
Shrinkage Mining of the West Reef Resource, Prestea Underground Mine, Ghana

Effective Date: December 18, 2014

Golden Star Resources Ltd.
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Qualified Persons
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Executive Summary (Item 1)

Golden Star Resources Ltd. ("GSR") is a Canadian Federally-incorporated, international gold mining and exploration company producing gold in Ghana, West Africa. GSR also conducts gold exploration in South America.

The Prestea Underground Mine is an inactive underground gold mine located 15km south of the Bogoso mine and adjacent to the town of Prestea. The mine consists of two usable access shafts and extensive underground workings and support facilities. Access to the mine site is via an unpaved road from Tarkwa.

The Prestea Underground Mine was mined from the 1870’s until 2002, when mining ceased following an extended period of low gold prices in the late 1990s and early 2000s. The Prestea mining area has produced approximately nine million ounces of gold, the second highest production of any mine in Ghana. The underground workings are extensive, reaching depths of approximately 1,450 meters and extending along a strike length of 9 kilometers. Underground workings can currently be accessed via two surface shafts, one near the town of Prestea (Central Shaft) and a second approximately four kilometers to the southwest at Bondaye.

Golden Star Bogoso Prestea Ltd (GSBPL) holds a 90% ownership in the Prestea Underground Mine with the Government of Ghana holding a 10% ownership interest in the Prestea Underground Mine as well as its 10% holding in GSBPL, resulting in an 81% beneficial ownership by Golden Star.

The West Reef mineral resource is developed on a steeply dipping, narrow vein structure between 17L and 24L, accessible through the Prestea Central Shaft.

PEA Highlights:

- Post-tax NPV5% of $121M with an IRR of 72% at $1,200/oz gold price;
- Initial capital $40M required to first production;
- $52M capital required to 50% production rate;
- $65M capital required to 100% production rate of 500 tpd;
- Payback period of 2.5 years from the start of project;
- The total capital cost of the project will be $94M over the life of the project including contingency, sustaining capital and mine closure;
- Total project life of six years, including one year of infrastructure rehabilitation;
- Life of Mine (LoM) cash operating costs of $370/oz;
- LoM all-in sustaining costs of $518/oz;
- LoM processing 649 kt of ROM at a diluted grade of 17.2 g/t and recovery of 90% for 323 koz sold;
- The resource will be exploited using underground shrinkage mining methods;
- Run-of-Mine (ROM) material will be transported by road about 15km to GSR’s operating Bogoso Processing Plant;
Geology

The Prestea-Bogoso mineralization occurs at the southern end of the Ashanti Belt, where eleven gold deposits, mined or under exploration, are localized principally along up to three steep to sub vertical major crustal structures. Rock assemblages from the southern area of the Ashanti belt were formed between a period spanning from 2,080 to 2,240 million years (“Ma”) with the Sefwi Group being the oldest rock package and the Tarkwa sediments being the youngest. The Ashanti belt is host to numerous gold occurrences, which are believed to be related to various stages of the Eoeburnean and Eburnean deformational events.

The geology of the Prestea Mine site is divided into four main litho-structural assemblages, which are fault bounded and steeply dipping to the west. This suggests that the contacts are structurally controlled and that the litho-structural assemblages are unconformable. These packages are from the eastern footwall to the western hanging wall, the Tarkwaian litho-structural assemblage, the tectonic breccia assemblage, the graphitic Birimian sedimentary assemblage and the undeformed Birimian sedimentary assemblage.

At Prestea, the principal structure is the mineralized quartz vein, known as the Main Reef which is relatively continuous and has been modelled and mined over a strike length of some 6 km and to a depth of approximately 1,450 m below surface (35 L). The subordinate West Reef and East Reef, in the immediate hangingwall and footwall respectively of the former structure, are discontinuous. West Reef occurs some 200 m into the hangingwall of the Main Reef structure and, at present is known to occur over a strike length of 800 m and has currently been defined by underground drilling between 550 to 1,150 m below topography as far as the 24 L.

Mineral Resources

The “reasonable prospects for eventual economic extraction” requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the Mineral Resources are reported at an appropriate cut-off grade, taking into account extraction scenarios and processing recoveries.

The Mineral Resource statement has been prepared using a block cut off grade of 4.74 g/t based on a US$ 1,400/oz gold price and appropriate costing data to produce a Mineral Resource which meets the requirements that the deposit should have “reasonable prospects for economic extraction” as defined by the CIM.

