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IAMGOLD Corporation :
Updated Feasibility Study - Essakane Gold Project
Burkina Faso

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IMPORTANT NOTICE

This report was prepared as a National Instrument 43-101 Technical Report, in accordance with Form 43-101F1, for Orezone Resources Inc. (Orezone) by G Mining Services Inc. The quality of information, conclusions, and estimates contained herein is based on: i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report.

As of February 25, 2009, IAMGOLD Corporation (IAMGOLD) had acquired all of the outstanding shares of Orezone pursuant to the arrangement agreement dated December 10, 2008, as amended as of January 12, 2009, between IAMGOLD, Orezone and Orezone Gold Corporation. Effective as of the close of business on February 25, 2009, the common shares of Orezone were delisted from the Toronto Stock Exchange and the NYSE Alternext. IAMGOLD has requested that G Mining Services Inc. readdress this report to it in order to support its own disclosure. Other than a correction to the title of Table 17.2, no other changes have been made to this report beyond addressing it to IAMGOLD and re-dating it.

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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 held by Essakane S.A., a Burkinabe anonymous company, created on January 24, 2008, to hold the mining permit for the Project. Essakane S.A. is owned at 90% by Orezone Essakane (BVI) Limited and 10% by the Government of Burkina Faso. In turn, Orezone Essakane (BVI) Limited is a wholly owned subsidiary of Orezone Inc. (BVI), which is in turn wholly-owned by Orezone Resources Inc. ("Orezone").

This report was prepared to allow the Directors of Orezone 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 ("Gold Fields"), a predecessor majority owner of the Project, and an Addendum to the Definitive Feasibility Study prepared by G Mining Services Inc. ("G Mining") in June 2008.

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

Gold Fields initially 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. Orezone subsequently purchased the 60% interest held in the Project by Gold Fields on November 26, 2007. Orezone subsequently retained the services of G Mining to manage the engineering, procurement and construction activities required to bring the Project to commercial production. G Mining confirmed the role of GRDMinproc as the EPCM contractor for the processing facilities and retained most of the other engineering firms.

Golder Associates Ltd were retained to review past technical studies and conduct detailed engineering work for tailings disposal, water supply and other geotechnical aspects of the Project.

Following the change in ownership, it was decided to produce an Addendum to the DFS to reflect some of the changed conditions, improved design basis and additional design work that occurred since July 2007. Those changes were meant:

1. to adjust the economics to a higher gold price environment and higher costs for supplies, equipment and labor;
2. to increase mill throughput in saprolite and transition ore and determine its impact on capital costs and project cash flows;
3. to evaluate impacts on water balance and tailings disposal; and
4. to reduce some of the risks perceived in the initial DFS.

Within this report, Orezone Essakane (BVI) Limited, Orezone Inc. (BVI) and Essakane S.A. 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 with mineral reserves of 58.1 M t at 1.67 g Au/t has been identified and the intention of the Project is to mine and process this deposit at a rate of 7.5 M t per year in the initial years when mill feed is predominantly saprolite and transition ore and 5.4 M tpy in subsequent years when mill feed is entirely fresh rock. The expected life of milling operations is 9.4 years and during this time a total of 2.96 M oz 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 required 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 construction and pre-production expenditures of the Project are US$420.4 Mto an accuracy of ±15%. Operating and capital costs are estimated on the basis of first quarter 2008 prices and no allowance has been made for escalation.

Construction of the mine and resettlement villages is planned to start in July 2008 and construction of the process plant in September 2008. Mining activities will commence in June 2009 and ore commissioning of the plant is planned for December 2009. The process plant will be handed over in January of 2010 ready for full production in 2010.

1.1 Summary of geology and mineralization

The Essakane Main Zone (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 and the Addendum are the culmination of exploration work that started with BHP in 1995. Ranger Minerals, Orezone and Gold Fields were also project operators before Orezone took over again as operator in November 2007.

The so-called main arenite is the economically most important rock type within the EMZ. The arenite occurs within the 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 mineralization 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 mineralization is progressively deeper. Quartz vein density decreases to the south and the opportunity to locate additional economic mineralization appears poor. Drilling has not closed off gold mineralization 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 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 mineralization.

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

The January 2007 and subsequent May 2007 block models are consequently 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 (2003-2005) and Gold Fields (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. Gold Fields developed practical sampling and analytical protocols that were applied in the 2006 diamond drilling campaign. In addition, Gold Fields' 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. Gold Fields 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-degradeable 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 Gold Fields 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.

Gold Fields 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 Gold Fields' 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 Au/t are also presented. Typically less than 7% of the unremediated BHP and Orezone data have grades above 1.0 g Au/t which illustrate the low grade of samples not re-assayed.

A count of 3,298 assays exceed 1 g Au/t within the unremediated dataset whereas 3,707 assays exceed 1 g Au/t in the corresponding remediated dataset. Remediation has thus changed 409 assays from less than 1 g Au/t to more than 1 g Au/t which is only 0.29% of the total May 2007 dataset. 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 Au/t to >1 g Au/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 Au/t cut-off grades.

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

1.3.1 Uniform Conditioning (UC)

Gold Fields 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 mineralization and that a diffusion process model is a more appropriate random function model than a mosaic process to describe the mineralization. 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 determine the lithostructural domains used for resource estimation.

The weathering profile consists of an upper Saprolite unit, a lower transition (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 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 centimetres 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.

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 25 m (X) x 50 m (Y) with local areas drilled at 25 m 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, two 25 m x 50 m x 6 m panels of saprolite ore would be mined in one day and a fresh ore panel in 1.5 day.

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 Au/t. Single fire assay of 2 kg BLEG tailings was clearly inadequate and the reason for this is an increase in the nugget effect caused by partial leaching, BLEG dissolved small particles leaving large gold particles in the tailings which were very difficult to detect in single 50 g subsampling.

Gold Fields made use of routine screen fire assay of 1 kg 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% 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 Gold Fields 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 percent 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 Whittle surface mine shell. Mine optimisation was initially completed by Snowden on the January 2007 Mineral Resource model and reported to Essakane on March 13, 2007. The classified mineral resource estimate is presented in Table 1.2. Orezone announced this resource estimate on April 10, 2007.

Snowden also completed pit optimization on the May 2007 model and the updated Mineral Resource constrained by a US$650/oz Whittle pit shell is reported in Table 1.3 by gold cut-off grade and classification. Orezone announced this resource estimate on September 19, 2007 at cut-offs of 0.5 g Au/t and 1.0 g Au/t respectively.

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

Total Mineral Resource					
Jan-07	**COG (g Au/t)**	**0.50**	**0.80**	**1.00**	**1.20**
Indicated	Tonnes (M t)	68.7	46.0	36.5	29.3
	Grade (g Au/t)	1.56	2.01	2.31	2.60
	Au (Moz)	3.44	2.98	2.71	2.45
	Tonnes (M t)	23.5	15.2	11.9	9.5
Inferred	Grade (g Au/t)	1.50	1.98	2.27	2.57
	Au (Moz)	1.13	0.96	0.87	0.79
Mineral Resource Constrained by US$650/oz pit shell					
	COG (g Au/t)	**0.50**	**0.80**	**1.00**	**1.20**
Indicated	Tonnes (M t)	63.2	43.3	34.6	28.0
	Grade (g Au/t)	1.60	2.05	2.34	2.63
	Au (Moz)	3.26	2.85	2.60	2.37
	Tonnes (M t)	14.7	10.4	8.4	6.9
Inferred	Grade (g Au/t)	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 September 19, 2007

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

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

Note: Tonnes have been rounded to the nearest hundred thousand

1.4 Summary of status of exploration, development and operations

Snowden provided external audit and inputs to QAQC, sampling protocols, geological modelling and estimation since January 2006 when Gold Fields undertook to complete a DFS by Q3-2007. 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.

Gold Fields 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 1.4.

Table 1.4 Activities carried out by various parties in the DFS under the co-ordination of GRD Minproc

Consultants	Description
Gold Fields	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	Hydrogeological study report 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 1.5 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%.

In November 2007, G Mining Services Inc. was mandated by Orezone to optimize the development plan for the Essakane Project and provide senior management services for the Project. G Mining Services Inc. prepared an Addendum to the DFS containing updated mining reserves on the basis of $600/oz and a modified mine plan to supply an increased processing throughput of 7.5 M tpy when ore feed is predominantly saprolite and transition ore. Golder Associates was mandated to review prior work and recommend improved design for tailings disposal and water management (including the diversion dam and reservoirs). GRD Minproc was mandated to confirm increased processing throughput and to update the design and capital costs of all processing facilities. G Mining Services Inc. coordinated the estimation of capital costs and operating costs under Essakane's direct responsibility and prepared the detailed financial models for the Project.

The time-critical equipment for the Project has been ordered and includes the following: gyratory crusher, grinding mills, mine equipment fleet, construction equipment including cranes, construction and permanent power generators. Initial contracts have been tendered and are ready for awards. The Project is ready to go into construction in July 2008.

Table 1.5 List of activities documented within the DFS

Activity	Description
Environmental management and statutory requirements	Coordinate the work carried out by Knight Piésold for the completion of the Environmental Impact Assessment (EIA) Report
Social, relocation and resettlement	Coordinate rePlan and Essakane activities regarding the resettlement villages and the required infrastructure. Preparation of the cost estimate for the resettlement village based upon designs supplied by rePlan.
Hydrological and hydrogeological investigations	Coordinate the activities of Knight Piésold and their sub-consultants, GCS, in the investigations into surface and underground water supplies and quality Provide input into the preparation of the overall water balance by Knight Piésold and GCS.
Geotechnical investigation	Coordinate the development of geotechnical information by Knight Piésold for the surface mine, roads, earthworks and civilworks
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 Project for integration into the DFS Review the designs and proposals forwarded by Knight Piésold for the tailing storage facility and the 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 Knight Piésold.
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, required for the Project, with an accuracy range of +15%.
Operating cost estimate	Prepare an operating cost estimate for the cost of the operating activities and services required for the Project with an accuracy range of ±15%.
Project implementation plan	Prepare a project implementation plan which will include preparation of a schedule showing critical activities for the completion of the Essakane Project.

1.5 QP conclusions and recommendations

G Mining Services Inc. has prepared an Addendum to the DFS for surface mining and gravity/CIL processing of the EMZ gold deposit at an initial rate of 7.5 M tpy for the first three years of operations, 6.5 M tpy for the fourth year, and 5.4 M tpy thereafter when mill feed is entirely fresh rock. The average gold production over the 9.4 years-mine life is 310,000 ounces per year, with an average of 330,000 ounces per year for the initial four years. Total gold production will be 2,960,000 ounces at a total operating cost of $358 per ounce including royalties and government charges.

Initial capital costs, without financing costs, are estimated at $420,444,000; it includes $41,870,000 in accuracy provisions and contingency or 11% of direct capital costs. This estimate is in 1Q-2008 constant dollars. Sustaining capital and working capital are estimated at $26,003,000 and $27,089,000 respectively.

Using a gold price of $800/oz and a Brent oil price of $85/barel for our base case, a detailed financial model shows an after-tax IRR (Internal Rate of Return) of 20.8% for Orezone on the investment subsequent to January 1st, 2008. The undiscounted Net Cash Flow to Orezone is estimated at $630M and its 5% Net Present Value is $373M.

The financial model shows that the Project's economics are most sensitive to gold and oil price. This is demonstrated in Table 1.6 which presents the summary of financial results for three separate sets of commodity price assumptions.

Table 1.6 Financial results for different scenarios of Gold Price/Oil Price

Gold Price ($/oz)	600	800	1,000
Brent Oil Price ($/b)	66	85	100
C1 ($/oz)(exc. Govt. charges)	297	310	321
C2 ($/oz) (incl. Govt. charges)	337	358	375
Gross Project – IRR (%)	14.2	28.6	41.5
Orezone after tax IRR (%)	10.0	20.8	30.6
Orezone after tax NPV-0% (000$)	289,401	630,167	984,671
Orezone after tax NPV-5%(000$)	112,839	372,751	637,599

In conclusion, the Essakane Project is economically justified at a long-term gold price in excess of $600 per ounce.

2 Introduction

This Technical Report has been prepared by G Mining Services Inc. for Orezone Resources Inc., in compliance with the disclosure requirements of the Canadian National Instrument 43-101 (NI 43-101). The trigger for the preparation of this report is the completion of an Addendum to the Definitive Feasibility Study (DFS) on the Essakane Gold Project; this Addendum contains significant changes since the DFS, such as Mineral Reserves, increased mill throughput in the initial four years of operation, a higher gold price partly negated by higher capital and operating costs and revised economics and financial analysis.

Unless otherwise stated, information and data contained in this report or used in its preparation has been provided by Orezone Resources Inc. and all currencies are expressed in US dollars (US$).

The Qualified Persons for preparation of the report are Dr L. Gignac who visited site during May 2007 and March 2008, Mr I Glacken who visited site during August 2006 and May 2008, Mr J Hawxby who visited the project site during October 2006 and September 2007, Mr Louis-Pierre Gignac who visited site during March 2008 and Mr P. Bedell who visited site in May 2008.

The responsibilities of each author are provided in Table 2.1.

Table 2.1 Responsibilities of each QP

Author	Responsible for section/s
Dr L. Gignac	1, 2, 3, 4, 5, 15, 18.2, 18.3, 18.5, 18.7, 18.8, 18.9, 18.10, 19.3, 20.3 Overall compilation
Mr I Glacken	6, 7, 8, 9, 10, 11, 12, 13, 14, 17.1, 17.2, 19.1, 20.1
Mr J Hawxby	16, 18.10.5 (Minproc)
Mr L.P. Gignac	17.3, 17.4, 18.4, 19.2, 20.2
Mr P. Bedell	18.1, 18.6

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 km². 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 Granted	Date Expiry	Surface area km^2
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

In April 2008, following the filing of the DFS, the Environmental and Social Impact Study and the obtention of the Environmental Permit, the Essakane Mining Permit was granted over an area of 100.2 km² containing the EMZ deposit.

4.2 Location

The Essakane Project straddles the boundary of the Oudalan and Seno provinces in the Sahel region of Burkina Faso and is approximately 330 km northeast of the capital, Ouagadougou as shown on Figure 4.1. It is situated some 42 km east of the nearest largest town and the provincial capital of Oudalan, Gorom-Gorom, and near the village of Falagountou to the east.

All the exploration 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.

Similarly, the perimeter of the Essakane Mining Permit is defined by the UTM coordinates of the corner points as listed in Table 4.3.

Table 4.2 Exploration Permits: boundary coordinates

Permit name	Point	Northing	Easting
Alkoma	A	1,582,851	177,115
	B	1,582,633	194,311
	C	1,572,484	194,187
	D	1,572,699	177,115
Dembam	A	1,623,457	177,115
	B	1,623,226	194,813
	C	1,613,078	194,686
	D	1,613,305	177,115
Gomo	A	1,607,850	194,621
	B	1,576,161	205,038
	C	1,576,161	194,232
Gossey	A	1,613,305	177,115
	B	1,613,078	194,686
	C	1,602,929	194,560
	D	1,603,154	177,115
Koriména	A	1,603,171	814,519
	B	1,603,171	822,612
	C	1,599,070	822,612
	D	1,599,070	178,012
	E	1,593,001	178,012
	F	1,593,001	177,115
	G	1,572,701	177,115
	H	1,572,701	818,352
	I	1,584,500	818,352
	J	1,584,500	814,519
Lao Gountouré	A	1,603,171	822,612
	B	1,602,929	194,560
	C	1,592,781	194,435
	D	1,592,990	178,012
	E	1,599,070	178,012
	F	1,599,070	822,612
Tassiri	A	1,593,001	177,115
	B	1,592,781	194,435
	C	1,582,633	194,311
	D	1,582,851	177,115

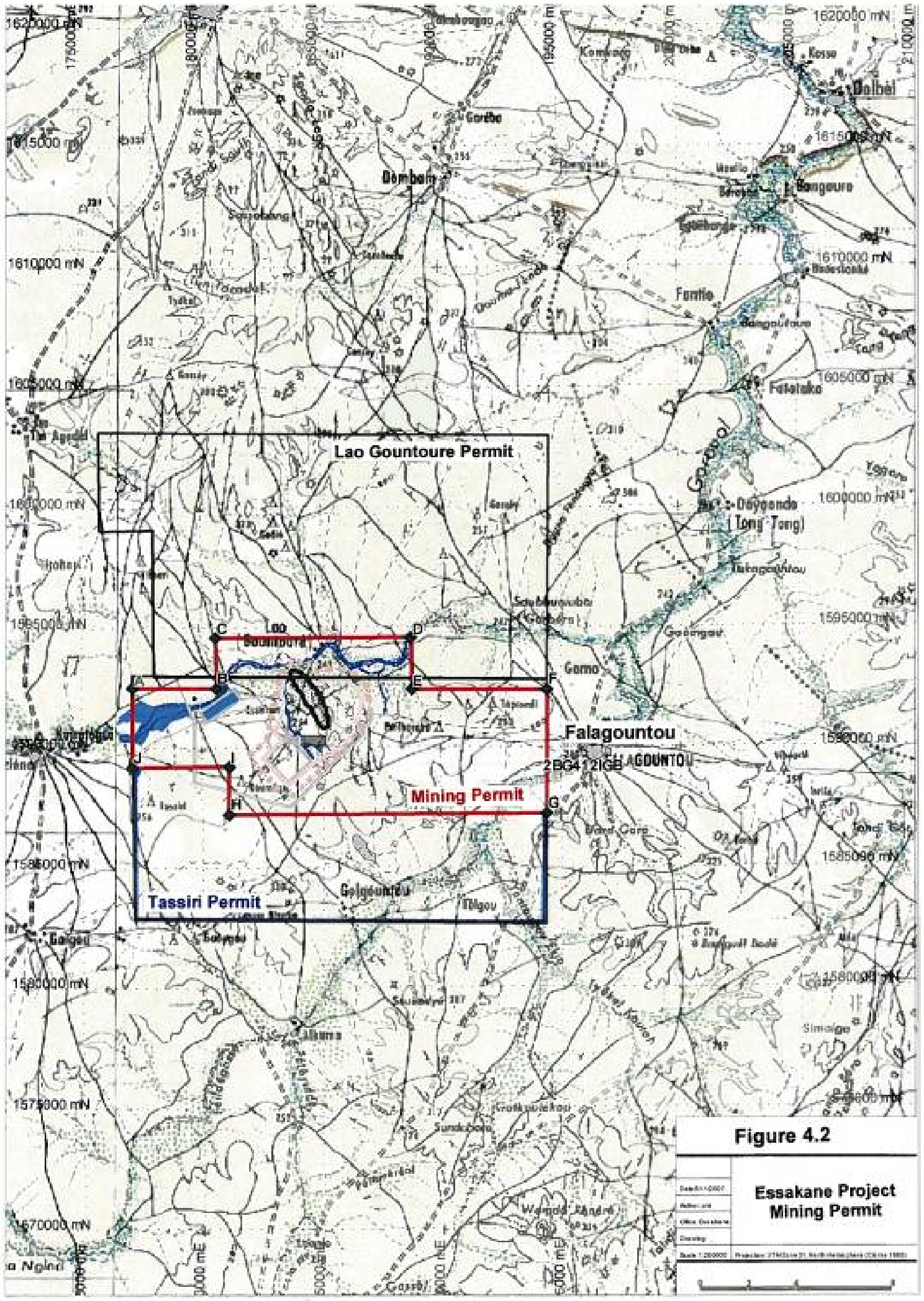
Table 4.3 Essakane Mining Permit boundary coordinates

Points	X	Y
A	177,115	1,592,488
B	180,607	1,592,488
C	180,607	1,594,564
D	188,770	1,594,564
E	188,770	1,592,379
F	194,430	1,592,379
G	194,367	1,587,187
H	181,104	1,587,187
I	181,104	1,589,186
J	177,115	1,589,186

Figure 4.1 Project Location

The location of the Essakane Mining Permit in relation to the Tassiri and Lao Countouré exploration permits is shown on Figure 4.2.

Figure 4.2 Essakane Project Mining Permit



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 Miniere, 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 4.1. All exploration permits except Korizéna have to be converted to mining permits by July 10, 2009 or be relinquished to the State with no residual interests. In terms of current Law, each mining permit application requires a separate feasibility study but there are precedents in Burkina Faso for variations to this rule (Etruscan's Youga project).

The Korizéna exploration permit is valid until November 21, 2009 and would expire on November 21, 2015 after two renewals of three years each (the permit 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 270,000 francs CFA per square kilometre (or about $650/km^2$). Burkina Faso's 2003 Mining Code provides for an Exploration Permit to be superseded by a Mining Permit.

The requirements are:

- Exploration permit holders must have observed all prior obligations under the Mining Code.
- Applications for conversion to a Mining 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.

In the Decret No. 2008-203 signed by the President, the Prime Minister, the Minister of Economy and Finance, the Minister of Mines, Quarries and Energy and the Minister of Environment on April 28, 2008, Essakane received the Mining Permit ("Permis d'exploitation minière") for the Essakane property. The Mining Permit is for an initial period of twenty years and can be renewed for successive periods of five years until depletion of the orebodies within the limits of the permit.

4.4 Issuer's interest

Orezone Resources Inc. owns 90% of the common shares of Essakane S.A. through its wholly owned subsidiaries Orezone Inc. (BVI) and Orezone Essakane (BVI) Limited, while the Government of Burkina Faso owns 10% of the common shares of Essakane S.A. Essakane S.A. is the Bukinabe company that owns the Essakane Mining Permit and the Project. It is expected that all past and future expenditures in the Project will be accounted as a Shareholder's Loan from Orezone Essakane (BVI) Ltd to Essakane S.A.

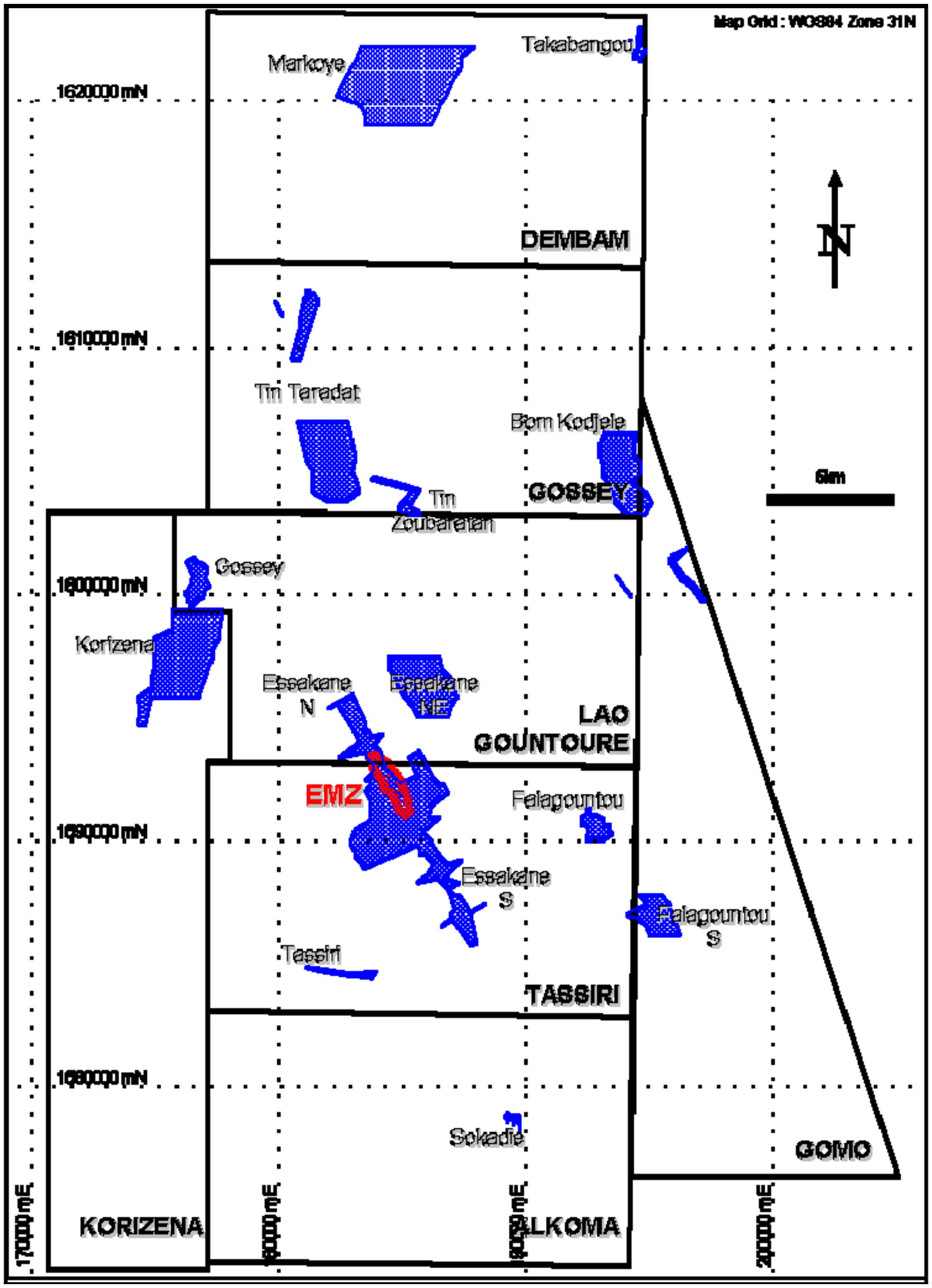
4.5 Location of property boundaries

Figure 4.3 presents the location of the US$650/oz EMZ pit shell (shown in red) in relation to all the exploration 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, the Addendum and this Technical Report.

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

The Government of Burkina Faso retains 10% (free-carried) of the common shares of Essakane S.A. In addition, the Government of Burkina Faso receives a 3%-royalty on the revenues from mineral production. The Government also collects various taxes and duties on the imports of fuels, supplies, equipments and outside services as specified in the Mining Code.

Figure 4.3 Project permits and location of the EMZ Mineral Resources and other gold prospects



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 the second quarter 2008. This is the last of four conditions that had 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.
- Agreement with local populations on resettlement plans and process.
- Grant of a "Mining Permit" by the Burkina Faso Minister of Mines.
- Grant of a "Mining convention" by the Burkina Faso Government.

In accordance with Burkina Faso's statutory requirements and International best practices, an Environmental and Socio-economic Impact Assessment was submitted to the Burkina Faso Minister of Environment on August 8, 2007. After review and public consultations, the Environemental Permit ("Émission d'avis conforme sur la faisabilité environnementale") for the Essakane Project was issued under the Arreté No. 2007-083/MECV/CAB by the Minister of the Environment on November 30, 2007.

In parallel with the planned process for the Environmental Permit, a broad consultation process was continued and formalized regarding the physical displacement and resettlement of communities directly affected by the Project and the economic impact on some other communities. A formal consultation forum was established on August 9, 2007 consisting of representatives of the affected communities, the government and Essakane S.A., jointly referred to as the Esssakane Consultations Committee (ECC). The work of the ECC resulted in a Resettlement Memorandum of Agreement (MOA) signed by the various parties in December 2007. The MOA contains terms for the compensation and entitlement packages, agreement on unauthorized activities on the project site, implementation process of the agreement, obligations and undertakings of the various parties and a monitoring and evaluation process.

Subsequent to the Environmental Permit and the Agreement with the local populations, the Mining Permit was issued on April 28, 2008 on the basis of the DFS submitted in August 2007, as discussed earlier in section 4.3. The Mining Permit was issued to Essakane S.A., a Burkinabe company created for the purpose of developing and operating the Essakane Project. Essakane S.A. was created and registered by Me Seydou Baloma on January 24, 2008.

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 all other approvals are granted. However, certain points require further review and clarification by Essakane and its lenders in order to:

- 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 are ongoing on these various aspects and final resolution is expected shortly.

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 relatively flat terrain sloping gently towards the Gorouol River to the north of the EMZ. 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 1 km east of the EMZ and contains approximately 1 M t of leached material.

5.2 Access

Access to and from the capital Ouagadougou is by paved road over 263 km to the town of Dori, and then by laterite road over some 63 km to Essakane. Access via the town of Gorom-Gorom to the west is also possible. Within the exploration 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. As part of the project, the laterite road will be upgraded at river crossings and site roads will be improved to enable regular traffic all year.

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. Some 2,562 households totalling 11,300 individuals live within the Project footprint or will be economically affected by the Project. A Resettlement Action Plan (RAP) describes the policies, procedures, compensation rates, mitigation measures and schedule, jointly developed by Essakane Project and the Project-affected communities to guide the compensation, resettlement and relocation of the people, households and communities that will be physically or economically displaced by the Essakane Project. 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. It is planned to build a private airstrip at Essakane early in the Project.

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 46ºC to 10ºC with annual pan evaporation rates of 3,000 mm/year. The mean annual rainfall is 503 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 M m^3/year. Rainfall is sporadic or absent during the rest of the year. Weather conditions are expected to have minimal impact on mining operations, but require proper planning to ensure sufficient water supply during the dry season.

5.5 Surface rights

Surface rights in the area of the Mining Permit belong to the State. Utilisation of the surface rights is granted by the Mining Permit under condition that the current users are properly compensated.

5.6 Infrastructure

The planned infrastructure required for the proposed Essakane 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. Offices, maintenance facilities and shelters required for exploration activities are also available.

5.6.1 Power

Public infrastructure such as electrical power, potable water, distribution, sewage treatment and disposal, and telecommunications does not exist in the Project area. The nearest grid-supplied power is at the town of Gorom-Gorom some 35 km to the west-northwest. Electricity to the exploration site is provided by on-site diesel generators; satellite communication is also available at the exploration site. A 26 MWatt power plant, fuelled with heavy fuel oil (HFO) will be built for the production phase. Construction power will be supplied by three 800 KWatt diesel gensets, which will become emergency generators during the production phase.

5.6.2 Water

Water is pumped from wells (boreholes) in sufficient quantities for exploration drilling and the exploration camp. A much larger supply of water mainly for milling operations, but also for dust control and human consumption will be required by the Project. The main sources of water will be the Gorouol River during the rainy season and well fields around the Essakane pit and near the Gorouol River. A diversion dam and weir will be built across the Gorouol River west of the mine and water will be diverted by gravity into an Off-Channel Storage facility (OCS) with sufficient storage capacity to provide water to the Project net of evaporation losses during the dry season.

5.6.3 Mining personnel

The Project will result in the displacement of 11,563 people living in 2,562 households and an initial Resettlement Action Plan (RAP) has been developed, in consultation with the community, to address the resettlement of these people. 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, but also elementary courses in carpentry, masonry, mechanical and electrical trades. It is envisaged that unskilled labour will be sourced locally with skilled labour drawn from Burkina Faso at large. It is expected that some 90-100 expatriates from North America and Europe will be required in the initial years of production, but that number will decrease as Burkinabe workers acquire the expertise and experience to replace the expatriate employees.

5.6.4 Tailings and waste storage areas

The EMZ is surrounded by ample flat and uninhabited land where tailings and waste dump storage can be located. The tailings storage facility (TSF) is to be located southwest of the surface mine and processing plant. The mine waste storage facility will be located mainly west of the surface mine, but a smaller waste dump will also be located east of the surface mine.

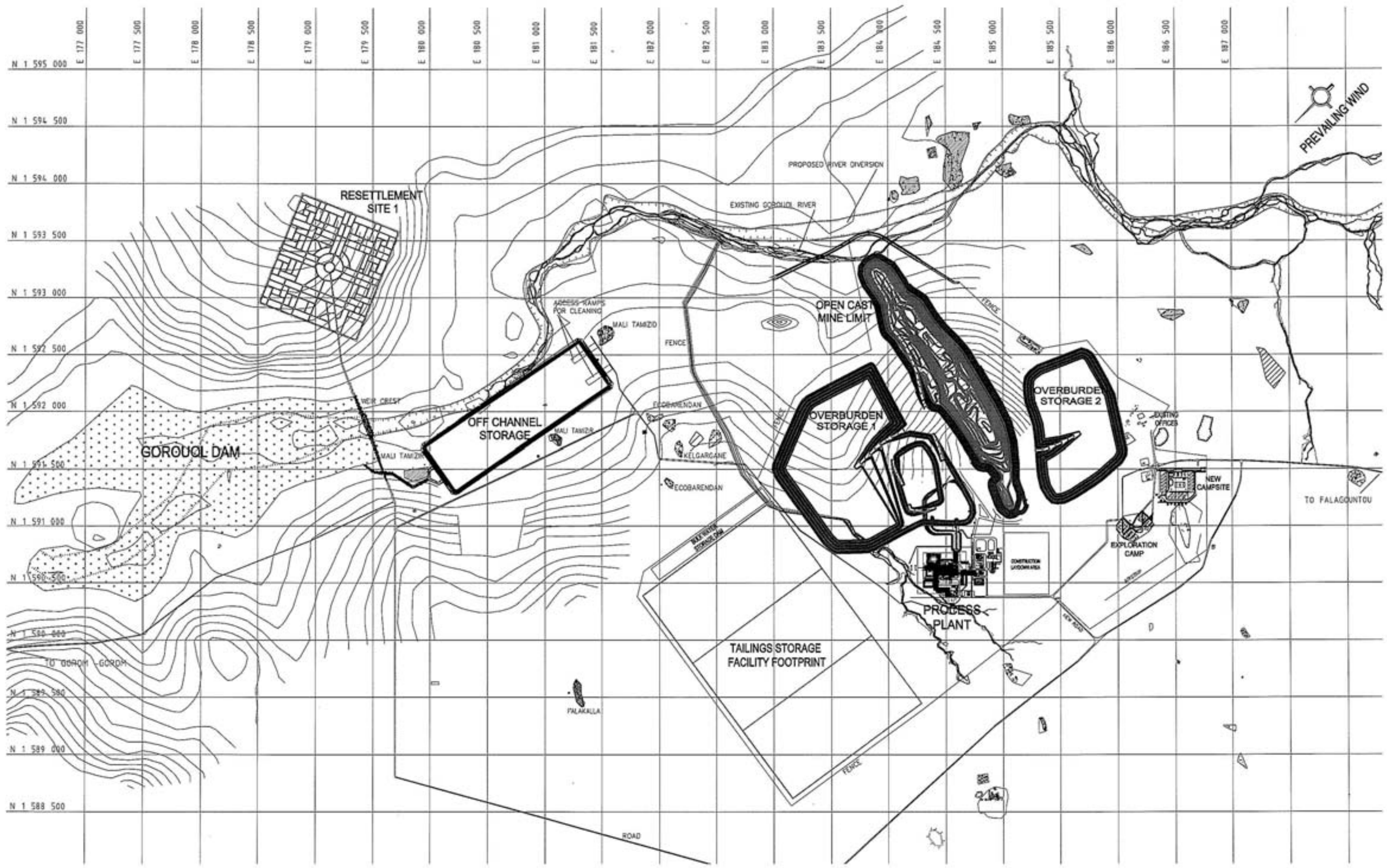
5.6.5 Heap leach pad areas

Approximately 1 M t of material on the existing CEMOB heap leach pad from previous mining operations may be processed during the production phase. 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 Essakane 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 M t of material at an average grade of 1.9 g Au/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 m 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 M t @ 2.52 g Au/t for 1.2 M oz (at 1 g Au/t cut-off grade) for the main arenite (based on due diligence documents provided to third 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 (AngloGold, Ashanti Gold Fields, 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 M 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 M t @ 2.14 g Au/t for 1.3 Moz (at 1 g Au/t cut-off) and an Inferred resource of 5.2 M t @ 1.76 g Au/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 M of the US$8 M 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 (M t)	Grade (g Au/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 July 19, 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 M on exploration over five years; (b) Orogen, or a nominee, could earn a further 10% in the project by sole funding and completing 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 April 1st, 2007, Orezone Resources Inc, Orezone Inc. (BVI), 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 M on exploration, it increased its ownership to 60% in the Essakane Project when it gained a further 10% interest in Essakane (BVI) Limited having completed a bankable feasibility study on September 11, 2007.

In October 2007, Orezone Resources entered into an agreement with Gold Fields to acquire their 60% interest in the Essakane Project in consideration for $200 M, with $150 M in cash and $50 M in shares. The transaction closed on November 26, 2007 and Orezone became the operator and owner of a 100% interest in the Project.

After obtaining the Environmental Permit, and concluding a MOA with the local population, Essakane was granted the Mining Permit, which resulted in the transfer of the Project to Essakane S.A., a Burkinabe anonymous company, created for the purpose of owning and operating the Project. Essakane S.A. is owned at 90% by Orezone Essakane (BVI) Limited and 10% by the Government of Burkina Faso. In turn, Orezone Essakane (BVI) is owned by Orezone (BVI), a wholly-owned subsidiary of Orezone Resources Inc.

6.2 Previous exploration and development work

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

- Trenching by CEMOB in the early 1990's: a total of five 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 60º 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 60º 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 Gold Fields 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 Gold Fields

		Holes (Metres)			
Drill type	**Number of holes**	**DD**	**RC**	**AC**	**TOTAL**
AC	1,336			16,069	**16,069**
DD	126	20,145			**20,145**
RC	205		16,363		**16,363**
RCD[1]	73	11,574	5,101		**16,675**
Total	**1,740**	**31,719**	**21,464**	**16,069**	**69,252**

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 m x 25 m (50 m spaced lines with holes 25 m apart on section) with one high grade central area (Panel F) drilled to 25 m x 25 m. The RC holes were drilled to the water table and sampled at 1.0 m intervals (producing 25 – 30 kg 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 3–5 kg 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 m x 50 m nominal grid.

Ranger and BHP's fire assaying demonstrated poor reproducibility of economic gold assays. Orezone thus introduced cyanide - saturated 2 kg BLEG as the standard method to improve assay reproducibility. The concentration of NaCN was set at 5 kg/tonne (10 g per 2 litres) and the sealed bottles were rolled for 24 hours. Residues (tailings) for all BLEG solution grades greater than 1 g Au/t were fire assayed for gold. Although BLEG is a partial dissolution method, the results showed improved reproducibility (for duplicate 2 kg 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 Au/t. SGS Lakefield reported significantly higher values but ABILABS reported lower grades for all samples >0.7 g Au/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.

Gold Fields combined these findings with promising results from gravity and rapid cyanide leach tests, and in January 2006 it replaced Orezone's 2 kg BLEG bottle roll process with LeachWELL rapid cyanide leach on 1 kg subsamples (the "LWL69M" method). By this time, Gold Fields had accepted the need to re-assay 40 000 pulp rejects by LeachWELL from the BHP, Ranger and Orezone programs because the 2 kg BLEG and 50 g fire assay data showed significant biases when compared with the 1 kg 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 M t @ 2.0 g Au/t for 1.91 Moz and an Inferred resource of 4.4 M t @ 2.0 g Au/t for 0.29 Moz (at 1.0 g Au/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%.

Table 6.4 EMZ Mineral Resource estimate completed by SRK in 2004

Classification	Tonnes (M t)	Grade (g Au/t)	Gold ('000 oz)
Measured	-	-	-
Indicated	30.5[1]	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.

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.

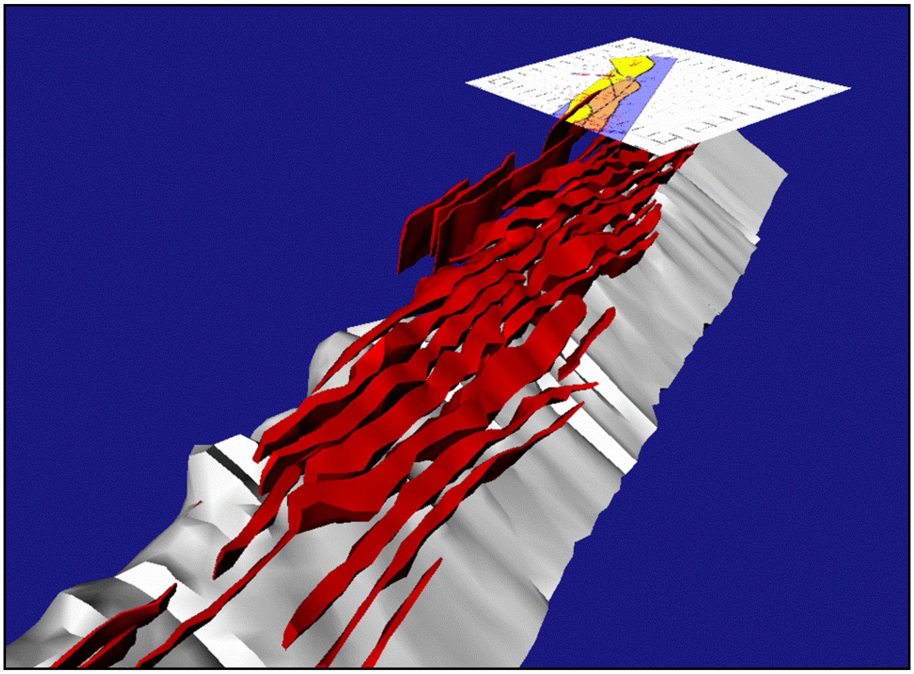
Orezone presented a new grade domain and geological fold model to RSG Global in December 2005. The basis of this model was that (a) the main gold-bearing vein sets are steep west dipping and parallel to the fold axis, and (b) strike continuity of sets of the main vein sets as seen in the trenches could 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 April 10, 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 6.5 January 2006 UC resource estimate – RSG Global

	Cut-off	0.5 g Au/t			1.0 g Au/t		
	Category	Tonnes (M t)	Grade (g/t)	Gold '000oz	Tonnes (M t)	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

Figure 6.1 December 2005 steep structure EMZ grade model



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 volcaniclastic 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.

