from the University of New South Wales, Sydney Australia and have also attained a Masters of Science (Mineral Economics) from Macquarie University, Sydney, Australia.

I am a Fellow of the AusIMM (membership # 102495) and member of the SME (membership # 3031520).

I have worked as a mining engineer continuously for a total of 32 years since my graduation from university as a mining engineer/senior mining engineer/chief mining engineering for 13 years with several Australian mining companies (ZC/NBHC mines, Metals Exploration Ltd, Peko-Wallsend Ltd); as Manager-Operations/Resident Manager/General Manager Operations for ten years with ERA Limited, Western Mining Corporation Limited, Selwyn Mines Limited and Dragon Mining Limited; as Manager Technical Services/Development for four years with North Limited; four years as an independent Mining Consultant and at Gold Fields Australia Ltd on the Essakane Project.

I have read the definition of 'qualified person' set out in National Instrument 43-101 ('the Instrument') and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements of a 'qualified person' for the purposes of the Instrument. During my career to date, I have been involved with several feasibility studies including The Peak, Northparkes, Gecko Mine Recommissioning and Las Pascualas (Chile) for at least 5 years.

(d) I have made two visits to the Essakane Project site in Burkina Faso. The first was in April 2007 and the second was in October 2007.

(e) I am responsible for the preparation of the sections of the Technical Report as detailed in Table 2.1.

(f) I am independent of the issuer as defined in section 1.4 of the Instrument.

(g) Other than assistance in the preparation of the Definitive Feasibility Study, I have not had prior involvement with the property that is the subject of the Technical Report.

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

(i) As of the date of this certificate, to the best of my knowledge, information and belief, the Technical Report contains all the scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Johannesburg this 9 October 2007.

Signed

Simon Solomons, BE (Mining) Hons Cl 1, MSc (Mineral Economics)