The statement was prepared by Mr. Mitch Wasel who is a Qualified Person pursuant to National Instrument 43-101. Mr Wasel is employed by GSR as Vice President Exploration and is not independent of the company. The effective date of the Mineral Resource Statement is 31 December 2013. The Mineral Resource Statement for Prestea Underground, is given below in Table ES 1.

In declaring the Mineral Resources for the PUG deposits, GSR notes the following:

- The identified Mineral Resources in the block model are classified according to the CIM definitions for Measured, Indicated and Inferred categories and are constrained by a block cut-off grade calculated using a gold price of US$ 1,400 /oz and below the end of year topographic surface. The Mineral Resources are reported in-situ without modifying factors applied;
- The Mineral Resource models have been depleted using appropriate topographic surveys and underground stope data, to reflect mining until the 31 December 2013;
• The Mineral Resources were estimated using block models. The composite grades were capped where this was deemed necessary, after statistical analysis. Ordinary Kriging was used to estimate the block grades. The search ellipsoids were orientated to reflect the general strike and dip of the modelled mineralization;

• Block model tonnage and grade estimates were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014). The basis of the Mineral Resource classification included confidence in the geological continuity of the mineralized structures, the quality and quantity of the exploration data supporting the estimates, and the geostatistical confidence in the tonnage and grade estimates. Three-dimensional solids were modelled reflecting areas with the highest confidence, which were classified as Indicated Mineral Resources;

• All figures are rounded to reflect the relative accuracy of the estimate. All composites have been capped where appropriate; and

• Mineral Resources are not Mineral Reserves and do not necessarily demonstrate economic viability.

### Table ES 1 Prestea Underground Mineral Resource Statement 31 December 2013

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<thead>
<tr>
<th>Orebody</th>
<th>MEASURED</th>
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<tr>
<td></td>
<td>Tonnes</td>
<td>Grade (g/t Au)</td>
<td>Content (koz Au)</td>
<td>Tonnes</td>
<td>Grade (g/t Au)</td>
<td>Content (koz Au)</td>
<td>Tonnes</td>
<td>Grade (g/t Au)</td>
<td>Content (koz Au)</td>
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<td>Grade (g/t Au)</td>
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<td>West Reef*</td>
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<td>Footwall</td>
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<td>363</td>
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<td>3,289</td>
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* The section 15 of this report, mining method, considers the West Reef resources only.

### Mineral Reserves

No Mineral Reserves have been estimated for the purposes of this PEA.

Recent Mineral Reserve estimates have been prepared by:


• GSR – Year-end reserve estimate effective December 31, 2013.

Both of these recent reserve estimates relate to mechanized mining of the West Reef resource and are superseded by this PEA.

### Underground Mining

There is an extensive infrastructure of vertical shafts, inclined shafts, horizontal development, raises and stoping developed along the 9km of strike length of the various orebodies from Prestea in the north to Tuapim in the south. Figure ES 1 shows a long section view of this development and indicates the extent of historical stipping operations on the Main Reef, West Reef and Footwall Reef throughout the area.
For the purposes of this PEA, only the West Reef resource between 17L and 70m below 24L has been evaluated for potential mineability.

The West Reef project consists of an underground mine with a production mine life of five years and a modified Bogoso processing plant. The maximum mill feed rate is set at 175,000 tonnes per annual (t/a), or a nominal 500 tonnes per day (t/d).

The proposed mining method is traditional shrinkage mining with the application of modern rock bolts and traditional wood stulls/props to support the stope walls in order to maintain stope stability and control waste dilution.

The initial stope design was based on a cut-off grade defined at a gold price of US$1,200 per ounce, a royalty of 5%, a process recovery of 90%, and a site operating cost obtained from GSR’s recent in-house concept study work. An in situ cut-off grade of 7.0 g/t gold was used to define mining shapes in the resource block model. Mining shapes were interrogated with the mine planning software and checked against a cut-off grade of 7.0 g/t gold that includes an allowance for an initial estimated total dilution of 20% at zero grade.

Shrinkage stopes were generally planned at a 60 m strike length and a height of 40 m for the upper levels (above 4235L) or a height of 35 m for the lower levels (below 4235L). There are some areas where this standard size has been varied including around the perimeter of the Indicated mineral resources.

