Red5Limited ABN 73 068 647 610

ASX REPORT TO SHAREHOLDERS

16 July 2009

Red5Limited
is a publicly listed company on the ASX
- ticker symbol RED

The Board strategy is to focus on the development of Siana.

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Siana Bankable Feasibility Study shows cash flow of US$228 million and an IRR of 38%

The Red 5 Limited board has approved the Siana Joint Venture open pit and underground bankable feasibility study using a base case gold price of US$800 per ounce. Cash flows are pre-tax pending finalisation of tax benefits applicable to mineral developments in the Philippines.

During the mine life, and based on reasonable assumptions of Inferred Resource conversion into the mine plan during underground mining, production from known resources is forecast to reach 849,000 ounces over a ten year operating mine life at a cash cost of US$351 per ounce.

Funding requirement to sustainable positive cash flow is forecast at US$72.5 million, comprising US$56.5 million in capital development to first gold pour, weighted capital contingency of US$6.0 million and working capital for the first six months of operation of US$10.0 million. Underground development expenditure will be funded from cash flow. Total life of mine costs, including capital, average US$473 per ounce.

The Ore Reserve is estimated to 400 metres vertical and the Mineral Resource to 500 metres vertical. The orebody remains open at depth and to the north and south. Not included in the Resource data base is SMDD 135 intersection 3 metres at 31.6 g/t gold at 220 metres vertical and accessible from early underground development, and SMDD134 intersection 5 metres at 25.4 g/t gold, 48 g/t silver and 2.4% zinc – the deepest northern orebody intercept.

At the recently granted Mapawa MPSA, approximately 30 km north of Siana, there is historic evidence of near surface mineralisation with potential as a future ore source for Siana.

Funding - the Company has prepared an Information Memorandum to be forwarded to financial institutions with whom confidentiality agreements have been signed.

Greg Edwards
Managing Director

Background

In April 2007 the Board announced the scope of the Siana Joint Venture open pit development based on the results of a Bankable Feasibility Study, together with the acceleration of detailed studies for an integrated underground operation. The latter initiative resulted from a positive underground scoping study based on an Inferred Resource below the proposed open pit.

Deep infill drilling continued throughout late 2007 and 2008, together with studies on underground geotechnical characteristics, confirmatory metallurgical testing, preliminary underground mining technique selection and Mineral Resource estimation.

Subsequently an underground mine design was completed defining a Probable Ore Reserve, together with refreshed capital and operating cost estimates for the construction and development of an open pit and underground mine.

Financial modelling indicates a robust project with strong return on investment at gold prices well below current levels.

Siana Mineral Resource Statement

Mineral Resource estimates described in this report are consistent with the guidelines and definitions of the 2004 Australasian Code for Reporting of Mineral Resources and Ore Reserves (the JORC Code).

They are assumed to have a reasonable prospect for eventual economic extraction according to the analysis of the known data, and from which Ore Reserves have been derived by the application of appropriate Modifying Factors including mining, metallurgical, economic, marketing, legal, environmental, social and governmental considerations.

Open Pit Mineral Resource

The previously reported open pit Mineral Resource is summarised in Table 1 by classification category, and in Table 2 by area.

The open pit Mineral Resource was estimated by Hellman & Schofield Pty Ltd (H&S) using the method of Multiple Indicator Kriging (MIK) with block support correction. MIK is one of a number of methods that can be used to provide better estimates of recovered grade for open pit mining than the more traditional methods such as Ordinary Kriging and inverse distance weighting. Estimation methods such as MIK provide an alternative approach based on large block sizes, often called panels, and are tailored to drill hole spacing.

The model estimated resources into panels with dimensions 10mE by 20mN by 6m vertical and assumes the selective mining unit for selecting ore and waste in an open pit will be approximately 5mE by 5mN by 3m vertical. MIK of gold grades used indicator variography based on the resource sample grades, with continuity of gold grades characterised by indicator variograms at fourteen indicator thresholds.

H&S do not recommend application of ore loss and dilution factors in quantifying open pit ore reserves based on this methodology.

Panels in the model have been classified as Indicated or Inferred Resources based on the number and location of samples used to estimate proportions and grade of each panel.

MINERAL RESOURCES

Table 1. Mineral Resource by Category

Category	Tonnes M (million)	Au g/t	Au '000 oz	Ag g/t	Ag '000 oz
Indicated Resource					
Open Pit	3.07	3.4	336	8.5	839
Stockpiles	0.08	1.3	3	10.7	29
Underground	2.00	6.7	430	10.2	655
Total Indicated Resource	**5.15**	**4.6**	**769**	**9.5**	**1,523**
Inferred Resource					
Open Pit	0.16	2.9	15	13.6	70
Underground	1.38	7.6	338	11.3	503
Total Inferred Resource	**1.54**	**7.1**	**353**	**11.5**	**573**
Total Mineral Resource	**6.69**	**5.2**	**1,122**	**9.7**	**2,095**
Indicated/Total Resource	77%		69%		73%

Note: - reported February 2009
- contains minor rounding errors
- Open pit cut-off grade 1.1g/t Au
- Underground Resources are defined as the region below the designed Open Pit (nominally below -170mRL) and nominal +2g/t Au model

Table 2. Mineral Resource by Area

Category	Tonnes M (million)	Au g/t	Au '000 oz	Ag g/t	Ag '000 oz
Open Pit and Stockpiles					
Indicated	3.15	3.3	339	8.5	868
Inferred	0.16	2.9	15	13.6	70
Total Open Pit and Stockpile Resource	**3.31**	**3.3**	**354**	**8.8**	**937**
Underground					
Indicated	2.00	6.7	430	10.2	655
Inferred	1.38	7.6	338	11.3	503
Total Underground Resource	**3.38**	**7.1**	**768**	**10.7**	**1,158**
Total Mineral Resource	**6.69**	**5.2**	**1,122**	**9.7**	**2,095**
Indicated/Total Resource	51%		68%		55%

Underground Mineral Resource

Cube Consulting Pty Ltd (Cube) provided a revised underground Mineral Resource estimate and classification based on drill data available at December 2008. Results were published in February 2009.

The underground Indicated and Inferred Resource total of 3.4 million tonnes at 7.1 g/t gold (768,000 ounces) and 10.7 g/t silver (1.16 million ounces) represents a nine percent increase in ounces of contained gold compared with the initial underground Inferred Resource (2.9 million tonnes at 7.4 g/t Au and 13.1 g/t Ag for 706,000 ounces of gold and 1.25 million ounces of silver) reported in April 2007.

The Indicated category accounts for 56 percent of the total underground Mineral Resource and is the basis for conversion to a Mining Reserve in the underground feasibility study.

Individual panels (lenses) within the underground Resource are detailed in Table 3.

The underground database was augmented by twenty two diamond drill holes for 11,078 metres completed since the initial 2007 estimate.

Drilling, sampling, and quality control procedures for these holes were consistent with those applied for the open pit and preliminary underground feasibility studies and are as described in the Appendix.