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. Gold Fields 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 7.1 Regional geological setting of the Essakane Gold Project

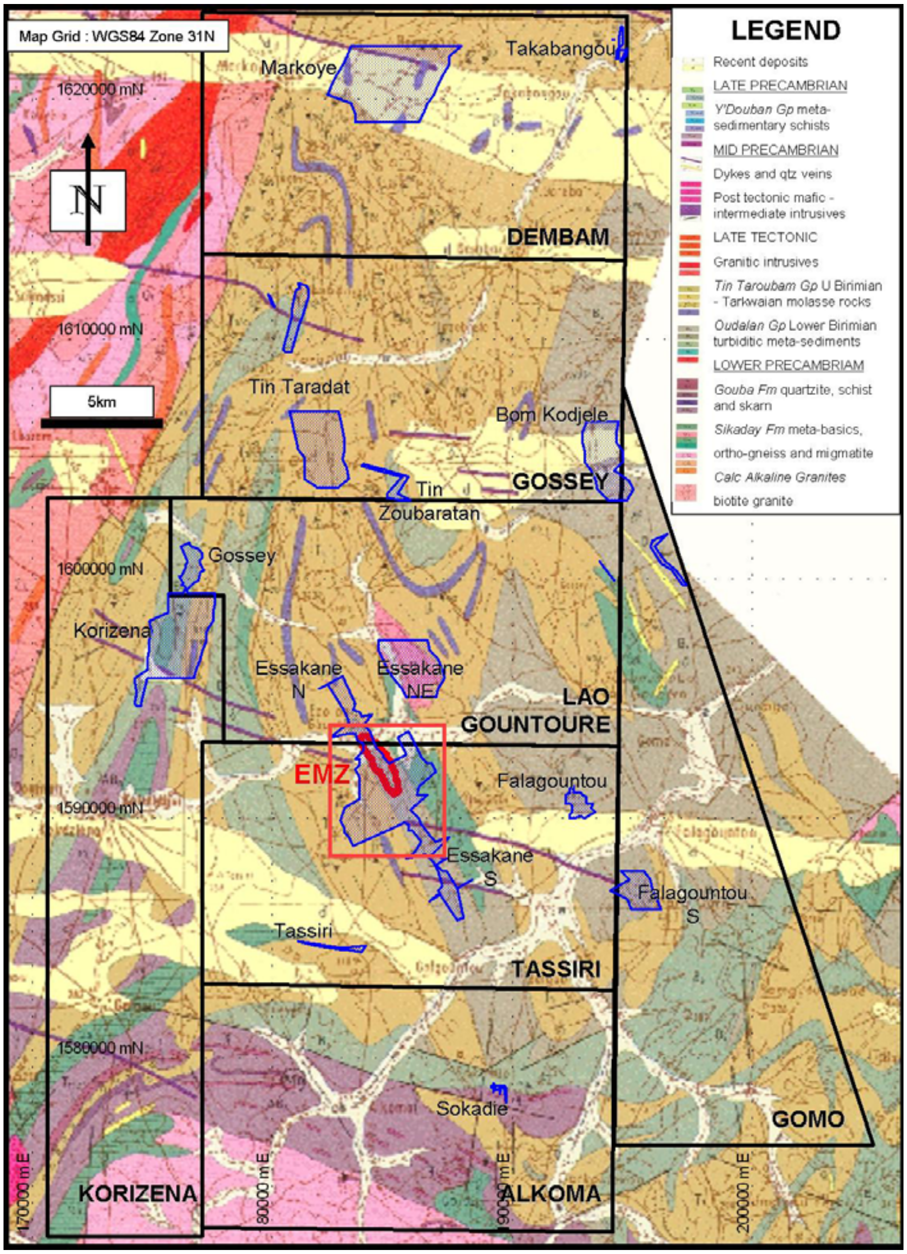


Figure 7.2 Surface geology of the Project area showing transported cover

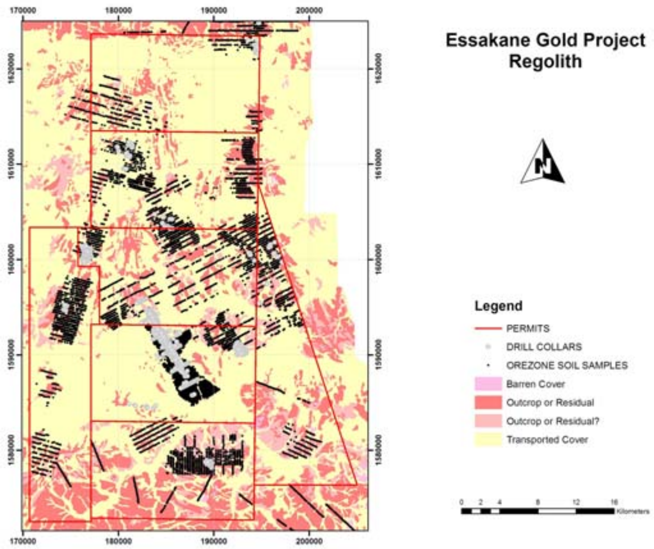


Figure 7.3 Au and As in soils for the Project area

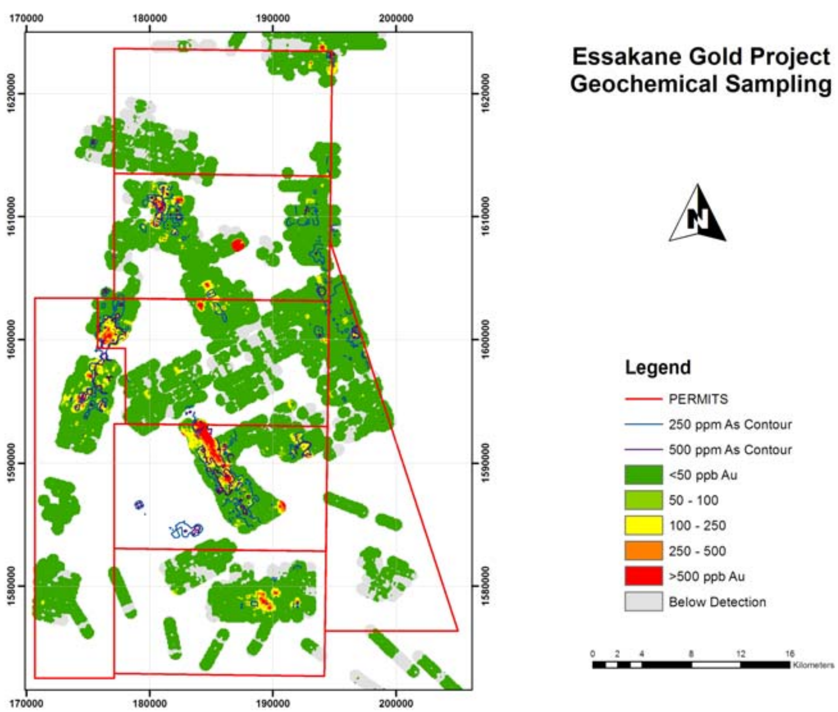


Figure 7.4 Fold model developed for the Project area

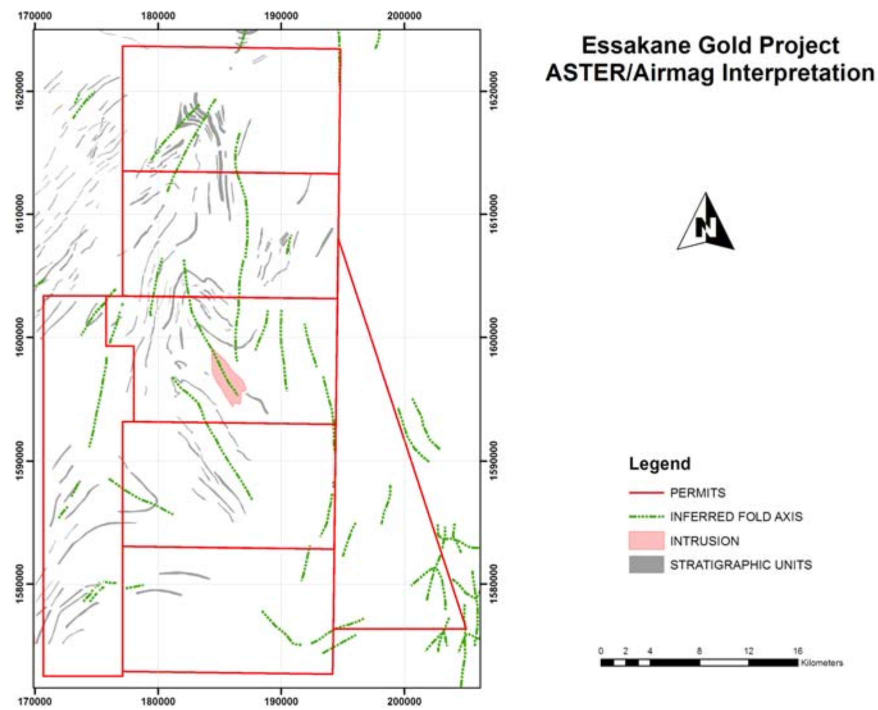
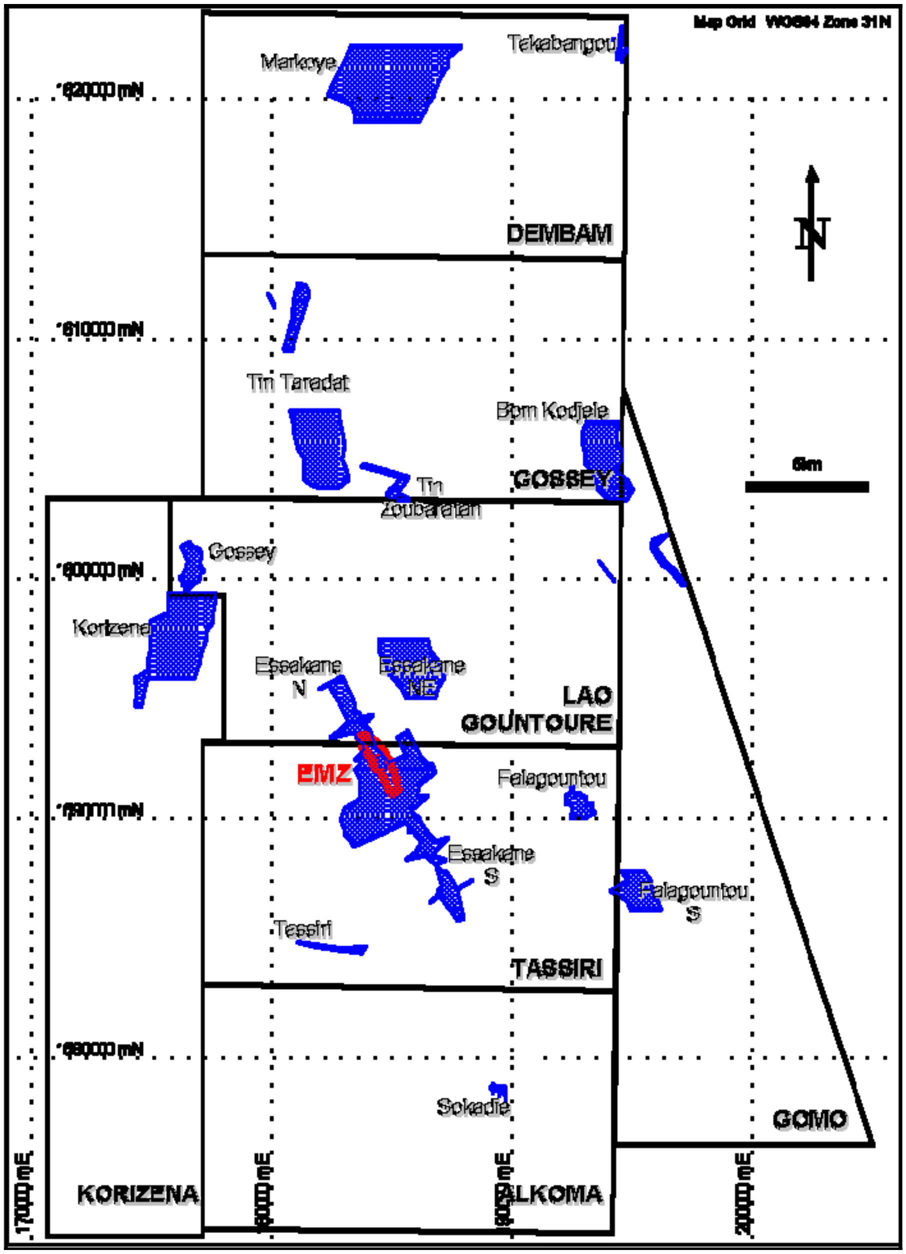


Figure 7.5 Location of all known mineralized zones in the Project area



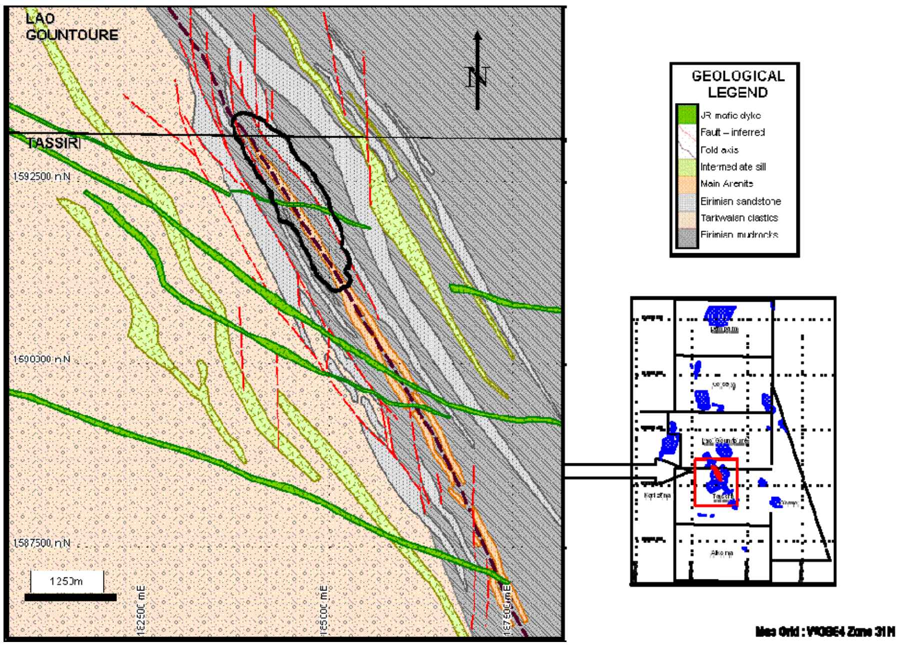
7.3 Property geology

The subsurface geology of the area surrounding the EMZ on the Tassiri permit is presented in Figure 7.6. This map was developed by Gold Fields 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 Pre-Feasibility Study (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 Orezone 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 Orezone. 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.

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.

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

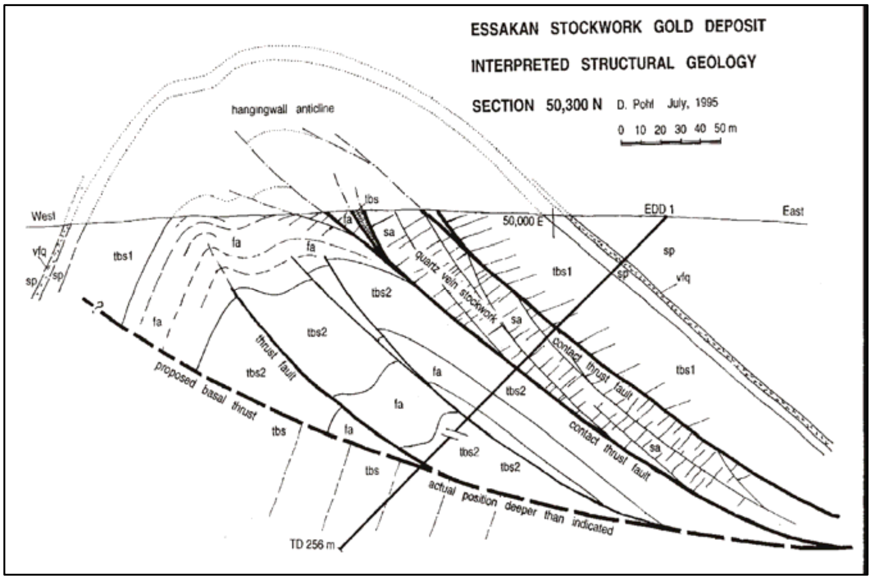


Figure 8.2 Cross-section showing the 2005 PFS thrust domain model

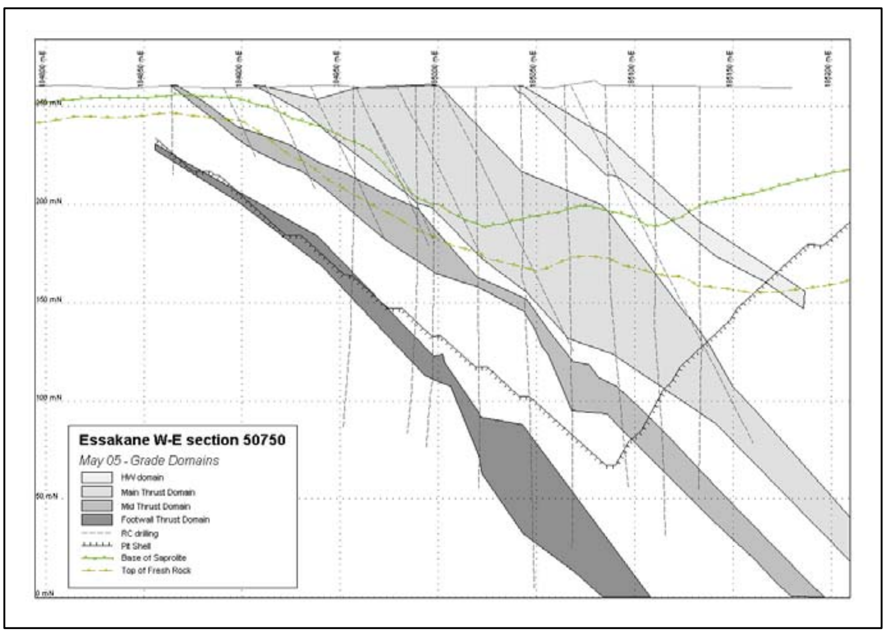
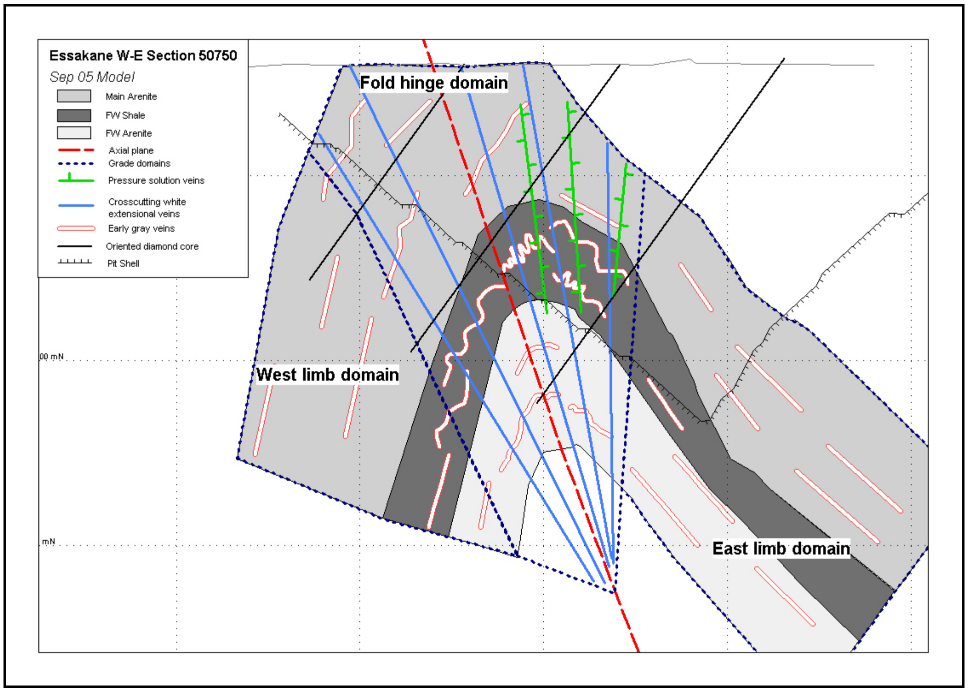


Figure 8.3 Cross-section showing the EMZ fold geological model



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.

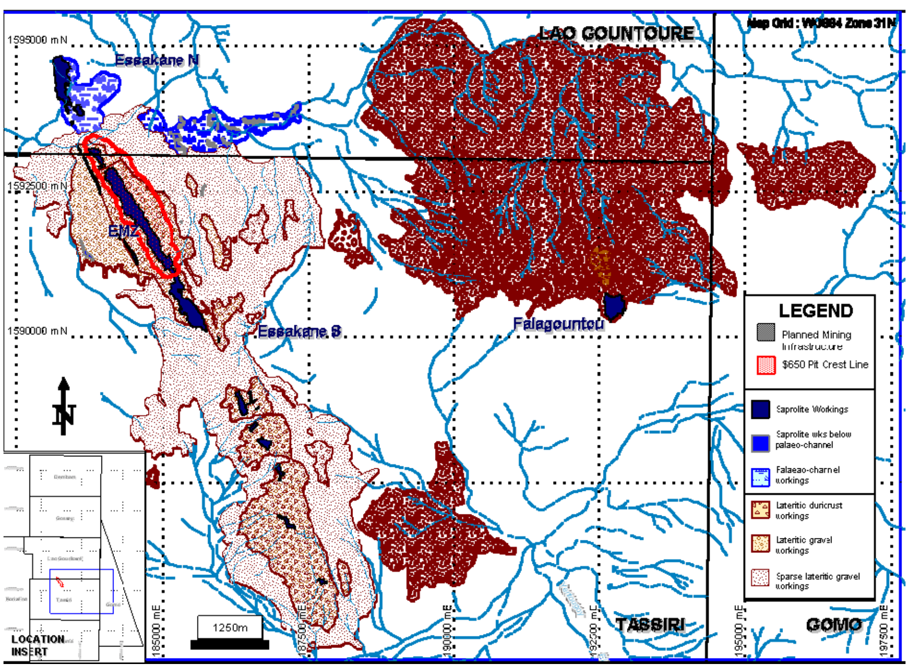
9 Mineralization

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.

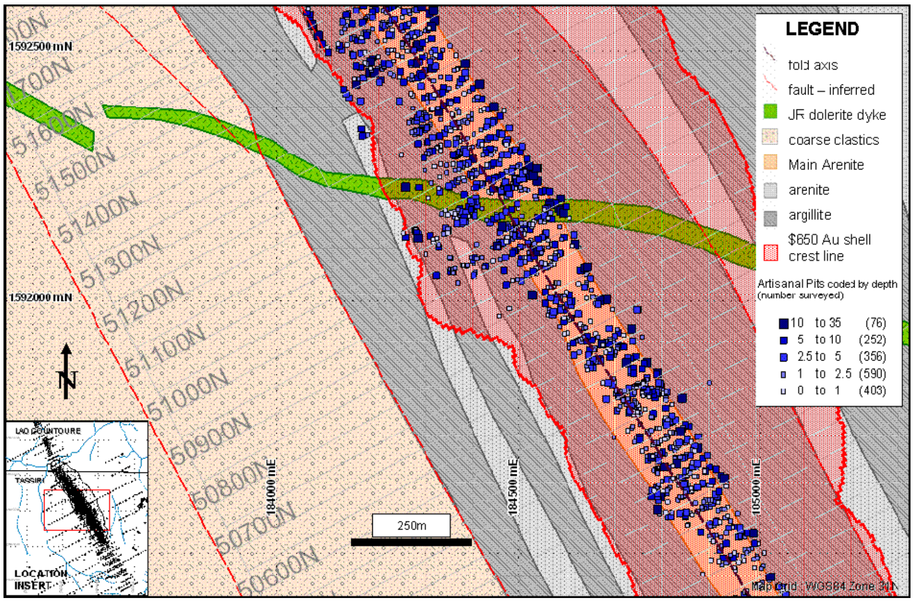
Figure 9.1 Artisanal workings surrounding the EMZ



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 cm. 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

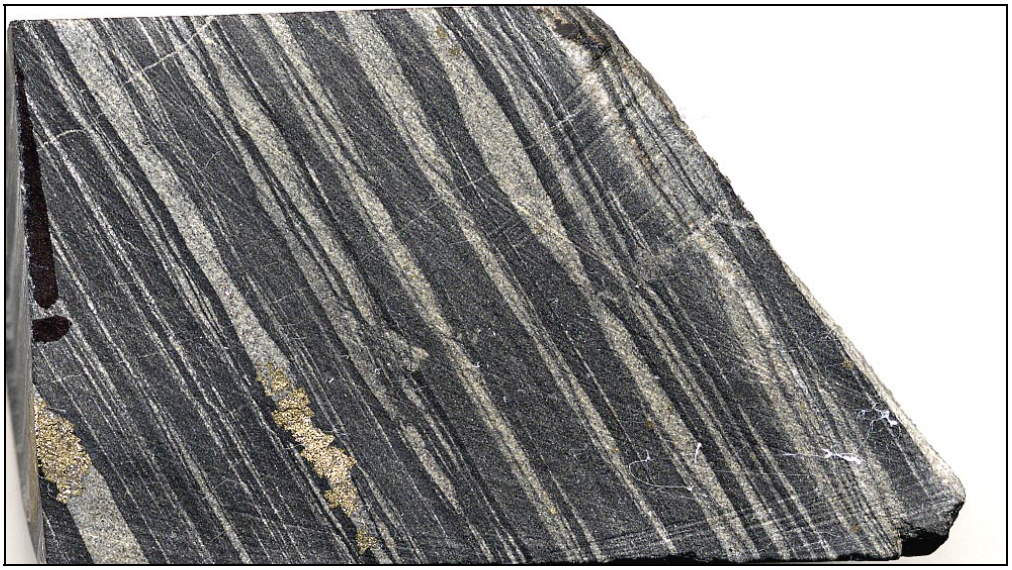
Stratigraphic Succession	Lithology	Comments
HW Argillite	Tourmalinized mudstone and siltstone with a basal 5m thick graphitic argillite. The base of the HW argillite is marked by a bedding parallel fault. Intrusive sills are common.	Largely barren on the east limb of the fold but the vertical west limb 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. Arsenopyrite is common along the top and bottom contacts marked by flexural slip deformation.	Secondary gold bearing unit. High grade pressure solution structures can occur in the fold hinge with abundant and coarse visible gold.
FW Arenite	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 ± albite ± arsenopyrite ± 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 mineralization 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 absorbtion was used from January 2006 to pick out the base of upper and lower saprolite (transition zone). 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 (or Saprolite Zone)	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 (or Transition Zone)	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.

Gold Fields 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		Gold Fields 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 Gold Fields 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)

9.2 Gold deportment

The EMZ is a coarse gold deposit. The rule-of-thumb definition for coarse gold is when particles are larger than 100 microns in diameter. This is reflected in Figure 9.5 which shows screen fire assay data for 96 x 1 kg 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 >5 g Au/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 (transition) samples and shows that most samples assaying >2 g Au/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 Au/t), medium (2.0 g Au/t) and high grade (>5.0 g Au/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.

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.

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

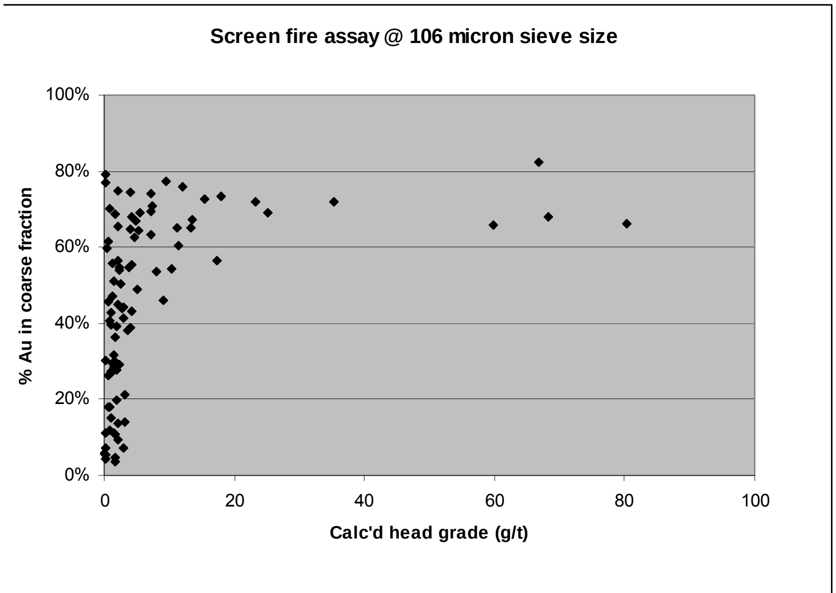


Figure 9.6 Proportion of gold reporting to Falcon gravity concentrates

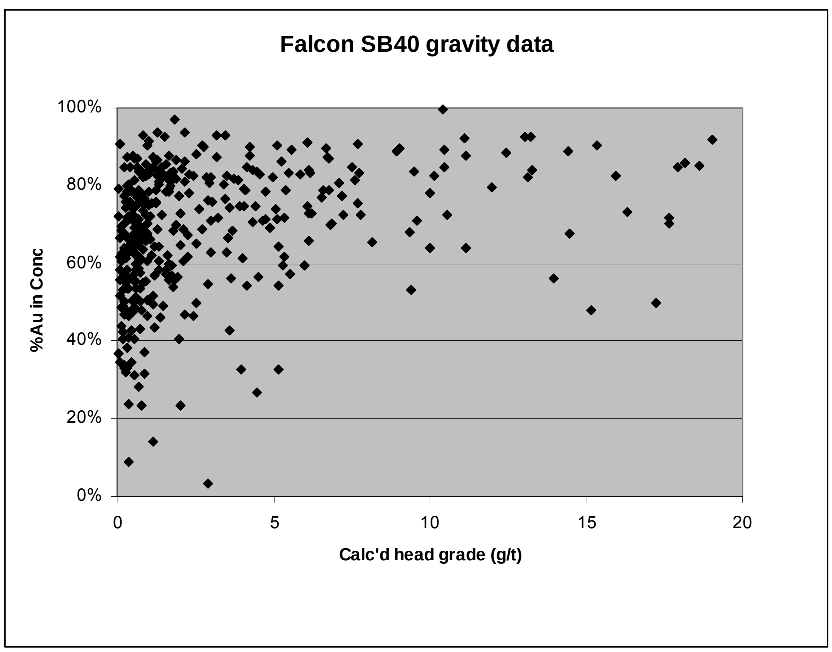


Table 9.6 Gold distribution from nine EMZ test samples

#	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 (Transition)					
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

9.3 Structural controls on gold mineralization

Mapping of surface trenches and logging of oriented diamond core boreholes by Gold Fields 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

The 2006 project development exploration program on the EMZ was carried out by Gold Fields 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. Gold Fields 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. Gold Fields 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. Gold Fields took over as operator of the project in January 2006 with new senior project managers appointed by Gold Fields. Reporting to Orezone was by monthly progress reports and inter-office meetings in Ouagadougou. Periodic site visits by senior Orezone managers also took place.

11 Drilling

11.1 Introduction

Orezone and Gold Fields 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 (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[1]	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
Gold Fields	DD	58	8,843			W - E and N - S
Gold Fields	RC	44	3,911			W - E and N - S
Gold Fields	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

Gold Fields 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 Gold Fields, 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 300 m (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 (4,186 m 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.

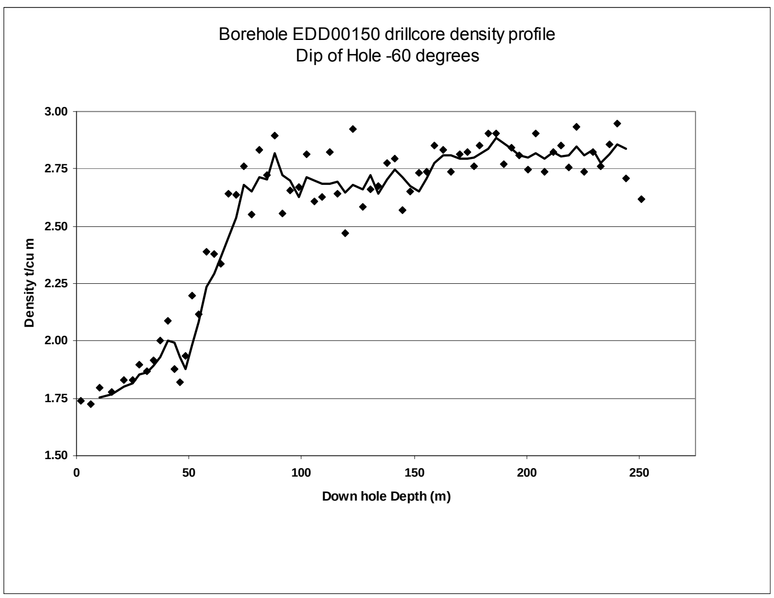
Essakane experienced serious delays in assaying the drill core samples and, coupled with quality assurance problems, Gold Fields and Snowden failed the on-site SGS assay laboratory in August 2006. This meant that reliable assays were not available to guide further drilling until well after the EMZ program was completed. The drilling program ended in June 2006 but final assays were only reported between

August 6, and November 30, 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^3 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 or transition 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. Gold Fields 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 Gold Fields 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 1 kg assay subsample was split out only after the sample had been crushed to 80% passing 2 mm. The entire 1 kg 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 20-40 kg RC rig sample down to 3-5 kg with an 8 : 1 riffle splitter.

Gold Fields 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 1 kg sample which was pulverized to 90% passing 75 microns and assayed by LeachWELL rapid cyanide leach.

The 1 kg 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. Gold Fields 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 = 1 kg; (ii) Liquid : solid ratio = 2 : 1; (iii) one LeachWELL tablet added; (iv) Bottle roll duration = 10 hrs, (v) Agitation by adding glass beads; (vi) Gold analysis by AAS, (vii) Fire assay of 1 : 10 tails.

Figure 12.1 2006 sampling protocols for DD samples

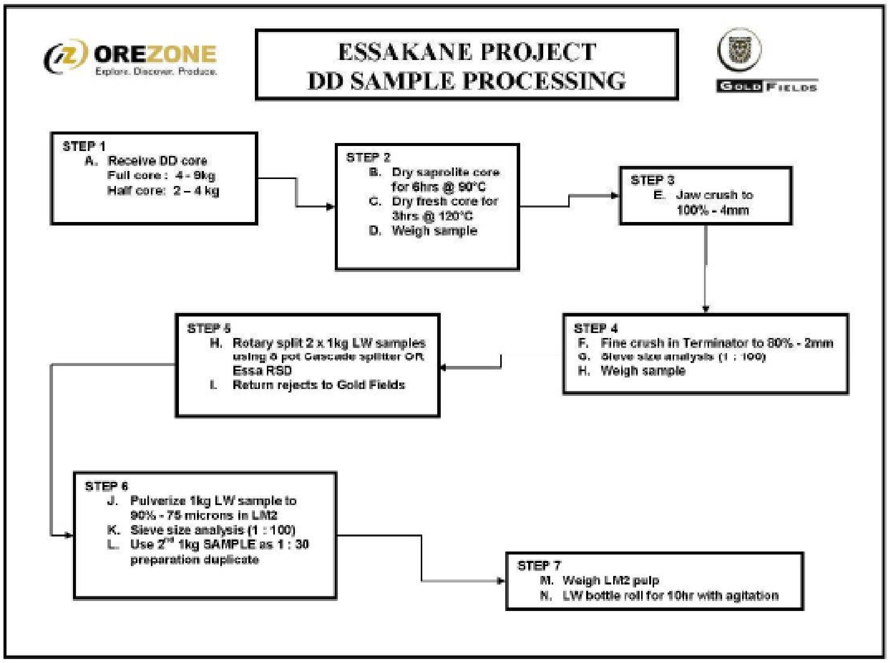
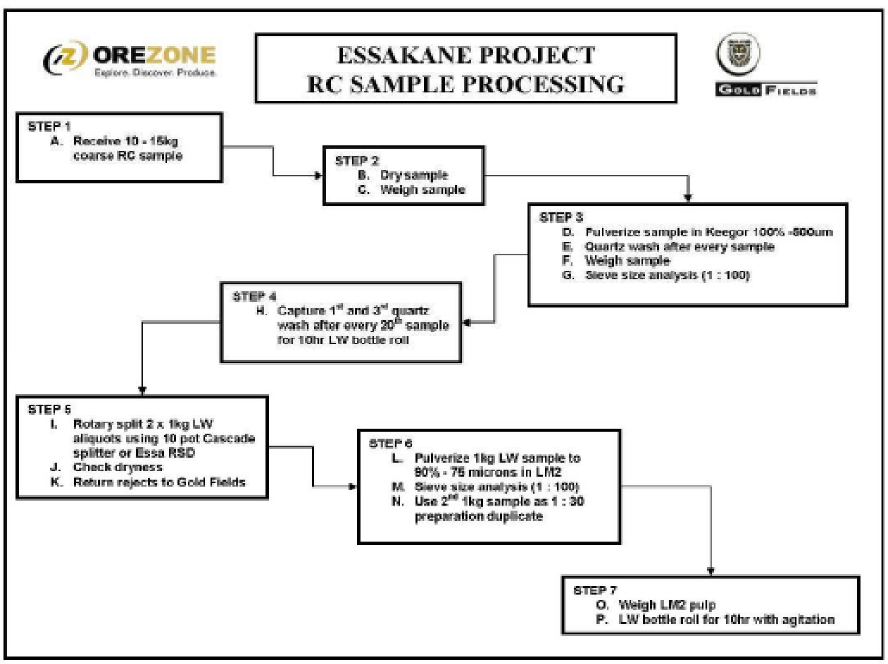


Figure 12.2 2006 sampling protocols for RC samples



12.2 Re-assay program

Gold Fields 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 1 kg splits were re-assayed using exactly the same LWL69M method at SGS Tarkwa. Eriez rotary splitters were used to split out the 1 kg 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 Au/t was used as a guideline for selecting the limits of mineralized economic intervals for re-assay. Below a grade of 0.3 g Au/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 1 kg 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 Au/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

Gold Fields 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 Gold Fields' 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.

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

- Gold Fields 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 20-40 kg in weight were split in a 1 : 1 stainless steel riffle splitter to reduce the sample weight to approximately 10 kg.
- The 10 kg RC samples were then split into 10 x 1 kg 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 1 kg 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.

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

SGS scheme code	Method	Description
PRP86	Dry and Split pulp sample	1. Empty "As received" pulp reject into drying pan 2. Turn the old sample bag inside out and make sure all sample falls into the pan 3. Only use the new Sample ID tags 4. Dry the sample at 110°C 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. 6. The same rules apply if a 50 : 50 Jones riffle splitter is used. Split the sample once and process the whole split. 7. Process the entire "as received" pulp if the as received weight is <1500 g. 8. DO NOT mat roll under any conditions. 9. Store any rejects for 60 days then return to Essakane
SCR32 **1 : 100**	Wet screen 100 g "as received" sample at 106µm	1. Wet screen 1 : 100 "as received" samples to evaluate the historical grind by Keegor 2. Report weight of coarse and fine fractions
PUL47	LM2 pulverize the whole split which will be between 700 and 1,500 g.	1. Pulverize the whole split in an LM2 mill to 80% passing 75 microns 2. Care must be taken to avoid over - grinding and loss of gold by smearing
SCR34 **1 : 100**	Wet screen 100 g of the LM2 pulp at 75 microns	1. Wet screen 1 : 100 of the LM2 pulps to evaluate the LM2 grind performance 2. Report weight of coarse and fine fractions
LWL69M	LeachWELL bottle roll the whole split (this will be approximately 1,000 gs). DO NOT tamper with the split sample by adding or removing material to get precisely 1,000 g.	1. NaCN Leach period = 10 hours 2. Liquid : Solid ratio = 2 : 1 3. No. of LW tablets = 1 4. Add four glass balls to assist agitation 5. Allow to settle and decant the pregnant solution 6. Analyze by solvent extraction AAS finish 7. Report as LWL69M_ppm
FAS31K **1 : 10**	For 1 : 10 Preparation Duplicates by screen fire assay	1. Randomly select the 2nd half of a split as per PRP86 2. Weigh the whole split (TOTWT) in grams 3. Pulverize the whole split to 90% passing 75 microns in an LM2 mill 4. Dry screen the whole pulp through a 106 micron sieve cloth 5. Weigh CORS fraction 6. Fuse the CORS fraction and the sieve cloth in a Pb collection fire assay 7. Duplicate fire assay the FINE fraction as FA50g by Pb collection 8. Report TOTWT, CORSWT, CORS_AU, AU (x2), CALC_ppm
GFL Standards / Blanks	LWL69M	Report results as LWL69M_ppm

13.2 Certified Reference Materials and blanks

Gold Fields 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 Au/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 is 95-105% of Expected Value.