Internal dilution on the total plant feed averages 1%. External dilution on the total plant feed averages 18%. A conservative assumption is made that both internal dilution and external dilution carries no grades.

The Prestea underground mine rehabilitation and preproduction period is defined as a 24-month period from January 2015 (start of underground rehabilitation which requires a period of 12 months) to December 31, 2016. This is dependent on GSR securing the necessary financing and approvals to develop the project. In H1 2017, an average production rate of 404 t/d is planned, which is more than 80% of the designed underground mine capacity. The full production period extends from January 1, 2017 to December 2020 for a period of four years. At full production, the planned mining rate is 500 t/d (175 kt per year).

During the first year of pre-production, the infrastructure of the mine will be improved and upgraded. The principal infrastructure upgrade projects are as follows:

- Central and Bondaye shaft rehabilitation;
- Central and Bondaye hoist upgrades;
- Electrical infrastructure;
- Compressed air;
- Pumping;
- Ventilation
- 17L and 24L track and ground support;
- 25L loading pocket;

**Processing**

The existing Bogoso processing plant consists of separate processing routes and equipment for oxide and refractory ores. The refractory processing route will be suspended by the time PUG is producing. This presents an opportunity to utilize some of the existing equipment and achieve an overall cost effective capital solution. The principal applied is to utilize certain selected equipment from the Bogoso refractory plant (which will be on care and maintenance), and supply and install new equipment where no suitable existing equipment can be sourced from site or refurbished.

The plant is designed to treat 500 tpd of non-refractory RoM material. The processes involved are crushing, milling, CIL, elution and electrowinning.

**Tailings**

Processing tails will be deposited on the currently operated TSF2 tailings storage facility and/or later approved facilities. TSF2 is a conventional tailings storage facility.

**Environmental and Community Considerations**

The Prestea community is supportive of the effort to reopen the underground operation; this support will pave the way for easier permitting with the Environmental Protection Agency, which considers community input to project development as a key factor.

Due to the limited footprint associated with the development, environmental considerations are deemed manageable and environmental approval for an operating permit should be granted within the normal timelines. The development of the project is not expected to contribute to the water already removed from the underground. Therefore, the existing environmental indemnity for the dewatering water is expected to remain in place.

**Capital Costs**

Capital costs for the project total $94M, including:

- $30M for infrastructure upgrade
- $7M for West Reef development
- $7M for mining equipment
- $24M for care and maintenance during development period
- $2M engineering and owner costs
- $2M for closure
• $22M sustaining capital

Initial capital requirements are as follows:

• Initial capital $40M required to first production;
• $52M capital required to 50% production rate;
• $65M capital required to 100% production rate of 500 tpd;

Cash drawdown for the project is $47M.

**Operating Costs**

PEA level operating costs were estimated as follows:

• Mining costs – zero based personnel, equipment and maintenance, supplies and power estimate, including 35% contingency;
• Haulage costs – based on current haulage contracts in place for the Bogoso Mine;
• Processing costs – based on GSR and MDM estimate for operating the plant as described in Section 16;
• G&A – zero based personnel, equipment and maintenance, supplies and power estimate.

Table ES 2 summarizes the operating costs incorporated in the economic analysis.

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<th>Item</th>
<th>Cost</th>
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<tr>
<td>Processing cost</td>
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<td>Haulage cost</td>
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<td>G&amp;A cost</td>
<td>27</td>
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**Indicative Economic Results**

Project net present value at a discount rate of 5% is US$121M with an internal rate of return of 72%. The payback period is 2.5 years from the time of initial investment. Cash operating cost is estimated to be US$369 per ounce produced.

The project is most sensitive to changes in gold price and grade, and least sensitive to capital and operating cost.

**Conclusion**

The Prestea Mine has a long history of underground stoping production and the West Reef area has good potential as a consistent, 500 tpd production area to supply high-grade material to the Bogoso Plant.

**Recommendations**

The findings of this PEA provide compelling arguments to move the study to the FS design stage. The following are the recommendations for advancing the project in the individual areas:
Resource

- Further drilling along strike and at depth could expand the current resource and upgrade the inferred material to indicated. This drilling will be conducted once project construction commences.