Table 3. Underground Mineral Resource Panels*

Classification	Panel	Tonnes ('000)	Au g/t	Au oz	Ag g/t	Ag oz
Indicated	1	296	8.6	82,085	9.0	85,799
	2	850	5.9	161027	10.1	277,492
	3	614	7.1	141,141	12.0	237,867
	4	34	3.6	3,885	6.5	7,032
	5	12	11.9	4,503	9.7	3,677
	6	53	5.7	9,818	11.8	20,289
	7	58	5.7	10,601	3.2	5,986
	8	26	3.5	2,889	3.0	2,464
	9	54	8.0	14,042	9.2	16,071
	10	-	-	-	-	-
Total Indicated		**2,000**	**6.7**	**430,000**	**10.2**	**657,000**
Inferred	1	255	8.1	66,238	9.6	78,765
	2	453	7.1	104,044	14.0	203,716
	3	396	8.6	109,991	12.6	161,226
	4	21	4.2	2,887	7.6	5,227
	5	12	12.4	4,923	8.6	3,413
	6	35	5.3	6,094	12.5	14,203
	7	33	5.0	5,322	3.3	3,457
	8	67	3.9	8,395	2.2	4,868
	9	70	7.2	16,156	7.7	17,414
	10	39	10.7	13,481	8.1	10,247
Total Inferred		**1,400**	**7.6**	**338,000**	**11.3**	**503,000**
Grand Total		**3,400**	**7.1**	**768,000**	**10.7**	**1,160,000**

* *contains minor rounding errors; no cut-off grade applied*

Interpretation

The underground mineralisation interpretation includes three dominant carbonate hosted lenses (Panels 1 to 3) defined by geological structures and nominal plus 2 g/t gold outlines, extending to a known depth of approximately 500 metres below surface, striking NNW and steeply dipping to the east (Figure 1). These three panels represent the 20 metre to 80 metre wide central carbonate zone mineralisation generally comprising soft clay rich breccias in fault zones with disseminated pyrite and minor polymetallic sulphide veinlets (Figure 2).

Seven additional lenses (Panels 4 to 10) are hosted in a basalt domain to the east of the central carbonate zone.

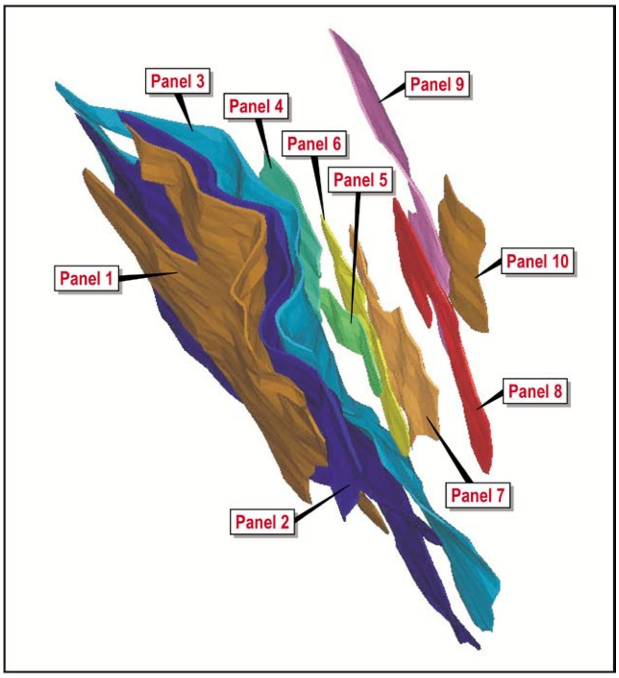

Figure 1. Siana mineralised panels - oblique NE view

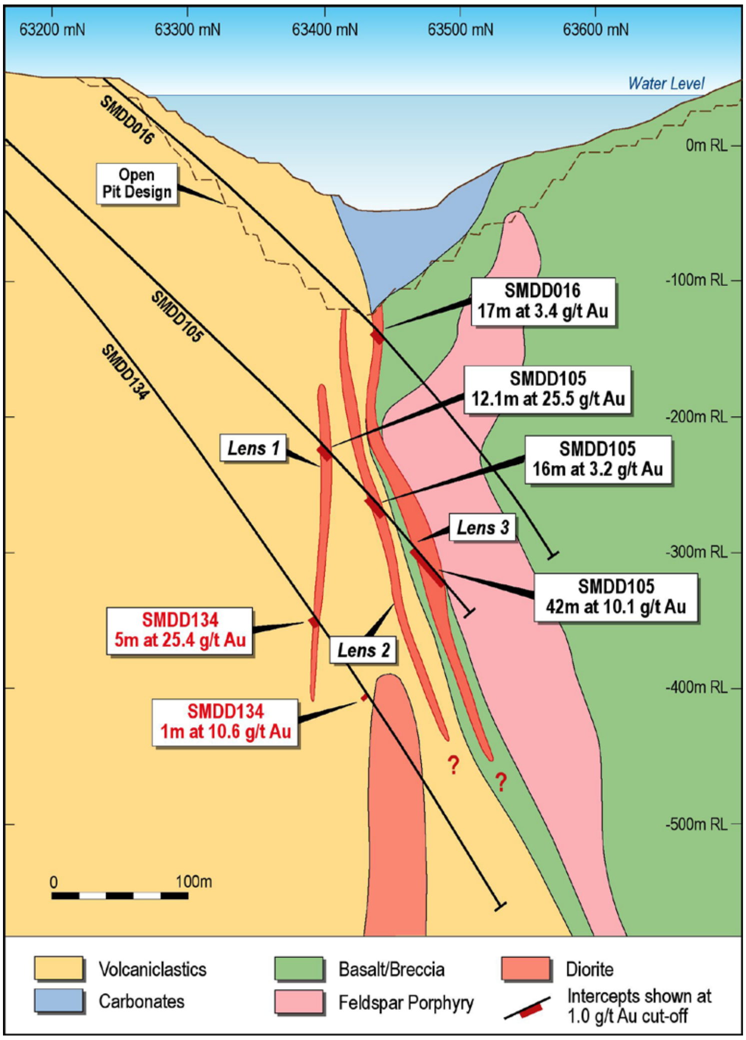

Figure 2. Siana cross section 55200N - SMDD134

Estimation methodology

Down-hole samples (generally one metre) were uniquely coded to respective host panels after which intercept composite grades were calculated. Based on statistical analyses of the composited interval grades Cube applied upper grade cuts of 30g/t Au and 60 g/t Ag to the major tonnage Panels 1 to 3. No upper grade cuts were necessary for Panels 4 to 10.

An average bulk density was determined for each Panel with an overall mean of 2.6 (t/m3) based on a 1,272 sample subset of the underground bulk density database of almost 14,600 determinations.

Cube adopted a 2D longitudinal modelling approach based on metal accumulation variables incorporating panel horizontal width and intercept grade, prior to conversion to a 3D model.

Cube's 2D metal accumulation estimation technique is based on two variables - intercept grade and thickness. The accumulation variable (grade x horizontal thickness) and horizontal width were independently interpolated into blocks using Ordinary Kriging. Final block grades were back calculated by dividing the kriged accumulation by the kriged horizontal width.

All block grade estimates were based on interpolation initially into 20m x 20m parent cells in the longitudinal plane. Data spacing was the primary consideration taken into account in selection of an appropriate estimation block size. For the main Panels 1 to 3, the average drill spacing is approximately 40m.