Table 13.2 List of certified Rocklabs reference materials

Selected CRMs supplied by Rocklabs Ltd.			
Type	CRM	EV (g Au/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	1,086
	OXN49	7.635	62
Sulphide	SE19	0.583	23
	SH13	1.315	42
	SH24	1.326	1,039
	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 Gold Fields 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 (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. 1 kg 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 200 g were used if a Rocklabs blank was used.

Analytical blanks used in 2006/07 programs	EV (g Au/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	1,959
FALQ002	0.005	6,174

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 1 kg split at the rotary splitting of minus 2 mm crushed material for DD and RC samples. In the case of re-assays the preparation duplicate was a second 1 kg 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

Gold Fields started transporting 1 kg 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.

Gold Fields has represented that there are no known issues relating to tampering with samples. A fulltime Gold Fields 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.

Gold Fields 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 Gold Fields' 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 Gold Fields 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

Gold Fields 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 Gold Fields.

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 1 kg 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. Two quartz washes were used after every grind and the material was discarded into bins placed at the LM2. Gold Fields 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 Gold Fields. Periodic technical reviews were also carried by SGS auditors on request from Gold Fields.

13.7 Check assay methods

By agreement with Snowden, Gold Fields introduced an umpire check assay process by analysing 1:10 preparation duplicate samples by 1 kg 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 10 hrs 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 50 g fire assay. This step allowed estimation of leach efficiency as a % recovery factor.

Gold Fields also introduced analysis by gravity using on-site SB40 Falcon concentrators. Remaining sample rejects up to 30 kg in weight were pulverized to 90% passing 425 microns in vertical spindle Keegor mills and passed through the concentrators. The gravity concentrates and 1 kg 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, (ii) whole borehole check assays of reject samples. The results confirmed that 1 kg 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, Gold Fields 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 1 kg 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 the 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 Au/t; this suggests that tails samples should be taken for all BLEG samples > 0.5 g Au/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 1 kg SFA is also described together with work on replicate LWL69M assays. This analytical process has been accepted by Gold Fields 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 1 kg SFA on the duplicate. The 1 kg preparation duplicates for SFA were collected from the rotary splitters, then pulverized separately in an LM2 mill to 90% passing 75 microns. 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 Gold Fields 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 analyzed as a duplicate 50 g fire assay. The undersize tailing was not assayed to extinction.

Three hundred and forty eight pairs were analyzed using 1 kg LWL69M cyanide leach and 1 kg 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 Au/t and SGS Tarkwa SFA=211.27 g Au/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 Au/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 Au/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

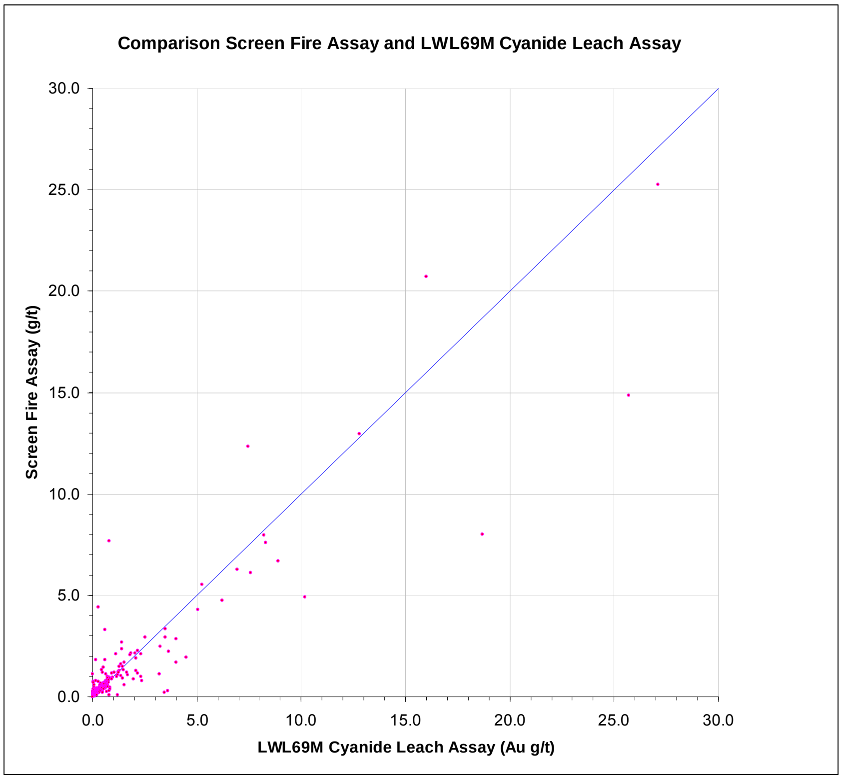


Figure 14.2 Preparation duplicates: QQ plot of LWL69M vs SFA results

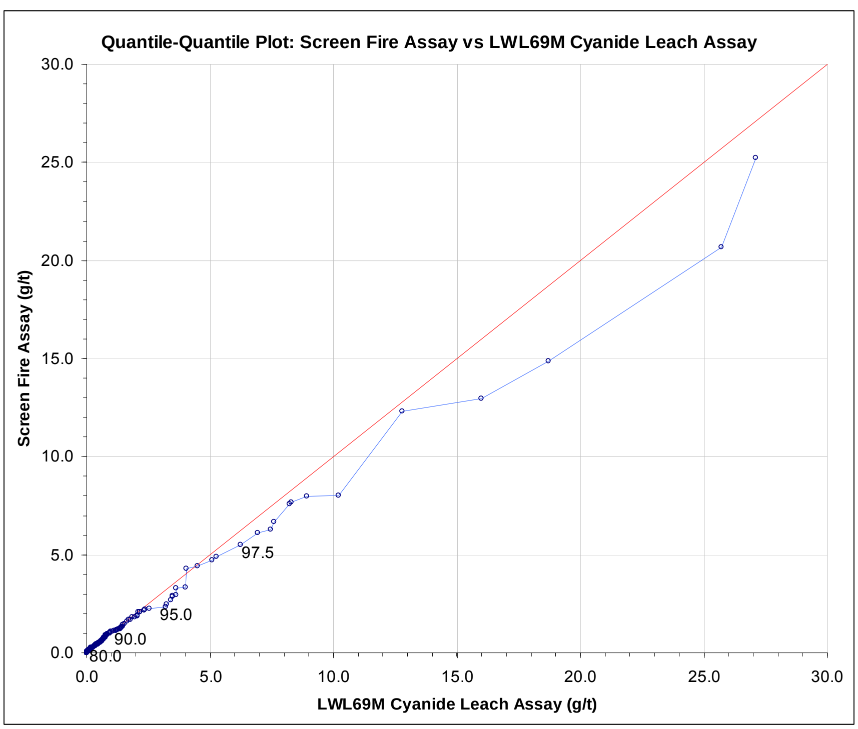
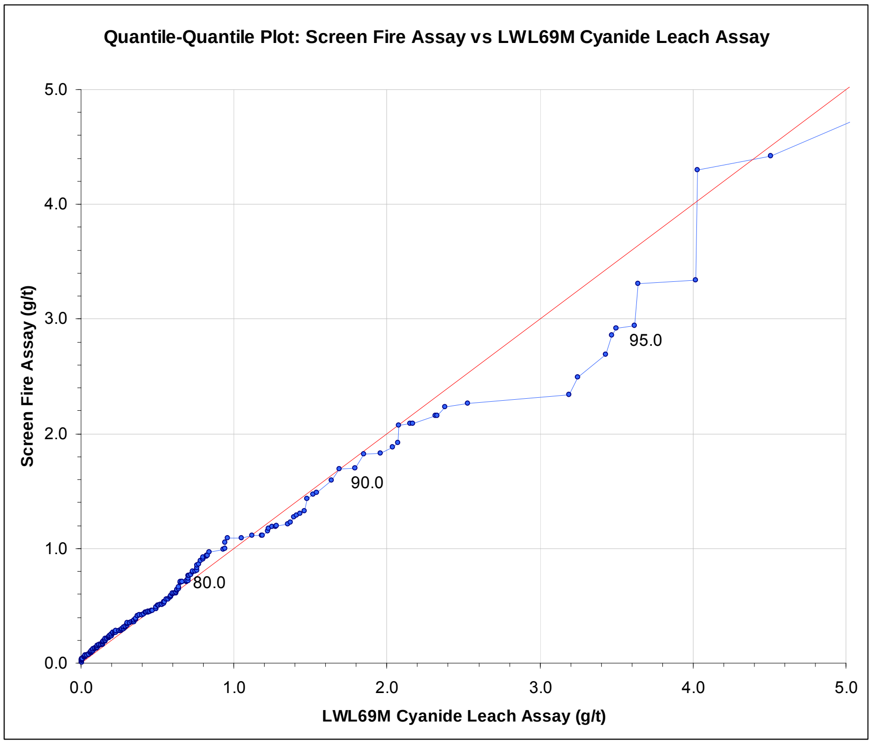


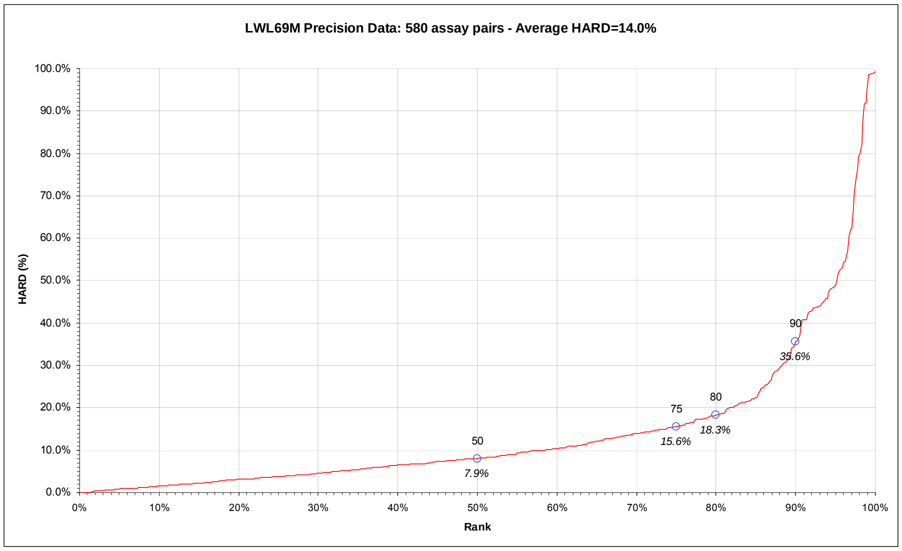
Figure 14.3 Close-up of Figure 14.2: QQ plot of LWL69M versus SFA results



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 1 kg samples. The large RC drill samples were crushed and then pulverized within an open circuit vertical spindle mill to P_{100} less than 425 microns. Assay aliquots of 1 kg were split out from the pulverized sample using a Cascade rotary splitter. Each 1 kg aliquot was then pulverized separately within a closed circuit LM2 mill to P_{90} 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 ±20% on 90% of pairs is desirable, whilst for pulp duplicates agreement of ±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 ±10% agreement on 90% of pulp duplicate pairs is equivalent to ±20% agreement on the HARD plot. In the same way, Long's ±20% agreement for 90% of coarse reject pairs is equivalent to ±40% HARD. The sampling protocol for these preparation duplicates split the aliquots at a nominal grain size of P_{100} -425 microns. This is coarse for normal pulps but fine for normal coarse rejects, implying that neither of Long's recommended values of ±20% and ±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 (±30 kg) was passed through a 1 : 1 riffle splitter to yield ±15 kg. This 15 kg sample was dried and pulverized within a vertical spindle mill to P_{100} -500 microns. Rotary splitters separated the sample into equal 1 kg 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_{100} -4 cm in a Bruno crusher and then passed through a Terminator crusher to achieve P_{80} -2 mm. Rodding was a problem from time to time with damp samples: when this occurred the sample (which failed P_{80} -2 mm) was dried and pulverized in a Keegor mill to P_{100} -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 1 kg samples were pulverized to P_{90} -75 microns in an LM2 mill and the entire 1 kg 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 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 one 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 1,000 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

Gold Fields 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.

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 is 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.

14.3.1 Abilabs fire assay

Table 4.2 compares fire assays from Abilabs to LWL69M assays from SGS.

Table 14.2 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 14.5 supports a bias between the Abilabs FA and the LWL69M re-assay data. The raw data QQ plot in Figure 14.5 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; LWL69M reports higher grades for the majority of the data distribution, but LWL69M reports lower grades for the highest grade samples (+5 g Au/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 Au/t, at which point Abilabs FA report higher grades for the highest value samples.

14.3.2 ITS fire assay

Table 14.3 details a comparison of fire assays from ITS and LWL69M assays from SGS.

The QQ plot in Figure 14.6 shows a complex bias pattern between ITS FA and SGS LWL69M. At low grades (<2.2 g Au/t) the LWL69M reports higher values than the ITS FA assays; above this value ITS FA systematically reports higher grades than the LWL69M assays.

Table 14.3 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 14.5 QQ plot of Abilabs FA vs SGS LWL69M re-assays

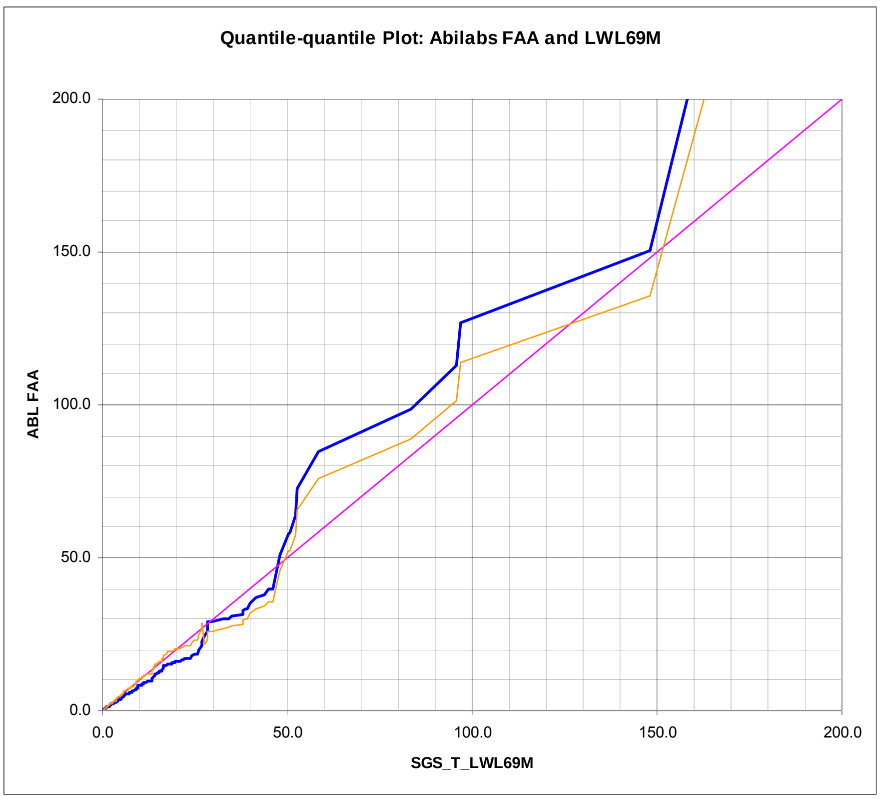
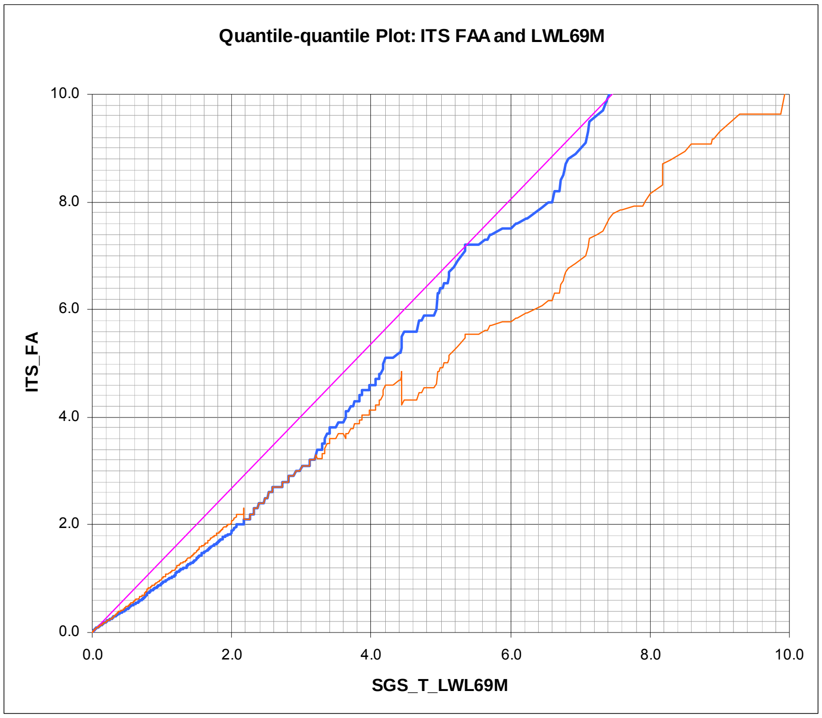


Figure 14.6 QQ plot of ITS FA vs SGS LWL69M re-assays



14.3.3 SGS Tarkwa BLEG

Table 14.4 shows a comparison of bulk leachable gold (BLEG) assays from SGS Tarkwa and the LWL69M assays from the site laboratory.

Table 14.4 Statistics of SGS BLEG assays vs SGS LWL69M re-assays

SGS Tarkwa BLEG	LWL69M	SGS Tarkwa BLEG	Remediated SGST BLEG 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 14.7 supports a systematic bias between the 2006 LWL69M assays and the historical SGS BLEG assays. LWL69M reports higher grades for all but the highest grade samples.

14.3.4 TransWorld BLEG

Table 14.5 shows a comparison of BLEG assays from TransWorld in Ghana and the SGS LWL69M assays.

Table 14.5 Statistics of TransWorld BLEG vs SGS LWL69M re-assays

TransWorld BLEG	LWL69M	TWLBLEG	Remediated TWL BLEG 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 14.8 supports a consistent and systematic bias between LWL69M assays and the TransWorld BLEG assays, with BLEG showing a negative bias compared to the LWL69M assays.

Figure 14.7 QQ plot of SGS BLEG vs SGS LWL69M re-assays

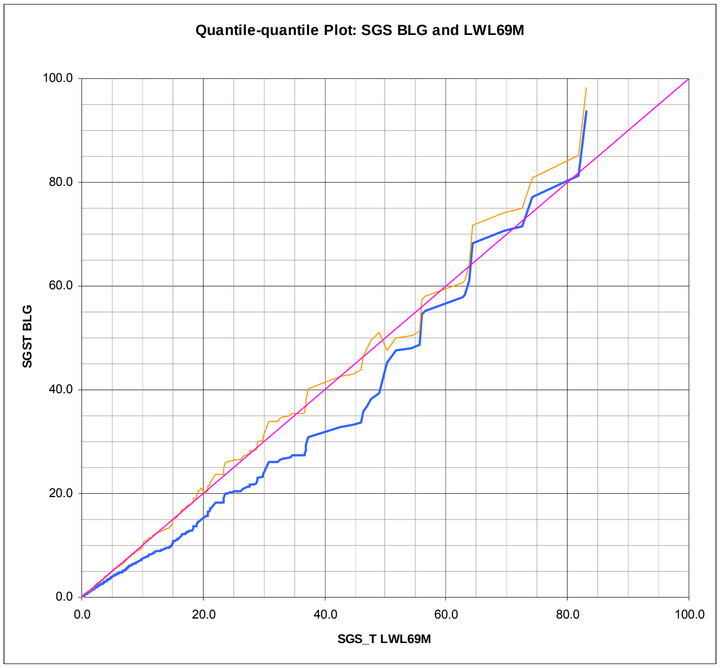
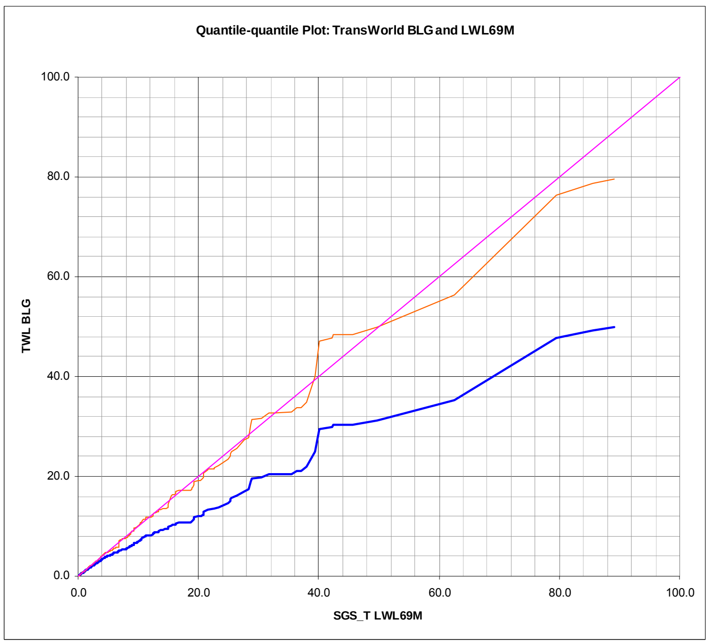


Figure 14.8 QQ plot of TransWorld BLEG vs SGS LWL69M re-assays



14.3.5 TransWorld LeachWELL

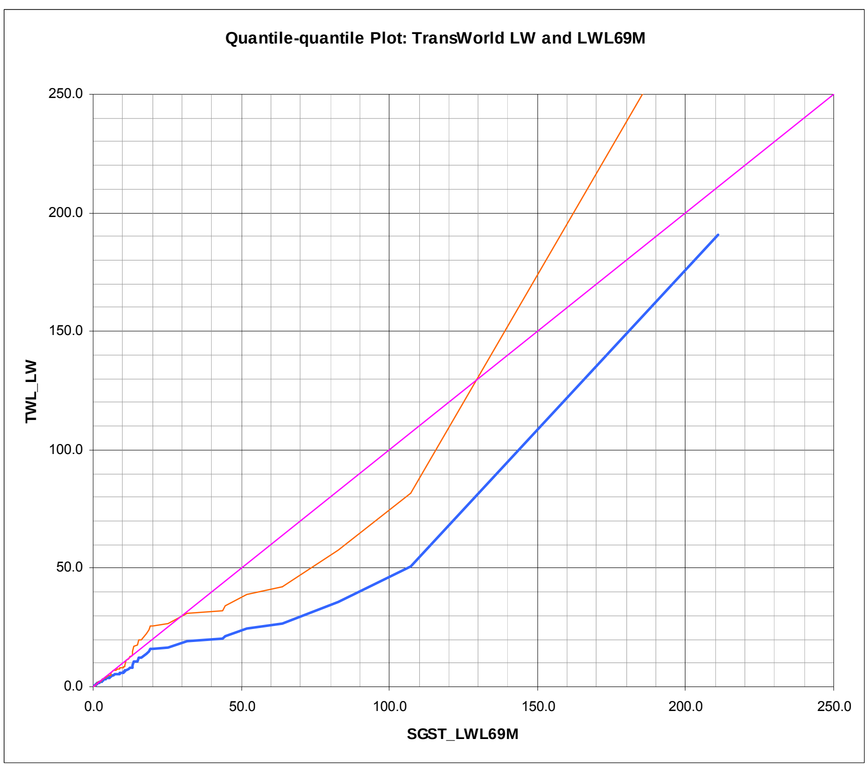
Table 14.6 details a comparison of TransWorld LeachWELL (LW) assays with the SGS LWL69M assays.

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

TransWorld LW	LWL69M	TWL LeachWELL	Remediated TWL BLEG 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 14.9 supports a systematic bias with SGS LWL69M reporting systematically higher grades than the corresponding TransWorld LW assays.

Figure 14.9 QQ plot of TansWorld LW vs SGS LWL69M re-assays



14.3.6 Ranger Minerals twin hole validation program

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.

Ranger fire assayed 21,844 samples at TransWorld Laboratories in Ghana. Sample rejects were stored on site in bio-degradeable 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 Au/t and std dev = 2.82, TWLFAA average = 0.95 g Au/t and std dev = 2.86).

Statistics of the twinned hole assay data are presented in Table 14.7. 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 14.10. The QQ plot comparing the SGS Tarkwa and TransWorld Fire Assays is presented in Figure 14.11.

Table 14.7 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 14.10 Twinned holes - scatterplot of Ranger FA vs SGS LWL69M

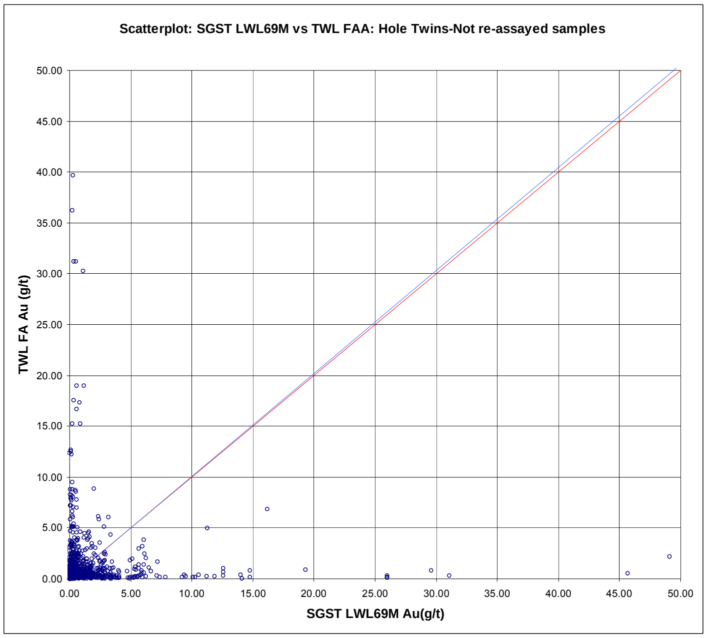
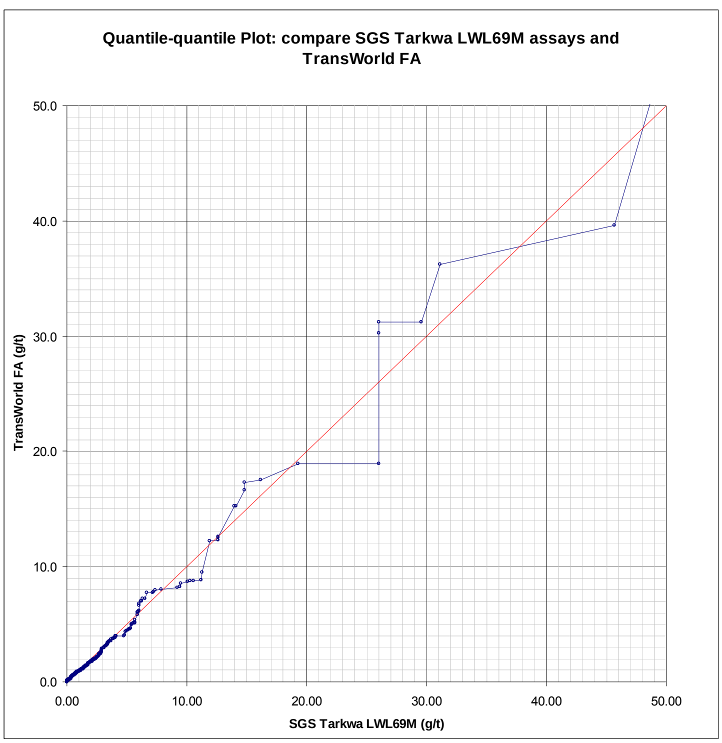


Figure 14.11 QQ plot of Ranger FA vs SGS LWL69M re-assays



14.3.7 Remediation Tables

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 1 g Au/t are compared in Table 14.10 before and after remediation. Remediation results in a comparatively small change in the proportion of samples exceeding 1 g Au/t.

Table 14.8 Factors applied to historical assay data

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

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 BLEG [remediated]	16,663	0.43	5.58	31.17	0.001	537.84	12.97	12.0%
SGS Tarkwa BLEG [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%
Total	**138,855**	**0.75**						100.0%

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 BLEG	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 BLEG	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

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 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 JV. 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 JV. Cyanidation tests on four EMZ-4 Oxide samples.
- Gold Fields Ghana Tarkwa Mine (2005) on behalf of Essakane JV. Cyanidation and column leach tests on an EMZ-3 sample.
- SGS/Lakefield Johannesburg (2005) on behalf of Essakane JV. 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 Cassiday and Associates (2005/6) on behalf of Essakane JV. Heap leach amenability testing, and gravity concentration, cyanidation and CIL testing on two Fresh ore composites (EMZ-12 Arenite and EMZ-13 Argellite) 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 JV. Comminution testing and mill circuit modelling on samples provided by McClelland Laboratories.
- 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 JV. 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 Piesold, and Patterson & Cooke to characterize the geochemical and geophysical properties of CIL tailings.

16.2 Ore types and samples

Metallurgical test samples were selected by Gold Fields during 2006 from a large number of RC and DD drillholes which intersected oxide, transitional and Fresh ore 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.

More recently in 2008, two saprolite ore samples were generated from test pits (P13 and P14) to conduct particle size analysis of in-situ saprolite.

16.3 Testwork results

Comminution test results on Fresh ore dictated the design of the grinding circuit. Comminution parameters determined from the test work and used for design purposes are summarized in Table 16.1. Using this data, the mills were selected as further described in Section 16.

Extensive leaching tests were conducted on the various ore types. A common characteristic of all Essakane ores is slow leaching kinetics if whole ore is cyanided without removing the coarse gold particles in a gravity concentrate. In this respect, Essakane ores are reported similar to the ores at Gold Fields' Tarkwa mine in Ghana. Figure 16.1 and Figure 16.2 demonstrate the impact of removing the gravity gold from the leaching circuit. While leaching is still on-going after 50 hours if coarse gold is present in the ore feed, leach extraction reaches a plateau after less than 20 hours if gravity gold is removed prior to the leaching stage. This is even better demonstrated when we look at results of cumulative recovery versus leach time for saprolite ore shown on Figure 16.3.

Optimization studies focussed on grind size and recovery versus operating costs concluded that the economical optimum grind size for hard rock was P80 minus 125 microns. The presence of activated carbon during leaching showed improved leaching kinetics and ultimately recoveries. This observation led to the use of CIL circuit as opposed to a Leach-CIP circuit.

Gravity concentration test work was included in the programs by SGS (Johannesburg) 2004 and 2005, Kappes Cassiday & Associates 2005/6, and McClelland Laboratories 2006 and 2007. Rougher and cleaner concentrate gravity recoveries are shown as a function of mass pull in Figure 16.4. The method of rougher and concentration varied considerably from laboratory to laboratory. The graph also does not discriminate between different ore types, their head grades or the initial grind size at which concentration was effected. For these reasons, the data are not necessarily internally comparable and should be considered only as an indication that the gold in Essakane ores concentrates readily. However, it must be noted that gold recovered in the rougher concentrate varied from 40% to 90%, which is relatively high for gold deposits.

Table 16.1 Comminution Parameter Summary

Samples #	Ai	Cwi	Rwi	Bwi	A*b	UCS	Lab	Rock Type	Sample Designation
1	0.046	19.7	16.9	14.6		203	SGS Jo	Fresh Argellite	EMZ8 Bbwi@106um
2				14.2			SGS Jo	Fresh Argellite	EMZ8 Bbwi@150um
3	0.322	23.4	16.3	15.6		205	SGS Jo	Fresh Arenite	EMZ9 BBwi@106um
4				13.4			SGS Jo	Fresh Arenite	EMZ9 BBwi@150um
5				8.8			SGS Jo	Oxide Saprolite	EMZ10 Levin @150um
6	0.197	3.8		7.8		47	Phillips	Saprock	3096 EDD 0140B (78-87m)
7			8.8	9.1	62.3		SGS	Saprock	3096 EDD 0140B (78-87m)
8				9.1			Phillips	Saprolite/Arenite	3096 EDD 0154 (39-47m)
9	0.036			8.1			Phillips	Saprolite/Arenite	3096 EDD 0154 (65-72m)
10	0.260	6.7		12.0		62	Phillips	Fresh Arenite	3096 EDD 0154 (129-137m)
11			13.7	12.6	34.7		SGS	Fresh Arenite	3096 EDD 0154 (129-137m)
12	0.109	7.1		13.8		82	Phillips	Fresh Argellite	3096 EDD 0154 (201-208m)
13			17.5	14.6	29.3		SGS	Fresh Argellite	3096 EDD 0154 (201-208m)
14	0.202	4.7		10.7		23	Phillips	Fresh Arenite	3096 EDD 1671D (110-118m)
15			11.9	10.6	48.1		SGS	Fresh Arenite	3096 EDD 1671D (110-118m)
16	0.158	6.3		13.2		90	Phillips	Fresh Argellite	3096 EDD 1671D (145-153m)
17			16.0	13.5	31.3		SGS	Fresh Argellite	3096 EDD 1671D (145-153m)

	Ore Types	Ai	Cwi	Rwi	Bwi	A*b	UCS	From Samples	
1	oxide	0.036	10.0	8.8	8.7	80.0	47	5, 8-9	Oxide design (above 78m)
2	transition	0.200	15.0	10.4	9.6	55.2	47	6-7, 14-15	Transition (78-118m)
3	fresh avg	0.183	21.6	15.4	13.2	35.9	204	1-4, 10-17	Fresh Average Design (110-210m)
4	fresh hard	0.179	21.6	16.1	13.8	31.8	204	1-4, 10-13, 16-17	Fresh Hard Design (129-210m)

estimate

data not used

Figure 16.1 Kappes Cassiday & Associates (2005) Whole Ore Leach

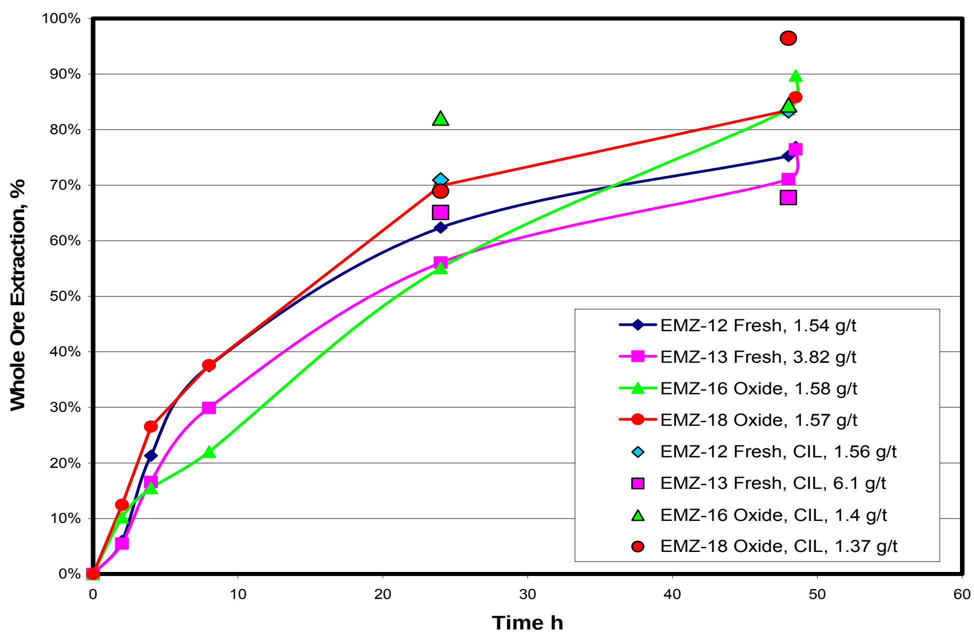


Figure 16.2 Kappes Cassiday & Associates (2005) Gravity Tailing Leach

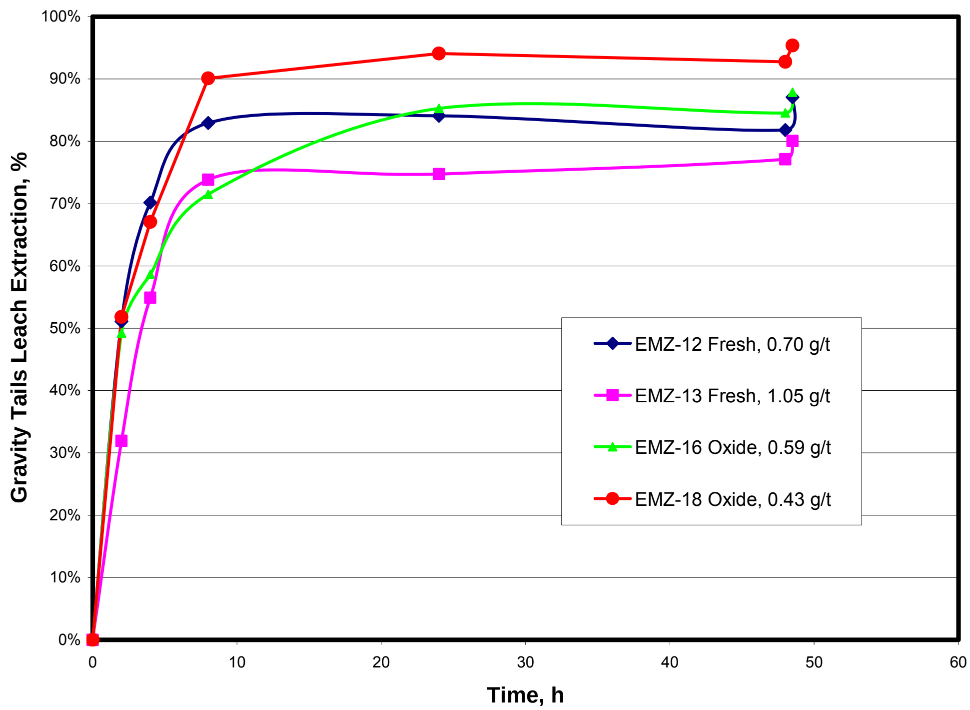


Figure 16.3 McClelland Laboratories (2006) Oxide Arenite Ore

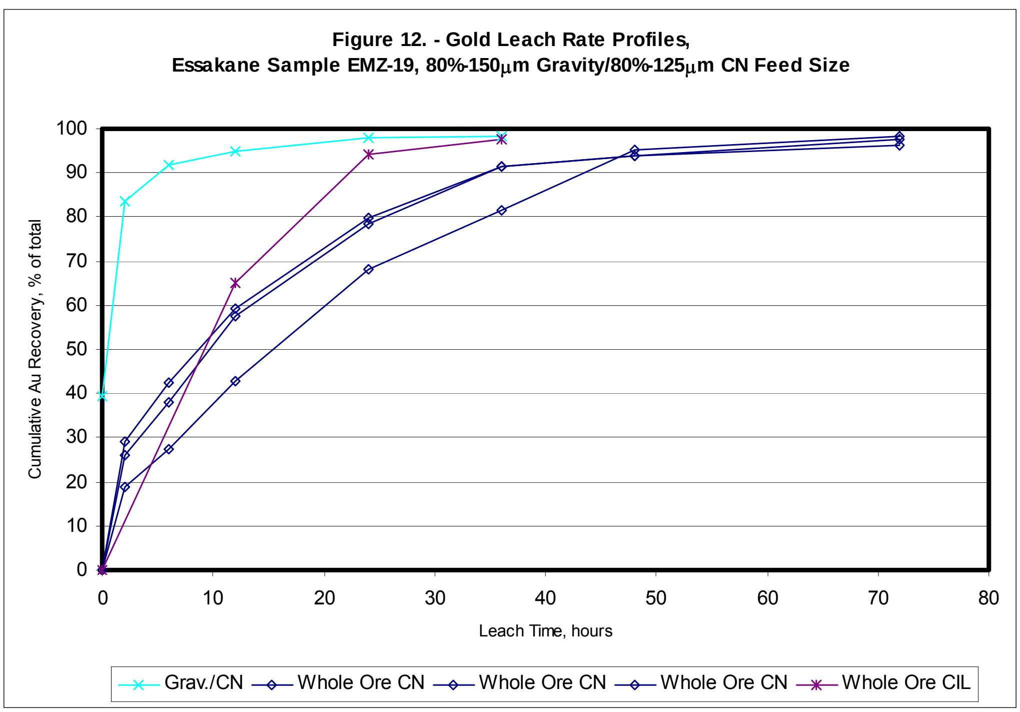


Figure 16.4 Summary of Test Work Gravity Data

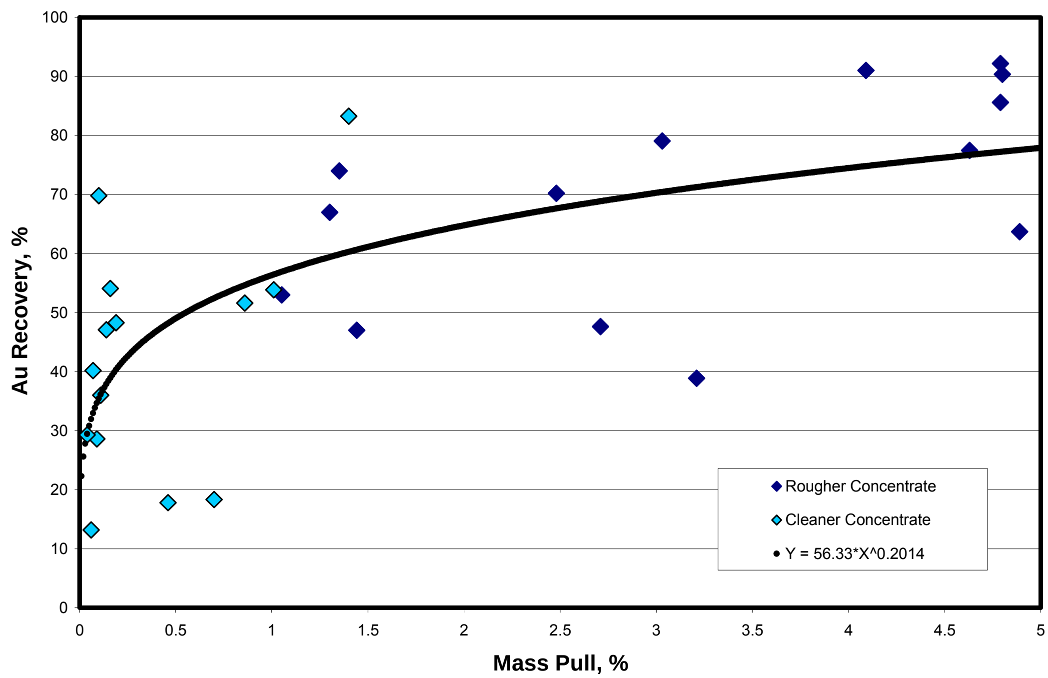
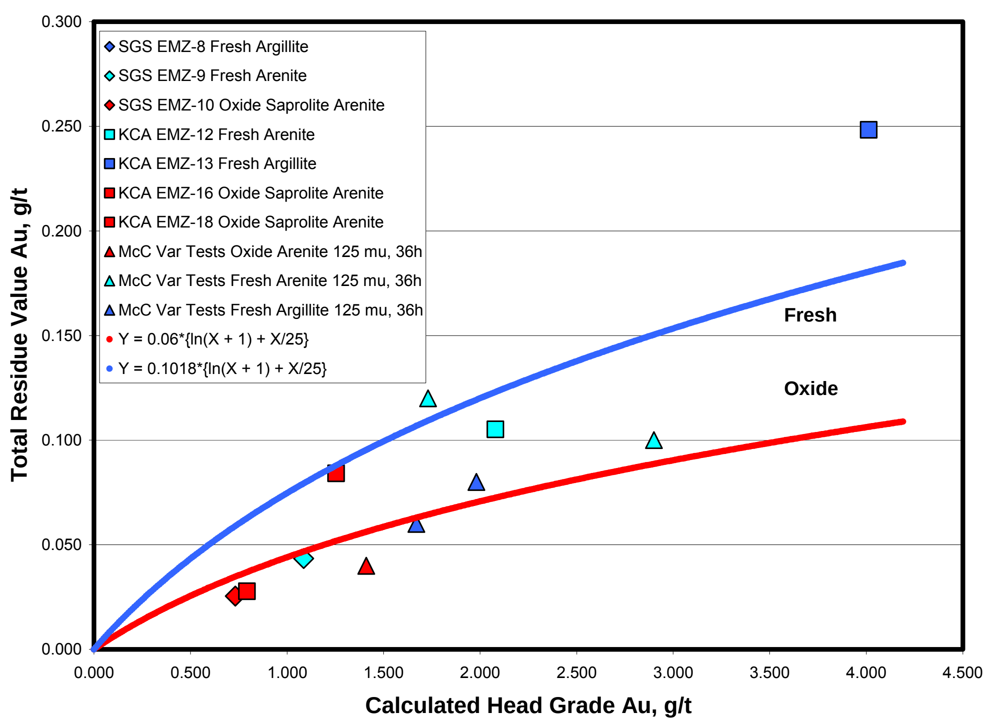