Underground Mining

- Advance the geotechnical evaluation of the shrinkage mining to a FS level
- Advance the mine design and scheduling to a FS level
- Undertake FS level capital and operating cost estimates

Process and Tailings

- Develop processing strategy considering processing through the Bogoso CIL Plant and the Wassa CIL Plant in addition to the option considered in this PEA

Environment and Infrastructure

- Initiate community consultation for the development of the underground operation and determine the methods for including appropriate community concerns within the project design and operation
- Develop the environmental and socioeconomic baselines for the operation including a community health assessment
- Complete the environmental permitting process with the appropriate involvement of stakeholders and regulators such that appropriate stakeholder concerns are addressed in the design and environmental management plan for the project.
# Abbreviations

<table>
<thead>
<tr>
<th>Abbreviation</th>
<th>Description</th>
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<td>AAS</td>
<td>Atomic absorption spectroscopy</td>
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Units

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<td>g/t</td>
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Introduction (Item 2)

Golden Star Resources

Golden Star Resources Ltd. (“GSR”) is a Canadian Federally-incorporated, international gold mining and exploration company producing gold in Ghana, West Africa. GSR also conducts gold exploration in South America.

The Prestea Underground Mine (PUG) is currently on care and maintenance (“C&M”) having ceased operation in 2002. GSR holds a 90% interest in the subsidiary company / operating entity in Ghana, known as Golden Star Bogoso Prestea Ltd (“GSBPL”).

The Bogoso/Prestea mining complex consists of several open pit and underground operations along 30 km of the Ashanti Trend. PUG is located about 15 km south of the Bogoso mine and adjacent to the town of Prestea. The property consists of two currently operational access shafts and extensive underground workings and support facilities. Access to the mine site is via an unpaved road from Tarkwa.

GSBPL is planning to process the Prestea West Reef Project material in the Bogoso Processing plant situated at the Bogoso operation some 15 km away from Prestea, close to the Chujah open pit operation.

Bogoso is located on the main road from Tarkwa to Kumasi and there is a tarred road between Bogoso and Prestea. The property has paved/unpaved road access to Accra (6 hours), Tarkwa (1 hour) and the major port at Takoradi (3 hours). There are airports at Kumasi (3.5 hours) and Takoradi, which provide daily services to the International Airport at Accra.

Terms of Reference and Purpose of the Report

This PEA is intended for the use of GSR to further the evaluation of the Prestea Underground Mine by estimating resources in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) classification system. A mining method and access development strategy is proposed, in addition to infrastructure and processing options. The capital and operating costs and production parameters have been used to generate indicative economics.


This PEA is based on Mineral Resources, not Mineral Reserves. Mineral Resources do not have demonstrated economic viability.

This PEA is preliminary in nature. It includes Inferred Mineral Resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Mineral Reserves. There is no certainty that the PEA will be realized. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The metric (SI System) units of measure are used in this report unless otherwise noted to describe quantities in this report. All monetary values shown in the report are US Dollars (“US$”) unless otherwise stated.
1.3 Sources of Information

In compiling this report, GSR has utilized data from the following sources:

- NI 43-101 compliant resource estimation reports prepared by SRK (UK) Ltd.;
- Geotechnical assessment prepared by SRK (UK) Ltd.;
- Mine design, scheduling and mining operating cost estimates prepared by SRK (Canada) Ltd.
- Metallurgical assessment report prepared by SGS Canada Inc.;
- Process design and cost estimates prepared by MDM Engineering (South Africa);
- Electrical design and cost estimate prepared by PPE Technologies (South Africa);
- Ventilation design prepared by MVS Ltd (USA);
- Various in-house and external geological reports and papers;
- Data archives at the Prestea Underground Mine site offices;

1.4 Qualified Persons

Dr. Martin Raffield is QP responsible for the Executive Summary and sections 1 through 4, 12 and 14 through 25 of this report. He is based in Bogoso, Ghana and employed by GSR as Senior Vice President, Project Development and Technical Services. Dr. Raffield last visited the site in November 2014. Dr Raffield has overall responsibility for the report.