The 2D block centroids were converted and imported into a 3D block model suitable for formulating a reportable resource and reserves.

Cube reported global resources for each of the panels with no lower cut-off grade but within the constraints of the nominal +2g/t gold model wireframes.

The model included estimates for the minor elements Ag, Cu, Pb, Zn, As and Sb.

Depletion

Previous underground mine workings summarised on historic mine plans were utilised to create 3D models of regions known from drilling to contain underground mine workings, impacting volume estimates in a portion of Panels 2 and 3.

Low-Grade Stockpile Resource

Mill feed over the life of mine (LOM) will include two low grade stockpiles remaining from previous operations that total 83,000 tonnes at 1.33 g/t Au and 10.7 g/t Ag. These stockpiles have been the subject of a detailed sampling programme to arrive at the reported estimate of grade and tonnage and are classified in the Indicated category.

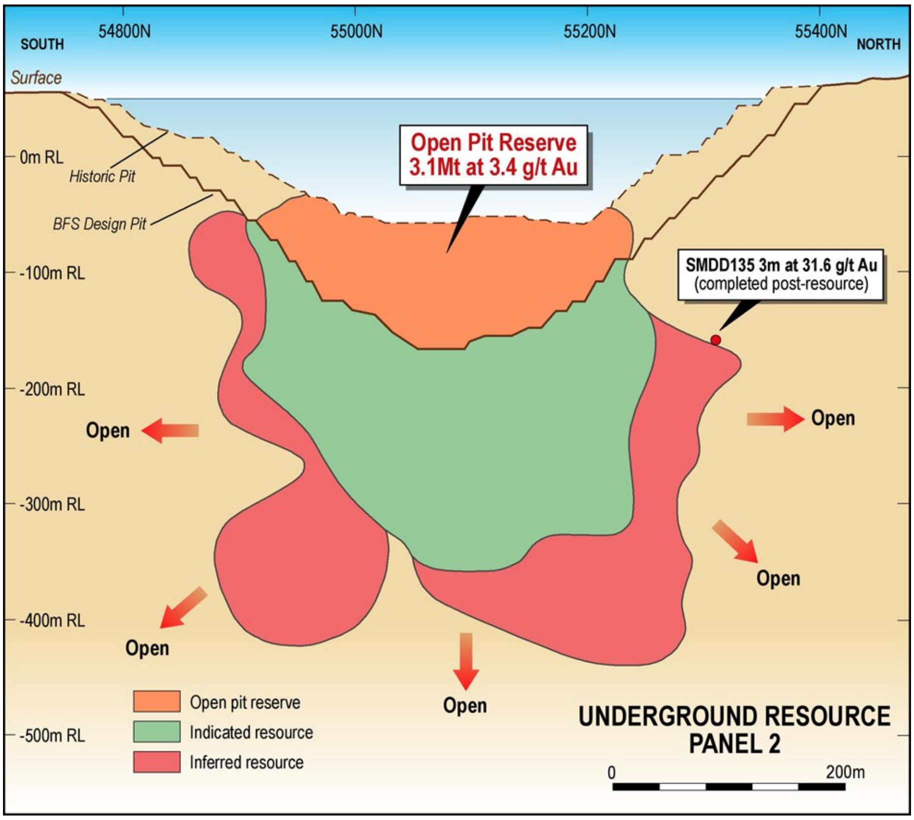

Figure 3. Indicated and Inferred Resources - Panel 2 (longitudinal projection)

SIANA ORE RESERVES

Probable Ore Reserves derived from the economically mineable part of the open pit, underground and surface stockpile Indicated Resources are summarised in Table 4.

In addition, a long term mine planning inventory has been estimated that includes inferred mineralisation adjacent to the underground Indicated Resource with potential for future upgrade and conversion of portion of the underground Inferred Resource of 1.4 million tonnes at 7.6 g/t gold (350,000 ounces).

The Mining Reserve extends to approximately 400 metres below surface (-360m RL). The Mineral Resource remains open to the north, south and at depth below 500 metres.

The relevant Modifying Factors and the methodology used to analyse the feasibility of the extraction of this mineralisation are described below.

Open Pit Ore Reserve

Apart from small increases in contained gold and silver that are attributed to minor modifications to the pit design there is no material difference from the open pit Probable Ore Reserve reported in April 2007 (3.07 million tonnes at 3.4 g/t gold (336,200 ounces) and 8.5 g/t silver (839,900 ounces).

Pit design modifications since 2007 included widening of the main pit ramp and reduction of bench heights from six metres to five metres. The waste to ore ratio by volume increased marginally to 7.0:1 (from 6.4:1) and total material movement increased to 25.7 million tonnes (from 22.8 million tonnes).

Within the open-pit design there are additional Inferred Resources amounting to 156,000 tonnes at 2.9 g/t Au and 13.6 g/t Ag containing 14,500 ounces Au and 68,200 ounces Ag that under JORC Code guidelines cannot be included in the Ore Reserve and are summarily classified as waste.

High grade ore is treated as it is mined, and hence bears the full cost of production. However, there is also low grade mineralisation which does not bear the full cost of production and will be stockpiled for treatment at the end of the mine life. Within the mine design the high and low grade cut-off grades are 1.25g/t Au and 1.1g/t Au respectively. Their combined tonnage constitutes the Open Pit Mining Reserve stated in Table 4.

Open Pit Optimisation and Mine Design

Optimisation

The H&S Mineral Resource model was used as the basis of a mining evaluation study by RSG Global Pty Ltd (RSG Global). A Whittle 4D pit optimization was prepared using a US$650 per ounce gold and US$10 per ounce silver price and a processing throughput of 750,000 tonnes per annum.

Pit wall slope angles were based on recommended parameters from the BFS geotechnical study. Metallurgical recoveries for gold and silver were determined on a constant tails grade basis established from the metallurgical test work (described below).

Mine Design

The RSG Global open-pit mine design is based on recommendations from the bankable feasibility study geotechnical study. It is proposed to exploit the deposit using conventional open-pit mining techniques. Benches will be blasted in 5m lifts but excavated at 2.5m intervals using a hydraulic excavator loading 6WD articulated trucks.

These units allow a pit ramp slope of 1:8 to be incorporated in the design which extends to a maximum depth of 215 metres below surface. The upper section of the ramp is located wholly on the west side of the pit due to superior geotechnical conditions and exits in the north-western corner to minimise haul distances to the ROM (run of mine) stockpile and waste dump.

The main waste dump will be located north and east of the pit on the site of an existing dump.

A smaller temporary dump west of the pit may be used during construction of the tailings dam embankments. The dumps are designed with 20 metre lifts, 10 metre wide berms and 30° batter angles with a 1:10 haul ramp.

The waste dumps and site layout have been specifically planned to minimize the environmental impact and do not substantially increase the existing waste dump footprint. The mine plan includes progressive rehabilitation with stockpiled topsoil to reduce the requirement for major works at mine closure. Particular attention has been paid to site drainage and strict control of surface run-off.