Figure 16.5 Residues vs head grades for relevant testwork programs



Metallurgical recoveries, as characterized by the final residue values for given ore types and head grades, are summarized in Figure 16.5. The data in Figure 16.5 were used to generate recovery curves versus head grades models. These curves allow for a 0.01 g Au/t tailings solution loss and a 0.01 g Au/t tailing contingency loss at a head grade of 2.0 g Au/t and therefore should represent practical extractions achieved in the plant. Translated into gold recovery terms, these models are:

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

Fresh Ore (Arenite, Argillite): Gold Recovery % = 99.59 – 10.18 x {ln[Au + 1]}/[Au]

The recoveries calculated from these relationships will vary with the head grade. At a 2.0 g Au/t head grade, for example, the Saprolite ore recovery projected from the model is 96.5%, while that for Fresh ore is 94.0%. Transition ore was considered a 50 : 50 blend of Saprolite and Fresh ore. These recovery models were used for ore Mineral Reserve estimation and for all financial evaluations.

16.4 Design criteria

The design criteria for the gold processing plant in the DFS was based entirely on the basis of 100% Fresh Ore feed:

- Plant throughput 5.4 million tonnes per year.
- 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 of 26 hours with a feed density of 40% solids.
- Leach time for fresh ore of 36 hours with a feed density of 50% solids.

Because the mine plan calls for predominantly saprolite and transition ore feed in the initial four years of operations followed by almost entirely fresh ore feed for the remaining mine life, Essakane saw an opportunity to increase mill throughput for those initial four years of operations, increase gold production, reduce unit operating costs and improve financial returns in spite of a marginal increase in capital costs.

Particle size analysis on two saprolite ore samples (in-situ) are presented in Figure 16.6 and Figure 16.7. On the basis of a targeted grind size of 80% minus 125 microns, it can be concluded that saprolite ore does not require any grinding since already finer that the targeted grind size. The same situation has been observed at many operations with similar oxidation pattern of the gold deposit; Saprolite ore requires only minimal grinding for the residual mineralized quartz veins within the saprolite mass and can be processed at a much higher rate and require significantly less grinding energy.

Furthermore, by taking advantage of grinding in cyanide, optimized slurry density by thickening ahead of leaching and inherent faster leaching kinetics of saprolite, it was concluded that no additional CIL tankage was required and no significant impact on recovery would be observed at a plant throughput of 7.5 M tpy for predominantly saprolite and transition ore feed.

Figure 16.6 Grain Size Distribution of In-Situ Saprolite (Pit 13)

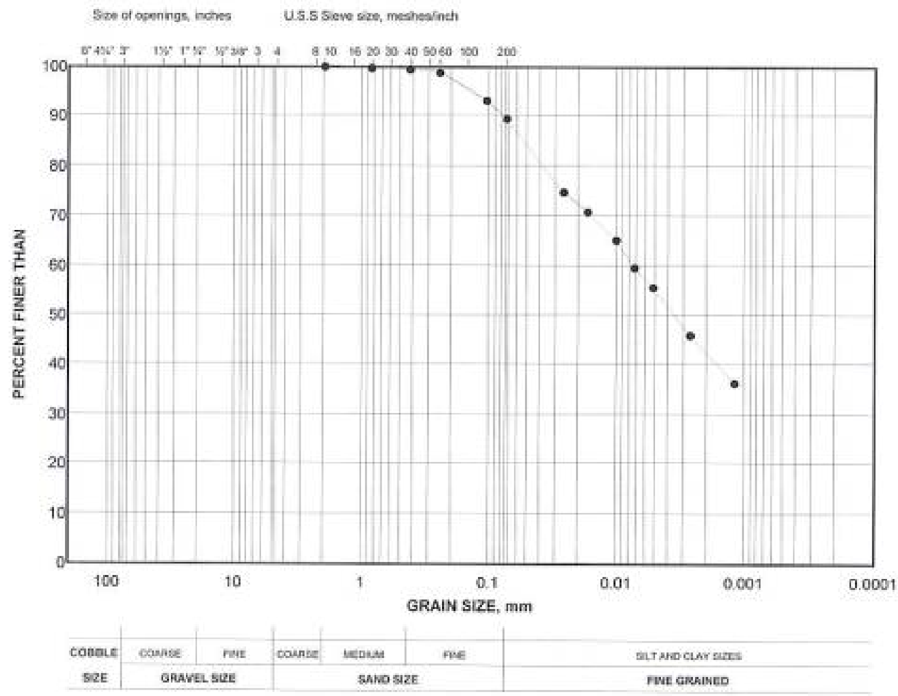
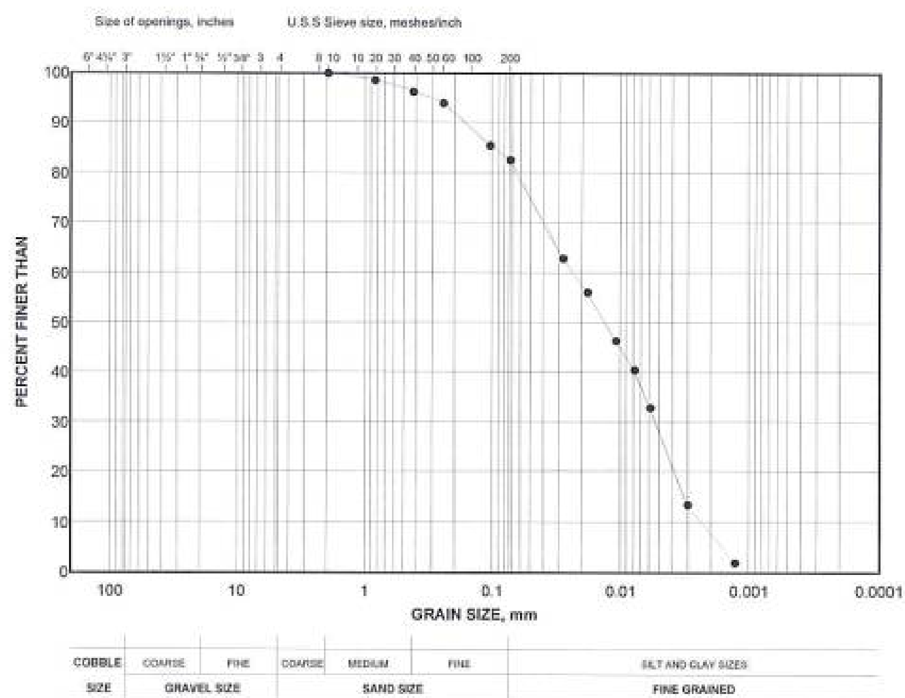


Figure 16.7 Grain Size Distribution of In-Situ Saprolite (Pit 14)



Consequently, design criteria were modified as follows:

	Saprolite/Transition Ore	Fresh Ore
Plant Operating Schedule	365 days	365 days
Plant Availability	95%	95%
Plant Throughput	7.5 M tpy 900 tph	5.4 M tpy 650 tph
Leach Slurry Density	45%	50%
Leaching Time	22 hours	36 hours

Finally, GRDMinproc was instructed to size pumps, pipes and screens with a capacity range allowing a throughput of 1,050 tph. The processing facility will be capable of a higher throughput in the early years if the mine feed is available at a cost of slightly lower recovery.

16.5 Process flow-sheet development

16.5.1 Design philosophy

The process flow-diagram is presented in Figure 16.8 and the process plant layout is shown in Figure 16.9.

The GRD Minproc process design accommodates the sequential processing of ores with widely differing physical characteristics whilst keeping the plant as simple as possible. Obviously, certain components of the circuit will have surplus capacity for some period of time during the mine life. For instance, the crushing and grinding circuit will be underutilized during the initial period of operations.

All components in the process flow sheet are based on well-proven technology, with equipment well suited for the application and material processed. This criterion led us to replace the mineral sizers in the DFS by a gyratory crusher. Furthermore, the addition of crushed ore stockpile (buffer) between crushing and grinding and a thickener between grinding and leaching, when compared to the DFS, will enable optimal operating conditions on a continuous basis.

By adopting this philosophy towards the development of the flow-sheet it was possible to design a plant that will achieve a dependable and 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 without experimentation and disruption in operations.

Figure 16.8 Process Flow Diagram

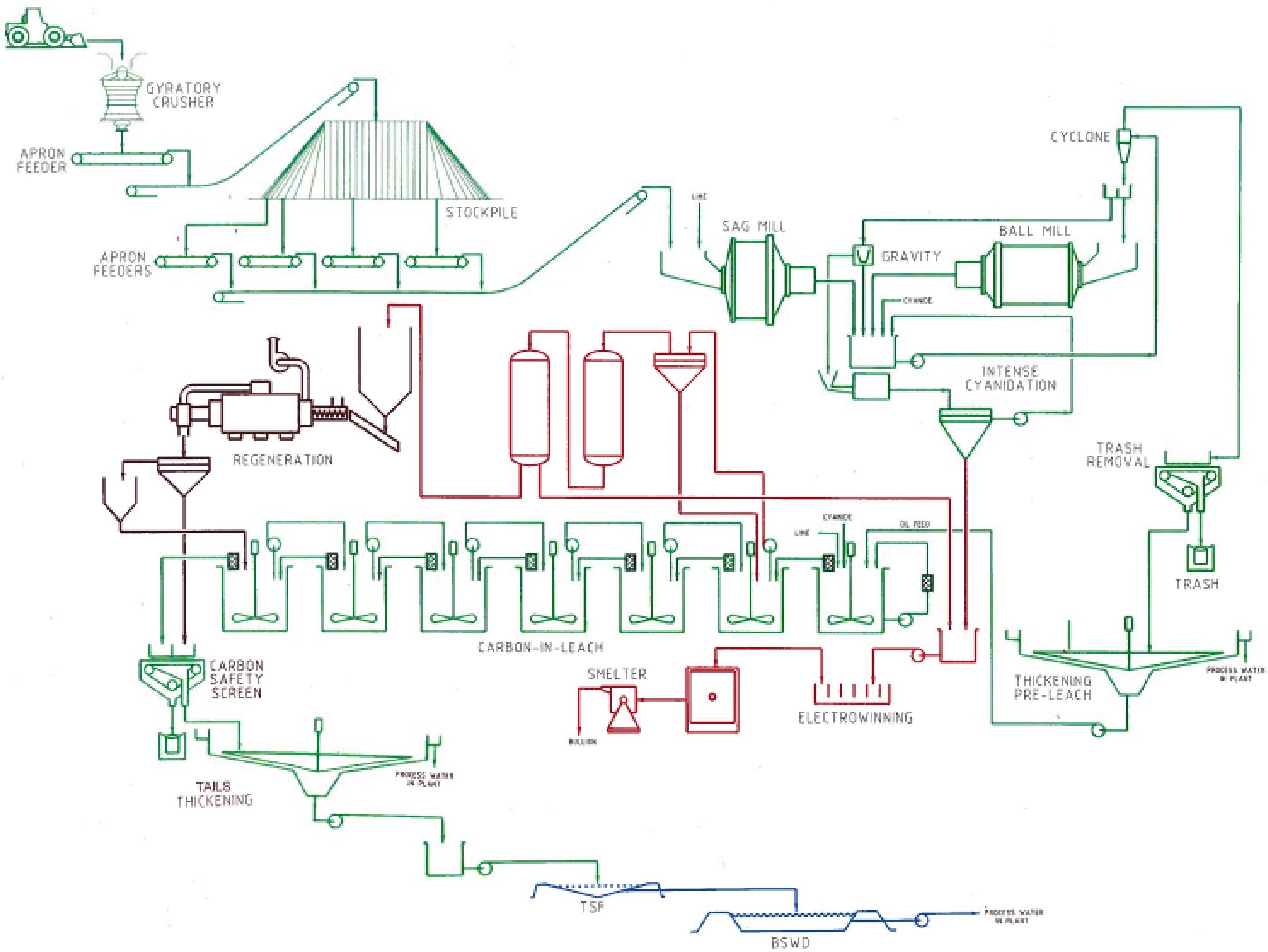
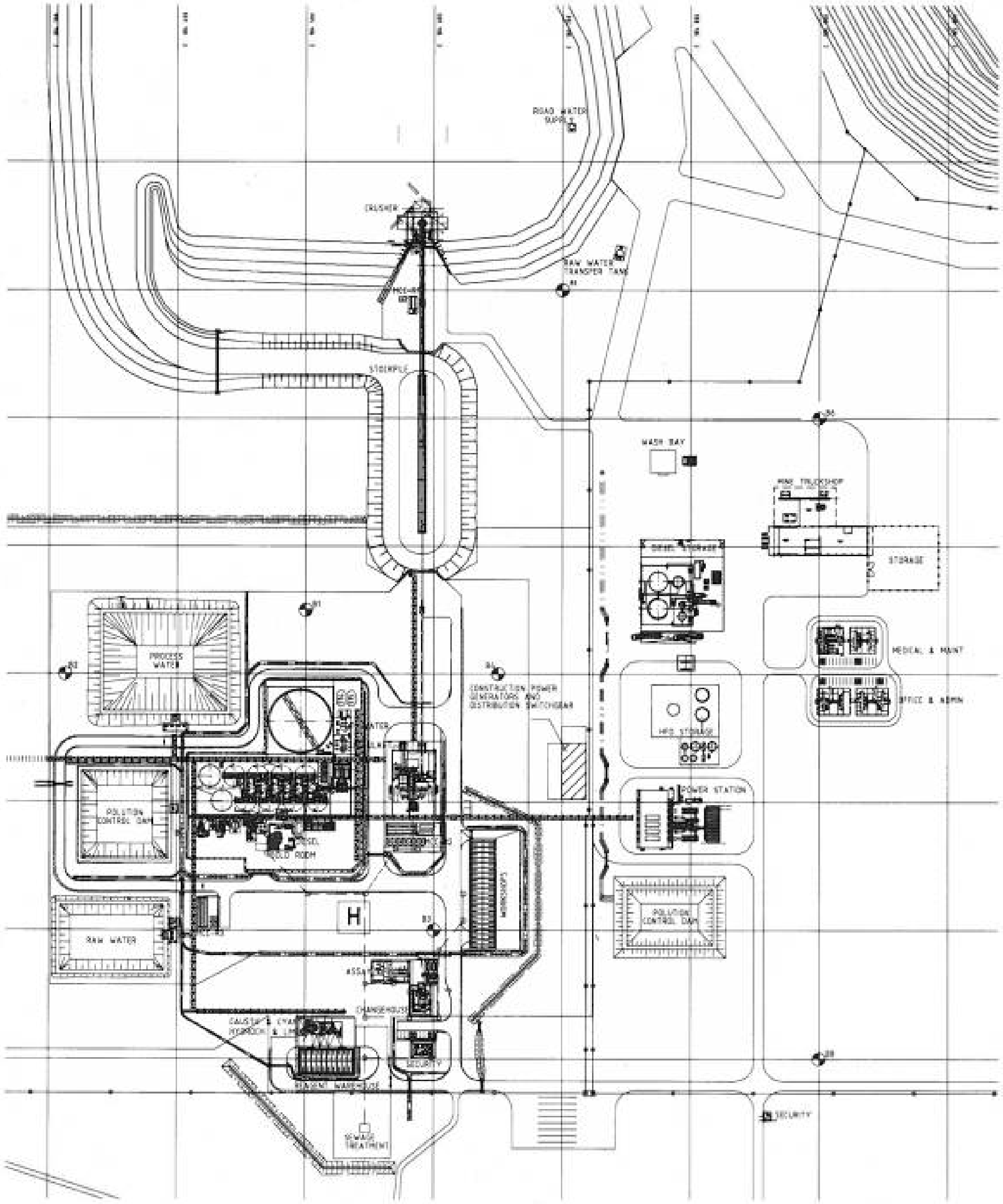


Figure 16.9 Process Plant Layout



16.5.2 Crushing

The crushing plant will use a Metso 50 X 65 gyratory crusher equipped with a 60-ton overhead crane and an impact breaker. An apron feeder under the crusher bin will feed a stacker conveyor via a sacrificial conveyor equipped with a belt magnet for the removal of tramp metal. The stacker conveyor will create a 15,000 t live crushed ore pile on top of a tunnel containing four apron feeders equipped with variable speed drives, each with a feeding capacity of 600 tph. A separate location will enable the stacker conveyor to accumulate crushed waste rock as required for roads or other constructions. If or when saprolite would be too wet to flow by gravity to the reclaim apron feeders, it has bee anticipated that mining trucks could bring the saprolite ore near the reclaim feeders, which could be fed continuously with a backhoe or dozer.

16.5.3 Grinding

The grinding circuit 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. Spillage in the grinding area is contained within a bunded area with dedicated spillage pumps.

16.5.4 Gravity concentration

The SAG and Ball mill discharge will be combined and pumped to a cyclone cluster. The cyclone overflow will be directed towards the leaching circuit via a pre-leach thickener. The cyclone underflow will be recirculated to the Ball Mill, with a fraction being fed through a gravity circuit composed of spiral and falcon concentrators. Shaking tables will further concentrate the rough gold concentrate. Since there is a high percentage of free gold in the ore, the removal of coarse gold prior to leach reduces the possibility of gold lock-up and improves the leach kinetics of the ore.

16.5.5 Carbon in leach (CIL)

Ore slurry to the CIL circuit will be thickened to ensure optimal and continuous slurry density through the leaching circuit.

The CIL circuit will have eight tanks, one leach tank and seven CIL tanks, each with a capacity of 4,000 m^3. As seen earlier in Section 16, the Essakane ore leaches rapidly once the coarse gold is removed. The CIL tanks are equipped with inter-stage screens to prevent the migration of carbon between tanks and with carbon transfer pumps. The CIL and pre-leach thickener areas are contained within a bunded area with dedicated spillage pumps.

16.5.6 Elution, electro-winning and regeneration

Loaded carbon is stripped of gold using the Zadra process and gold is recovered by electrolysis in conventional electro-winning cells. The elution system will be heated by diesel fired heaters and the stripped carbon will be reactivated in a diesel heated re-generation kiln and fed back to the CIL circuit.

16.5.7 Tailing thickening and pumping

Golder has recommended proceeding with thickened tailings to reduce water losses and maximize cyanide recirculation, while ensuring deposition of non-segregated tailings. A pilot plant test is planned during the final design stage to determine slurry viscosity and achievable slurry density in the thickener underflow especially for saprolite ore. The thickener selection for this application is a high compression-high density thickener (HCT). Costing is based on a HCT thickener 40 m in diameter located near the south-east sector of the tailings pond site. Tailings will be pumped at 40-45% solids to the tailings thickener through a dual HDPE tailing pipeline installed within a bunded area to contain any loss of slurry.

16.5.8 Gold room and smelt

The monthly gold production will fluctuate between 25,000 oz to 30,000 oz and it is anticipated that a gold smelt will be undertaken every second day. Gold will be smelted in an electric induction furnace and gold shipments are planned on a weekly basis.

16.5.9 Reagents

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

- Flocculants for the thickeners.
- Lime for pH control.
- Cyanide for gold leaching and elution.
- Caustic soda for elution.
- Hydrochloric acid for washing the carbon.
- Compressed air for CIL sparging.

Aside from compressed air, reagents will be delivered to site by road transport and will be transferred to the mixing tanks outside the processing plant fenced area. Once prepared, the reagents will be pumped to the day tanks within the high-security fenced area.

16.6 Process equipment selection

During the DFS several vendor specific types of equipment were identified. Subsequent to flow sheet changes caused by the increased throughput and other design modifications, new vendor quotes were obtained and capital costs were adjusted accordingly. The main processing equipment is listed here;

- Crushing:Metso Gyratory Crusher.
- Milling:FLSmidth SAG and Ball mills.
- Classification:Multotec cyclone clusters.
- Gravity concentration:Spirals, Falcon concentrator and shaking tables.
- CIL circuit:Kemix agitators and carbon screens.
- Thickening:Outokumpu pre-leach thickener.

The orders for the very long lead items have been placed and deliveries are timed to enable mill commissioning with ore in December 2009; the long lead items include the gyratory crusher, SAG mill, Ball mill and their ancillary equipment. The tower cranes for the grinding sector and the leaching sector are also on order and will be delivered in time for the start of mill construction.

16.7 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 crusher and 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 Schneider 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 May 2007 Mineral Resources reported at a cut-off grade of 0.5 g Au/t within a US$650/oz Whittle pit shell

Category	Tonnage (M t)	Grade (g/t Au)	Contained gold (M oz)
Indicated	73.4	1.62	3.82
Total Indicated	**73.4**	**1.62**	**3.82**
Inferred	16.1	1.66	0.86
Total Inferred	**16.1**	**1.66**	**0.86**

Table 17.2 June 2008 Mineral Reserve estimate based on US$600/oz

Category	Reporting cut-off (g/t Au)	Tonnage (M t)	Grade (g/t Au)	Contained gold (M oz)
	Oxide (0.44)	13.92	1.36	0.61
Probable	Transition (0.55)	12.39	1.58	0.63
	Fresh (0.66)	31.82	1.84	1.88
Total Probable		**58.12**	**1.67**	**3.12**
Proven		-	-	-
Total Proven		-	-	-
Total		**58.12**	**1.67**	**3.12**

Note: Mineral Resources are inclusive of Mineral Reserves. Tonnes and ounces have been rounded and this may have resulted in 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, Principal Consultant for Optiro. No change has occurred to the data and methodology that resulted in the Mineral Resources determination presented in the NI 43-101 Technical Report "Orezone Resources Inc.: Update on Essakane Gold Report, Burkina Faso" filed with the Canadian regulators in October 2007. Chapters 5 to 15 of this report are practically identical to the October 2007 Report and reflect only the change in ownership.

Mineral Reserves reported in Section 17 were based on surface mine optimization and mine planning studies undertaken by Mr. Louis-Pierre Gignac, ing., Sr Mining Engineer of G Mining Services Inc., and reviewed by Dr. Louis Gignac, President and Principal of G Mining Services Inc. Mineral Reserves calculations were updated from the October 2007 NI 43-101 to reflect a higher gold price of $600/oz (instead of $500/oz), a larger mining block size, and different operating costs used in cut-off grades calculations and pit design. Mineral Reserves reported in this NI 43-101 differ materially from the previous October 2007 report because of the higher gold price, but also because the October 2007 ultimate pit design was established with the January 2007 Resource Model (a smaller resource), then superimposed over the May 2007 Resource Model. G Mining Services worked entirely with the May 2007 Resource Model.

All persons named above are Qualified Persons as defined in NI 43-101. Snowden, Gold Fields International Services Limited, and G Mining Services Inc., 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 and G Mining Services are 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 obtained a mining permit for the property.
- Essakane has represented that there are no outstanding legal issues; no legal actions or injunctions are pending against the Project.
- Essakane has represented that the mineral and surface rights have secured 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. Mineralization is controlled by a combination of lithological and structural controls and post-mineralization 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.

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.

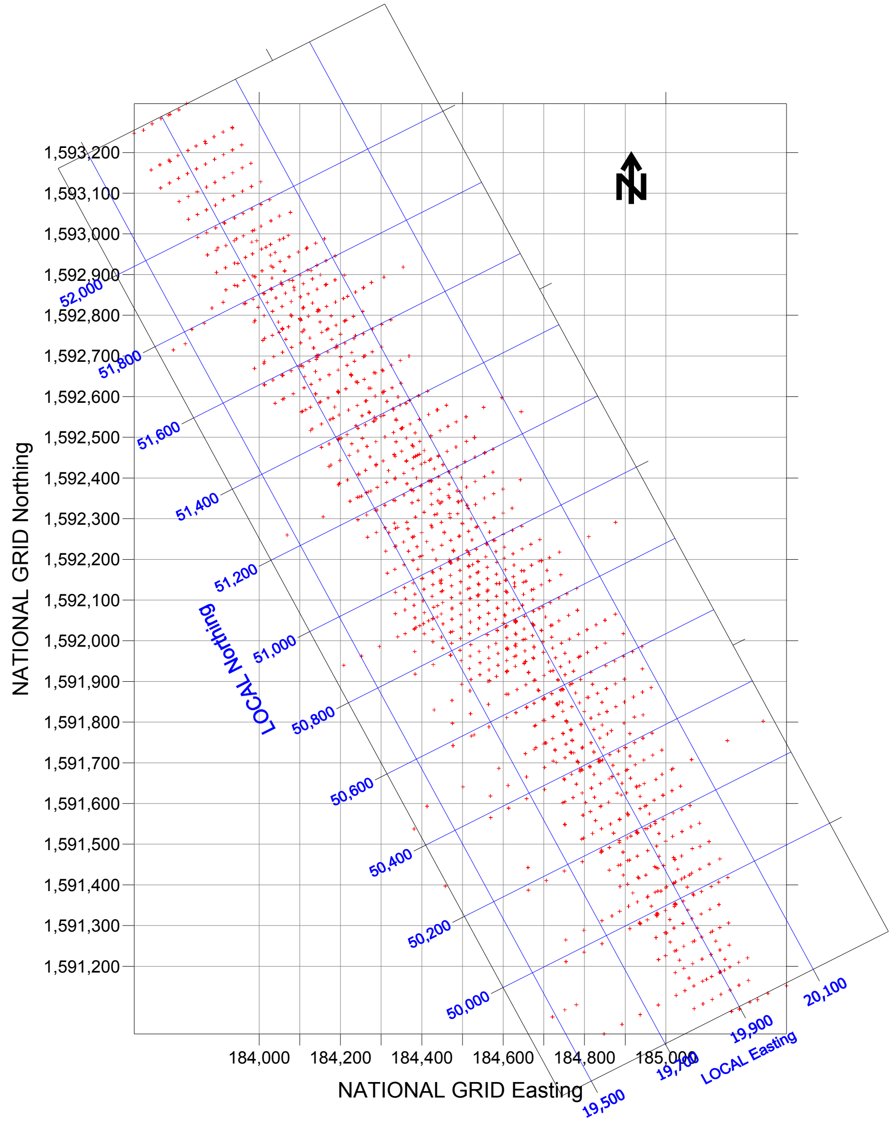
An additional waste model, representing all materials not described by the main arenite, FW argillite and FW arenite block models, was created at the panel scale. This waste model contains the HW argillite (Rock Code 100) and unmineralised dyke material (Rock Code 600) and has no grade estimates. Relative 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 = standard deviation/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 17.3.

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



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 3 m and 1 m 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 (3 m composites)

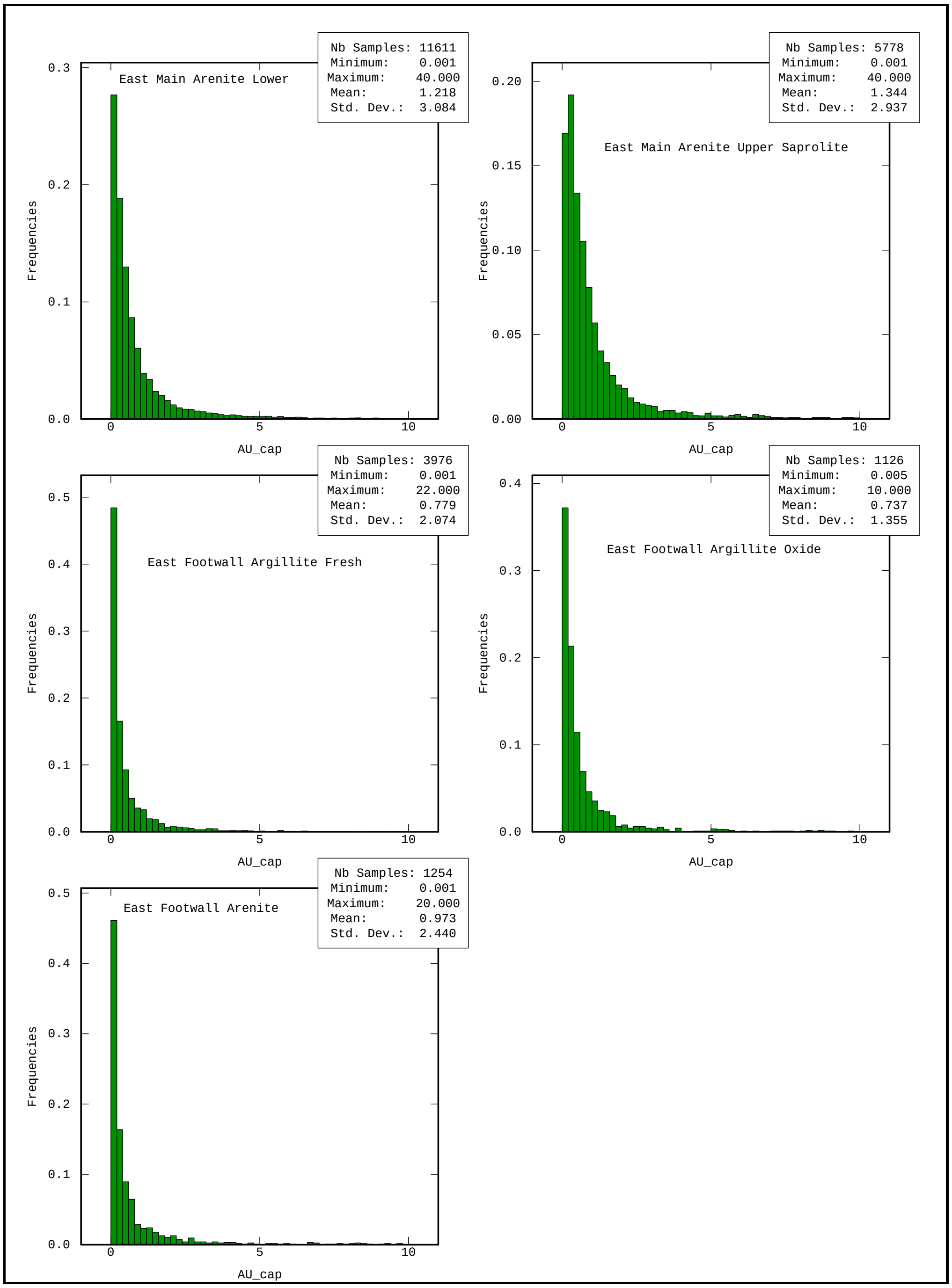
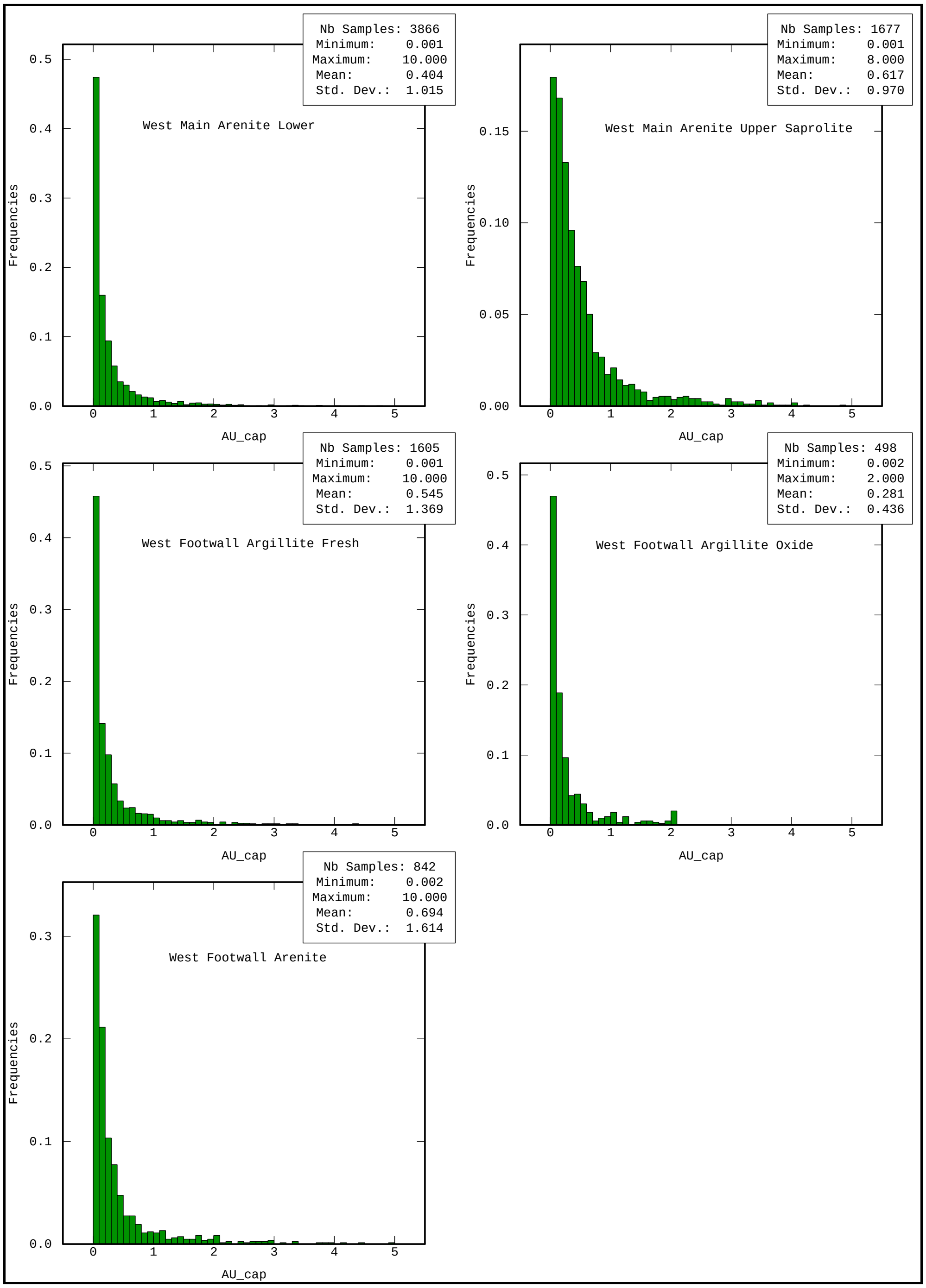


Figure 17.3 EMZ West Limb - Histograms & data statistics for capped gold grades (3 m composites)



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 (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 (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 Lower	Au	11,611	1.27	4.23	3.33			40	
	Au-capped	11,611	1.22	3.08	2.53	4.3%	46.9%	99.82%	20
East main Arenite Upper Saprolite	Au	5,778	1.39	3.95	2.83			40	
	Au-capped	5,778	1.34	2.94	2.19	3.5%	44.6%	99.81%	10
East FW Argillite Fresh	Au	3,976	0.83	3.33	3.99			22	
	Au-capped	3,976	0.78	2.07	2.66	6.7%	61.2%	99.70%	11
East FW Argillite Oxides	Au	1,126	0.77	1.67	2.17			10	
	Au-capped	1,126	0.74	1.35	1.84	4.1%	34.3%	99.56%	4
East FW Arenite Fresh	Au	1,254	1.06	3.44	3.26			20	
	Au-capped	1,254	0.97	2.44	2.51	7.9%	49.8%	99.52%	6
West MainArenite Lower	Au	3,866	0.44	1.47	3.37			10	
	Au-capped	3,866	0.40	1.01	2.51	7.2%	52.3%	99.64%	13
West Main Arenite Upper Saprolite	Au	1,677	0.66	1.46	2.22			8	
	Au-capped	1,677	0.62	0.97	1.57	6.5%	56.0%	99.46%	9
West FW Argillite Fresh	Au	1,605	0.62	2.07	3.35			10	
	Au-capped	1,605	0.55	1.37	2.51	11.9%	56.3%	99.07%	14
West FW Argillite Oxides	Au	498	0.33	0.85	2.57			2	
	Au-capped	498	0.28	0.44	1.55	15.6%	74.0%	97.99%	10
West FW Arenite Fresh	Au	842	0.77	2.19	2.86			10	
	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	19,600.0	20,250.0	25 m
Northing	49,525.0	52,475.0	50 m
Elevation	-49.0	275.0	6 m

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 (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:

a) 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.

b) 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).