S. Mitchel Wasel is the QP responsible for Sections 5, 6, 7, 8, 9, 10, 11 and 13 of this report. Mr. Wasel is based in Takoradi, Ghana and is employed by GSR as Vice President of Exploration. Mr Wasel last visited the site in November 2013.
2 Reliance on Other Experts (Item 3)

The preparation of this PEA has been undertaken by GSR staff, however there are disciplines where GSR was not the sole author or relied on specialists in a particular field. In these cases, GSR QP’s have reviewed and approved the work of other experts as follows:

- NI 43-101 compliant resource estimation reports prepared by SRK (UK) Ltd.;
- Geotechnical assessment prepared by SRK (UK) Ltd.;
- Mine design, scheduling and mining operating cost estimates prepared by SRK (Canada) Ltd.
- Metallurgical assessment report prepared by SGS Canada Inc.;
- Process design and cost estimates prepared by MDM Engineering (South Africa);
- Electrical design and cost estimate prepared by PPE Technologies (South Africa);
- Ventilation design prepared by MVS Ltd (USA);
- Tailings will be stored in existing storage facilities at the nearby Bogoso operation. The design of these facilities was undertaken by consultants from Knight Piésold Ltd who also undertake regular reviews;
3 Property Description and Location (Item 4)

3.1 Location and land Tenure

The Prestea concession is located in the Western Region of Ghana approximately 200 km from the capital Accra and 50 km from the coast of the Gulf of Guinea. Bogoso and Prestea comprise a collection of adjoining mining concessions that together cover a 40 km section of the Ashanti gold district in the central eastern section of the Western Region of Ghana (Figure 3-1), with the processing facilities situated approximately 10 km south of the town of Bogoso. GSR currently holds five mining leases as well as several prospecting licenses to the southwest, northeast and west of Bogoso.

![Figure 3-1 Location of Bogoso/Prestea in the Regional Context of Ghana and West Africa (Source: United Nations 2008)](image-url)
Figure 3-2 shows the location of the mining licence boundaries in relation to the location of the main GSBPL mine workings at Bogoso, Prestea and Pampe including:

- **Bogoso mining lease.** Chujah Main and Bogoso North are the current major producing open pit mines for the GSBPL deposits located within the Bogoso mining lease. The other main deposit includes Dumasi (immediately north of the Chujah Main pit). The processing facilities are located just south of the Chujah Main open pit; and

- **Prestea mining lease.** The Buesichem deposit and PUG lie to the north of the Prestea lease with the Beta Boundary South, Bondaye and Tuapim deposits (collectively, Prestea South) located south east, in the central part of the lease.

The map in Figure 3-3 illustrates the outline of the Prestea mining lease along with geographic latitude and longitude of each points of the mining lease.

### 3.2 Mineral Titles and Agreements

A detailed investigation into the legal tenure of the Prestea Mine concession was beyond the scope of the FS and this technical report. However, the concession boundaries have been reviewed and it is concluded that the Mineral Resources lie inside of the current licence area.

The Prestea concession is a mining lease that was issued to Prestea Gold Resources Limited ("PGR") on 29 June 2001 by the Government of Ghana with land registry number 2799/2001. The agreement granted PGR the exclusive right to operate underground mining within the Prestea Concession lower than a depth of 150 m below sea level for a period of 30 years effective from the date of Mining Lease. The strike of the underground lease extends from the Ankobra shaft in the north to the Tuapim shaft to the south and covers an approximate area of 11.27 km², which represents only a portion of the entire 129 km² Prestea Mining Lease. A joint operating agreement was signed in January 2002 between Bogoso Gold Limited ("BGL"), a subsidiary of GSR incorporated under the laws of Ghana, and PGR. An amount of US$ 2.1 million was paid to PGR as a first option payment. This agreement granted Bogoso Gold Limited the right to develop and operate the PUG while also setting out the protocols and procedures to be observed by BGL and PGR in the day-to-day operations of the surface and underground mining operations.

A second agreement entitled Memorandum of Agreement ("MoA") was signed on 14 March, 2002 between PGR, BGL, Prestea Goldfields Limited, the State Gold Mining Company Limited ("SGMC"), the Ghana Mineworkers Union of Ghana and the Republic of Ghana. This agreement was formed to create a joint venture agreement between all parties who had an interest in the PUG at the time and to consolidate the management of the underground mine. The agreement also defined the conditions for the PUG to be put under C&M, which includes mine dewatering and shaft maintenance along with the number of employees required. The PUG has remained under C&M since the signature of the agreement.
Figure 3-2  Location of Principal Operations in Relation to Mining Licence Boundaries at Bogoso/Prestea (Source: GSR 2011)
Figure 3-3  Prestea Mining Lease (Source: Golden Star Exploration)
3.3 Royalties and Encumbrances

Royalties associated with the Prestea Mining lease are defined under Section 21 of the mining lease that was issued to PGR on 29 June 2001. The agreement stipulates that the company shall pay a 5% royalty on gross revenue on a quarterly basis to the government as prescribed by the legislation. Royalties are based on production and are to be paid through the Commissioner of Internal Revenue within thirty days from the end of the quarter.