Pit De-watering

The existing flooded pit is estimated to contain 8.2 gigalitres (GL) of water. Data recorded during the previous mining operations and from three monitoring boreholes provided the basis for the estimate of expected water flows into the pit area. Groundwater inflows and rainfall will total an estimated 6.8 GL/yr.

The pit will be dewatered in two stages using in-pit pumps and external bores. Stage one is to progressively dewater the pit ahead of the pre-production waste cut-back using pontoon mounted electric drive pumps, with power from the main grid. Bores abstracting 3.2GL/yr will be located at the southern end of the pit to dewater the southern limestone which historically was a major source of groundwater inflow. Smaller bores will also be installed to depressurise and stabilise the eastern pit wall. Stage two comprises ongoing pumping from the bores and in-pit sumps.

Mine Costing Analysis

The capital and operating cost assumes contract mining. The fleet includes conventional diesel powered mining and ancillary support equipment. Heavy equipment prices were quoted by an in-country supplier.

In addition, it will be necessary to augment the fleet during the pre-strip period with an additional excavator and trucks on a lease basis.

The operating conditions at site are subject to heavy rain and potential flooding. The selection of 6WD articulated trucks will match the operating environment and allows the pit ramp gradient to be steepened. Hydraulic excavators will be used for truck loading and the types and sizes have been specifically matched to the haulage trucks.

Blasting patterns and powder factors have been adjusted to allow for variation of rock type hardness and operating conditions. Based on the previous mining history the presently designed provision for blasting might be reduced once mining commences and more geotechnical information is available.

The diesel price has been estimated from the Singapore Pricing model and the discounted Philippine Wholesale Posted Price (WPP) based on the quantities involved during the life of the open pit mine.

Underground Ore Reserve

The February 2009 underground Mineral Resource was used as the basis for an underground mine design, scheduling and costing analysis by Red Rock Engineering (RRE), to derive both a Probable Reserve and a target mine extraction plan incorporating a proportion of the adjacent Inferred Resource. For clarity, the latter does not relate to the Ore Reserve statement in Table 4.

A range of six production scenarios was considered, ranging from 250,000 to 500,000 tonnes per annum, using either conventional jumbo drill and blast or road headers for waste and ore breakage. All production scenarios indicated the exploitation of the underground Resource as viable.

The preferred option utilizes road headers with targeted annual production of 300,000 tonnes from the Probable Reserve of 1.94 million tonnes at 5.8 g/t Au and 9.1 g/t Ag (Table 4).

The underground Probable Reserve accounts for approximately 84% of the contained gold within the underground Indicated Resource.

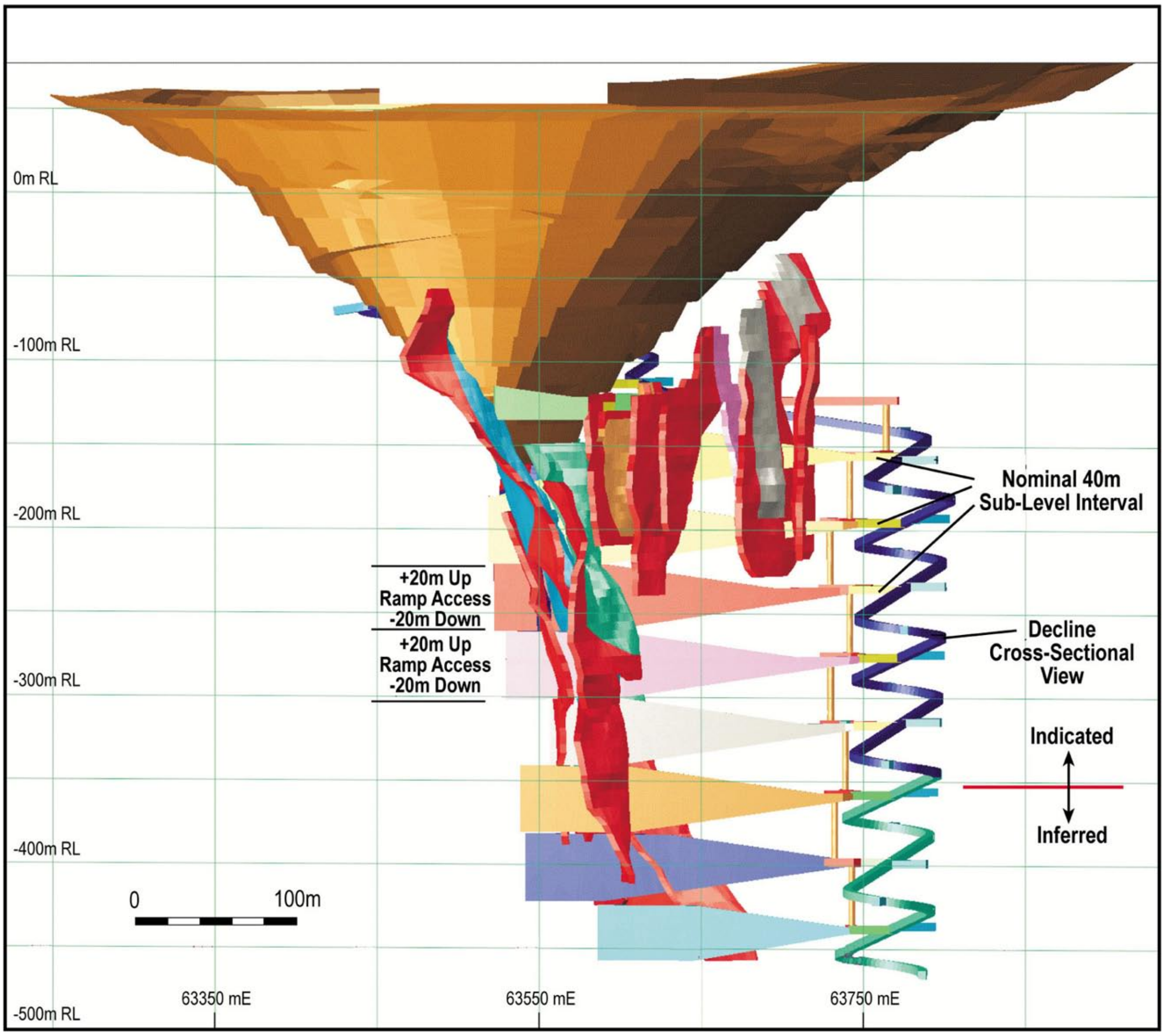

Figure 4. Siana open pit and underground mine design (view NNW)

Table 4. Probable Ore Reserve Statement

	Stockpile	Open Pit	Underground*	Total^
Tonnes	83,000	3,109,000	1,938,000	**5,130,000**
Grade g/t Au	1.33	3.42	5.82	**4.3**
Grade g/t Ag	10.7	8.7	9.1	**8.9**
Ounces Au	3,500	341,400	362,800	**708,000**
Ounces Ag	28,500	870,000	566,000	**1,465,000**

** 300ktpa road header option*

^ contains minor rounding errors

Underground Mine Design

Cut-off grade, stope design and dilution

A lower cut-off grade of 3.0 g/t gold was applied to the Mineral Resource block model to define stoping limits.