Table 17.5 Variogram parameters

Backtransformed Models	Rotation				Structure 1				Structure 2				Structure 3			
Domain	X	Y	Z	C_0	C_1	Range X	Range Y	Range Z	C_2	Range X	RangeY	Range Z	C_3	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

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 in Table 17.6 for illustration.

Table 17.6 Mean of declustered drillhole data vs OK large panel estimates

	3m drillhole composites		Volume weighted OK large panel estimates	
Domain	**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 3 m 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 modelling within the UC process was validated in the following way:

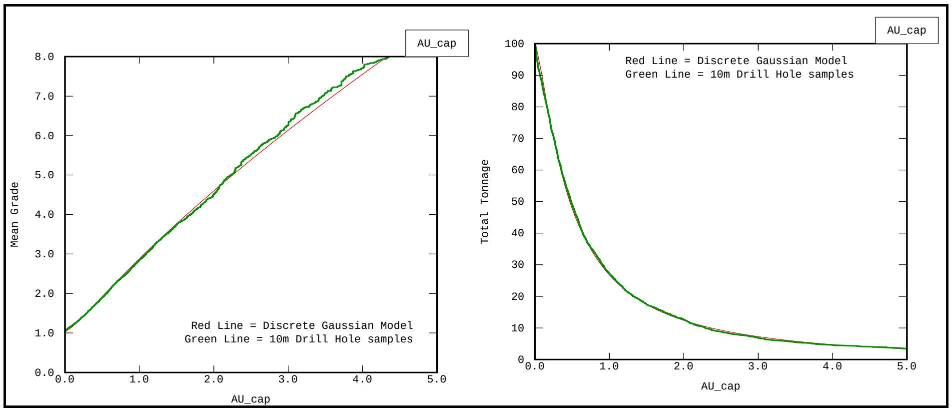
a) In the East Main Arenite lower domain, the samples were composited to 10 m downhole lengths.

b) The histogram of 10 m downhole sample grades was then estimated using the same change-of-support model employed in the uniform conditioning.

c) 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 g Fire Assay of LWL69M tailings to calculate % Leach in the following way:

% Leach = LWL69M-solution grade/(LWL69M-solution grade + Tailing grade) 100

The leach efficiency was modeled within each of the major RCODE lithology domains. These data are presented in Table 17.7 by domain and weathering type. There are sufficient data to derive factors for all units except for FW argillite upper saprolite which daylights at the southern end of the design pit shell.

Table 17.7 Summary statistics for % Leach of LWL69M assays

RCODE	Data	Upper Saprolite	Transition	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 rock type and weathering domains as a set of step-wise conditional means. The final models are presented in Table 17.8 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 17.8 %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
	90.02%		**97.63%**		**98.12%**		**98.31%**		**98.96%**	
FW Argillite Fresh	0.00	0.70	0.70	2.00	2.00	10.00	10.00	100.00		
	90.68%		**94.91%**		**97.20%**		**98.76%**			
FW Arenite Fresh	0.00	1.00	1.00	3.50	3.50	100.00				
	93.09%		**95.66%**		**98.36%**					
Main Arenite Upper Saprolite	0.00	0.20	0.20	0.40	0.40	1.30	1.30	2.00	2.00	100.00
	92.32%		**97.66%**		**98.12%**		**98.40%**		**99.05%**	
Main Arenite Lower Saprolite	0.00	0.20	0.20	0.70	0.70	1.50	1.50	5.00	5.00	100.00
	84.74%		**97.88%**		**98.25%**		**99.11%**		**99.26%**	
FW Argillite Upper Saprolite	Use Main Arenite Upper Saprolite Curve									
FW Argllite 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.

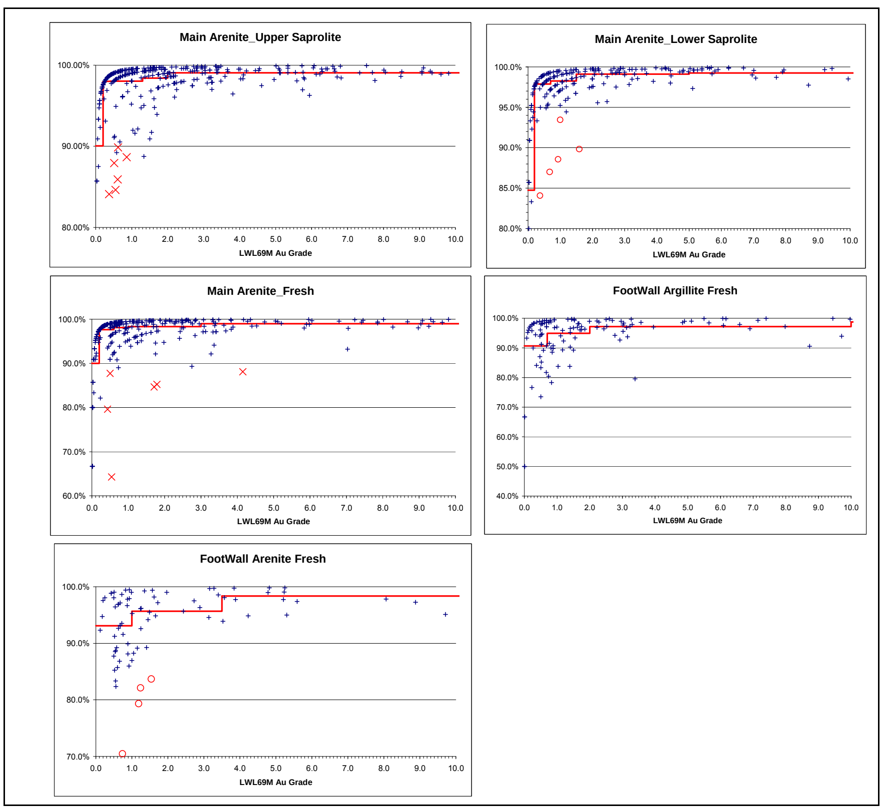
17.2.11 Density

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[2] 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 Mineral Reserves determined by a final pit design at US$600/oz.

17.2.14 Mineral Resource reporting

Snowden reported Mineral Resources for the May 2007 models constrained by US$650/oz Whittle pit shells using both Indicated and Inferred Resources. The Whittle input parameters were the same as used in the DFS design.

Table 17.9 reports the Total Mineral Resource and the May 2007 Mineral Resources constrained within a US$650/oz Whittle pit shells 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).

Table 17.9 May-07 resource estimate for the EMZ

	Cut-off Grade	0.5 g Au/t			1.0 g Au/t		
	Category	**Tonnes (M t)**	**Grade (g/t)**	**Total Au ('000 oz)**	**Tonnes (M t)**	**Grade (g/t)**	**Total Au ('000 oz)**
Total Mineral Resources unconstrained by US$650/oz pit shell	Indicated	78.4	1.58	3,993	42.2	2.33	3,160
	Inferred	27.4	1.44	1,272	13.7	2.17	960
Mineral Resource within a Whittle US$ 650/oz pit shell using Indicated and Inferred Resources	Indicated	73.4	1.62	3,822	40.5	2.36	3,064
	Inferred	16.1	1.66	860	9.5	2.31	706

1. Mineral Resource estimates have been developed as Uniform Conditioned estimates using 50m x 25m x 6m panels estimated by ordinary kriging. Application of a Discrete Gaussian change-of-support model has been made to derive conditional estimates of 5m x 2.5m x 3m selective mining units (SMUs). Resource are reported in terms of these SMU estimates.
2. Mineral Resources include an Information Effect correction to account for future grade control procedures assuming a 5m x 5m grade control sample spacing.
3. Some historical assay data have been remediated in reference to paired assay results derived using a LeachWELL rapid cyanide leach process.
4. Cyanide leach assay data have been used within the estimation. These results have been adjusted to reflect in situ gold grades in preference to cyanide-leachable gold values.

17.3 Assumptions, methods and parameters – Project reserve estimates

Open pit optimization was performed using the Whittle software based on the Lerch-Grossman algorithm. As part of the Whittle pit optimization process, a series of nested optimal pit shells were generated by incrementally increasing a revenue factor that varies the gold price with all variables either held constant or changing as a function of the revenue factor. The final pit shell used for mine design corresponds to the one that maximizes the net present value (NPV) of the project.

17.3.1 Pit optimization

The Whittle optimization was run on a re-blocked version of the May-07 Resource Model. This constitutes already a major difference over the DFS and the NI 43-101 published in October 2007. The pit design in the DFS was generated on the basis of the January 2007 Mineral Resources, and the pit shell was superimposed over the May 2007 Mineral Resources; this shortcut was caused by time pressure at the time. The May 2007 Mineral Resources are compared to the January 2007 Mineral Resources in Table 17.10. Obviously, the indicated resources increased by 9.7 M t (14%) and 540,000 oz (16%) from the January 2007 Model to the May 2007 Model and created a significant difference in the size of the Gold Fields (DFS) optimized pit when compared to the current optimized pit. Only mineralized blocks classified as indicated were considered for optimization. The resulting optimal Whittle pit shell was used as a guideline for mine design.

Table 17.10 Comparison of January 2007 and May 2007 Resource Model

	January 2007 Total Resource Model (COG 0.5 g Au/t)	May 2007 Total Resource Model (COG 0.5 g Au/t)
Indicated		
Tonnage (M t)	68.7	78.4
Grade (g Au/t)	1.56	1.58
In situ gold (M ozs)	3.45	3.99
Inferred		
Tonnage (M t)	23.5	27.4
Grade (g Au/t)	1.50	1.44
In situ gold (M oz)	1.13	1.27

The base case gold price used for pit optimization and design is US$600/oz. This compares to US$500/oz used in the DFS and the October 2007 NI 43-101. Based on the guidelines stated by the SEC in the United States, it is usual for the gold producers to use the average gold price of the previous three years to calculate mineral reserves. On December 31, 2007, this trailing-3 year average gold price was US$582/oz; at year-end 2007, the majority of gold producers used US$600/oz for reserves calculation as reported by PriceWaterhouse Coopers. It can be noted that the 36-month trailing average gold price on May 1st, 2008 is US$637/oz.

A range of pit shells at various gold prices were generated and used in subsequent scheduling of the pit. A 3% royalty is payable to the government of Burkina Faso and a $2.50/oz transportation and refining charge was assumed.

The optimization parameters used to generate optimal pit shells are summarized in Table 17.11.

Table 17.11 Summary of Optimization Parameters

Gold price ($/oz)	600		
Royalties (3%) ($/oz)	18		
Transp. & refining ($/oz)	2.50		
Total selling costs ($/oz)	20.50		
Net gold price ($/oz)	579.50		
Fixed ore based costs	$11.6M	$10.8M	$10.0M
Processing rate (t/y)	7,500,000	6,450,000	5,400,000
Parameters	Saprolite	Transition	Fresh rock
Metallurgical recoveries[1]	variable	variable	variable
Mining recovery[2]	100%	100%	100%
Mining dilution[2]	0%	0%	0%
Ore based costs			
Processing & Power	6.67	8.52	10.36
G&A	1.55	1.67	1.85
Total	8.22	10.20	12.21
Recovered Cut-off grades	0.44	0.55	0.66
Note 1: Calculated on a block by block basis according to ore type and in situ gold grade.			
Note 2: Included in block size and re-blocking			

Metallurgical recoveries were characterized by a Global Head Grade Model with metallurgical recovery rates as a function of head grade. As described in section 16, these curves allow for a 0.01 g Au/t tails solution loss and a 0.01 g Au/t tails contingency loss at a head grade of 2.0 g Au/t and therefore should represent practical recoveries achieved in the plant. Translated into gold recovery terms, these model relationships are:

- Saprolite (Oxyde) Au rec (%) = 99.7 – 6 x {ln(Au+1)}/(Au)
- Transition (Saprock) Au rec (%) = (Saprolite Au rec % + Fresh Au rec %)/ 2
- Fresh Au rec (%) = 99.5 – 10.18 x {ln(Au+1)}/(Au)

Saprock or Transition ore is considered a 50:50 blend of saprolite and fresh ore. These recovery models were used for mineral reserve estimation and for all financial evaluations.

The Resource Model used for mine design is the May 2007 Resource Inventory Model prepared by Gold Fields as part of the DFS work. The block size for this model is 2.5 m x 5 m x 3 m.

This model was imported into the Gemcom software for mine planning. The small block dimensions of the Resource Model were deemed too small in relation to what can be selectively mined considering the equipment selected to achieve a high mining rate. This mismatch required adjustments to the Resource Model for mine planning purposes. A small mining unit (SMU) size of 10 m x 10 m x 3 m was determined appropriate for the digging equipment selected and according to the grade control grid pattern to be implemented (8 m x 8 m).

The approach of combining blocks from the Resource Model to create larger SMU appropriate for the mining equipment in effect diluted the grades by 8.2%. To construct this re-blocked model, eight of the small blocks in the Resource Model were combined together. Mineralization at the contact of waste blocks is diluted through the re-blocking process; if the larger SMU is below cut-off grade, then some of the ore is lost.

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 6% discount factor.

Operating costs were estimated based on fuel costs of $0.81/litre for HFO and $1.09/litre for diesel delivered to the site and including government duties and taxes; these fuel costs are based off a crude oil Brent price of $65/bbl.

A total cost of US$ 10.77 M based on 32 M t mined per year was estimated for Mining Fixed Costs which includes:

- Mine supervision.
- Maintenance supervision.
- Equipment damage.
- Surface Mine dewatering.
- Ancillary mining equipment.

Variable mining costs for load, haul and dump were calculated using cycle times and fuel consumption 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 crusher pad. The 3 m flitches were combined into 12 m benches for this exercise. Haul profiles were digitized with a maximum 10% gradient. Loading cost for ore is slightly more, as mining will be accomplished with a 10 m^3 excavator, and waste will be loaded by a 15 m^3 shovel.

The ore based costs used to calculate the marginal cut-off grade is variable according to the weathering status. The ore based cost consists of processing (including power generation) and G&A costs. Saprolite has a processing cost of $6.67/t and a G&A cost of $1.55/t. Transition ore has a processing cost of $8.52/t and a G&A cost of $1.67/t. Fresh rock has a processing cost of $10.36/t and a G&A cost of $1.85/t.

Unit mining cost used for optimization consist of a fixed portion and a variable portion in relation to volume for drilling, blasting and grade control and a variable portion for haulage at depth in the pit. This first variable cost based on volume is $0.54/BCM for Saprolite, $0.978/BCM for Transition and $1.42/BCM for Fresh rock. The waste loading and hauling reference cost from pit rim is $0.772/t for Saprolite, $0.763/t for Transition and $0.759/t for Fresh rock. In addition, an incremental bench cost of $0.0286/t per 6m bench is factored to increase haulage costs at depth.

17.3.2 Pit design

Mine design criteria is based on a conventional surface mine operation using 15 m^3 shovels for waste loading and a 10 m^3 excavator for ore loading. Haulage is accomplished by a mixed fleet of 100 t-class and 140 t-capacity 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 17.12.

The haul road was designed for the largest vehicle being the 140 t-capacity haul trucks. For double lane traffic the minimum width is 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 is 10%.

The final surface mine design is 2,500 m along strike and approximately 550 m in width. Its highest elevation is 266 mRL and its lowest elevation is 56 mRL, which gives a maximum depth of 210 m.

The final surface mine design is shown in Figure 17.6.

Table 17.12 Geotechnical Configuration for 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	72	85
Berm width (m)	4	4	6.44	4	4	6.44
Ramp width (m)	26	26	26	-	-	-
Bench stack angle (degrees)	32	47	57	32	47	60
Overall Slope Angle (degrees)	31			49		

17.3.3 Mine phases

Mining will be accomplished with four phases to achieve the final pit limits. The objective of pit phasing is to improve economics by feeding the highest grade during the earlier years and to defer some stripping towards the latter years.

The starter pit (Phase 1) is centered in the middle of the deposit and is completely internal to the other pit limits. It is centered on high grade saprolite material. This starter pit has no common walls with the other phases.

The Phase 2 pit starts to establish the final limits on the upper benches. The ramp access established is the same as the final pit ramp which will eliminate re-work in this regard. The Phase 2 pit continues to sink the pit deeper but defers some stripping on the east hangingwall which is barren of mineralization. These benches could subsequently be mined with 12 m benches to reduce costs.

The Phase 3 pit achieves pit bottom for the south part of the pit and defers the mining of the north part where lower grade material is found. The ramp is common to the final pit design to a depth of 93 meters.

For the mining of Phase 4, the possibility of in pit dumping exists as the southern half of the pit will be completely mined out. Reduced waste hauling costs are possible for this last phase but were not factored in the mining costs for the present study.

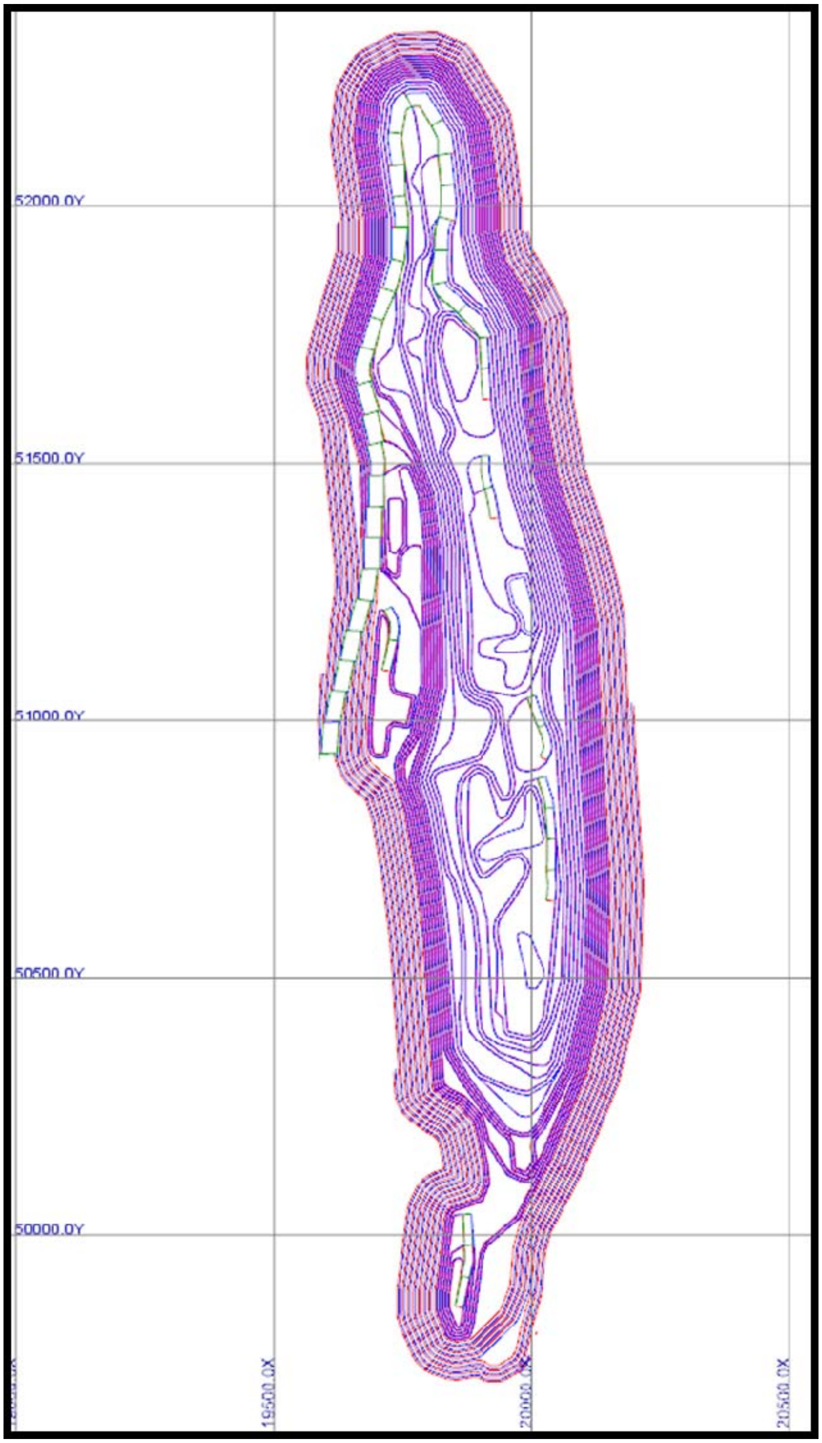
Mining of the phases was planned with a 60 m lag, equivalent to 10 benches.

17.4 Mineral Reserve

The reserves for the surface mine design are reported according to the Canadian Institute of Mining, Metallurgy and Petroleum's (CIM) standards. According to these standards, Resource Model blocks classified as Measured and Indicated are reported as Proven and Probable Reserves respectively. Owing to the above reporting standards, the Inferred Resources cannot be included as Reserves and consequently were not used in pit optimization and production scheduling.

Since no Measured Resources were classified in the Mineral Resources Inventory, the Mineral Reserves are classified as Probable only.

Figure 17.6 Plan of the final surface mine design



Mineral Reserves for the Project at US$600/oz are 58.1 M t of ore at an average grade of 1.67 g Au/t for an in-situ content of 3.12 M oz with a strip ratio (waste/ore) of 2.71.

The mineral reserves are presented in Table 17.13. Non-reserve material listed represents Inferred resources contained within the US$ 600/oz pit design but not used in the pit optimization process.

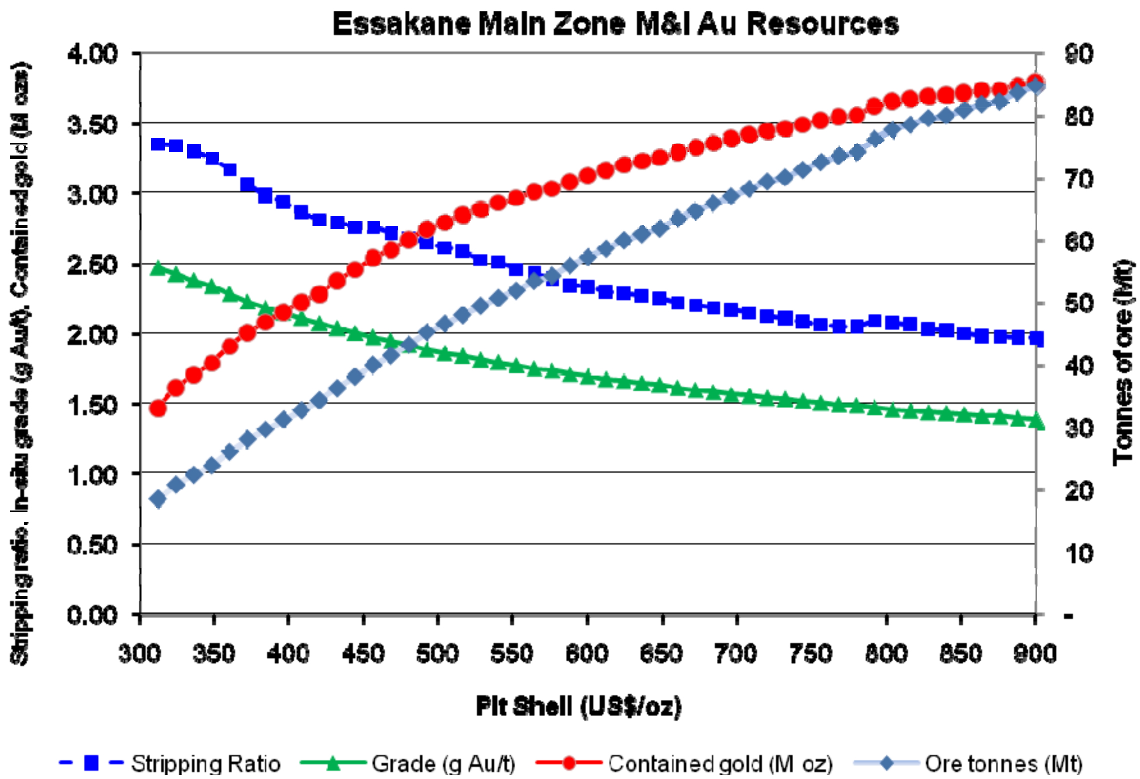
Table 17.13 Essakane Mineral Reserves

Weathering Type		COG	Total				
			Tonnage (000t)	Grade (g Au/t)		Au (koz)	
		g Au/t		Rec	In-situ	Rec	In-situ
Probable Mineral Reserves	Saprolite	0.441	13,915	1.313	1.363	587	610
	Transition	0.547	12,389	1.506	1.581	600	630
	Fresh Rock	0.656	31,817	1.733	1.839	1,773	1,881
	Total		58,122	1.584	1.670	2,960	3,121
Non-Reserve Material	Saprolite	0.441	139	0.847	0.907	3.8	4.1
	Transition	0.547	12	1.247	1.272	0.5	0.5
	Fresh Rock	0.656	1,019	1.868	1.972	61.2	64.6
	Total		1,170	1.740	1.838	65.5	69.1
Waste (including non-reserve material) (000t)			157,467				
Strip Ratio			2.71				

These Mineral Reserves compare to 46.4 M t at 1.78 g Au/t with a stripping ratio of 3.05 in the DFS and October 2007 NI 43-101 Report; the increase in the current reserves (May 2008) is due to the larger May 2007 Mineral Inventory and a higher gold price.

In order to evaluate the possible impact of gold price on mineral reserves, many Whittle pits at various gold prices were ran on the May 2007 Total Resources model after reblocking to the 10 m x 10 m x 3 m SMU. All design parameters, like operating costs and slope angles, were kept constant but cut-off grades for the three ore tyes were modified by the gold price. Figure 17.7 presents the evaluation of ore tonnage and grade, contained gold, and stripping ratio within Whittle designed pits at various gold prices.

Figure 17.7 Essakane main zone M&I Au Resources



18 Other relevant data and information

18.1 Water Supply

Because the Essakane Project is located in a semi-desertic area, water supply and management is a critical aspect of the Project.

18.1.1 Hydrology

The Project is located in the north east of Burkina Faso and the temperature ranges from 10 – 46 ºC with evaporation rates of 3,000 mm per year. The mean annual rainfall is 503 mm with an estimated 100 year maxima of 143 mm in a 24 hour period.

The main source of water for the Project is the Gorouol River, which flows immediately north of the EMZ deposit. The mean runoff in the Gorouol River is conservatively estimated to be 91 M m^3/year and is concentrated in the three months July to September.

Several options were considered in the DFS before deciding to build a diversion dam with spillway on the Gorouol River that will divert water, during the wet season, into an off channel reservoir. The pond upstream of the diversion dam will have a storage capacity of 1.37 M m^3 but will decrease in time by siltation and annually by evaporation. This water will be available for use by the local population.

Recent work by Golder considers the off-channel reservoir as part of a network of at least three reservoirs, with two storing water from the Gorouol River and the third one, the Bulk Storage Reservoir, being used also to accumulate and recirculate decant water from the tailings pond. The strategy developed for the network of reservoirs is to ensure full reservoirs at the end of the rainy season and to sequentially draw down each reservoir, the Bulk Storage Reservoir being last, to satisfy demand during the dry season and consequently reducing evaporation surfaces. Alternatively, the Bulk Storage Reservoir can be fed to the mill continuously to optimize cyanide recovery.

18.1.2 Mine hydrogeology and pit dewatering

The water table is approximately 20 to 80 m below surface in the vicinity of the surface mine (higher at north end which is close to the Gorouol River). A water aquifer exists in the saprolite and saprock zones and consequently, dewatering of the proposed surface mine is planned prior to start-up. Numerous boreholes will be drilled and the groundwater will be extracted using submersible pumps. The groundwater flow is expected to reduce over the life of the mine since the pumping capacity will be greater than the recharge. The initial dewatering volume is predicted to be 90,000 m^3 per month.

Approximately 25 boreholes will be drilled around the circumference of the surface mine to an average depth of 65 m and separate submersible pumps installed in each borehole. All boreholes will be connected to a common pipeline around the surface mine perimeter that will deliver water to the plant clean water pond for use in the process. A branch line from the perimeter main will also provide water to a small pond near the mine perimeter for use by water trucks for dust suppression when water is not available from the in-mine pumping system. Water requirements for dust control are estimated at 630,000 m^3 /y.

To dewater the pit bottom, a sump will be created in the bottom of the surface mine to hold approximately 500,000 m^3 water from wet season rains on the surface mine catchment area. A pair of two submersible pumps and two slurry pumps (one operating, one standby), the latter on a skid frame for ease of movement, will be installed in the bottom of the surface mine to pump any seepage and wet season excess water from the mine. Additional slurry pumps will be added over time to stage pump the water with increasing pit depth. At the commencement of operations, the in-mine pumping system will be designed to pump approximately 500,000 m^3 (planned surface mine wet season storage capacity) in a period of two months (100 l/s) over a total head of 50 m. Water will be pumped to the plant dirty water pond for use in the process with a branch line to the pit reservoir for use by water trucks for dust suppression.

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

18.1.3 Potable water and sewage treatment

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

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

18.1.4 Water balance and management

Golder prepared a model based on the water flow diagram shown in Figure 18.1. The Essakane project site will be a zero-discharge operation.

Water that has to be collected at the Essakane site includes:

- water that accumulates in the tailings storage facility (TSF), including direct precipitation.
- precipitation pumped from the mine (mine infrastructure, rock/overburden dumps, open pit mine).
- direct precipitation on the dust control pond.
- direct precipitation on the Bulk Storage Reservoir.
- direct precipitation on the off-channel storage reservoir.
- dewatering well yields.

The inflows to the TSF are the water in the tailings discharged from the mill and runoff from precipitation. In order to minimize the water storage in the TSF, the tailings water discharged from the mill and the runoff will be diverted to the Bulk Storage Reservoir on a flow through basis. The losses to the system are water retained in the tailings, evaporation and seepage. The difference between the inflows and losses is the water available for recirculation to the mill via the Bulk Storage Reservoir. Process water is also recirculated from the tailings thickener to the mill. Because the tailings will be non-segregating thickened tailings, the volume of water available from tailings storage for return to the process plant is significantly reduced in the later years of production when the ore is predominantly rock.

The dewatering wells combined are designed to yield 125 m^3/hr (GCS, 2007). A portion of this flow will be used to meet the mill fresh water requirements. Any excess water from the wells can be used as process water or for dust control.

The system must be in balance year over year. If there is a deficit of water in the system, the required volume is taken from the Gorouol River through the off-channel storage reservoir. If there is a surplus of water in the system during the wet season, the water can be stored in the Bulk Storage Reservoir to be used in the dry season.

The water in the Bulk Storage Reservoir could be potentially contaminated because it receives the net inflow from the TSF. The Bulk Storage Reservoir must be designed with adequate capacity, including containment of inflow from a PMP event, to prevent discharge to the environment.

The precipitation pumped from the surface mine will be stored in the dust control pond. Any deficit in water for dust control from the pond will be compensated with water from the dewatering wells or the off-channel reservoir. In a wet year, runoff from precipitation can be held in the pit sumps, if needed. Any excess of water not used for dust control can be used in the process plant.

Figure 18.1 Water Flow Diagram

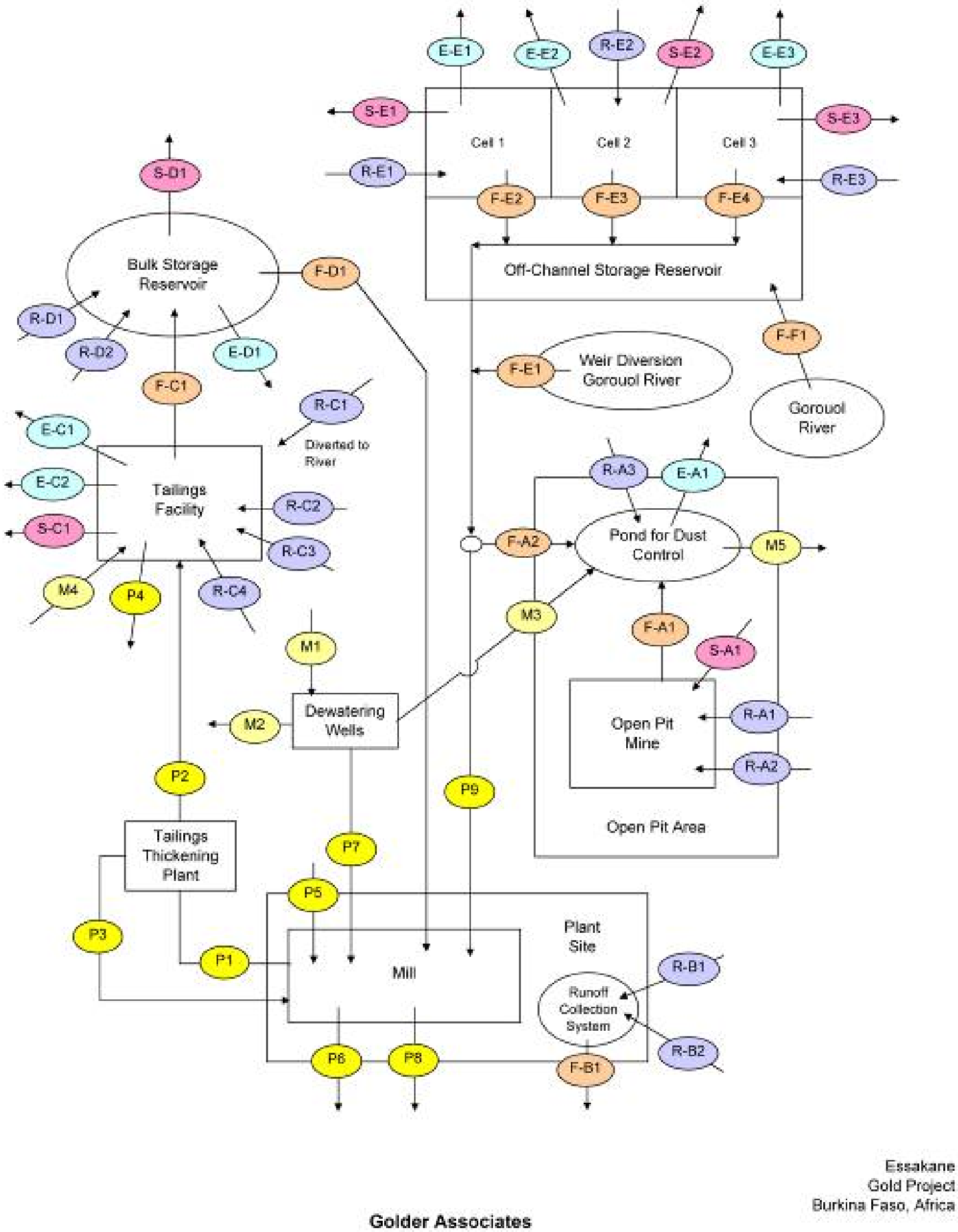


Figure 18.2 List of Flows

Area	Flow No.	Description	
Flows associated with processing the ore and tailings production (P flows)	P1	Discharge from the mill to the tailings storage facility	
	P2	Discharge from the tailings thickener to the disposal facility	
	P3	Process water that is recycled to the mill	
	P4	Water retained in the tailings in the storage facility	
	P5	Moisture (water) in the ore going into the mill	
	P6	Water leaving the mill in the concentrate	
	P7	Fresh (clean) make-up water required in the mill	
	P8	Losses in the mill to evaporation and spillage	
	P9	Required make-up water to run the mill	
Flows associated with runoff from precipitation (R flows)	R-A1	Open Pit Mine	From natural ground
	R-A2		From pit walls
	R-A3		From pond surface (direct precipitation)
	R-B1	Plant Site	From natural ground
	R-B2		From prepared surface
	R-C1	Tailings Storage Facility	From natural ground (diverted to the Gorouol River)
	R-C2		From wet tailings beach
	R-C3		From dry tailings beach
	R-C4		From pond surface (direct precipitation)
	R-D1	Bulk Storage Reservoir	From natural ground
	R-D2		From pond surface (direct precipitation)
	R-E1	Off-Channel Storage Reservoir	From pond surface Cell 1 (direct precipitation)
	R-E2		From pond surface Cell 2 (direct precipitation)
	R-E3		From pond surface Cell 3 (direct precipitation)
Evaporation from ponds (E flows)	E-A1	From the Dust Control Pond	
	E-C1	From the Tailings Storage Facility pond surface	
	E-C2	From the Tailings Storage Facility wet beach	
	E-D1	From the Bulk Storage Reservoir pond surface	
	E-E1	From the Off-Channel Storage Reservoir pond surface - Cell 1	
	E-E2	From the Off-Channel Storage Reservoir pond surface - Cell 2	
	E-E3	From the Off-Channel Storage Reservoir pond surface - Cell 3	
Seepage (S flows)	S-A1	Into the Open Pit Mine	
	S-C1	From the Tailings Storage Facility	
	S-D1	From the Bulk Storage Reservoir	
	S-E1	From the Off-Channel Storage Reservoir pond - Cell 1	
	S-E2	From the Off-Channel Storage Reservoir pond - Cell 2	
	S-E3	From the Off-Channel Storage Reservoir pond - Cell 3	
Miscellaneous flows (M flows)	M1	Water from dewatering wells	
	M2	Potable water	
	M3	Excess water from dewatering wells	
	M4	Sewage water	
	M5	Water for dust control	
Flows between elements (F flows)	F-A1	Flow from the Open Pit to the Dust Control Pond	
	F-A2	Flow from the Off-Channel Storage Reservoir for dust control	
	F-B1	Flow from the Plant Site	
	F-C1	Flow from the Tailings Storage Facility to the Bulk Storage Reservoir	
	F-D1	Flow from the Bulk Storage Reservoir to the Process Plant	
	F-E1	Flow from the Weir Diversion - Gorouol River	
	F-E2	Flow from the Off-Channel Storage Reservoir - Cell 1 to the Process Plant	
	F-E3	Flow from the Off-Channel Storage Reservoir - Cell 2 to the Process Plant	
	F-E4	Flow from the Off-Channel Storage Reservoir - Cell 3 to the Process Plant	
	F-F1	Flow from the Gorouol River to the Off-Channel Storage Reservoir	

Essakane
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Burkina Faso, Africa

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The model compares inflows (precipitation, process water, etc.) with losses to the system (evaporation, seepage losses, water retained in the pore spaces of the tailing, etc.) on a monthly basis to determine pond volumes and decanting/discharge requirements needed to keep inflows balanced with outflows.

The model set-up is transparent. The input data can be changed for ranges of operating and hydrological conditions. Changes can be easily made to investigate worst case and what if scenarios. The flow model is a mathematical design tool that can be used, combined with engineering judgement, to establish flows over a specific range of operating and climate conditions.

The simulation results obtained from the model indicate that Years 1 to 4 of operations are critical for the project because the water requirements are maximized due to a higher ore processing rate during the first four years of operations and predominantly saprolite ore in the first two years; Year 1 has the maximum water requirement. The storage capacity of the off-channel storage reservoir is defined based on the annual water requirements of this early period. The annual volume required from the off-channel storage reservoir equals the annual make-up water required to run the mill minus the annual volume of water collected on site from runoff, direct precipitation on ponds and well dewatering.

One of the main challenges at the project site is to minimize losses due to evaporation. In order to achieve this objective, the surface of the reservoir should be minimized. One way to achieve this is to maximize the reservoir depth and to subdivide the reservoir into a number of internal cells that can be drained one at a time. Once a cell is emptied, its impact on evaporation ceases.

Unfortunately, detailed soil investigation will be required to determine optimal location and surface to volume ratio. Based on conservative estimates at this time, it is proposed to have a total storage capacity of 3,800,000 m^3 for average precipitation and 4,100,000 m^3 for 20-year dry precipitation in our network of reservoirs, of which the Bulk Storage Reservoir should have a minimum of 729,000 m^3 of capacity. It is expected that these requirements will be optimized through detailed engineering work over the next month.

18.2 Fuel Supply

Petroleum-based fuels are a major input to the Essakane Project, not only for the mobile fleet but for the generation of electricity. At a Brent crude price of $85/barrel, the cost of fuels represents about 23% of total operating costs before government charges.