The Prestea mining lease is subject to a rent for land usage, which is defined under Section 20 of the mining lease that was issued to PGR on 29 June 2001. The rent is paid bi-annually before the first day of January and on or before the first day of July.

Another financial obligation related to the Prestea mining lease is rent payable to the government of Ghana for the Central Shaft headframe and other mine infrastructure. The rent is US$0.5 million per year.

3.4 Environmental Liabilities

The existing environmental liabilities associated with the Prestea underground operation were included in an indemnity granted to Prestea Gold Resources (now part of GSBPL) by the Republic of Ghana. The indemnity document is dated 21 December 2001 and titled ‘Prestea Gold Resources Indemnity Against Pre-Existing Environmental Liabilities’. This indemnity suggests that the WRP will be allowed to function independently of the rest of the PUG and GSR will not be obliged to treat the water emanating from the wider Prestea Mine.

3.5 Operating Permits Required

To re-start the PUG, an environmental approval for the WRP is required. An outline of the permitting process is as follows:

- Submission of form EA2 to the Environmental Protection Agency (“EPA”) to register the project (completed);
- Submission of a draft Environmental Scoping Report and Terms of Reference for EPA comment (completed);
- Completion of an EIA with the submission of a draft EIS for EPA comment;
- Revision and submission of the EIS; and,
- Issuance of the Environmental Permit.
4 Accessibility, Climate, Local Resources and Infrastructure (Item 5)

4.1 Infrastructure and Physiography

Local population centers are located at Bogoso town in the northern half of the Bogoso Concession, and Prestea town, which is about in the center of the Prestea Concession. Bogoso is on the main road from Tarkwa to Kumasi and there is a paved road between Bogoso and Prestea. The town of Prestea is located adjacent to the backfilled workings of the Plant North open pit. The Central Shaft complex and offices for PUG are within the town limits.

4.2 Topography, Elevation and Vegetation

The topography of the area within which the GSBPL assets are located is characterized by a series of northeast-southwest trending sub-parallel ridges. The mineralization tends to occur on the western slopes of the ridges with the intervening valleys used by farming communities and containing ephemeral streams. Vegetation in the area has been largely disturbed by the various activities and consists of a patchwork of small farms, and urban / community development, with some secondary forest growth on the steep slopes and hilltops that are not suitable for farming.

4.3 Climate and Length of Operating Season

As PUG is an underground mine, the climate has no major impact on the operations. In the tropical environment, work on the surface can continue year round, with short breaks during the mostly short-lived storm events.

4.4 Access to the Property

Access to the property by road is a 6 hr drive from Accra via the port city of Takoradi. The road is paved from Accra to Tarkwa with the last 1 hr to Prestea being unpaved road. There are airports at Kumasi and Takoradi, which provide daily services to the international airport at Accra. Kumasi is situated approximately a 3.5 hr drive from the PUG. Road surfaces in the area vary from poor (on the section between Bogoso and Tarkwa) to good (Accra to Takoradi). There have been government plans to re-surface the road between Bogoso and Tarkwa for several years; however, it remains in poor condition but is passable throughout the year.

4.5 Local Resources and Infrastructure

PUG is in an area where mining has occurred more or less continuously since the late 1800’s so local skilled underground workers are available. The following services and infrastructure are relevant to the assessment the Project:

- Surface access to PUG is via the public road network that extends to the Project;
- Electricity and water are available;
- Processing and tailings storage will be carried out at Bogoso; and
- Any waste rock generated at the site will be stored such that acid generation is prevented.
5 History (Item 6)

5.1 Prior Ownership

Recorded production for the Prestea mine began in 1912 under the British company Ariston Mining, which operated the mine until the 1950’s and was responsible for the majority of the underground development including shaft sinking, ventilation and level development. The mine was nationalized in the late 1950’s, following the independence of Ghana, when all mining operations in the Prestea region were consolidated under the management of Prestea Gold Limited, a subsidiary of the SGMC.