Stope designs are based on a minimum mining width of 4m allowing twin boom jumbos or smaller road headers to be utilised. Designs were applied to the ten Resource panels defined in the Mineral Resource model, and attributed for Indicated or Inferred status.

Where the Mineral Resource panels were less than 4 metres wide the dilution out to the stope margin was assigned zero gold grade. In reality this material is mineralised and therefore the approach used is conservative.

Detailed geotechnical investigations by Peter O'Bryan and Associates identified a large proportion of the underground rock mass as having low strength. In order to take advantage of this low strength RRE evaluated the economics of using tunnel boring road headers for rock breakage compared to conventional drill and blast methods.

In the final analysis the road header option is preferred on the basis of efficiency, cost and diminished impact on the stability of the wall rocks.

Critically, the geotechnical assessment also influenced the selection of an underhand cut and fill method (i.e. top down mining) utilising small open spans during the development cycle and cemented paste fill in voids created during the production cycle in order to support the rock mass.

The cut and fill method offers the opportunity of minimising the unsupported span widths and maximising ore recovery.

Disposing of most of the mill tailings in the underground paste fill has an environmental benefit in that it reduces the size of the surface tailings dam required.

Initial access would be via a 5m x 5m decline at 1:7 gradient commencing from a portal off the new open pit approximately 125 metres below surface.

The decline is scheduled to commence during the open pit mining phase in time to allow establishment of sufficient underground development for approx. two years of overlapping ore production prior to exhaustion of the open pit Ore Reserve.

Harder material in the decline development will be mined utilising jumbo drill and blast while remaining development and ore production will utilise a road header.

Cross cuts from the decline provide ramps up and down to levels spaced at 40m intervals, and access for detailed underground diamond drilling.

Hand held stoping methods may be more applicable in narrow sections of the mineralisation (less than 4 metres) and would add to the current mine plan inventory.

Long term mine plan

It is likely that with additional detailed underground drilling a portion of the current underground Inferred Resource would be upgraded to Indicated Resource status and ultimately, conversion to a reserve category.

To estimate the potential for eventual extraction similar modifying factors to those used in the current underground Ore Reserve estimate were applied to the known Inferred mineralisaton. A mine design using road headers and an annual production rate of 400,000 tonnes would yield up to an additional 1.4 million tonnes at 5.8 g/t gold (267,000 ounces) and 9.2 g/t Ag (420,000 ounces) from underground.

If this plan was to eventuate the total mine extraction including surface stockpiles and the open pit mineralization is estimated as 6.5 million tonnes at 4.6 g/t gold (975,000 ounces) and 8.9 g/t Ag (1.9 million ounces), or approximately 87% of the current total Mineral Resource of 1.12 million ounces gold (Table 5).

For the avoidance of doubt, this target mine extraction plan does not constitute an Ore Reserve according to the JORC guidelines but is estimated for long term mine planning purposes.

Table 5. Target Mine Extraction Plan

	Stockpile	Open Pit	Underground*	Total^
Tonnes	83,000	3,109,000	3,362,000	**6,554,000**
Grade g/t Au	1.33	3.42	5.83	**4.6**
Grade g/t Ag	10.7	8.7	9.1	**8.9**
Ounces Au	3,500	341,400	630,000	**975,000**
Ounces Ag	28,500	870,000	986,000	**1,885,000**

** 400ktpa road header option*
^ contains minor rounding errors

Metallurgical Studies

Metallurgical testing has been completed by Independent Metallurgical Laboratories (now Amdel), Outukumpu Technology, Orway Mineral Consultants and AMMTEC, in conjunction with process engineering group Intermet Engineering.

Comprehensive metallurgical testing of the open pit ores has been completed encompassing comminution, gravity concentration, flotation, cyanide leaching, carbon kinetics, thickening and slurry viscosity measurements, and cyanide detoxification testing.

Testing of underground ores included verification of gravity and leaching characteristics, tailings diagnostics, recovery variability and zinc sulphide flotation optimisation.

Mineralogical analysis and core logging indicated that the major mineral present is pyrite as euhedral crystals ranging from very fine to greater than 2mm in size. Sphalerite, galena and to a lesser extent chalcopyrite are also present, either as discrete aggregates, or as inclusions within the pyrite.

Native carbon is present in the limestone / carbonate ores from the central Panels 1-3.

Visible gold occurrence within drill core was found to be low. Gold down to several microns grainsize is present in metallurgical concentrates.

The open pit feasibility study established that a standard gravity and CIL process flowsheet would yield average metal recoveries of 85.4% Au and 76.0% Ag with constant tails grades of 0.49 g/t Au and 2.1 g/t Ag from ore milled to (P80) 75µm.

Diagnostic tests to identify the gold association in the open pit tails indicated that 69% to 79% occurs with sulphides. Sodium cyanide consumption averaged 1.2 kg/t of open pit ore milled.

Testing of representative underground mineralisation using the same process confirmed gold recoveries ranging from 87.5% (upper level 'low zinc' domain) to 84.1% (lower level 'high zinc' domain). Silver recoveries ranged from 52.7% (upper level 'low zinc' domain) to 61.6% (lower level 'high zinc' domain).

Based on the testwork, constant tails grades of 0.7 g/t Au and 2.1 g/t Ag were adopted for the underground. Diagnostic tests on the underground tailings indicated 52% to 64% of the gold occurs with sulphides. Sodium cyanide consumption averaged 2.2 kg/t for low zinc underground ore and 2.4 kg/t for the high zinc ore.

The deeper level underground ore is defined as a 'high zinc' domain with frequent grades in excess of 2% zinc accompanying gold mineralization.

Flotation test work showed that a saleable zinc concentrate can be produced from 'high zinc' gold ore at high recovery. Typical flotation performance produced zinc concentrate grades of 45% to 50% with recoveries in the range 93% to 96%.

The zinc concentrate can be produced with no significant reduction in gold recovery from the high zinc ore. After the gravity separation and zinc flotation, leaching of the concentrate and tailings yielded 83.4 % Au recovery with sodium cyanide consumption of 2.45 kg/t.

With current low zinc metal prices there is no initial plan to produce zinc concentrate from the gold processing plant. However a preliminary flotation circuit design has been included within the overall plant design to facilitate later addition should zinc prices return to viable levels.

Process Flowsheet

The standard design plant comprises single stage crushing, SAG milling, gravity concentration and high intensity cyanidation, leaching and adsorption (CIL), followed by carbon elution and electrowinning to produce combined gold and silver doré. The tailings from the cyanide leach area will be treated in a detoxification circuit to minimise cyanide concentration prior to discharge to the tailings storage facility.

Production Schedule

Life of mine mill throughput, head grade and recovered ounces are summarised in Table 6.

Infrastructure Design

Many of the infrastructure requirements of the site have been designed to definitive level and costed, including:

- tailings storage facility
- a 50 man permanent camp
- administration building and permanent medical facilities on site
- relocation housing
- construction of a 1.0 km access road and causeway from the National Highway
- power distribution from the National 138KV line 1.2km from the site
- site drainage and run-off mitigation
- potable water and site waste disposal facilities

Environmental approval

The Environmental Impact Statement (EIS) prepared by BMP Environment & Community Care, Inc., Philippines included detailed baseline studies on both the natural and socio-economic environment, prepared to Philippine and Equator Principles standards. The report identified and ranked potential impacts, and presented key elements of the Environmental Management Plan.