18.2.1 Fuel Supply and Storage

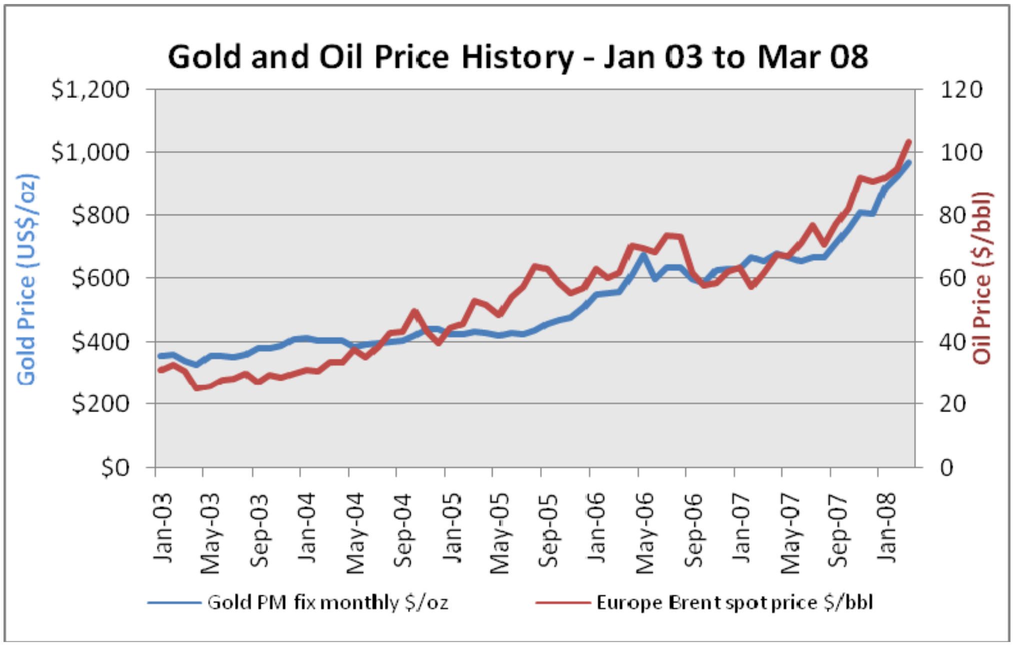
Burkina Faso has no domestic oil production and must import all of its fuels through ports in Ghana or Togo. Even though fuels can be purchased from large producers, like Total or Shell, or smaller producers, pricing of fuels and its storage and transportation is regulated by the SONABHY, an agency of the Burkinabe government. Typically, fuels are imported and stored temporarily in large reservoirs at ports in Togo and Ghana, and then transported by contract tankers to Burkina Faso. Essakane will need both diesel (LFO) for the mobile fleet, and heavy fuel oil (HFO) for the power generators.

At Essakane site, tank farms will be built with total capacity of 1,535,000 litres of HFO and 1,050,000 litres of LFO. Temporary storage will be installed for construction and subsequently integrated to the permanent arrangement.

18.2.2 Fuel Pricing

The fuel price assumption is an important parameter greatly influencing operating costs. It has been observed that the price of crude oil, which is the base to derive all fuel prices, was correlated with the gold price, as shown on Figure 18.3 which compares the average monthly gold price to the average monthly Brent crude oil price.

Figure 18.3 Gold & oil price history



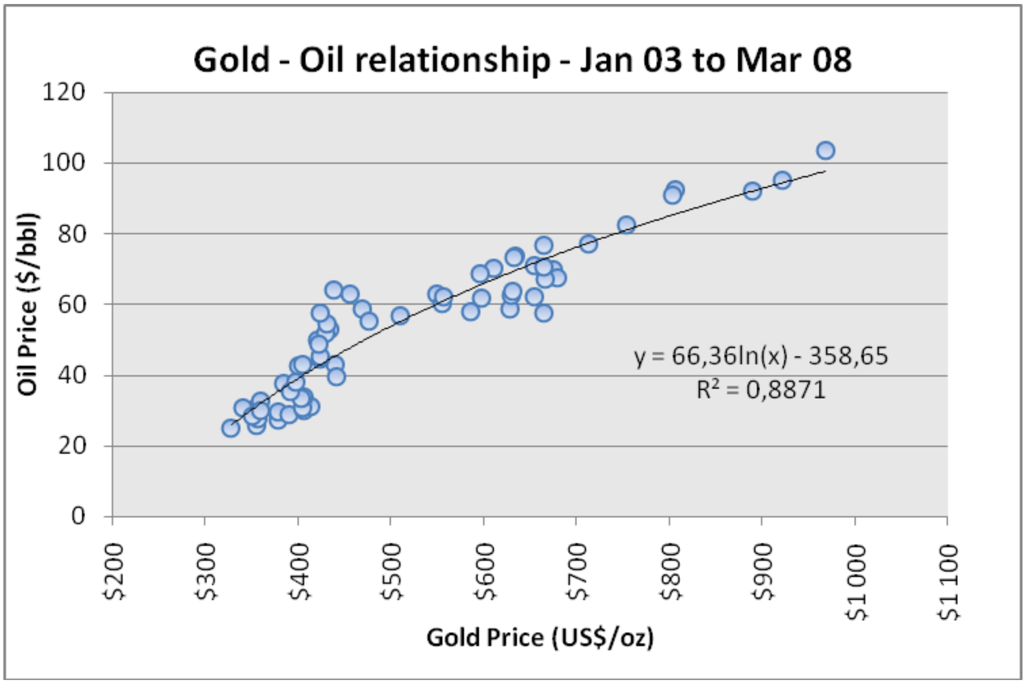
A correlation was established by using the pairs of monthly data points, as shown on Figure 18.4, and produced the following formula:

Brent oil price (US$/bbl) = 66.36 x LN (Gold Price) -358.6

The application of the formula gives the following corresponding data:

Gold Price ($/oz)	Brent Oil Price ($/barrel)
600	66
800	85
1000	100

Figure 18.4 Gold / oil relationship



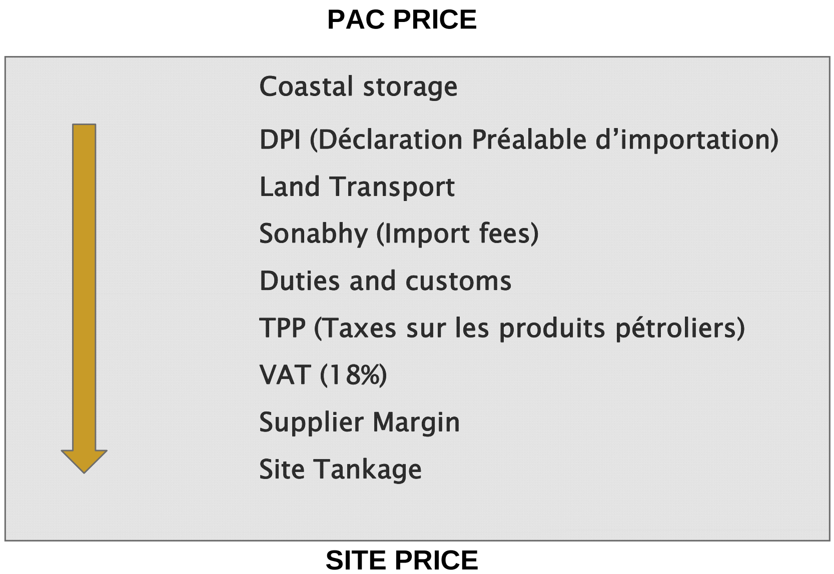
Historical data and detailed supplier quotes were available to establish accurate pricing for LFO delivered at Essakane. Figure 18.5 illustrates the components and multiples that are possibly added to the market coastal price for fuels (PAC) to get to a delivered price on site. Fortunately for Essakane, a major portion of taxes and duties are waived for power generation and industrial applications such as mining. The fuels coastal price (PAC) corresponds to market prices, which in turn correlates to crude oil prices as follows:

PAC Diesel ($/b) = [1.122 x BRENT ($/b)] + 4.56

PAC HFO 180 ($/b) = [0.78 x BRENT ($/b)] + 5

PAC HFO 380 ($/b) = [0.75 x BRENT ($/b)] + 5

Figure 18.5 From Coastal market price to delivered site price



Delivered fuel prices at Essakane were calculated prior to government taxes and duties, and these later charges were identified separately. The estimated fuel prices for different Gold Price/Oil Price combinations are presented in Table 18.1. While there is several sources and sufficient supply of diesel fuel to Burkina Faso, demand is relatively small for HFO 18O and non-existent for HFO 380. Because of the large requirement for the power plant, supply will be organized, but final terms have to be confirmed by the SONABHY. All fuels will be transported to site by the fuel suppliers.

Table 18.1 Delivered fuel prices to Essakane (based on US$1.48 per 1 Euro)

	Diesel (LFO)		HFO 380	
Brent Oil Price ($/barrel)	**Delivered Direct Price ($/l)**	**Gov't Taxes & Duties ($/l)**	**Delivered Direct Price ($/l)**	**Gov't Taxes & Duties ($/l)**
66	0.744	0.110	0.587	0.084
85	0.838	0.117	0.677	0.090
100	0.944	0.125	0.747	0.096
115	1.050	0.133	0.818	0.101
130	1.156	0.141	0.856	0.106

18.3 Power Supply

18.3.1 Power demand and generation

Power demand for the Essakane Project was estimated based on the list of connected motors, utilization factors, scheduled ore feed, and benchmarking with other operations. The big variable is the grinding power requirement which will increase from 5.7 kWh/t in the first year of operation to 15.5 kWh/t when only fresh rock is fed to the mill. Similarly, power requirement for crushing will increase mainly with the proportion of fresh rock, and to a lesser extent, transition ore, in the mill feed. Finally, energy requirement for pumping slurries and water will be reduced when throughput decreases from 7.5 M tpy to 5.4 M tpy. Table 18.2 summarizes the estimated annual power consumption and the peak demand over the mine life; these estimates include a 4% internal consumption and losses within the power plant.

Table 18.2 Essakane estimated power consumption and power demand over mine life

Operating Year	1	2	3	4	5	6	7	8	9	10
Consumption (MWh)	111,369	122,417	140,464	160,259	155,939	161,684	162,000	162,000	162,000	63,646
Peak Demand (MW)	16.9	17.6	18.8	21.8	21.8	21.8	21.9	21.9	21.9	21.9

Regarding power generation, several alternatives with different genset sizes and fuel types were analysed. Based on the current energy pricing environment, HFO fuels proved to be the preferred energy source considering the location and peak demand for the project. Similarly, power generating efficiency and capital costs favored larger generating units. Finally, considering also delivery dates, the Wartsila 12V32 – V type turbo charged medium-speed (750rpm) generator with a capacity of 5.5 MW was selected as the base power generating unit. Power plants are typically built on the N+2 principle, where N is the number of units required on line to meet peak demand and for which we assume that at times one unit may be in major overhaul and one unit comes offline for unplanned mechanical downtime. Even though no major overhaul would normally be required until the genset reaches 24,000 operating hours, insurers indicated their preference for N+2 units from the first year of operation and five units will be purchased initially. A sixth unit is planned to be purchased and installed in the third year of operation to deal with the increased power demand caused by a high proportion of fresh rock in the mill feed in the fourth year of operation; this investment is part of sustaining capital. The major advantage of the N+2 design criteria is the very high power availability to the operation; with proper maintenance and based on benchmarking, a power plant availability of 99.5% is expected.

18.3.2 Construction power and emergency power

Construction power at Essakane will be provided through three CAT generator sets rated 800 kW with an output of 400 V at 50 Hz. After their implementation as primary power source during construction phase, these three generators will be relocated to the following areas for emergency power during the production period:

1) Mine Village: To insure constant power supply at mine village, one 800 kW generating unit will be located at proximity to the camp near the power line. In case of power outage, the mine village generator will supply power to the camps, kitchen and recreational center via the existing power line (a step-up transformer will be dedicated to the generator set). An isolating switch will be implemented such that power from this set is dedicated only to the mine village.

2) Plant site: During power outage, another 800 kW will be dedicated for the use of office buildings, maintenance shop and mine loads (dewatering pumps). Loads from LFO and HFO storage areas will also be serviced from this generator. The idea behind this strategy is to allow administration services and mining operations to continue during outages if required.

3) Mill: Critical loads will be fed from the remaining 800 kW unit. This strategy insures that critical loads that need to be continuously operating (lighting, leaching agitators, furnace) will be supplied by an emergency power source during power plant downtime.

Emergency power is not intended for operation continuity, but for sustaining limited service during downtime of power plant.

18.3.3 Power distribution

The power plant will feed five separate power lines at 6.6 kV, as follows:

1) Processing plant: this is the sector where most of the power is consumed. Consequently, the power plant is located as close as possible to the mill sub-station and grinding sector. Two feeders and breakers, one set being redundant, are planned between the plant and mill substation. The grinding motors will be fed at 6.6 kV while the remainder of the processing plant will require a step-down to 690 V or 400 V.

2) Mine Village: this line will bring power to all mine village infrastructures.

3) Mine Plant: this line will bring power to mine maintenance shop, warehouse, fuel storages, office buildings, gyratory crusher and pit dewatering pumps.

4) Water Management: water reclaim from the various reservoirs and ponds will be serviced by this line in addition to the tailings thickener.

The overhead power lines will be located such that there will be no exposure of live conductors to major mobile mining equipment. Connection to these overhead lines from the power plant will be by underground 6.6 kV cables and each circuit will be fed by a dedicated circuit breaker located in the power plant.

The permanent power lines will be an extension of those installed during the construction phase.

18.3.4 Power Operating Costs

Power operating costs are based on fuels costs developed in section 18.2 and the estimated power consumption presented in Table 18.2. Average fuel consumption specified by supplier is 0.2 litre of HFO per kWh to which we add 0.0045 litre of LFO per kWh for genset start-ups and emergency use. Operating labor, maintenance and lubrication add another $0.023 per kWh.

Power operating costs for the entire mine operation are presented in Table 18.3 for various gold price/oil price combinations, before government charges. If we consider the cost of power generation only, it can be summarized as follows:

Gold/Oil	**$ / kWh**
$600/$66	0.144
$800/$85	0.162
$1,000/$100	0.177

Table 18.3 Operating costs over mine life

Operating Year	1	2	3	4	5	6	7	8	9	10	Total
1) Gold- Oil ($600/oz - $66/b)											
Operating Costs ($000)	16,000	17,594	20,187	23,032	22,411	23,237	23,282	23,282	23,282	9,147	201,462
($/t milled)	2.13	2.35	2.69	3.54	4.15	4.30	4.31	4.31	4.31	4.31	3.47
2) Gold – Oil ($800/oz - $85/b)											
Operating Costs ($000)	18,081	19,874	22,805	26,017	25,317	26,249	26,300	26,300	26,300	10,333	227,574
($/t milled)	2.41	2.65	3.04	4.00	4.69	4.86	4.87	4.87	4.87	4.87	3.92
3) Gold – Oil ($1,000/oz - $100/b)											
Operating Costs ($000)	19,690	21,642	24,834	28,334	27,570	28,585	28,641	28,641	28,641	11,252	247,828
($/t milled)	2.63	2.89	3.31	4.36	3.68	5.29	5.30	5.30	5.30	5.30	4.26

18.4 Mining

18.4.1 Production Plan

As discussed in Section 16, it was decided subsequently to the DFS to take advantage of the excess grinding capacity of the processing facility in the early years of operations, when the ore feed was predominantly saprolite and transition ore, and schedule a milling rate of 7.5 M tpy for that period. Later, when the mill feed becomes entirely fresh ore, the milling rate will return to 5.4 M tpy.

There are two main rock types – arenite (sandstone) and argillite (primarily silstone and claystone) and three weathered states (saprolite, saprock or transition and fresh rock) that will be sequentially mined over the life of the operation. As discussed in Section 17, mining will be accomplished with four phases to achieve the final pit limits. The objective of pit phasing is to improve economics by feeding the highest grade during the earlier years and to defer some stripping towards the later years.

The US$600 mine design provides for 58.1 M t of ore at an average grade of 1.67/g Au/t and 157.5 M t of waste material will be mined over an 8.4 year pit production period. The strip ratio (on tonnes basis) is estimated at 2.71 : 1.

The mine production schedule includes a six month pre-production period during which time the overburden will be pre-stripped and some ore will be stockpiled. At the start of commercial production, the ore stockpile will contain 1.99 M t an average grade of 1.26 g Au/t that will be drawn down by the third year of production. During pre-production, a total of 6.0 M t of waste material will be mined. This waste material will be mainly saprolite material and will be used for the ROM pad construction and tailings dam construction.

Annual mining rates commence at 32 M t for the first four years of operation after which it decreases to 25 M t and stabilizes at 15 M t thereafter. The life of mine schedule is presented in Figure 18.6 and Table 18.4.

Figure 18.6 LOM Mining Schedule by Material Type

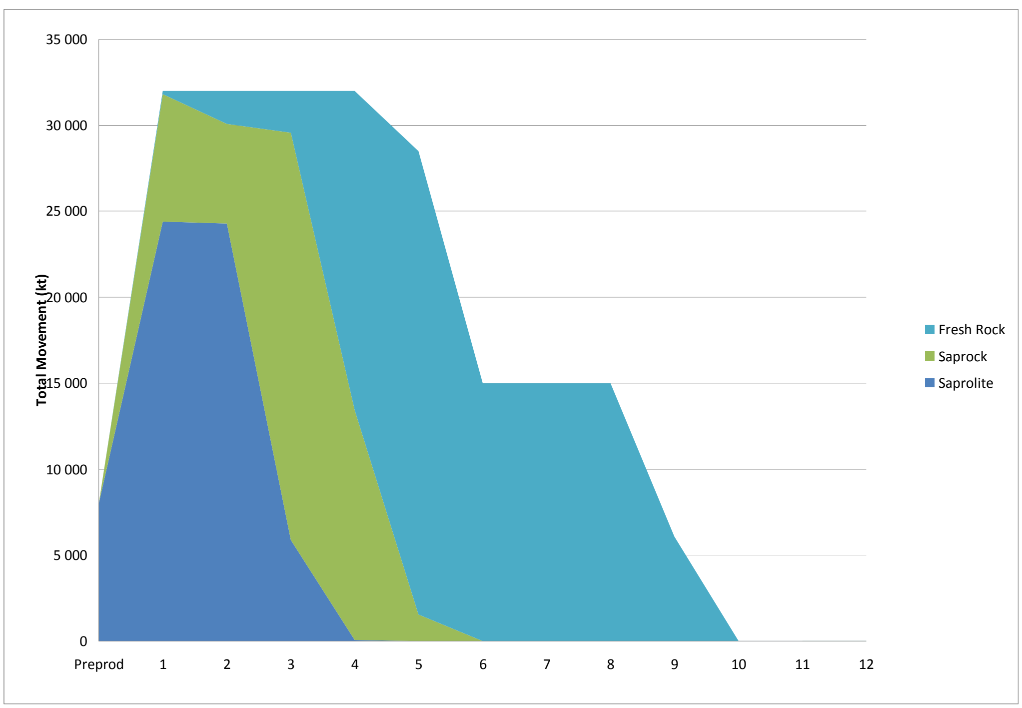


Table 18.4 Essakane LOM Mining Schedule

Period	Ore (kt)	Ore (g Au/t)	Waste (kt)	Ore Reclaim (kt)	Total Material (kt)	Strip Ratio
Preprod	1,989	1.258	6,012	-	8,000	3.02
1	8,483	1.610	23,517	535	32,535	2.77
2	5,409	1.393	26,591	2,091	34,091	4.92
3	6,655	1.375	25,345	1,273	33,273	3.81
4	6,876	1.578	25,124	1,169	33,169	3.65
5	6,858	1.811	21,642	355	28,855	3.16
6	5,470	1.937	9,530	115	15,115	1.74
7	6,638	1.944	8,362	-	15,000	1.26
8	6,751	1.777	8,249	-	15,000	1.22
9	2,993	1.829	3,095	2,894	8,982	1.03
10	-	-	-	2,122	2,122	-
Total	58,121	1.670	157,466	10,554	226,142	2.71

Ore to the plant will be a blend of the three weathering types commencing predominantly with saprolite and transition (saprock) for the first three years. Fresh rock mill feed will gradually increase from Year 4 onwards. This production profile by material type is shown graphically in Figure 18.7 and life-of-mine mill schedule details are presented in Table 18.5.

Figure 18.7 LOM Milling Schedule by Material Type

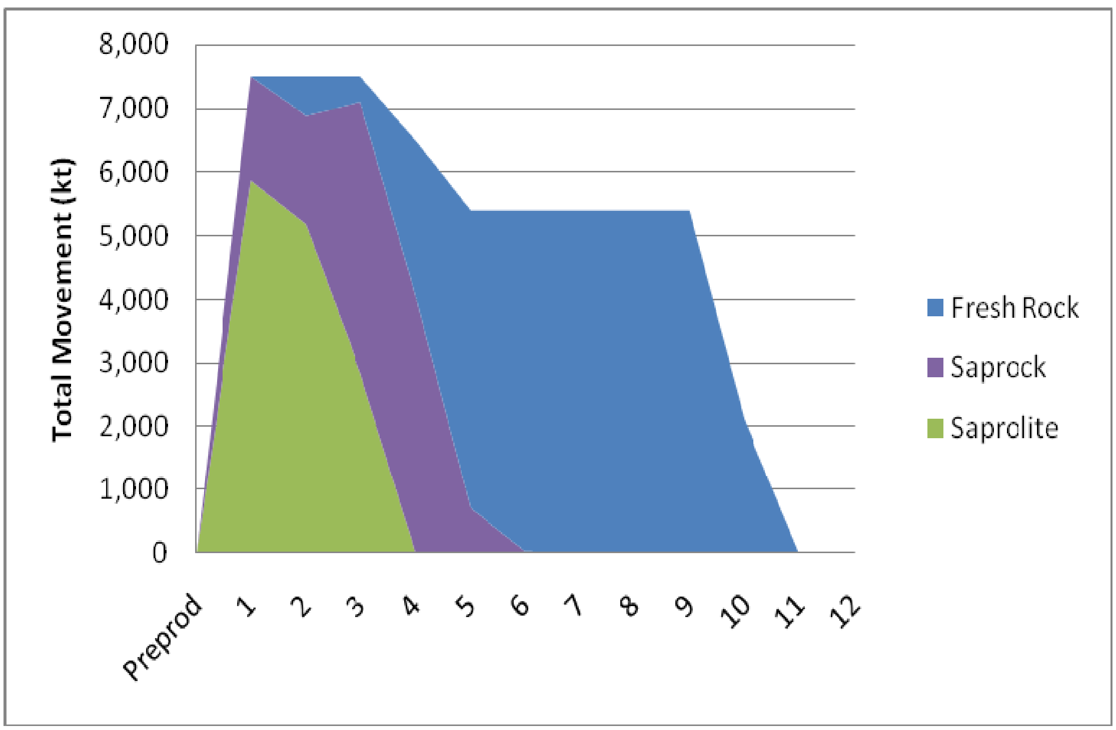


Table 18.5 Essakane LOM Mill Schedule

	Saprolite		Saprock		Fresh Rock		Total	
Period	(kt)	(g Au/t)	kt	(g Au/t)	(kt)	(g Au/t)	(kt)	(g Au/t)
1	5,870	1.53	1,629	1.90	-	-	7,500	1.61
2	5,182	1.17	1,705	1.81	613	2.04	7,500	1.39
3	2,848	1.37	4,249	1.36	403	1.49	7,500	1.37
4	16	1.10	4,056	1.58	2,429	1.57	6,500	1.58
5	-	-	713	1.58	4,687	1.81	5,400	1.78
6	-	-	37	2.05	5,363	1.93	5,400	1.92
7	-	-	-	-	5,400	1.96	5,400	1.96
8	-	-	-	-	5,400	1.80	5,400	1.80
9	-	-	-	-	5,400	1.82	5,400	1.82
10	-	-	-	-	2,121	1.82	2,121	1.82
Total	13,916		12,389		31,818		58,121	

The great majority of waste material will be stored in the main Overburden Storage Facility (OSF) which has been relocated on the west side of the pit such that it is on the same side as the ramp exit. The waste dump has been positioned in order to keep the dump toe line at least 500m from any surrounding village. To meet this requirement a small portion of waste will be stored in a satellite OSF on the east side of the pit.

Total overburden storage requirements from the updated mine plan are estimated at 93 M m^3 based on a placed density of 1.7 t/m^3. The storage of this material is planned as follows:

- West OSF = 84 M m^3
- In-pit dumping from last phase = minimum of 3 M m^3
- East OSF = 6 M m^3

The East OSF, as designed, can contain 20.4 M m^3 if additional storage capacity is required. The East OSF will be sited after completion of definition drilling at the south end of the EMZ.

An overburden characterization campaign was conducted for the DFS and concluded that a very limited amount of the Essakane overburden may have the potential to generate acid rock drainage. The non Potential Acid Generating (PAG) nature of this deposit results from the relatively low acid generating potential (maximum sulphide sulphur was 0.69% sulphur), the wide distribution of acid neutralizing potential, and the fact that Neutralising Potential (NP) tends to be higher in the deeper rock where Acid Generating Potential (AGP) also increases.

The material classified as saprolite was found to be material that is entirely oxidized. Most of the transition material (saprock) contained no detectable sulphide, and the few samples with detectable sulphide had very low net AGP. Virtually all fresh rock samples contained detectable sulphide sulphur.

Leachate or run-off from saprolite overburden will potentially contain arsenic at concentrations above the World Health Organization (WHO) drinking water guideline of 0.01 mg/l. Overburden and wall rock that contain sulphide sulphur is likely to act as a long-term source of soluble arsenic leaching in proportion to the oxidation of sulphide minerals.

Design of the OSF includes the placement of impervious saprolite at the base and a drainage system to collect any water percolating from the OSF and accumulate it in an evaporation pond.

18.4.2 Mining Operations

Mining at Essakane will be carried out using a conventional drill and blast and load and haul surface mining method with an owner fleet approach. The annual mining rate is initially set at 32 M t for the first four years of operation after which it decreases to 25 M t and stabilizes at 15 M t thereafter.

The weathering zones will be sequentially mined over the life of the operation starting with the friable upper saprolite. The saprolite material is assumed to be free digging while the saprock or transition material and fresh rock will require drilling and blasting.

Mining will occur on 6 m benches with some material potentially excavated in two discrete 3 m high flitches where additional ore selectivity is required. The recommended final bench height is 6 m in saprolite and transition and 12 m in Fresh rock.

Grade control samples will be collected from 5½" RC drill holes. Holes will be drilled on an 8 m along strike x 8 m across strike pattern, with a crawler type drill equipped with an RC sampling kit. Holes will be drilled at 60 degrees dip, with the hole azimuth inclined at 45 degrees to the strike of the Essakane Main Zone to maximize the sampling efficiency for both northerly striking and easterly striking vein sets. All material below the hanging wall contact of the main arenite unit will be subjected to grade control sampling. Grade-control drill meters and costs are based on eight 3 m flitches being drilled for a total drill hole length of 27.7 m. The grade control schedule assumes the same shift regime as other pit operations.

Drilling will be accomplished by six Atlas Copco Roc L8 drill rigs. The Roc L8 drill rig is versatile as it is capable of drilling production blastholes, RC holes, pre-split holes as well as dewatering holes. Orezone will have the ability to perform all its drilling requirements with a single drill rig model which will simplify maintenance activities and parts management. It is determined that production blast holes will be 152 mm in diameter. In saprock, the production drill pattern is estimated to be 6.0 m x 6.5 m (burden x hole spacing) while in fresh rock it is assumed to be 4.5 m x 5.0 m. The drill factors for saprock and fresh rock are 83 t/m and 56 t/m drilled respectively. The estimated overall production rates for production blast hole drilling is 26 m/h in saprock and 19 m/h in fresh rock. For pre-split drilling, 102 mm diameter holes will be used which is the low range that can be drilled by the selected production drill. Pre-split drilling will only be undertaken in fresh rock with a hole spacing of 2.0 m with a planned production rate of 24 m/h.

Charging and blasting will be carried out by the blasting crew that will use an explosives pump truck to load all blast holes. Production blasting will use a system of in-the-hole and surface Nonel delays with cast primers and an emulsion explosive. It is estimated that the powder factor for saprock and fresh rock will be 0.35 kg/bcm and 0.68 kg/bcm (0.15 kg/t and 0.25 kg/t) respectively. Blasting will be done only in day light hours and when there is no work being done in the surface mine (meal break or change of shift). Pre-split blasting will use a similar initiation system but all holes will be fired on the same delay using a more appropriate emulsion explosive or a specific pre-split explosive.

Loading will be accomplished with hydraulic excavators and a wheel loader will be added as an auxiliary loading unit. For the first five years, the mining rate is established at 32 M t/yr and requires the operation of two 15 m3 shovels, one 10 m^3 excavator and one 12 m^3 wheel loader. Specific use and costing of the 12 m^3 wheel loader for mining in the pit for the first four years of mining has been factored in the loading costs.

The truck fleet will be composed of fourteen 140 t-class and five 100 t-class trucks. The 140 t-class trucks are matched with the 15 m^3 shovels (3.7 m wide) which takes five to six passes depending on material type. The 100 t-class trucks are matched with the 10 m^3 (3.0 m wide) excavator which loads in five passes for the transport of ore to the mill. The 100 t-class trucks provide an ideal match with the 10 m^3 backhoe. The 100 t-class trucks have an earlier delivery date and will be the preferred units for dam building and other civil earthwork during the pre-production period.

Ancillary equipment includes motor graders for roadway maintenance, tracked dozers with single shank ripper for bench face and stockpile management, road building and grade control, wheel dozer for bench floor maintenance, water truck (76 kl capacity) for bench face and roadway dust suppression, a custom tool carrier with forklift and tire changing attachments for maintenance services and a mobile fuel and lubrication vehicle.

A buffer stockpile will be created in front of the primary crusher. However, it is anticipated that most ore feed will be directly tipped into the primary crusher by the haulage trucks. A crushed-ore stockpile with a live capacity of 15,000 t will be available between the primary crusher and the milling circuit.

Should ore feed from the pit be interrupted for extended periods, the plant will be fed solely from stockpiled ore on the ROM pad by a 12 m^3 wheel loader.

18.4.3 Mining Equipment

The mine fleet required over time is presented in Table 18.6. Most of the equipment will be purchased during pre-production. Only production drills need to be purchased in Year 2 to 4 as more transition and fresh rock material require blasting.

Because of the relatively short mine life and excess equipment in Year 5 to 6, no equipment replacement will be required with the exception of the RC drills.

Table 18.6 Mine fleet status over time

Equipment	PreProd	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9
Production drill	0	0	2	4	6	6	4	4	4	3
RC drill	2	2	1	2	1	2	2	2	2	1
15 m^3 shovel	2	2	2	2	2	2	1	1	1	1
10 m^3 backhoe	1	1	1	1	1	1	1	1	1	1
140t Truck	14	14	14	14	14	9	7	7	7	5
100t Truck	4	4	5	5	5	4	4	5	5	4
Front end loader	2	2	2	2	2	2	2	2	2	2
Wheel dozer	1	1	1	1	1	1	1	1	1	1
Tracked dozer	2	2	2	2	2	2	2	2	2	2
Grader	2	2	2	2	2	2	2	2	2	2
Water truck	1	1	1	1	1	1	1	1	1	1
Tool carrier	1	1	1	1	1	1	1	1	1	1
Fuel/Lube trucks	2	2	2	2	2	2	2	2	2	2
Field maintenance truck	3	3	3	3	3	2	2	2	2	2

18.4.4 Mine facilities and maintenance operations

Other mining infrastructure involves a mine office complex (mine offices, change house and canteen), equipment workshop with overhead cranes integrated to the main warehouse 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.

Essakane will be performing directly the maintenance of its mobile equipment fleet. Consequently, the maintenance department will include experienced expatriates that will be responsible for managing this function, performing maintenance planning and training the national employees.

A maintenance control system will be used to manage maintenance and repair operations. This system will keep up to date status, service history and maintenance needs of each machine. The specific software package will be selected to interface with the parts management and inventory system. The Information Management System (accounting, purchasing and inventory) will be Great Plains, which is a Microsoft Support Business Solution. This is the system used currently at the project site.

The regional dealer for the Caterpillar, O&K and Atlas Copco equipment has maintenance facilities in Ouagadougou and will provide service during the warranty period and support afterwards. Component rebuilds are planned to be performed by the regional dealer.

18.4.5 Mining Operating Costs

Using a gold price of $800/oz and a Brent crude oil price of $85/barrel, operating costs were estimated on the basis of a diesel price of $0.84 per litre before governmental taxes and duties; these government charges add $0.12/l to the cost of diesel and are accounted separately. Mining operating costs are presented in Table 18.7.

In constant dollar, the average mining cost is $1.58 per tonne mined. Operating costs increase steadily from $1.17 per tonne in the initial year of production to $2.43 per tonne in the final year of production for the following reasons:

- increasing costs for drilling and blasting as mining progresses from saprolite to transition to fresh rock.
- increasing haulage costs as the pit gets deeper.
- lower annual tonnage beyond the fifth year of production.

Table 18.7 Mining Operating Costs – Base Case

Year of operation	1	2	3	4	5	6	7	8	9	10	Total
Tonnage mined (000t)	32,000	32,000	32,000	32,000	28,500	15,000	15,000	15,000	6,088	---	215,589
Operating Costs ($000)	37,406	40,602	46,026	51,495	51,047	31,544	33,339	33,553	14,822	1,158	340,990
($/t mined)	1.17	1.27	1.44	1.61	1.79	2.10	2.22	2.24	2.43	---	1.58

18.5 Processing

18.5.1 Production schedule

Based on the mine production plan detailed in section 18.4.1, the mill production schedule presented in Table 18.8 was generated. A mill throughput of 7.5 M tpy is maintained for the first three years of operations, while the fourth year is an intermediate year at 6.5 M tpy because of increasing hard rock in the mill feed before coming down to 5.4 M tpy when mill feed is entirely fresh rock. The mill will produce some 2,960,000 ounces of gold over 9.4 years of operations for an average gold production of 330,000 ounces per year for the first four years and 315,000 ounces per year over the mine life. The average mill recovery is 94.8% over the mine life, with gravity recovery being possibly as high as 50-60% of the total production.

Table 18.8 Mill Production Schedule

Year of operation	Saprolite (000t)	Transition (000t)	Rock (000t)	Total (000t)	Grade (g Au/t)	Recovery (%)	Gold Prod (000oz)
1	5,870	1,630	-	7,500	1.62	96.3	375
2	5,182	1,705	613	7,500	1.39	95.3	319
3	2,849	4,247	403	7,500	1.38	95.5	316
4	16	4,056	2,430	6,500	1.58	94.6	312
5	-	713	4,687	5,400	1.78	94.3	291
6	-	37	5,363	5,400	1.94	94.4	317
7	-	-	5,400	5,400	1.96	94.4	321
8	-	-	5,400	5,400	1.80	94.2	294
9	-	-	5,400	5,400	1.82	94.2	297
10	-	-	2,122	2,122	1.82	94.2	117
Total	13,915	12,389	31,817	58,122	1.67	94.8	2,960

18.5.2 Processing operating costs

The processing flowsheet was explained in details in Section 16.

The processing operating cost estimate is for a plant capable of treating 7.5 M t/year when saprolite and saprock constitutes the majority of the mill feed and 5.4 M t/year of fresh rock ore constitutes the total mill feed. The operating costs for the process plant were developed from metallurgical test work, calculations of consumables such as grinding media, wear liners and reagents and actual reagent costs incurred currently for Gold Fields' Tarkwa gold plant in Ghana plus a transport component differential estimate from Tarkwa, Ghana to the Essakane site in Burkina Faso. The operating costs have been determined for two ore types, namely the saprolite ore and fresh rock arenite/argillite ore. The operating cost for the transition ore type is determined by averaging the process cost for the oxide and fresh ore.

CIL reagent consumption was based on laboratory scale test work conducted by various metallurgical test work service providers. The reagent consumptions for chemicals used in the acid wash and elution/electro-winning process have been determined from industry averages.

Grinding media and liner consumptions have been developed from wear rates factored from the Bond abrasion index using GRD Minproc's in-house correlations. Bond abrasion indices were 0.036 for the saprolite ore and 0.179 for the fresh rock arenite/argillite ore, indicating a significant variation. Initially, liner sets have been calculated to last two to three years, due to the low abrasion index of the saprolite ore. Practical experience, however, suggests that this is rarely the case because factors such as liner breakage, concentrated wear areas or the need to perform extensive auxiliary work means that, as a minimum, liner changes should be planned on an annual basis.

Table 18.9 presents the consumption and unit costs of consumables and reagents for saprolite and fresh rock ore, before government charges. Power costs were covered in section 18.3. The main difference in operating costs between ore types is the consumption of liners and grinding media.

Table 18.9 Consumption and unit costs of consumables and reagents for saprolite and fresh rock

	Saprolite		**Fresh Rock**	
	kg/t milled	**$/t milled**	**kg/t milled**	**$/t milled**
SAG mill liners	0.011	0.034	0.075	0.232
Ball mill liners	0.021	0.065	0.062	0.187
Grinding steel 120mm	0.063	0.057	0.418	0.378
Grinding steel 50mm	0.244	0.221	0.774	0.701
Cyanide	0.558	1.156	0.45	0.965
Lime	1.000	0.393	1.00	1.383
Carbon	0.050	0.108	0.05	0.108
Acid 32%	0.135	0.066	1.135	0.066
Caustic soda	0.043	0.031	0.043	0.031
Flocculant	0.020	0.069	0.020	0.069
Borax	0.090	0.126	0.090	0.126
Silica	0.059	0.019	0.059	1.019
Sodium Nitrate	0.026	0.026	0.026	0.026
Soda ash	0.031	0.024	0.031	0.024
Diesel (l)	0.070	0.072	0.070	0.072
Total		**2.457**		**3.387**

The Burkinabe labor cost has been based on three crews. The work schedule being two weeks at 12 hours per day followed by one week leave. The expatriate schedule will be 30 days of work, 12 hours per day, followed by 26 days leave.

The employee numbers are based on the number of personnel required to supervise and conduct the operational activities per section based on similar operations in Western Africa. The labor cost used is consistent with a survey conducted in Burkina Faso for Burkinabes and in Canada for expatriate workers. The total number of personnel for the process plant and power plant is 242 comprising 30 expatriate personnel and 212 Burkinabe personnel.

The management philosophy requires having on site all the personnel required for the operation and maintenance of the plant. This diverges from the approach taken in the DFS where most of the maintenance was planned to be done by contractors. Reviews of contractor availability in Burkina Faso have shown that no experienced contractors are currently available and the operation will have to rely on its internal workforce for all regular maintenance work.

Expatriate personnel will steadily reduce during the operation as Burkinabe personnel will gain experience in process plant operation. While 30 expatriates will be required for the startup, the number will be reduced to 19 after two years of operation and eight by the end of operations.

Processing operating costs are summarized in Table 18.10; these costs are prior to government charges and exclude power costs calculated separately. As expected, the unit processing costs increase by about 18% as mill feed becomes mostly fresh rock and throughput is reduced.

Table 18.10 Processing Operating Costs

Year of operation	1	2	3	4	5	6	7	8	9	10	Total
Processing Costs ($000)	28,665	28,202	28,675	27,080	24,504	24,684	24,540	24,216	23,838	9,430	243,834
($/t mined)	3.82	3.76	3.82	4.17	4.54	4.57	4.54	4.48	4.41	4.44	4.20

18.6 Tailing disposal

18.6.1 Geotechnical investigation

In general terms, the soil profile over the project, as described in the DFS, is as follows from surface to depth:

- A relatively thin layer varying from a mixture of gravel within a sand and silt matrix to laterite to silty-to-clayey sand. This material would normally have a medium to high permeability (10^{-4} m/s).
- Saprolite material from residual arenite and argillite composed predominantly of dense silts and clays with very low permeability (10^{-7}-10^{-8} m/s).
- Transition material or saprock with changing geomechanical properties from top to bottom and showing relatively high permeability. This is the target horizon for groundwater supply.
- Fresh rock where permeability depends entirely on main fault and shear zones or other structural features.

The thickness of each horizon can vary greatly over the project site. Anomalies like intrusive dykes can also cut through the soil sequence and outcrop on surface.

Preliminary findings for the DFS were based on a first-pass drilling investigations to site the various project facilities. Detailed engineering will be based on a more extensive campaign of drilling and testing of the soils underlying the main project facilities, mainly the tailings storage facilities, the various reservoirs and the mill facilities.

18.6.2 Tailing pond and deposition method

In the DFS, Knight & Piesold had selected a method where tailings would 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 would decant the liquor and deliver it to the Bulk Storage Reservoir. The penstock would be raised as the TSF increased in height.

The starter walls of the TSF ranged from 5 to 14 m in height and would be constructed from suitable borrow material until the raises could be achieved using the deposited tailing. The outer slopes of the TSF would be clad with selected overburden rock placed concurrently with the tailing deposition. The site for the TSF was chosen taking into account the suitability of the ground conditions and the effect that the TSF would have on operations, the local population and the environment. The TSF had been designed for a final capacity of 60 M t and would 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 southwest and northeast corners respectively and would be 24 m and 34 m.