In the early 1990’s, the government of Ghana reopened the mining industry to foreign companies and a joint venture agreement was formed between Barnex JCI Ltd., Prestea Gold Ltd., the SGMC and the government of Ghana. Barnex JCI Ltd. withdrew from the joint venture in 1998 due to low gold price and aging infrastructure. A consortium supported by the Ghana Mine Workers Union was then founded to operate the mine under the name PGR. The mine operated for 3 years until closure in early 2002 due to depressed gold prices and financial difficulties. The mine has remained under C&M since the 2002 closure.

5.2 Past Exploration and Development

Ariston Mining established most of the current infrastructure and underground development prior to nationalization in the late 1950’s. The PUG workings extend over a distance of 6 km along strike and down to a maximum depth of about 1450 m below surface. The two primary shafts of the Prestea Mine are the Central Shaft and Bondaye Shaft.

Central Shaft is the primary access to underground mining levels, extending to a depth of 1238 m below surface to 30 L. Numerous levels were developed off the shaft to provide access to the Main Reef stoping areas. Traditional narrow vein mining methods were employed; primarily shrinkage stoping and captive cut and fill. RoM material and waste were trammed to the Central Shaft to loading pockets located below 20 L, 25 L, and 30 L, which served to load the RoM into skips for conveyance to the surface bins. The total capacity of the system at its peak may have been around 1300 to 1600 t/day.

Bondaye Shaft extends to a depth of 1103 m but unlike Central Shaft, there is no dedicated rock handling system at Bondaye, cars were loaded into the cages and raised to surface.

In addition to Central and Bondaye shafts, there are several internal shafts. The No. 4 and No. 6 shafts are located to the south of Central Shaft. No. 4 shaft extends from 23 L to 35 L and was used as the primary access to 35 L, the lowest developed level in the mine. No 6 Shaft which extends from 24 L to 31 L.

During the Ariston Mining period, exploration consisted mainly in driving crosscuts from the main footwall drive across the orebody fault structure and collecting channel samples across the fault-filled veins. The first drilling campaign was conducted in 1938, a total of 17 holes were drilled that year and consisted of short holes at Alpha Shaft which were targeting a subsidiary footwall structure. Exploration drilling ramped up in the 1960’s after the nationalization of the mine; over 350 holes were drilled during that decade mostly targeting subsidiary structures of the Main Reef.
The focus of this PEA is the West Reef orebody, which is parallel to and located in the hanging wall to the west of the Main Reef structure. Exploration drilling targeting the West Reef structure started in the 1970’s and continued until the closure of the mine in 2002.

5.3 Historic Mineral Resource and Reserve Estimates

Previous operators of Prestea, Barnex JCI Ltd., have historically classified a certain quantity of mineralised material as a Mineral Resource. This historical resource was reviewed by SRK, who reported this data in a NI 43-101 technical report to GSBPL in 2003. The historical resource consists of simple volumetric estimates based on vertical longitudinal projection block grades and thickness and this material had been included in the latest Mineral Resource statement. The historical resource is classified as Inferred material under the category ‘JCI Blocks’, but is not included as a mining target in this technical report.

Recent Mineral Reserve estimates have been prepared by:

- GSR – Year-end reserve estimate effective December 31, 2013.

Both of these recent reserve estimates relate to mechanized mining of the West Reef resource and are superseded by this PEA.

5.4 Historic Mine Production

Mining in the Prestea area dates back several centuries. The first direct involvement by Europeans in the area occurred in the 1880’s with the establishment of the Gio Apanto Gold Mining Company and the Essaman Gold Mining Company.

These companies changed to the Apanto Mines and Prestea Mines Limited in 1900. Both companies merged to become Ariston Gold Mines in 1927. Companies associated with Ariston carried out exploration and some mining to the north east of Prestea at Quaw Badoo and Brumasi. The company also prospected concessions immediately to the south west of Prestea at Anfargah.

Prospecting and some mining had been carried out independently on the adjacent Ekotokroo, Bondaye and Tuapim concessions located to the south of Anfargah. These concessions were acquired by Ghana Main Reef Limited in 1933 and operated continuously until 1961.