Following a Public Open Forum in December 2008 the EIS was approved by the Environmental Management Bureau and an Environmental Compliance Certificate ensued, issued by the Department of Environment and Natural Resources.

Table 6. Summary production schedule*

	Yr1	Yr2	Yr3	Yr4	Yr5	Yr6	Yr7	Yr8	Yr9	Yr10	Total
Ore Milled (tonnes)	594,000	750,000	921,000	1,092,000	887,000	558,000	516,000	507,000	481,000	250,000	**6,556,000**
Grade (g/t) - Gold	2.88	3.48	3.56	4.21	4.42	5.86	6.15	6.09	6.27	6.62	**4.63**
Grade (g/t) - Silver	15.0	10.1	6.2	5.9	6.0	8.7	10.0	11.9	12.2	11.1	**9.0**
Plant Recovery - Gold (%)	83.0	85.9	85.7	86.1	86.2	88.3	88.6	88.5	88.8	89.4	**87.1**
Gold Produced (ounces)	45,700	72,200	90,300	127,100	108,700	92,800	90,500	87,900	86,200	47,500	**849,000**
Silver Produced (ounces)	245,800	191,900	120,000	134,600	111,900	117,600	131,600	159,900	156,800	72,500	**1,443,000**

** contains minor rounding errors*

Additional statutory documentation is in preparation including; SDMP (Social Development Management Programme), EPEP (Environmental Protection and Enhancement Programme), FMRDP (Final Mine Rehabilitation and Decommissioning Plan) and Three Year Mine Plan, to be submitted in July 2009.

In terms of the natural environment there are no major risks identified and the planning for progressive rehabilitation and eventual closure will ensure that this former mine-site can be fully utilised to benefit future generations from the local communities.

Since project inception in 2002 the Joint Venture partners have maintained close communication with the host communities and implemented a number of social development projects well in advance of mine development. The aim is to expand on these initiatives via operational funding through the SDMP.

The host communities remain supportive and enthusiastic about the proposed development and the economic benefits that would flow to the region.

Capital Cost Estimate

The capital cost estimates for the open pit and underground phases of the operation are based on written quotes for equipment, labour, services and consumables provided to the various consultants by reputable suppliers in the Philippines, Australia and elsewhere in the Asia-Pacific region. These estimates have been consolidated and variable contingencies (5-20%) added to the total capital budget.

Table 7. Capital to first gold production

Capital Item	US$m*
Mining Open Cut	
Dewatering	2.5
Equipment & pre-strip	17.3
Total Mining Open Cut	**19.8**
Process Plant	**35.0**
Infrastructure	**4.3**
Misc Capital	**3.4**
Total	**62.5**

** including contingency*

The total direct and indirect capital cost from the beginning of detailed design to first gold pour has been estimated at US$62.5 million including $US6.0 million in contingencies, plus US$10 million in working capital until the project attains sustainable positive cash flow six months after first gold production. Hence the financing requirement is approximately US$72.5 million excluding any provision for cost over-runs which exceed the contingency.

Capital costs including sustaining capital for the life of the project total US$104 million (including escalation in underground development costs and US$10 million contingencies) and assuming owner purchase of underground plant and equipment from the positive cash flow of the operation.

A provision for the salvage value of the plant, mill and heavy equipment is not included.

The capital expenditure estimate is based on supply of new equipment, as normally required in Bankable Feasibility Studies, with the exception of the SAG mill that is already purchased and to be refurbished. Opportunities exist to reduce the capital costs via prudent purchase of selected used equipment and heavy earth moving machinery.

Operating Cost Estimate

Operating cost estimates are based on written quotes for labour, services and consumables provided by reputable suppliers in the Philippines, Australia and elsewhere in the Asia-Pacific region.

Power and fuel costs represent the largest components of the project operating costs being 12% and 7% respectively. The fuel price is the six month average of the discounted Wholesale Posted Price in the Philippines as supplied by several of the larger fuel companies based in-country. The power cost is based on a detailed written proposal from the local power cooperative that also includes the construction of appropriate power infrastructure.

Operating costs for the Base Case (long term mine plan - 400,000 tonnes per annum underground) are summarised in Table 8. The government excise tax is 2% of the value of the gold produced.

Table 8. Summary Operating Costs

	LOM
Mining Operating	28.34*#
Processing	12.64
Administration	4.03
Excise Tax**	2.53
Total Cost (US$/t)	**47.54**
Total Cash Cost (US$/oz^)	**351**

Notes:

* *Open pit mine operating costs $16.41 per tonne, underground mine operating costs $39.94 per tonne, includes escalation on underground mining capital cost at 2% per annum from project commencement*

\# *Base Case long term mine plan at 400,000 tonnes per annum underground rate*

** *Includes 1% community tax on processing/admin costs*

^ *Cash costs per ounce based on Au produced net of Ag by-product credits*

Cash costs are estimated and presented using the "Gold Institute Production Cost Standard" and are applied consistently for all periods presented.

Cash costs exclude depreciation, depletion and amortization, corporate general and administrative expense, exploration, interest, and feasibility costs.

Financial Analysis

The ungeared financial model derived from the Siana BFS indicates that at US$800 per ounce gold and US$10 per ounce silver the Base Case pre-tax IRR is 38% and the NPV US$145.5 million at an 8% discount rate. Pre-tax financials for a range of gold prices are shown in Table 9.

Table 9. Pre-tax Financial Summary

Gold Price ($US)	600	700	800	900	1000
NPV US$M (8%)	39.2	92.4	145.5	198.6	251.7
IRR (%)	17	28	38	47	55

Note: Silver price assumed as 1/80th of gold price

The Company will seek certain tax benefits applicable to mineral development in the Philippines. If a four year tax break was approved the Base Case post tax IRR would be 36% and the NPV US$120.3 million.

Implementation Schedule

The Project Management team will call for tenders from several experienced and reputable ngineering companies before appointing an Engineering, Procurement and Construction Management (EPCM) Contractor to be responsible for the detailed design and construction of the treatment plant, associated infrastructure and services.

The Project schedule is based on a fast track approach with design/engineering, fabrication/ equipment supply and construction all overlapping to create as short a time frame for the Project development as possible, expected to take approximately 17 months from the award of the EPCM Contract to the first gold production.

The EPCM contract is likely to include an early design and engineering phase that will be funded from current equity capital ahead of senior financing and commencement of Procurement and Construction.

Commencement of earliest infrastructure (road access) is scheduled in the September Quarter 2009, and dependent on financing, plant construction is targeted for January 2010 with first gold production twelve months later.

DATA COLLECTION AND SAMPLING PROCEDURES

Introduction

The Joint Venture continued to generate drilling data following the Bankable Feasibility Study reporting date (April 2007). Drilling, sampling, and quality control procedures applied throughout the PFS and BFS were maintained during the additional programme.