Golder conducted a review of previous studies and engineering work for tailings disposal at Essakane and considered alternative tailing disposal strategies to the conventional spigot system, as recommended in the DFS for the Essakane Project; alternatives included thickened tailing, paste tailing, filter cake and paste rock. Relative impacts on dewatering equipment selection, tailing transport and placement were considered. Mainly because of the presence of saprolite, the non-segregating thickened tailing disposal system was selected. The tailing thickener was discussed in section 16. Detailed engineering will be based on pilot plant testing, including dewatering and pump loop test, at Paste Tec. At this time, the tailings thickener underflow was conservatively estimated at 50% solids by weight for saprolite, 60% solids for transition and 70% solids for fresh rock.

The area of the TSF and Bulk Water Storage reservoir (BWS) is shown on the Location Plan, Figure 18.8. Also shown on Figure 18.8 are potential off-channel reservoir locations previously discussed.

The recommendations made below are subject to confirmation of suitable soil conditions and the availability of suitable clayey fill materials. The design is based on the Observational Design Method and provides maximum flexibility to incorporate design changes or modifications based on actual site conditions, constructability and changes in production rates.

Figure 18.8 Location plan

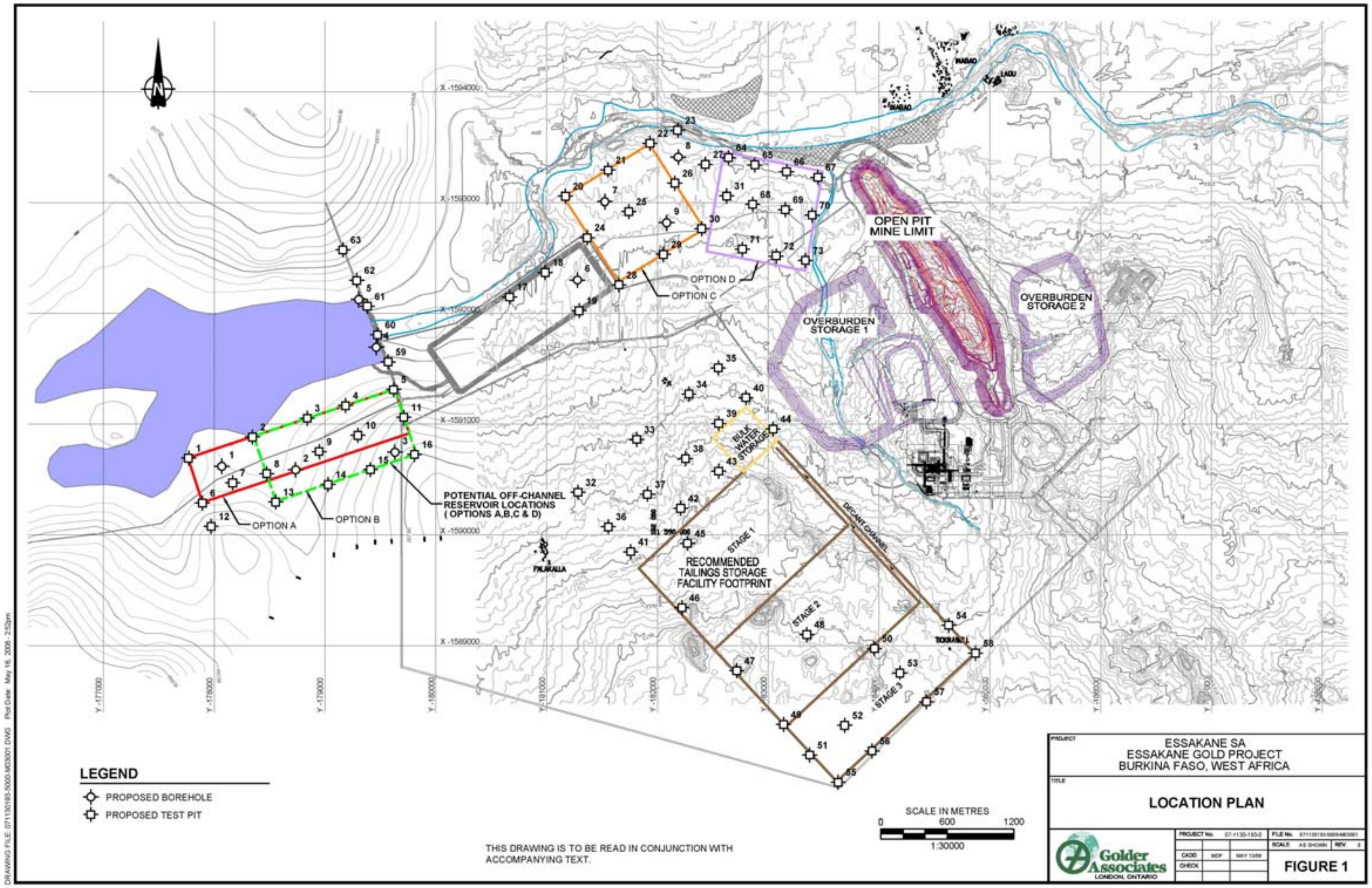
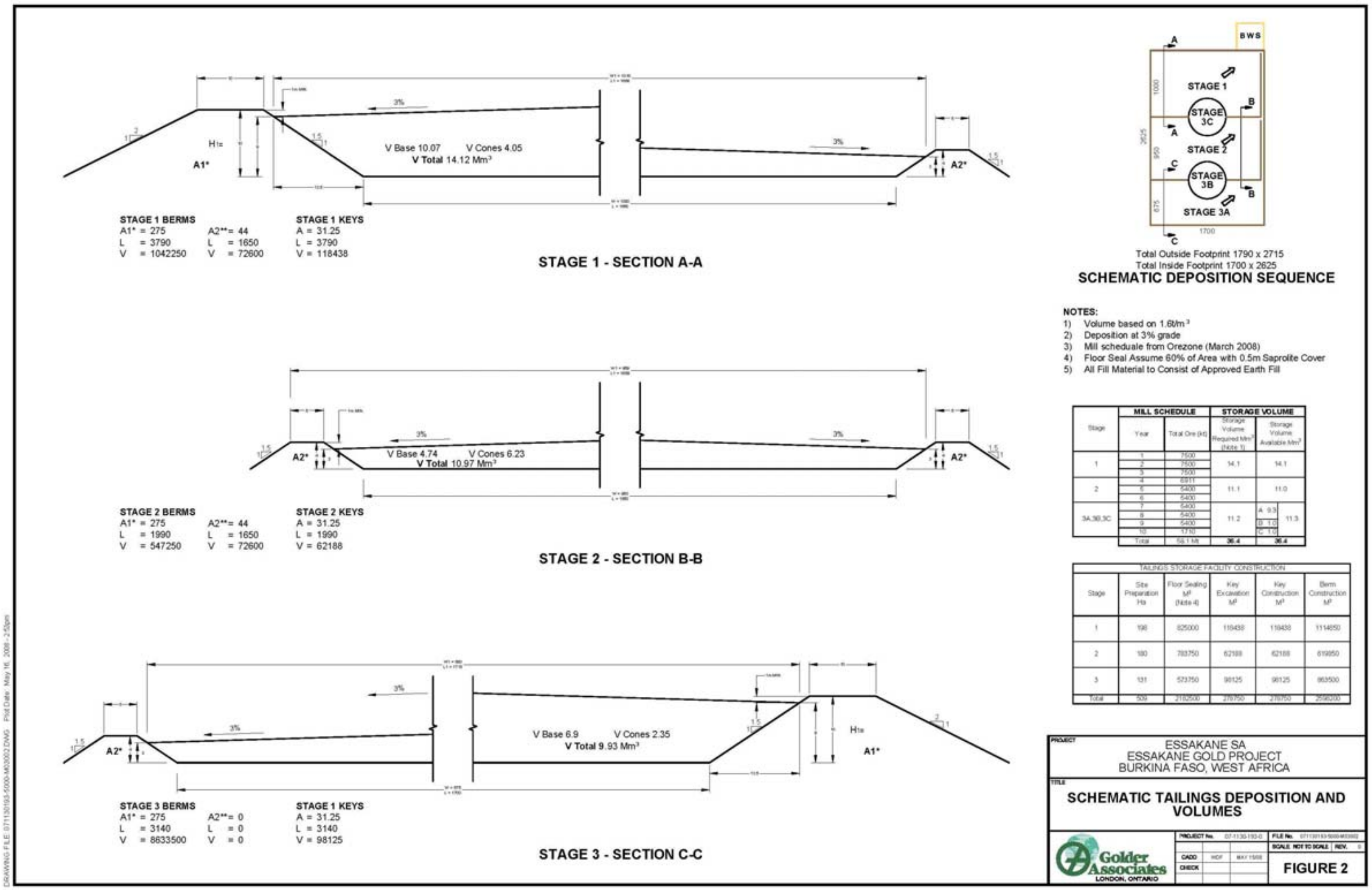


Figure 18.9 Schematic tailings deposition and volumes



Stage	MILL SCHEDULE		STORAGE VOLUME	
	Year	Total Ore (kt)	Storage Volume Required Mm³ (Note 1)	Storage Volume Available Mm³
1	1	7500	14.1	14.1
	2	7500		
	3	7500		
2	4	6911	11.1	11.0
	5	5400		
	6	5400		
3A,3B,3C	7	5400	11.2	A 9.3 / B 1.0 / C 1.0 — 11.3
	8	5400		
	9	5400		
	10	1710		
	Total	56.1 Mt	36.4	36.4

TAILINGS STORAGE FACILITY CONSTRUCTION					
Stage	Site Preparation Ha	Floor Sealing M³ (Note 4)	Key Excavation M³	Key Construction M³	Berm Construction M³
1	198	825000	119438	119438	1114850
2	180	783750	62188	62188	619850
3	131	573750	98125	98125	863500
Total	509	2182500	279750	279750	2598200

The proposed TSF is located in the same area as in the DFS, but has an overall base area of about 1,700 m by 2,625 m or some 450 ha. However, the TSF will be developed in stages concurrent with mine production. Three stages are proposed for the TSF as shown on Figure 18.9 and on the Schematic Tailings Deposition Sequence on Figure 18.9. The design optimizes the site topography which generally slopes downward from southwest to northeast and promotes drainage to the BWS. The three stages are detailed further in the tables on Figure 18.10. The stages are based on the projected milling rates for the indicated periods of the mine life. Stage 1 construction covers the initial three years of mine production when mostly saprolite ore will be milled resulting in more process water reporting to the tailings facility. Stage 2 covers Years 4, 5 and 6 during which time the ore will predominantly be a blend of transition and fresh ore. Stage 3 covers the remaining four years of operation during which time fresh ore will be milled. Stage 3 is broken down in sub-stages A, B and C, as shown on the schematic deposition plan. Stages 3B and 3C are infilling of spaces between deposition stacks. The main stages are separated by low internal berms. To accommodate the tailings disposal for the 58 M t of ore, some 36 M m^3 of storage is required based on an average tailings density of 1.6 t/m^3 (range 1.47 - 1.8 t/m^3).

Perimeter berm heights of 10 m are planned with crest widths of 10 m. Typical cross-sections are shown on Figure 18.10. The slope inclinations are at 1.5 horizontal to 1 vertical upstream and 2 horizontal to 1 vertical downstream. A foundation cut-off key down to the saprolite layer is required for the perimeter berms. Rip-rap erosion protection would be provided on the downstream slopes as suitable material becomes available from various sources. The internal berms are 4 m high with a crest width of 5 m. The upstream and downstream slopes are at 1.5 horizontal to 1 vertical inclinations. No key construction or rip-rap is required for the internal berms. The berms should be constructed of approved clayey earth fill materials placed in maximum 0.3 m loose lift thicknesses and uniformly compacted to 95% of standard Proctor maximum dry density (ASTM D698). Based on the anticipated site conditions, watering of the fill materials will probably be required to achieve adequate compaction.

Blinding or sealing of permeable layers such as gravels, sands and silts or rock outcrops in the TSF floor would be required. It is considered that these areas may be effectively sealed with a 0.5 m layer of clayey saprolite. Recognizing that the saprolite will be subject to drying and desiccation cracking, some maintenance will likely be required until such time as the floor is covered with tailings. For preliminary estimating, it has been assumed that 60% of the floor may require sealing. The saprolite floor seal should be uniformly compacted to 95% of standard Proctor maximum dry density.

Additional, detailed geotechnical investigation and testing is required for final design of the Tailings Storage Facility and the Bulk Water Storage Reservoir to confirm subsurface soil and groundwater conditions. Confirmation of the availability of suitable clayey saprolite fill in adequate quantities is also required. This confirmation work needs to be integrated with the characterization of the mine waste material since waste mining will be the major source of fill.

18.6.3 Tailings water management

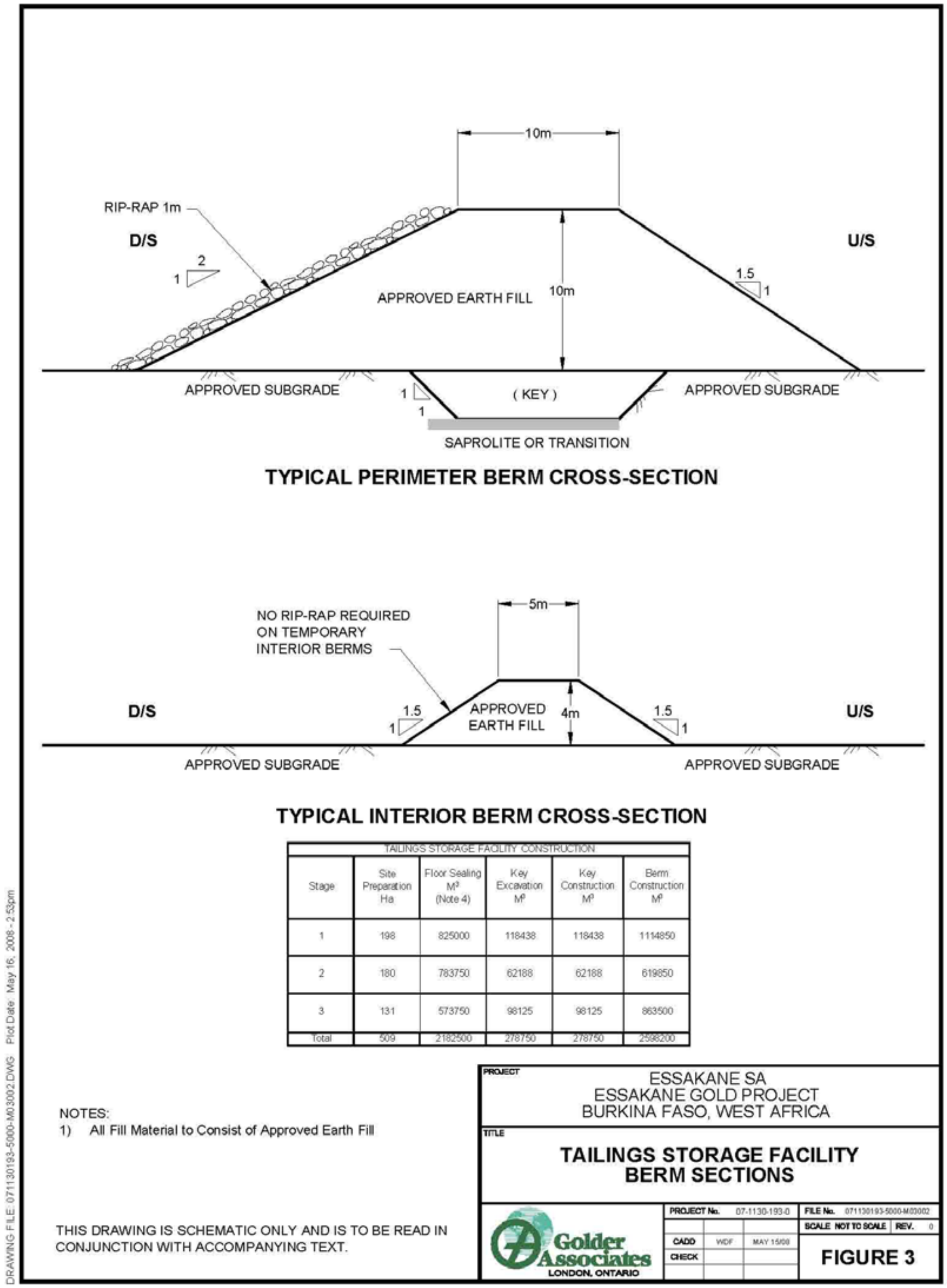
The TSF design is based on non-segregating, thickened tailings with an average slurry density of 60% solids (range 50 - 70%) and a 3% slope deposition. As noted above, the TSF can be constructed with an overall base area of about 450 ha (overall footprint 1,700 m by 2,625 m). The tailings bleed water will not be retained in the facility but will flow through to the BWS from where it will be pumped back to the mill. The design incorporates a decant channel to direct bleed water, as well as any runoff, by gravity to the·BWS by constructing nominal 4 m high berms along the east side of the TSF facility as shown on the drawings. The channel is common to the three Stages of the TSF.

It is presently proposed to construct the BWS contiguous to the TSF as shown on Figure 18.8. The design is based on no discharge to the environment and provides sufficient capacity to ·accommodate one month of process water and direct precipitation and run-off, including a probable maximum precipitation (PMP) event of 157 mm over a 24-hour period, from both the BWS and TSF, and allows one metre of freeboard. Based on the above, some 729,000 m^3 of storage volume is required.

The BWS would be constructed in cut and fill with berm construction utilizing suitable clayey saprolite fill. A 5 m deep basin excavation is proposed with 5 m high perimeter berms. The berms would have upstream and downstream slope inclinations of two horizontal to one vertical. Berm construction would be carried out in the same manner as the TSF berms as described above. The BWS berms will require downstream toe drains and filters (19 mm crushed rock) enveloped in a geotextile surround. The upstream slopes will require geotextile for erosion and scour protection in addition to the rip-rap. Typical details are shown on Figure 18.11.

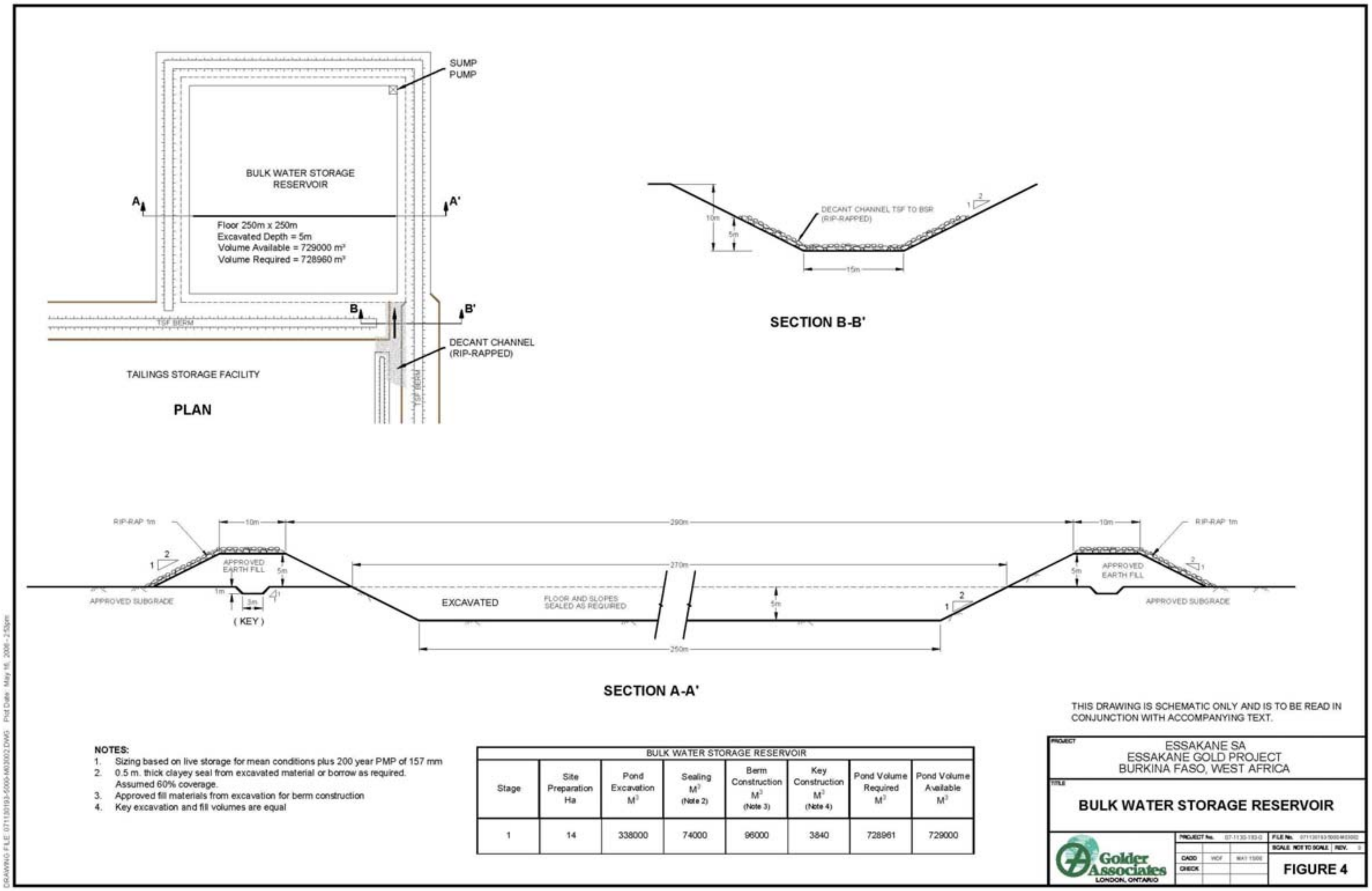
The BWS floor and portions of the cut slopes may require sealing in some areas. This can be achieved with clayey saprolite in the manner as described above for the TSF.

Figure 18.10 Tailings storage facility berm sections



TAILINGS STORAGE FACILITY CONSTRUCTION					
Stage	Site Preparation Ha	Floor Sealing M³ (Note 4)	Key Excavation M³	Key Construction M³	Berm Construction M³
1	198	825000	118438	118438	1114850
2	180	783750	62188	62188	619850
3	131	573750	98125	98125	863500
Total	509	2182500	278750	278750	2598200

Figure 18.11 Bulk water storage reservoir



BULK WATER STORAGE RESERVOIR							
Stage	Site Preparation Ha	Pond Excavation M^3	Sealing M^3 (Note 2)	Berm Construction M^3 (Note 3)	Key Construction M^3 (Note 4)	Pond Volume Required M^3	Pond Volume Available M^3
1	14	338000	74000	96000	3840	728961	729000

18.7 General Services

18.7.1 Infrastructures and facilities

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 laterite construction. The haul road will be 20 m wide and will be constructed by the Mining department 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 or treated by dedicated spillage pumping systems, as a policy of zero discharge has been adopted.

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 exploration camp. Other structures within the mine village will be the recreational and dining facilities which will also be prefabricated.

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.

Two offices complex will be located in the mine plant area, one to service mine operations and maintenance and the other one reserved for construction management and administrative services. An office complex in Ouagadougou is currently available to house general management, accounting and finance, procurement and logistics and human resources; some of these functions will be partly relocated to Essakane site in time.

The main warehouse is also attached to the mine maintenance shops and will include a sizeable storage yard.

18.7.2 Services operations

General services are an essential component to the success of the Project. Because of the remoteness and complex logistics of the Project coupled to the limited services available in Burkina Faso, the scope and extension of the General Services department to support construction and subsequently production is very substantial.

It will include the general management and accounting and finance functions. Because of the importance and visibility of the Project in Burkina Faso, government and public relations will require an important attention and structured program from senior management. The adoption by the Government of Burkina Faso of European-inspired controls and regulations towards all economic activity increases the administrative burden in all aspects of the Project and multiplies the required interactions with government officials. This is particularly severe in the Procurement and Logistics areas since most equipment and supplies have to be imported on a worldwide basis and transported through neighboring countries.

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 are laterite. 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.

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

Purchasing, receiving, warehousing and accounting will be facilitated by the Great Plains, a management information system already in use by Orezone and currently being updated and expanded for the Project. A third-party maintenance management module will need to be selected and added later to the system.

Construction manpower will eventually peak in excess of 2,000 workers in 2009 and manpower during production will be approximately 800 workers. A large basin of population is available for the project and the challenge for human resources personnel will be to find the best qualified personnel quickly when construction start is authorized. All personnel will attend site induction and general information and training sessions. Specific training based on a combination of in-class and on-the-job training, will have to be organized and structured by each operating department through a modular approach. All training will have an important focus on health and safety with the objective of achieving an accident-free project site; this objective will require a major emphasis considering the rapid staff build-ups and extensive training required. The main responsibility for health and safety will reside with qualified expatriate and national supervision. A continuous process of instruction, observation and feed back for all workers, combines with specific health and safety tool-box and regular formal meetings should forge a safety-first mentality. The Health and Safety department will coordinate all health and safety programs and activities, manage and/or inspect the safety equipment and fire protection systems and conduct inspection and reporting activities to senior management.

The environmental group will be responsible to implement management and operating procedures to ensure certification of the operation activities under ISO 14001. This will include training of employees and monitoring of the project site. Even if the environmental situation is simplified by the absence of effluent, the transportation, storage and handling of dangerous goods and petroleum products are always the source of potential impacts and incidents. The social group will be particularly challenged by the extensive population resettlement program and the organization and monitoring of sustainable programs for the local population, as discussed in section 18.8. Involvement of NGO organizations in the sustainability aspect of the Project will add to the management complexity but will have a beneficial multiplicator impact.

Camp management, catering and transportation is based on 300 employees lodged in camp and supplied with three meals per day. An additional 500 employees will be transported from the local villages and supplied with a mid-shift meal.

18.7.3 General Services Operating Costs

General services costs were estimated in details on a yearly basis and are presented in Table 18.11; they are inclusive of dore transportation and refining costs, but without governmental charges. Yearly costs benefit from a reduction in expatriate labor during the initial four years. However, economies of scale are reversed and unit cost increases in Year 5 as the throughput is reduced to 5.4 M tpy.

Table 18.11 General Services Operating Costs

Year of operation	1	2	3	4	5	6	7	8	9	10	Total
Operating Costs ($000)	13,235	12,424	12,114	10,793	11,043	10,729	10,578	10,510	10,517	4,259	106,202
($/t mined)	1.76	1.66	1.62	1.66	2.05	1.99	1.96	1.95	1.95	2.01	1.83

18.8 Environmental and Social Aspects

The Environmental and Social Impact Assessment (ESIA) has been prepared by Knight Piesold (KP) and rePlan and includes baseline data of the relevant environmental and social 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.

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 while 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. The Essakane Financial Model assumes an annual reclamation funding based on the tonnage proportion of mineral reserves processed yearly; the funding will total $15 M before accumulated interest at the end of operations.

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 regular elections for a President, a Prime Minister and Cabinet 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.

18.8.1 Environmental and social baseline information

The environmental study for the DFS covered an area of 50 km^2 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 500 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 46ºC to 10ºC with humidity from 98.5% in the wet season to 0.5% in the dry season 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 certain boreholes that intersect the EMZ ore body. A total of 25 dewatering boreholes around the surface mine will yield 6 m^3/hr per borehole and will provide 90,000 m^3 /month.

The overburden is considered to be non-acid generating with tests indicating pH above 7.5. Run-off from the saprolite and saprock may 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 previous activities of the artisanal miners. In general, seven vegetation classes can be encountered in the region. Similarly, there is little in the way of wildlife 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. Artisanal 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. The social 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.

18.8.2 Environmental and social impact assessment

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 dam and off channel reservoir will impact on a riverbank forest although careful planning is expected to reduce the 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 capacity of 60 m t. The outer slopes will be clad with selected rocks during operations and the beach surfaces will be allowed to consolidate. Upon closure the Bulk Storage Reservoir 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 Gorouol River dam and the off channel reservoir 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.

The people who will be 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 ha 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.

Direct and indirect employment by the Project is expected and must be generated proactively. In addition, health and education services will benefit from the existence of the Project.

18.8.3 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 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.

18.8.4 Resettlement and relocation

The Project will result in the displacement of 11,563 people living in 2,562 households and an initial 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.

Compensation policies and procedures were established 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.

Householders identified for resettlement will select their own resettlement house in the planned resettlement village north of the Gorouol River, 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.

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.

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.

For individuals that wish to relocate outside the area, a relocation package will be offered. Relocation compensation is where eligible owners are paid cash for their assets; indications are that about 10% of the affected households will opt for the cash payout. Those receiving cash compensation can choose how they would like the payment made and financial management advice will be provided.

18.8.5 Social management program

A Relocation Action Plan (RAP) will be implemented by a team of professionals comprising senior members from Essakane and Government Ministries. 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.

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.

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.

18.9 Project Organization and Execution

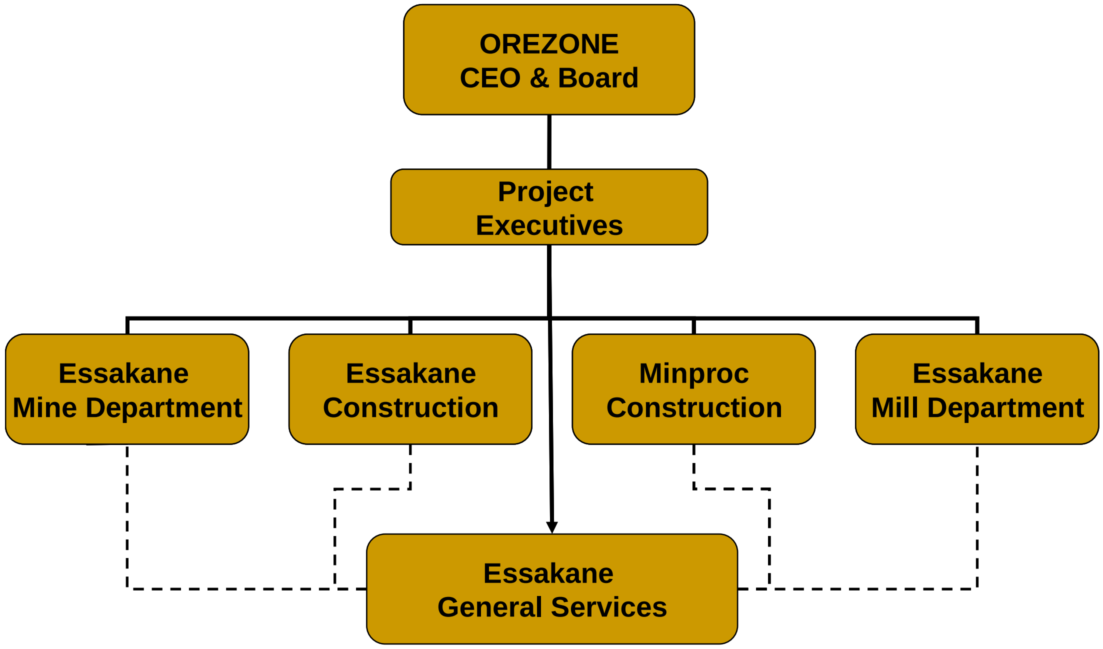
18.9.1 Project organization

Figure 18.12 presents the overall project organization for the Essakane Project. This organizational structure is viewed as the optimal structure experienced over more than twenty-five years of mining projects in both developed and developing countries. It is optimal in regards to capital costs to the owner, start-up and transition to operations and ability to achieve the operating objectives as forecasted.

Obviously, the overall authority on the Project lies with Orezone Resources, the Project Sponsor, and its CEO and Board of Directors based in Ottawa, Canada.

The Project will have essentially three project executives who share the responsibility for the preparation and execution of the project, which includes organizing, staffing, hiring, purchasing, supervising and controlling all engineering, procurement, construction and pre-production activities required to attain commercial production. These responsibilities are delegated to five different sub-groups with each their specific areas.

Figure 18.12 Project organization



Minproc was retained on an EPCM contract to perform the engineering, procurement and construction management for the processing facilities. In addition to the mill, their battery limits also include the assay lab, mill shops and warehouse, other service buildings within the processing area, power distribution for the entire plant site and tailing and water pumping and piping throughout the plant site. However, all supplies and equipment will be purchased by the owner (based on POs) and all construction activities will be governed by contracts signed between the owner and sub-contractors. Sub contractors envisioned at this time for the processing area would be for the following construction packages:

- Foundation preparation and miscellaneous earthworks.
- Concrete preparation, forming and placement plus aggregate preparation.
- Tankage , structural steel and cladding.
- Piping and mechanical installation.
- Electrical installation, instrumentation and process control.

The mill construction is on the critical path and requires more sophisticated and specialized engineering and contractors. For this reason, Minproc was selected as the EPCM contractor and it is expected that all sub-contractors will be foreign-based and will employ only the most qualified national workforce available. However, it is anticipated at this time that the Mine Department will perform mass filling around the primary crusher and the crushed ore pile.

The Mill Department personnel initially reviews and approves the mill engineering and procurement activities performed by Minproc. It will subsequently expand to monitor and supervise the mill construction activities, particularly piping and mechanical installation, electrical installation, instrumentation and process control. It will also be involved in recruiting and training the mill workforce and prepare the inventory of parts and supplies ahead of production. Finally, the Mill Department will take over the mill commissioning activities.

The Essakane Construction Group has the overall responsibility for the engineering, procurement and construction management of the mine village, the relocation village, administration and mine offices, mine shops and main warehouse, LFO and HFO tank farms, construction power and permanent power plant. The power plant is obviously a critical item and an EPC contract is planned for the engineering, purchasing, mechanical and electrical installation, and process control; the HFO gensets were purchased from Wartsila and the EPC contract is being negotiated. The other sectors generally require simpler engineering and construction activities and are not time critical for the project. Consequently, efforts will be made to solicit more involvement from the Burkinabe contractors and direct labor from the local population; this will enable Essakane to meet its social commitments and expectations from the local population.

The Essakane Mine Department will start receiving its equipment in July 2008 and will progressively build its total fleet over the next 12 months. This will enable it to progressively build its workforce while completing the construction of roads, water reservoirs, tailings pond (Phase 1), while pre-stripping some 8 M t of material from the Essakane Pit. In addition, the Mine Department will be responsible for the major fills required at the mill. Because the work described previously requires a mix of cuts and fills and that the pit stripping will generate a large quantity of saprolite, which is the best material to build impervious liners and dams, it was logical and cost-effective to assign this work to this group.

Finally, all the previous groups require support and services that will continue during the production phase. It was also logical to progressively build and organize those services in Essakane and avoid duplication and dislocation later when moving to the production phase. These services include general administration, finance and accounting, control and treasury, purchasing and logistics, human resources and training, security, social and environmental management, transportation, camp costs and plant maintenance, health and safety, receiving and warehousing in addition to the supply of power, fuel and water during construction. Some external contributors will also provide support, like UTI, an international freight forwarder, and rePlan, a social consultant. The Essakane General Services group will recruit heavily in the local and national labor pool and will be an important contributor towards our social and economic commitments.

All equipment and supplies will be tendered by Minproc, Wartsila or Essakane and obtained through purchase orders issued by Essakane or by Minproc for and on behalf of Essakane SA, the Burkinabe company owner of the project. Specifications are established by the various engineering firms retained for the project and accepted by owner's representatives.

Construction mandates and services will also be tendered. These activities will be controlled through contracts. These contracts will be based on the standard documents prepared by FIDIC (Fédération Internationale des Ingénieurs-Conseils) , with addendum specifics to our project.

18.9.2 Construction schedule

The latest construction schedule assumes an official construction release on July 1st, 2008, and aims for commercial production in January 2010 for a 18-month construction and commissioning period. Any further delay in the construction release will cause an identical delay in the start of commercial production. The construction schedule has already been compressed as much as feasible. The Project milestones for selected site activities are presented in Table 18.12.

This aggressive schedule was made possible by pre-funding for the following time-critical equipment:

- Gyratory Crusher.
- Grinding mills.
- Mine mobile equipment.
- Construction mobile equipment.
- Construction gensets.
- Cranes.
- HFO gensets.
- Pre-fabricated buildings for camps and offices.

Similarly, contract tendering for the initial construction work like earthwork, concrete and foundations, power distribution, is completed and contract award is pending construction release. Detailed engineering by Minproc is 45-50% completed for the processing plant; similarly, detailed engineering is complete for the mine village and is well advanced for the mine plant.

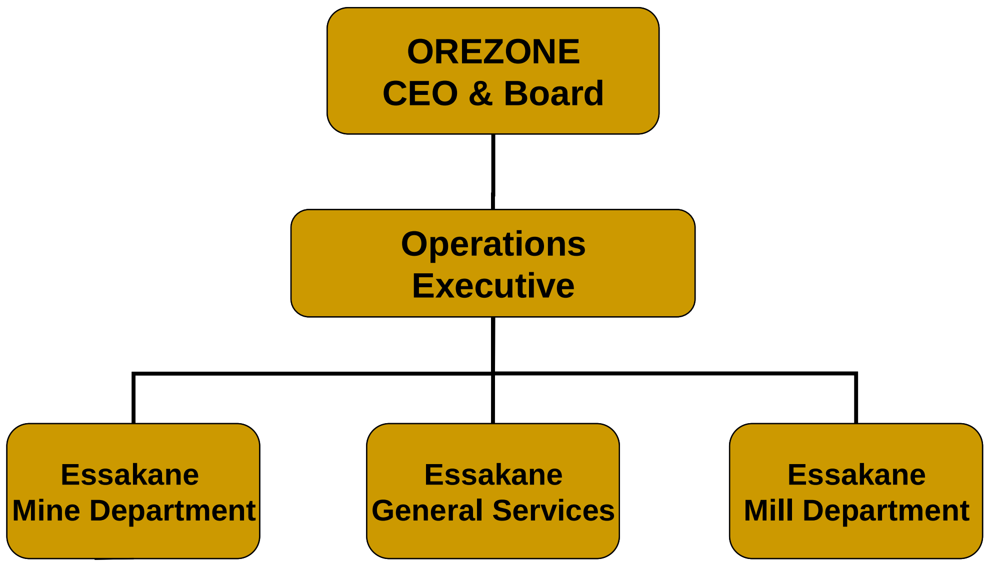
Table 18.12 Project milestones for selected site activities

	Start	**End**
Mine village	July 2008	January 2009
Offices	July 2008	November 2008
Truckshop & Warehouse	September 2008	January 2009
Power Plant and tank farms	February 2009	December 2009
Processing Plant	August 2008	October 2009
Processing Plant Commissioning	November 2009	December 2009
BWS – Tailing Dam	August 2008	December 2009
River dam and off channel reservoir	November 2008	May 2009
Pre-Production mining	April 2009	December 2009
Resettlement village	August 2008	October 2009

18.9.3 Production organization

The planned organization for the production phase is illustrated in Figure 18.13. An operation executive will report to Orezone's CEO and Board of Directors. Operations will be regrouped in three main sectors, Mine, Mill and General Services, that were staffed and organized progressively during the construction and development phase. This arrangement will result in a seam-free start of operations and achieve optimal results during the initial years of production, which are extremely important to the financial performance of the project.

Figure 18.13 Mine organization



18.10 Financial Analysis

18.10.1 Gold market and sales

The gold market is highly liquid and benefits from terminal markets (London, New York, Tokyo, Hong Kong) on almost a continuous basis.

Gold prices were in downward trend from 1980 to 2000 where it hit bottom and initiated a rapid upward trend that moved spot prices in excess of $1,000/oz momentarily in recent months. Analysts forecast continued strength in the gold price based on little/no growth in global mine production, weak US currency, higher inflation and new investment demand facilitated by the recent introduction of gold ETFs.

A constant gold price of $800/oz was used for our Base Case, but sensitivity at $600/oz and $1,000/oz are also presented for sensitivities. The DFS used four different cases of gold price/oil price combinations in its financial analysis as follows:

Case	Gold Price (US$/oz)	Oil Price ($/b)
1	580	50
2	460	40
3	650	60
4	720	80

Essakane's gold production will be transported by a combination of chartered planes and commercial flights to European-based refineries for sales through bullion banks or intermediaries. Shipping and refining costs of $2.50 per ounce were included in general services operating costs.

18.10.2 Currency exchanges and cost basis

Equipment and supplies are practically all imported to Burkina Faso from world wide sources. Quotes are generally made in the currency of the country of origin. Since the Project uses the US dollar currency as a basis, the exchange rates used in our cost estimates and financial analysis are compared to the DFS basis as follows:

	Update	DFS
US$/Euro	1.48	1.10
Rand/US$	7.40	7.29
US$/GBP	2.00	1.93
US$/$Can	1.00	N/A

The Burkinabe currency is the FCFA, which has been fixed to the Euro and previously the French Franc (656 FCFA/1 Euro) since 1994; this makes Burkina Faso essentially an Euro-based country. The difference in the Euro exchange rate since the DFS caused a 35% increase in European and Burkinabe based costs.