Origin and Validation of Historic Data

All available data from historic surface drilling, underground sampling, open pit mapping, open pit grade control sampling, survey pickup of dumps, tailings ponds and infrastructure have been captured from hardcopy drill logs, level plans, surface plans, cross sections and long sections, technical reports, files and Suricon annual reports. Most data were converted into digital form by Snowden Mining Industry Consultants (Snowden) or the Joint Venture partners (Red 5/Merrill Crowe Corporation).

Survey Control

The accuracy of drillhole collar data and other accuracy dependent data collected on site using a survey grade Sokkia GSR2650 differential GPS instrument is computed to be +/-0.25 metres.

Site Topographic Model

A digital terrain model (DTM) for use in mine planning and resource estimation was constructed from 3D point data derived from three sources:

- ground survey measurements recorded by Joint Venture personnel (32,940 points)
- pit and waste dump surveys from historic site plans (2,377 points)
- a digital terrain model constructed from stereo-pair Ikonos satellite imagery (sub-sampled at 50mx50m, 2,247 points).

The DTM was constructed using the local mine grid coordinate system.

The Joint Venture ground survey data were collected between November 2004 and March 2005. Surveys were collected at nominal 5m x 5m and 10m x 10m spacing, referenced daily to a local base station. Data were recorded in UTM zone 51N projection, using WGS84 as the horizontal and vertical datum, and converted to the local mine grid.

Orientation and Spacing of Drilling

The mineralisation at Siana occurs over broad widths (up to 80m in the central carbonate zone) but the deposit envelope is orientated approximately north-south. The drilling grid was orientated at 090 °– 270 ° (magnetic), a less than one degree variance from the original Siana Mine Grid. The majority of the resource holes were drilled toward magnetic east or west at moderate to shallow angles, with several notable exceptions drilled off grid for specific access related reasons, or were dedicated geotechnical holes.

The drill section spacing is at nominal 20 metre intervals along the strike of the deposit.

Drill Hole Planning and Collar Surveys

Consideration was made of the collar locations with respect to existing access and finally designed where possible to intersect both the main zone carbonate and eastern zone basalt mineralisation. Allowance was made for an increase in dip with depth. The vast majority of holes were designed to intersect the mineralised target with PQ3 or HQ3 diamond core.

Drill holes were sited and surveyed after completion using a Sokkia GSR2650 DGPS unit with horizontal and vertical accuracy of approx. 0.25 metres.

Drilling Techniques

Joint Venture diamond drilling was undertaken using United Philippines Drilling (UPD) sled portable CS1000 6PL diamond drill rigs. These rigs are capable of drilling depths of ~350m, ~600m and ~1,000m of PQ3, HQ3 and NQ3 diamond core respectively.

During the drilling operations, a geological aide was present at the rig at all times (rigs ran 24 hours per day continuously) specifically to record drilling progress, core recovery and down hole surveys.

Early holes were pre-collared to a depth of between 30 and 100 metres using tricone roller bit/ mud rotary drilling and cased off with PW casing before PQ3 diamond drilling. In the latest programme PQ3 coring commenced immediately after setting the collars.

Down hole surveying

Drill holes were down hole surveyed using a Reflex single shot electronic survey tool supplied by UPD, on a nominal 30m basis. The survey tool was checked on surface for accuracy on a periodic basis. Where results from the survey tool were considered substandard, the particular portion of the hole was resurveyed where possible.

Core Orientation

Up to and including drill hole SMDD055 all core orientation used a crayon spear method of marking the bottom of the core. Since SMDD056 orientation of drill core used a commercial core orientation system (Ballmark, or a Reflex digital instrument).

Core recovery

Core recovery was measured at the drill site. Markers were placed in trays where core was lost, or where the hole passed through minor voids due to previous mining.

Bulk density

Bulk density determinations were carried out routinely at site. All mineralised zones were measured as well as the footwall and hanging wall waste material. Samples of core were taken from each metre sample interval, weighed and the SG determined using the "Archimedes Principle" water displacement method.

A total of approximately 14,600 SG determinations were included in the latest estimation model (mineralisation and waste) with a subset of 1,272 determinations used for the mineralised panels.

Geotechnical Logging

Geotechnical logging of diamond core was overseen by Mining One Pty Ltd for the open pit drilling, and Peter O'Bryan and Associates for the latest phase of underground resource drilling. Holes were systematically logged, including routine RQD measurements, and a number of other parameters from oriented sections of core including Q, Q', RMR and MRMR.

Geological Logging

Core was logged by senior Filipino geologists and coded data were entered into a standard format spreadsheet, using two data entry clerks. Key fields are lithology, alteration and mineralisation; minor fields include colour, texture, structure, weathering and comments.

All diamond drill core was systematically photographed at high resolution before sampling.

Sampling

Altered and mineralised sections of the holes were sampled on a one-metre basis after splitting with a circular diamond-tungsten saw. PQ3 (83mm) diameter core was sampled by taking approximately one-quarter fillet, and HQ3 diameter core (54mm) was sampled by taking a one-third fillet for analysis. NQ3 diameter (46mm, rarely drilled), was split into equal halves. Further splits were later taken from selected holes for metallurgical purposes – these were taken from a central slab of core.

Soft sections of core, particularly in the mineralised zones, were wrapped in tape before cutting to effectively maintain sample competence. In a later phase of cutting for metallurgical sampling all the mineralised zone was wrapped with tape.

Transport and Security

Samples were stored in a locked and patrolled storage pen on site, prior to transport to Manila by ferry. Each transported batch was accompanied by a Joint Venture staff member until delivery and handover at the laboratory.

Audits and reviews

A detailed inspection of the laboratory facilities and procedures was conducted by the Management of the Joint Venture prior to commencement of resource drilling in February 2003. Spot inspections were later made to review lab cleanliness and procedures during processing of Siana core samples.

On each occasion the laboratory was observed to have maintained very high standards in the sample preparation area, fire assay facility and wet chemical section, and to follow accepted procedures in sample preparation and analysis.

Independent inspection and review of the site data collection, sampling methods and QA/QC procedures, and the McPhar laboratory sample preparation facilities and analytical techniques was undertaken and reported by Snowden Consultants in 2005 and found to be within standard industry practice. No changes to the procedures were made during subsequent programmes.

Data Verification

All Joint Venture drill hole planning, drill hole surveys, core recovery, specific gravity and magnetic susceptibility determinations, geological logging and geotechnical logging are first recorded on data entry forms and checked by the Geologist in Charge of the site.

These data are manually keyed to spreadsheets, checked and verified by the Geologist and transferred to Australia by email. Drill hole records were copied for site files and originals retained in Perth.

In Perth, data were checked by a senior database geologist prior to entry to a backup database and dispatch to ioDigital (a division of ioGlobal) for contracted database management and maintenance within acQuire software. IoDigital validated data and generated routine QA/QC reports on assay batches. IoDigital has provided this service for all drilling and sample data from the Siana Gold Project since inception.

QUALITY CONTROL

Accreditation

All routine samples have been processed at McPhar Geoservices (Phil.) Inc. located In Makati, Metro Manila. The laboratory is accredited with ISO 9001 certification, and is a regular participant in the Australian based Geostats Pty. Ltd. international laboratory quality monitoring scheme.

Umpire check analyses including fire assay (Au), AAS (multielements), sizing analysis, and screen fire assay (Au) were completed by Amdel Laboratory in Perth, (NATA registered for ISO/IEC 17025 and accredited for AS/NZS ISO 9001). Amdel is also a participant in the Geostats quality assurance survey.

The Joint Venture commissioned Geostats to report on the performance of both laboratories over the period April 2003 to April 2005. The regular surveys include distribution of sets of samples to over 120 laboratories worldwide. Elements of particular relevance include gold by fire assay, and silver, copper, lead, zinc and arsenic by AAS.

Over the surveys completed during the review period Geostats concluded that both laboratories performed very well for all elements (gold, silver, base metals and sulphur) and were capable of producing high quality results. Ninety percent of biases associated with both laboratories' results were within one standard deviation.

Gold Assay Method

Approximately 50g of sample pulp was used for fire assay gold analysis with AAS finish (Method PM-6, 0.005 ppm DL). Each charge of 30 crucibles contained 26 unknown samples, two replicates, one internal laboratory standard, and one blank.

Multielement Analytical Method

Routine analyses included silver (0.5ppm DL), copper (5ppm DL), lead (5ppm DL), zinc(5ppm DL) by AAS following concentrated HCl and HCl/HNO_3/ $HClO_4$ leach in latter stages on 1g sample, and arsenic/antimony (1ppm DL) by vapour generation/ AAS from the same acid leach. McPhar inserted two or three internal standards and one blank for every 100 samples.

The lab conducted 10% routine repeat analyses on a new 50g fusion (for gold), or new acid digest (for other elements) in addition to random repeat analyses.

Sizing Analysis

The quality of the McPhar sample preparation (nominal P 90-75 micron) was tested initially at Amdel by wet sizing analysis of bulk fines for random samples from 21 resource drill core batches. These data were supplemented by dry sizing results (-75 micron) from screen fire assay tests.

McPhar consistently achieved excellent sample pulverisation to nominal P90-75 micron, with rare cases falling within the 80-90% range. Results from the dry sizing tests are considered to be conservative, as adhering or agglomerated fines would inevitably report to the -75 micron fraction on wet screening.

Standards

Australian sourced gold standards (120g pulps, -75 micron, supplied by Gannet Holdings, Perth) were included in analytical batches from inception of drilling. At start-up, standards or blanks were inserted every 50 samples, but as the programme evolved the frequency of use was increased to 1:20 and additional gold standards were introduced to cover a wider grade range (0.4g/t to 6.0g/t).

McPhar

The same internal laboratory standards were used throughout the period of the drilling programme. Synthetic and Certified Reference Materials (CRM) were used in both the gold and base metal analytical procedures.

Blanks

At start-up, Joint Venture blank samples comprised screened local andesite aggregate which averaged ~0.02ppm Au.

A new commercial certified blank made from colour pigmented quartz sand was introduced for holes SMDD063 to133. Results for the commercial blank were consistently at or below the fire assay detection limit of 5ppb Au, confirming the excellent cleaning procedures used at the lab during the sample pulverisation process.

McPhar Precision and Accuracy

Excellent precision with minimal variance in accuracy is indicated for all standards used. Company policy is to repeat batches or partial batches where two (different) standards fall significantly outside a two standard deviation range – it has not been necessary to invoke the policy throughout the term of the resource drilling programme.

Multielement performance of the JV internal standards demonstrate consistent precision within 2SD tolerance limits. Performance of the McPhar internal gold and multielement standards indicated consistently high levels of accuracy and precision.

Resubmitted Replicates

Selected pulps (82) were repackaged, re-numbered and re-submitted for blind repeat analysis of gold and multielements. Scatter plots indicate good batch to batch precision for all elements, with only minor scatter at lower grade levels.

Umpire Check Assays

The accuracy of the McPhar analyses was checked at Amdel Laboratory in Perth on three occasions. Selected pulp samples (n=293) from resource diamond drilling with gold grades greater than 0.1 g/t were spatially representative of the Resource, and also the time interval over which the drilling was conducted. There is a high degree of correlation between the laboratories, with an insignificant positive bias in the McPhar results.

A fourth batch of umpire checks (109 samples) covering the 2007-2008 drilling was completed by UltraTrace Laboratory, Perth and reviewed by Cube Consulting. The check samples supported the original assays, and there were no material concerns with the accuracy and precision for Red 5 gold standards, blanks, or McPhar internal lab repeats.

Screen Fire Assay Tests

The occurrence and distribution of coarse gold was tested by re-submission of bulk fines samples for screen fire assay, representing a range of gold grade from 0.3g/t to 102g/t in both carbonate and basalt mineralisation from throughout the Resource. Samples from the area affected by previous mining were avoided. The tests were conducted at both McPhar and Amdel Laboratories. The results indicate that in general less than 20% of the gold is coarser than 75 micron, that there is a similar distribution of grade between the coarse and fine fractions, and that a high degree of confidence can be placed on the reliability of the routine 50g fire assays.

All the evidence from the testing indicates low sample variance in the deposit.

Duplicate Core Sampling

Field sampling precision was tested in a batch of 98 duplicate core splits selected from lithotypes unaffected by previous mining in holes SMDD061 to 085. The selection was made to represent a grade range above 0.3g/t Au, a range of rock types, and carbonate and basalt hosted mineralisation types from throughout the Resource to a depth of -200m elevation. Both PQ3 and HQ3 core sizes were represented. The duplicate split was taken from the opposite side of the core as the original split to emulate the original sample weight as closely as possible. The resulting central fillet was retained for reference.

Gold results indicated an acceptable level of precision between splits. The distribution of paired differences is similar for the PQ3 and HQ3 splits indicating no significant difference in the reliability of PQ3 splits compared with HQ3 splits.

COMPLIANCE

Competent Person Declarations

The information in this Public Report that relates to Exploration Results, Mineral Resources or Ore Reserves is based on, and accurately reflects, information compiled by Mr G C Edwards, Mr W Darcey and Mr A L Govey who are full-time employees of Red 5 Limited and who are Members of The Australasian Institute of Mining and Metallurgy.

Mr Edwards, Mr Govey and Mr Darcey have sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity which they are undertaking to qualify as a Competent Person as defined in the 2004 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (the JORC Code). Mr Govey and Mr Edwards consent to the inclusion in the report of the matters based on their information in the form and context in which it appears.

CORPORATE INFORMATION

Directors and Executive Management

Colin Jackson *(Chairman)*
Greg Edwards *(Managing Director)*
Lance Govey *(Executive Director - Tech)*
Peter Rowe *(Non-executive Director)*
Gary Scanlan *(Non-executive Director)*

Bill Darcey *(Project Manager)*
Frank Campagna *(Company Secretary)*
Lolot Manigsaca *(Philippines-based)*
Manny Ferrer *(Philippines-based)*
Attny E Panimogan *(Philippines-based)*

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Stock Exchange Listing

Australian Stock Exchange
Ticker Symbol: RED

Issued Capital

As at the date of this report,
issued capital – 659,288,043 shares
Unlisted options – 7,400,000

Substantial Shareholders

Mathews Capital Partners 19.0%
AngloGold Ashanti Australia 10.2%
Ross Stanley 8.0%

Shareholder Enquiries

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