18.10.3 Government charges

The government of Burkina Faso has a complex and sophisticated array of duties, royalties and taxes that imposes levies on the revenues, costs and profit of mining companies.

As specified in the Mining Permit, a royalty of 3% of revenues is payable to the Government.

The main governmental charges on costs include the TPA on salaries and taxes and duties on fuel, equipment and other supplies.

The TPA represents 8% of expatriate salaries and 4% of national salaries and is only applied starting at the mid-life of the operation, but not later than seven years after the start of production. Actual training expenses can be credited against the TPA up to a maximum of 50% of the tax amount. In the financial models, a net charge of 4% of all salaries starting in 2015 was assumed.

Starting with commercial production, the tax and duties on fuels represent a charge of 7.5% of the price of fuel at the coast (PAC) plus $0.07 per liter for diesel and $0.058/l for HFO. The taxes and duties are 4% on mill reagents and 9% on most other supplies and equipment. Services provided by suppliers not registered in Burkina Faso are taxed at 10%.

Profit-based charges include a 20% income tax on operating profit after deducting operating costs, depreciation, interest payments and other corporate charges from revenues. In addition, a 6.25% withholding tax is imposed on dividend payments.

18.10.4 Overall operating costs

Operating costs for mining, processing, power and general services have been discussed in previous sections and were presented prior to governmental charges. In the financial models, these mine operating costs are referred to as C1 and include all the mine costs but without the governmental charges composed of royalties, taxes and duties on fuels, supplies and labor. The total cash operating costs referred to as C2, include all the government charges.

Costing is based on Q1-2008 information from suppliers and corresponds to labor rates and market conditions prevailing at that time. Operating costs are expressed in constant dollars, based on exchange rates stated in section 18.10.2 and without inflation adjustment over time.

Table 18.13 presents operating costs for our Base Case with gold price of $800/oz and Brent oil price of $85/barrel. These operating costs are also summarized in Table 18.14 for the gold price/oil price combinations selected earlier.

A comparison with the DFS must be qualified for the following factors:

- The cost estimate in the DFS is based on 2Q-2007 information.
- Costs for many supplies in the DFS were discounted by 15% since God Fields believed that underlying commodity prices were temporarily high.
- Weaker Euro to the US dollar.
- No/little governmental charges on labor, supplies and equipment were estimated.

The DFS presented the following mine life C2 operating costs for their gold price/oil price scenarios:

	Case 1	Case 2	Case 3	Case 4
Gold Price ($/oz)	580	460	650	720
Oil Price ($/b)	50	40	60	80
Gold production (000oz)	2,507	2,507	2,507	2,507
C2 Operating Costs	298	269	321	356

In spite of the qualifiers and different estimating basis, Cases 3 and 4 of Gold Fields are similar to the first two updated scenarios with oil prices of $66/b and $85/b.

Table 18.13 Overall Operating Costs – Base Case

Year Mine Costs	1	2	3	4	5	6	7	8	9	10	Total
Mining (000$)	37,406	40,602	46,026	51,495	51,407	31,544	33,339	33,553	14,822	1,158	340,990
Processing (000$)	28,665	28,198	28,673	27,080	24,503	24,683	24,540	24,216	23,838	9,429	243,828
Power (000$)	18,081	19,874	22,805	26,017	25,317	26,249	26,300	26,300	26,300	10,333	227,574
General Services (000$)	13,235	12,424	12,114	10,793	11,043	10,729	10,578	10,510	10,517	4,259	106,202
Sub-total Mine Costs (C1)	97,387	101,098	109,618	115,385	111,910	93,205	94,757	94,580	75,477	25,179	918,591
Government Charges											
Royalties	8,978	7,675	7,580	7,591	7,490	6,992	7,612	7,715	7,061	7,129	71,034
TPA-Labor	-	-	-	-	-	564	555	540	418	130	2,208
Tax & Duties	6,697	7,094	8,027	8,859	8,647	6,940	7,122	7,175	5,485	1,646	67,692
Sub-total Govt	16,675	14,769	15,608	16,349	15,639	15,116	15,391	14,776	13,032	4,578	140,934
Total Cash Costs (C2)	113,061	115,868	125,226	131,734	127,548	108,321	110,148	109,356	88,509	29,755	1,059,530
C1 ($/t milled)	12.98	13.49	14.62	17.78	20.81	17.28	17.60	17.53	13.99	11.87	15.82
($/oz)	260	317	347	370	386	294	296	322	254	215	310
C2 ($/t milled)	15.08	15.45	16.70	20.29	23.71	20.08	20.45	20.26	16.40	14.02	18.25
($/oz)	302	363	396	423	440	342	344	372	298	254	358

Table 18.14 Overall Operating Costs for various gold price/oil price combinations

Gold Price ($/oz)		600	800	1,000
Brent oil price ($/b)		66	85	100
C1	(000$)	878,675	918,591	949,552
	($/t milled)	15.12	15.80	16.34
	($/oz)	297	310	321
C2	(000$)	998,894	1,059,525	1,110,538
	($/t milled)	17.19	18.23	19.11
	($/oz)	337	358	375

18.10.5 Capital costs and working capital

Capital cost estimates were finalized in April 2008 and included Minproc's revised estimate for the portion of the project covered by their EPCM contract and by the Essakane Project team for the remaining areas under direct owner control. The following parameters and qualifications must be made:

- Estimate was based on Q1-2008 prices and costs.
- Estimate is only for future capex starting on January 1, 2008; Capex prior to January 1, 2008 of $48.425 M are excluded. These past expenditures include mostly exploration, pre-feasibility and feasibility related studies and standby costs.
- Estimate excludes all exploration and development drilling expenditures post January 1, 2008
- Essakane SA corporate costs are included, but Orezone head office costs are excluded.
- Financing related charges, like fees, consultants, etc., are excluded.
- Capitalized interests and standby fees from third party lenders are excluded.
- There is no specific inflation factors added to the estimate, other than the general contingencies.

Table 18.16 provides a detailed summary of capital expenditures for the Project starting on January 1, 2008 until commercial production is achieved on January 1, 2010 and all construction activities and payments are completed in the first quarter 2010. Initial capital expenditures total $420,444,000 and include the following provisions and contingencies:

	Essakane (000$)	Minproc (000$)	Total (000$)
Accuracy provision	8,898	7,599	16,497
Contingency	15,352	10,021	25,373
Total	24,250	17,620	41,870

The Accuracy Provision is meant to reflect an estimating margin regarding quantities and unit costs for reasonable changes to a known and specified item. The Contingency covers unexpected additions to the scope or major scope change, in addition to external factors like exchange rate fluctuations and inflation. The total amount of $41,870,000 in Provision and Contingency represents 11% of total expenditures before Provision and Contingency.

Sustaining capital requirement for the Essakane production period is presented in Table 18.15 and total $26,003,000 over the mine life. The Mine Department will require additional production drills as the proportion of saprock and fresh rock increases; some RC drills will also be replaced. Construction of Phases 2 and 3 of the tailings pond are scheduled in 2012 and 2015. An HFO genset will need to be added to the power plant to meet increased power demand in 2013.

Table 18.15 Sustaining Capital Expenditures

	2010	2011	2012	2013	2014	2015	2016	2017	Total
Mine Equipment (000$)	1,135	1,135	3,194	4,036	-	-	750	-	10,249
Tailings Pond (000$)	-	659	3,162	-	-	3,491	-	-	7,312
HFO Genset (000$)	-	-	5,500	-	-	-	-	-	5,500
Serv. Vehicles (000$)	368	368	368	368	368	368	368	368	2,942
Total	1,502	2,161	12,224	4,403	368	3,859	1,118	368	26,003

Working Capital requirements were estimated as follows:

Cash:	$1,000,000 (minimal)
Receivables:	One week of revenues
Production Inventory:	5,000 oz at C1 costs
Supplies Inventory:	3 months of operating costs
Payables:	1 month of operating costs

Consequently, working capital will fluctuate in time and will also vary with our Gold Price/Oil Price Scenarios.

Gold Price ($/oz)	600	800	1000
Brent Oil Price ($/b)	66	85	100
Q1-2010 Working Capital (000$)	25,040	27,089	28,960

Total capital requirements for the Essakane Project, as estimated in this Update, can be compared to the DFS as follows:

	Update (000$)	DFS (000$)
Initial Capital	420,444	346,519
Sustaining Capital	26,003	17,733
Working Capital	27,089	33,093

The combined initial and sustaining capital for this update correspond to $151/oz produced versus $145/oz produced in the DFS.

The increase in the initial capital requirement is mainly due to the change in scope regarding throughput, but also changes to many other sectors like HFO and LFO tank farms (previously paid by suppliers), a conservative number of gensets in the early years, a primary crusher and crushed ore pile instead of mineral sizers and no stockpile, major underestimates in owner's costs in the DFS, unfavorable variation in exchange rates and 12-month inflation since the DFS.

Table 18.16 Essakane Project capital costs

Area Code	Area Name	Owner Costs incl. Accuracy Provision	Minproc incl. Accuracy Provision	Grand Total
100	**INFRASTRUCTURE**			
103	Mine Village	6,694,322		6,694,322
104	Site Roads	212,468		212,468
105	Mine Roads	247,591		247,591
106	Pit Dewatering	337,780		337,780
107	Relocation	19,929,000		19,929,000
109	Light Fuel Storage System (permanent)	1,748,644		1,748,644
110	Heavy Fuel Oil Storage	4,179,533		4,179,533
111	Warehouse / Storage (mill)	29,978	286,334	316,312
112	Engineering Workshops (mill)	142,031	966,487	1,108,518
113	Offices and Administration	1,576,777	1,036,302	2,613,079
114	Medical and Training Facilities	105,000		105,000
115	Security building		56,006	56,006
116	Site Power Reticulation		1,825,015	1,825,015
118	Off River Water Reservoir	11,477,813	260,930	11,738,743
119	Mine Effluent Disposal		284,148	284,148
120	Hazardous Waste Disposal	102,000		102,000
122	Mine Workshops/Warehouse/Offices	3,106,783		3,106,783
123	Bulk Water Storage Reservoir	3,604,099		3,604,099
125	Diversion Weir	1,305,165		1,305,165
209	Plant Mobile Equipment		2,107,245	2,107,245
210	Air Strip	77,899		77,899
300	**PROCESSING FACILITIES**			
300	General	1,026,774	9,241,641	10,268,415
301	Ore Receipt and crushing	1,250,592	15,657,829	16,908,421
302	Grinding	494,994	26,540,765	27,035,759
303	Gravity		1,034,578	1,034,578
304	Pre Leach Thickening	295,176	5,331,018	5,626,194
305	CIL	207,379	21,063,355	21,270,734
306	Screening and Final Tails		2,310,665	2,310,665
307	Acid Wash & Elution	4,953	1,313,058	1,318,011
308	Regeneration	40,334	1,356,507	1,396,841
309	Intense Cynidation		466,495	466,495
310	Electrowinning & Gold Room	30,721	1,318,622	1,349,343
311	Reagents - Caustic & Cyanide	54,899	663,164	718,063
312	Reagents - Hydrochloric Acid and Lime	35,921	721,448	757,369
313	Reagents - Flocculant		369,719	369,719
314	Air Services - Compressed Air		410,835	410,835
315	Air Services - Oxygen		1,906,120	1,906,120
316	Water Services - Raw Water		1,003,157	1,003,157
317	Water Services - Process Water		427,079	427,079
318	Water Services - Potable Water		454,864	454,864
319	Tailings Storage Facility	13,116,356	4,952,154	18,068,510
320	Plant Effluent Disposal		226,395	226,395
321	Assays Office	606,496	1,219,086	1,825,582
323	Fire Water	66,150	530,712	596,862
324	Commissioning and Vendor Representation		1,418,970	1,418,970
326	Earthworks Contractor Preliminary and General		5,271,857	5,271,857
327	Civilworks Contractor Preliminary and General		5,887,828	5,887,828
328	Platework/Steelwork Contractor Preliminary and General		4,185,204	4,185,204
329	Mechanical/Piping Contractor Preliminary and General		2,830,258	2,830,258
330	Plant Electrical		16,214,992	16,214,992
331	Electrical Contractor Preliminary and General		2,133,384	2,133,384
350	2007 Expenditures (credit)		(1,780,255)	(1,780,255)

Area Code	Area Name	Owner Costs incl. Accuracy Provision	Minproc incl. Accuracy Provision	Grand Total
400	**PLANT AND EQUIPMENT**			
401	Mining equipment	58,707,446		58,707,446
402	Light Vehicles	1,471,080		1,471,080
403	Explosives Plant	297,703		297,703
406	Support Equipment	4,683,204		4,683,204
415	Furnitures (office, camp)	1,219,008		1,219,008
420	Capital Spares	2,394,315		2,394,315
430	Power Plant	24,552,640		24,552,640
435	Operating systems	1,359,450		1,359,450
452	Power Factor correction	262,500		262,500
500	**CONSTRUCTION INDIRECTS**			
551	Construction Facilities	174,075		174,075
565	Temporary Facilities	1,111,383		1,111,383
580	Construction Equipment & Tools	1,278,000		1,278,000
584	Construction Engineering (consultants)	3,045,840		3,045,840
586	Construction Management	3,180,554	13,694,275	16,874,829
587	Construction Fuels & Lubes	4,728,163		4,728,163
588	Construction Freight	2,577,400		2,577,400
592	Vendors Representatives	157,500		157,500
595	Project Management Longueuil	5,002,068		5,002,068
600	**GENERAL SERVICES**			
601	Administration Burkina	4,066,846		4,066,846
602	Insurance	3,694,425		3,694,425
603	Human Resources, Training & Security	3,601,860		3,601,860
604	Procurement & Logistics	2,365,086		2,365,086
606	Social & Environmental Management	2,403,241		2,403,241
607	Transportation	1,248,727		1,248,727
608	Camp (Food & Lodging)	6,279,033		6,279,033
609	Customs & Duties	10,778,517		10,778,517
700	**PRE-PRODUCTION**			
701	Operations Management	2,313,112		2,313,112
702	Maintenance Management	3,382,254		3,382,254
703	Mine Engineering	1,275,011		1,275,011
704	Mine Geology	908,619		908,619
705	Pre-Stripping	7,385,592		7,385,592
706	Overburden Storage	736,001		736,001
707	Process Plant Management	819,211		819,211
708	Process Training and Plant Commissioning	2,868,042		2,868,042
710	Power Plant Training and Commissioning	789,640		789,640
712	Initial Fill	1,245,615		1,245,615
714	Pre Production Revenues	-4,929,540		(4,929,540)
800	**CORPORATE - ESSAKANE SA**			
800	Corporate - Essakane SA	332,000		332,000
	TOTAL	**239,873,249**	**155,198,246**	**395,071,495**
	CONTINGENCY	**15,351,888**	**10,020,801**	**25,372,689**
	GRAND TOTAL	**255,225,137**	**165,219,047**	**420,444,184**
	ACCURACY PROVISION (included)	8,897,974	7,599,000	16,496,974
	CONTINGENCY	15,351,888	10,020,801	25,372,689
	OVERALL PROVISIONS	24,249,862	17,619,801	41,869,663

18.10.6 Financial structure and project financing

The capital structure of Essakane SA is relatively simple since all past and future expenditures are accounted as Shareholders' Debt. The equity is minimal. Starting on January 1, 2008, past expenditures and future expenditures (as incurred) accumulate interest at an annual rate of 6.75% calculated monthly. The Government of Burkina Faso fixes the interest rate on Shareholders' loans at the BCEAO rate (constant at 4.75% since August 2006) plus 2%. However, there is no withholding tax on interest payments due on Shareholders' loan. The Financial Models perform a cash sweep at the end of each period to reduce the Shareholders' debt as fast as possible.

Once the Shareholders' debt is repaid, all free cash flow after tax is distributed as dividends, 90% to Orezone through Orezone Essakane (BVI) Limited and 10% to the Government of Burkina Faso. A withholding tax of 6.25% is paid by Orezone on its dividends.

18.10.7 Financial models and results

Two Financial Models were prepared for the Essakane Project; they differ only in the tax depreciation of the investments. Model A assumes that all past (prior to January 1, 2008) and future (post Jan. 1, 2008) expenditures are depreciated on a units-of-production basis. Model B uses past expenditures as loss-carried-forward and depreciates future expenditures on a units-of-production basis. Tax depreciation is still being clarified with the Government of Burkina Faso as part of the Convention Minière; these Financial Models are based on the most probable outcome of these discussions.

The financial results (NPV and IRR) do not take past expenditures ($48.425 M) into account; these are considered as sunk costs and the analysis is done on a forward basis. However, past expenditures are taken into account for tax calculations. The moment zero is taken as April 1, 2008.

The IRR on the Gross Project Cash Flow, which is the cash flow prior to any government charges and income taxes were also calculated. This allows a basic evaluation of the Project, before the value sharing formula selected by the State, owner of the mineral resources.

Table 18.17 presents the financial results generated by Model A, which are only slightly less favorable to Orezone than Model B. On the basis of these results, the investment to bring the Essakane Project into production is financially justified under current market conditions for gold and oil.

Table 18.17 Financial results for different scenarios of Gold Price/Oil Price

Gold Price ($/oz)	600	800	1,000
Brent Oil Price ($/b)	66	85	100
C1 ($/oz)	297	310	321
C2 ($/oz)	337	358	375
Gross Project – IRR (%)	14.2	28.6	41.5
Orezone IRR (%)	10.0	20.8	30.6
Orezone NPV-0% (M$)	289.4	630.2	984.7
Orezone NPV-5%(M$)	112.8	372.8	637.6
Orezone Payback Period (years)	5.9	3.4	2.3

Additional financial simulations were conducted to look at sensitivities of financial results to change in the major inputs. Table 18.18and Figure 18.14 present those sensitivities for gold price, operating costs, initial capital cost, head grade and oil price. Financial results appear most sensitive to gold price and head grade.

Table 18.18 Sensitivities of financial results

	Gold Price	NPV 0%	NPV 5%	IRR
	700	444,886	232,926	15.2
	800	630,167	372,751	20.8
	900	817,104	512,842	26.5

% Change	Cash Costs	NPV 0%	NPV 5%	IRR
0.90	829,991	689,334	418,191	22.7
1.00	918,591	630,167	372,751	20.8
1.10	1,007,191	571,085	327,322	19.0

% Change	Initial Capex	NPV 0%	NPV 5%	IRR
0.90	378,400	652,243	398,199	23.4
1.00	420,444	630,167	372,751	20.8
1.10	462,489	608,180	347,315	18.7

% Change	In-situ grade	NPV 0%	NPV 5%	IRR
0.90	1.50	482,404	261,183	16.3
1.00	1.67	630,167	372,751	20.8
1.10	1.84	778,919	484,442	25.1

	Oil price	NPV 0%	NPV 5%	IRR
	85	630,167	372,751	20.8
	90	622,899	367,231	20.6
	100	608,390	356,210	20.2
	110	593,881	345,189	19.7
	120	579,372	334,168	19.3

Figure 18.14 Sensitivities of financial results

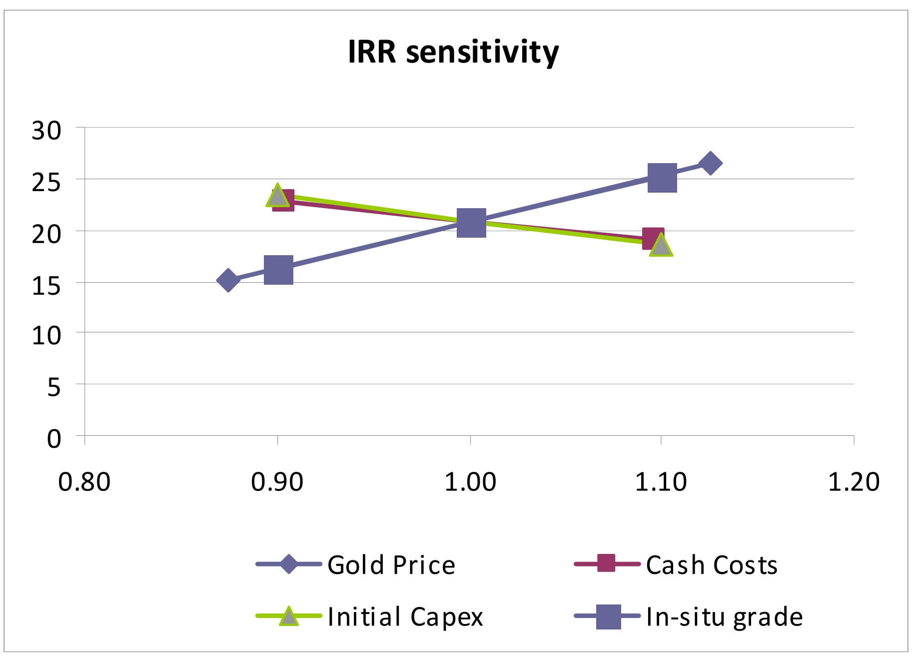
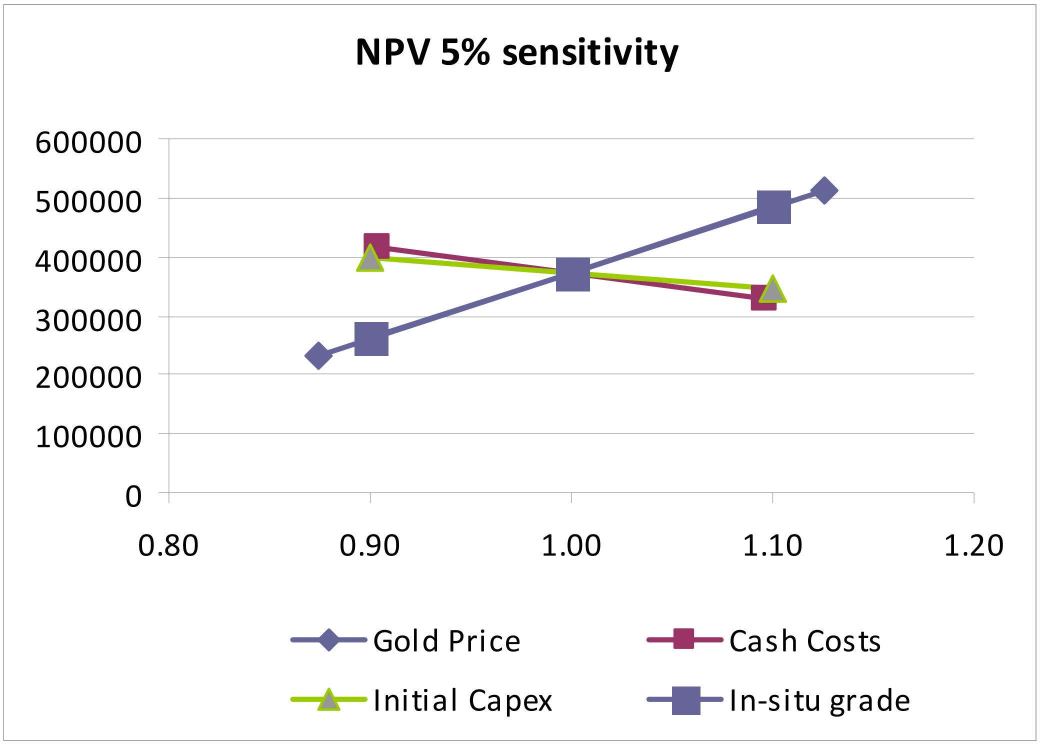
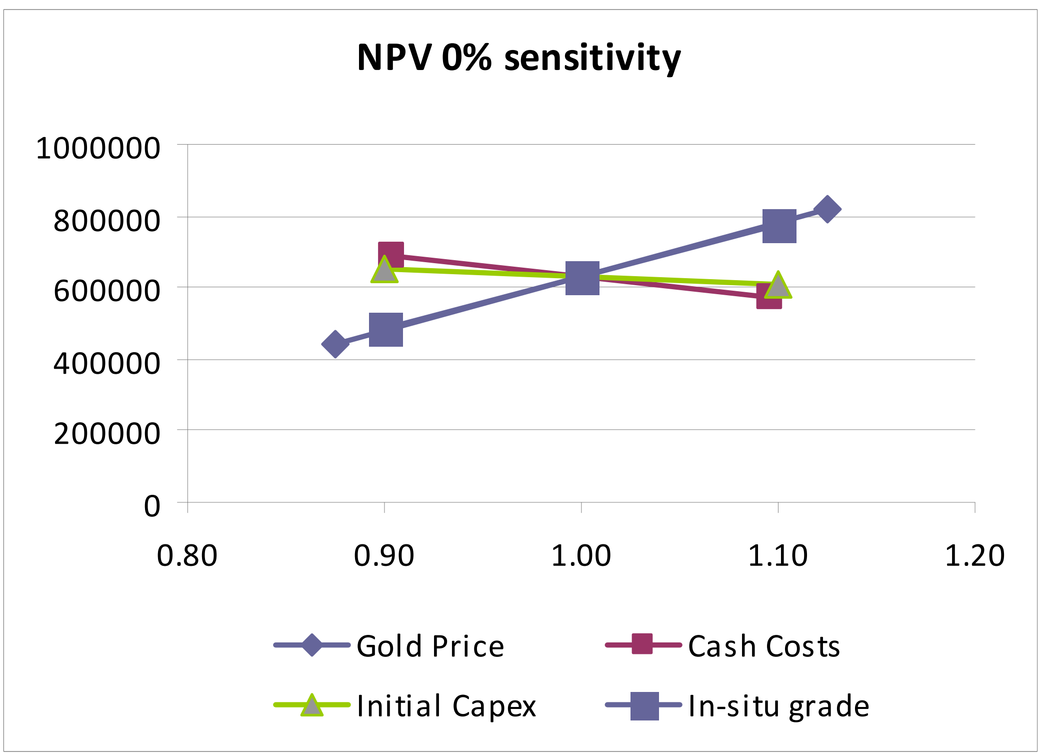


Figure 18.14 Sensitivities of financial results





19 Interpretation and conclusions

19.1 Geology and Mineral Resources

Starting in January 2006, Gold Fields 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 (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 Au/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 Au/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.

Gold Fields 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, Gold Fields' drilling, sampling of RC and DD drillholes, sample preparation and analytical procedures, and quality control procedures, meet industry standards. The procedures used by Gold Fields 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 both the DFS and the Addendum 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, G Mining Services Inc. 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$600/oz mine design shell. The total Probable Mineral Reserves for the Project are 3,121,000 oz of gold contained in 58.12 M t at an average head grade of 1.67 g Au/t and at a stripping ratio of 2.71 on a tonnage basis. These reserves yield a positive cashflow based on project economic assumptions and detailed technical evaluations of mine production and ore processing.

G Mining Services Inc. used the Mineral Resource block model to develop larger mining blocks to conservatively simulate mining dilution and ore losses. Operating costs differentiated properly between saprolite, transition and fresh rock which resulted in recovered cut-off grades of 0.44 g Au/t, 0.55 g Au/t and 0.66 g Au/t respectively.

The processing schedule plans for 7.5 M tpy for the first three years of production when the ore feed is mostly saprolite and transition ore, 6.5 M tpy in the fourth year of production when a significant portion of fresh rock is fed to the mill and down to 5.4 M tpy when mill feed is entirely fresh rock starting in the fifth year of production.

In order to mine the planned 32 M tpy of ore and waste during the first half of the mine life, a mining fleet of one 10 m^3 backhoe excavator, two 15 m^3 front-shovel excavator, one 12 m^3 wheel loader, five 100-t haulage trucks and fourteen 140-t haulage trucks with the usual support equipment of dozers, graders and miscellaneous service vehicles, was selected.

At the end of the mine life, overburden and tailings storage facilities will be recontoured and stabilized with rock cladding. On closure, the surface mine will 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

G Mining Services Inc. has completed a risk assessment of the Project as it stands in early June 2008. This assessment does not address issues related to political risk and changes in government legislation. On a positive note, the environmental and mining permits were issued on a timely basis and discussions towards finalizing the Mining Convention are progressing well.

In order of decreasing importance, the identified Project risks are as follows:

1. Gold price, oil price and price of commodities in general

The gold price over the medium to long term is the most important factor for the project. The rapid increase in gold price, oil price and commodity prices since 2005 and their increased price volatility has caused some uncertainty as to the medium-to-long term prices. Obviously, the gold price impacts revenues while oil and other commodity prices largely influence operating costs. Operating cash flow margins of gold mining producers have benefited only partially from the recent gold price increase because of the negative impact of other prices on operating costs. As long as the relationship between gold prices and other commodity prices remains the same, important variations in gold price will be attenuated by corresponding commodity prices and should maintain the economic viability of the Project.

Commodity prices will have a low to medium impact on initial capital expenditures because the capital cost estimate was updated in Q1-2008 and practically all equipment and supplies will be purchased in 2008. Labor and energy prices will be the main residual variables.

2. Timing of construction release

The Project has been scheduled and organized for a construction release on July 1, 2008 and for a start of commercial production in January 2010. The construction schedule has been compressed from 24 months to 18 months by pre-funding many time-critical equipments. Any further delay in construction release will cause corresponding delay in the completion date and will likely result in an increase in initial capital costs.

3. Exchange Rates

Evaluation for the Project was done on a US dollar basis. However, the Project capital cost, and to a lesser extent operating costs, has a significant exposure to the Euro, the South African Rand and the Canadian dollar. Because the US dollar has already experienced a large devaluation and this is reflected in the recent cost estimate, any further devaluation should be relatively small.

4. Logistics

Because all equipment and supplies will be procured on a world-wide basis, logistics and freight forwarding will be particularly demanding. In most cases, land transportation will be followed by ocean shipping to be followed again by land transportation before reaching Essakane. Shipments will go through a minimum of three jurisdictions, each with their own procedures and delays.

5. Water Management

Water supply being critical to the Project, further detailed engineering studies need to be completed to confirm the entire water management arrangement with particular emphasis on reservoir dimensions and quantities. Specific site conditions may result in adjustment to the location of some facilities and impact capital costs.

6. Manpower

Because of the global demand for experienced mining personnel, staffing of critical expatriates for construction and operations may be difficult.

Even though Burkina Faso now has three medium-sized mining operations, training of the national workforce will be demanding because of the large-size of the Project.

7. Local Population

Even though the preparation work, including the training of local residents, has been excellent, resettlement of a large number of people at Essakane represents a complex activity and will require an important contribution and focus from the Project team.

In addition, major efforts and programs will need to be implemented to provide employment to the local population considering their low-skill level. High expectations have been created and must be satisfied. Programs must be implemented to ensure a sustainable livelihood to the resettled population and prove to be effective and successful.

8. Geotechnical and Geochemical Investigation

Even though the general geotechnical and geochemical characteristics of the soils and materials at the Project site are known, a detailed investigation is required for the final siting and engineering of the various earth structures like dam, reservoirs, tailings pond and overburden storage areas, selection of fill materials and storage of overburden considering the arsenic content of some of the waste material in the Essakane pit. This detailed work may have an impact on capital and operating costs.

20 Recommendations

20.1 Mineral Resource evaluation

The following program is recommended by Snowden as important to the Essakane project development and planning process to extend the Life of Mine beyond 9.4 years:

- Infill diamond core drilling to upgrade Inferred Resources located below the US$600/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.
- Resource calculations for surface satellite deposits in the area.
- Exploration of resource extensions south of the May 2007 block model.
- Mapping of structural details within the pit benches 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 the pit bottom.
- Ongoing evaluation of gold prospects on the adjacent permits and development of a business plan for the District and renewal of exploration permits.

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 >200 m BS
Other permits	45	6,750	1,080	Upgrade of mineral resources + reserves inventory: Essakane North, Falagountou, Gossey
Total	188	35,350	5,665	

20.2 Mining

G Mining Services Inc. recommends the following additional studies:

- Detailed geochemical characterization of the materials (ore and waste, arenite and argillite formations, saprolite, transition and fresh rock) contained within the final pit to confirm the non-acid generating nature of the materials and the risks regarding arsenic leaching for certain materials, if applicable.

- Optimization studies regarding waste disposal within the overburden storage facility to minimize long-term environmental impacts.
- Identification of suitable construction materials among the pit waste materials.
- Engineering studies to replace emulsion explosives with cheaper ANFO.
- Sponsor studies on the long-term water quality of the water within the planned pit lake.

20.3 Processing and Tailings Disposal

GRD Minproc and G Mining Services Inc. recommend the following additional studies:

- Determination of the kaolinite content within the different types of saprolite ores and its impact on slurry density and viscosity within the leaching circuit and final slurry density for tailings disposal.
- Thickening tests for saprolitic ore.

21 References

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22 Dates and signatures

Name of Report: IAMGOLD Corporation: Updated Feasibility Study – Essakane Gold Project, Burkina Faso

Date: March 3rd, 2009

Issued by: IAMGOLD Corporation

Signed:	*/s/ Ian M. Glacken*	[ ]
	Ian M. Glacken	Date
Signed:	*/s/ John F. Hawxby*	[ ]
	John F. Hawxby	Date
Signed:	*/s/ Philip Bedell*	[ ]
	Philip Bedell	Date
Signed:	*/s/ Louis-Pierre Gignac*	[ ]
	Louis-Pierre Gignac	Date
Signed:	*/s/ Louis Gignac*	[ ]
	Louis Gignac	Date

23 Certificates

G Mining Services Inc.

CERTIFICATE OF QUALIFIED PERSON (QP)

1. I, Louis Gignac, President and Principal of G Mining Services Inc., 8250 Racine, Brossard, Québec J4X 1T8, certify that:
2. I have graduated from Laval University, Canada with a B.Sc. In Mining Engineering in 1973, from the University of Minnesota, USA with a M.Sc. in Mining Engineering in 1974, and from University of Missouri-Rolla, USA with a Doctorate of Engineering (D. Eng.) in Mining Engineering in 1979.
3. I am in good standing as a member of the Order of Engineers of Québec (No. 24041) and I am a member of the Canadian Institute of Mining and the Society for Mining, Metallurgy and Exploration.
4. I have worked in the mining industry continuously since my graduation from university.
5. I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that as a result of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I have been involved in mining operations, engineering, construction and development, financial evaluation and acquisition and senior management in the mineral industry for thirty five years.
6. I have made two visits to the Essakane Project in May 2007 and March 2008.
7. I am responsible for the preparation of Sections 1, 2, 3, 4, 5, 15, 18.2, 18.3, 18.5, 18.7, 18.8, 18.9, 18.10, 19.3 and 20.3 of the Technical Report entitled "IAMGOLD Corporation: *Updated Feasibility Study – Essakane Gold Project, Burkina Faso"* dated March 3rd, 2009.
8. I am independent of the issuer as defined in Section 1.4 of National Instrument 43-101.
9. I have read national Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
10. 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.
11. I consent to the filing of the Technical Report with any stock exchange or any regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated at Longueuil, this 3rd day of March 2009.

/s/ Louis Gignac

Signed: Dr. Louis Gignac, Ing.

CERTIFICATE of QUALIFIED PERSON

(a) I, Ian Martin Glacken, of 50 Colin St., West Perth, Western Australia, do hereby certify that:

(b) I am the co-author of the technical report titled "IAMGOLD Corporation: *Updated Feasibility Study – Essakane Gold Project, Burkina Faso*" (the 'Technical Report') effective June 3rd, 2008 and readdressed March 3rd, 2009 to IAMGOLD Corporation

(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 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 27 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 10 years, including gold deposit evaluation for all of that time.

(d) I have made current visits to the Essakane property in July 2006 for 6 days and in May 2008 for 4 days.

(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 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 8th day of March, 2009

CHARTERED PROFESSIONAL
THE AUSTRALASIAN INSTITUTE OF MINING & METALLURGY
IAN M GLACKEN
GEOLOGY

Ian M. Glacken, FAusIMM(CP), CEng

Principal Consultant, Optiro Pty Ltd.

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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 "IAMGOLD Corporation: *Updated Feasibility Study – Essakane Gold Project, Burkina Faso*" the 'Technical Report' effective June 3rd, 2008 and readdressed March 3rd, 2009 to IAMGOLD Corporation

(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 3rd of March, 2009

/s/ John F. Hawxby

Signed
John F. Hawxby (Pr.Eng, MIET, MSAIEE, BSc. Eng.)
Senior Project Manager
GRD Minproc (Pty) Ltd

CERTIFICATE OF QUALIFIED PERSON (QP)

1. I, Louis-Pierre Gignac, Senior Mining Engineer for G Mining Services Inc., 8250 Racine, Brossard, Québec J4X 1T8, certify that:
2. I have graduated from McGill University, Canada with a B.Sc. In Mining Engineering in 1999, and from École Polytechnique de Montréal, Canada with a M.Sc.A. in Industrial Engineering in 2002.
3. I am in good standing as a member of the Order of Engineers of Québec (#132995) and I am a member of the Canadian Institute of Mining.
4. I have worked in the mining industry continuously since my graduation from university.
5. I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that as a result of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101. I have been involved in mining operations, engineering and financial evaluation for 8 years, including pit optimization, surface mine design, mineral reserve calculations and mine scheduling. This has included work at Rosebel Gold Mines NV, the Camp Caiman Project and work on other pre-feasibility and feasibility studies.
6. I have made one visit to the Essakane Project in March 2008.
7. I am responsible for the preparation of Sections 17.3, 17.4, 18.4, 19.2 and 20.2 of the Technical Report entitled "IAMGOLD Corporation: *Updated Feasibility Study – Essakane Gold Project, Burkina Faso"* dated March 3rd, 2009.
8. I am independent of the issuer as defined in Section 1.4 of National Instrument 43-101.
9. I have read national Instrument 43-101 and Form 43-101F1, and the Technical Report has been prepared in compliance with that instrument and form.
10. 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.
11. I consent to the filing of the Technical Report with any stock exchange or any regulatory authority and any publication by them, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.
12. A copy of the "IAMGOLD Corporation: *Updated Feasibility Study – Essakane Gold Project, Burkina Faso"* is submitted as a computer readable file. The requirement of electronic filing necessitates submitting the report as an unlocked, editable file. I accept no responsibility for any changes made to the computer file after it leaves my control.

Dated at Longueuil this 3rd day of March 2009.

/s/ Louis-Pierre Gignac
Signed
Louis-Pierre Gignac, Eng.

PHILIP R. BEDELL

CERTIFICATE

As an author of a portion of this report, I, Philip R. Bedell do hereby certify that:

1. I am employed as the Senior Principal Geotechnical Engineer by, and carried out this assignment for, Golder Associates Ltd, 309 Exeter Road, Unit 1, London, Ontario N6L 1C1, tel (519) 652-0099 fax (519) 652-6299.
2. This certificate applies to Section 18 "Other relevant data and information", specifically sections 18.1 and 18.6, of the Technical Report entitled "IAMGOLD Corporation: Updated Feasibility Study – Essakane Gold Project, Burkina Faso", effective June 3rd, 2009 and readdressed March 3rd, 2009.
3. I hold the following academic qualifications:

   B.E.Sc., Civil Engineering, The University of Western Ontario, 1966

   M.E.Sc., Soil Mechanics and Foundation Engineering, The University of Western Ontario, 1967.
4. I am a registered Professional Engineer of Ontario (membership number 03046018); as well, I am a Designated Consulting Engineer in Ontario (No. 1452);
5. I have worked as a geotechnical engineer and consultant for the last 41 years. My work experience includes being Senior Project Director, Manager and Design Engineer responsible for major civil engineering works including the design and construction of mining, transportation and municipal infrastructure projects. The mining projects have included the geotechnical investigation, design and construction monitoring for open pit mines and related tailings facilities and infrastructure, as well as responsibility for a review and report preparation for a forensic investigation of a tailings dam failure in South America;
6. I am familiar with NI 43-101 and, by reason of education, experience and professional registration, I fulfil the requirements of a qualified person as defined in NI 43-101.
7. I have visited the property which is the subject of this report in May 2008;
8. I managed the preparation by Golder Associates Ltd. of the Technical Memoranda on Water Supply and Management, Tailings Storage and Waste Rock Stabilization and Characteristics. The relevant portions of these memoranda have been summarized in the "IAMGOLD Corporation: Updated Feasibility Study – Essakane Gold Project, Burkina Faso" Technical Report.
9. I am independent of the parties involved in the transaction for which this report is required, other than providing consulting services;
10. I have had no previous involvement with this property.
11. I am familiar with NI 43-101 and I consider that the portions of the report pertaining to the water supply, tailings and waste rock management have been prepared in compliance with the instrument.
12. As of the date of this certificate, to the best of my knowledge, information and belief, the portions of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed so as to make the technical report not misleading.
13. I consent to the filing of the report with any Canadian stock exchange or securities regulatory authority, and any publication by them of the report.

Dated March 3rd, 2009

/s/ Philip R. Bedell
Signed
Philip R. Bedell, P.Eng.