Ghana State Mining Corporation was set up with effect from March 1961 under an Instrument of Incorporation signed by the President. From April 1963 the various Ghanaian gold operations, were regrouped and renamed Tarkwa Goldfields, the Ariston and Ghana Main Reef concessions which were combined to form PGL, Dunkwa Goldfields and Bibiani Goldfields. The SGMC was established under the SGMC Instrument 1965.

Both the Ariston Mines and Ghana Main Reef companies were purchased by the Government of Ghana and merged to form PGL. The Buesichem concession to the north east and along strike from Brumasi was subsequently added to the Prestea concessions. The Buesichem concession contained a small historical open pit, one of several operated by Marlu Gold Mining Areas until 1955.
Figure 5-1 summarises the total historical production from the various orebodies at PUG. The authors note that some production figures for the area are not available, particularly for the early years. Total production from the Ariston Mines and PGL was in the order of 16.8 Mt of RoM material for the recovery of 5.95 Moz Au. The average RoM grade is estimated to have been 11 g/t Au. In addition, the Brumasi Mine is reported to have produced 0.3 Mt yielding 0.23 Moz Au for an average grade of 23.3 g/t Au. Prior to the amalgamation with Prestea, Ghana Main Reef produced about 2.0 Mt of RoM for approximately 1 Moz Au at an estimated RoM grade of 15 g/t Au. Total underground production from the area, excluding the Buesicgem open pit is estimated to be in excess of 19 Mt of RoM and 7.18 Moz Au. The Ghana Chamber of Mines has recorded approximately 9 Moz of gold produced from the Prestea area since 1877 which also includes production from open pit mines.

Production from Prestea peaked at 446,372 t in 1964 when 166,973 oz of gold were obtained at an average grade of 11.6 g/t.

Recovered grade peaked much earlier in the life of mine with a grade of 20.4 g/t Au in 1927. Production endured a serious decline throughout the mid to late 1970’s due to a reduced number of stopes being developed and lack of underground development to access new ground. The mine closed down in 2002 and has remained under C&M since.

Figure 5-1 Historic PUG Production (Source: GSR)
6 Geologic Setting and Mineralization (Item 7)

6.1 Regional Geology

The regional geological setting of the Ashanti belt has been described by several authors previously. The most recent publication describing the geological setting of the sub-region was from Perrouty et al., in Precambrian Research in 2012.

The Ashanti greenstone belt in the Western Region of Ghana is composed primarily of paleoproterozoic metavolcanic and metasedimentary rocks that are divided into the Birimian Supergroup (Sefwi and Kumasi Groups) and the Tarkwa Group. Both units are intruded by abundant granitoids (Figure 6-1) and host numerous hydrothermal gold deposits such as the Obuasi and Prestea mines and paleoplacer deposits such as the Tarkwa and Teberebie Mines.

Allibone et al. (2002) separated the Paleoproterozoic Eburnean orogeny into two phases known as Eburnean I and II, this classification was revised by Perrouty et al. in 2012 and proposed two distinct orogenic events, the Eoeburnean orogeny and the Eburnean orogeny. The Eoeburnean orogeny predates the deposition of Tarkwaian sediments and is associated with a major period of magmatism and metamorphism in the Sefwi Group basement. The Eburnean event is associated with significant post-Tarkwaian deformation that affected both the Birimian Supergroup and overlying Tarkwaian sediments. The Eburnean orogeny is associated with major northwest to southeast shortening that developed major thrust faults, including the Ashanti Fault along with isoclinal folds in Birimian metasediments and regional scale open folds in the Tarkwaian sediments. These features are overprinted by phases of sinistral and dextral deformational events that reactivated the existing thrust faults and resulted in shear zones with strong shear fabrics.

The Birimian series was first described by Kitson (1918) based on outcrops located in the Birim River (approximately 80 km east of the Ashanti Belt). Since this early interpretation, the Birimian stratigraphic column has been revised significantly. Before the application of geochronology, the Birimian super group was divided in an Upper Birimian group composed mainly of metavolcanics and a Lower Birimian group corresponding to metasedimentary basins. Subsequent authors have proposed synchronous deposition of Birimian metavolcanics. Most recently, Sm/Nd and U/Pb analyses have reversed the earlier stratigraphic interpretation with the younger metasediments overlying the older metavolcanics. Proposed ages for the metavolcanics vary between 2,162 ± 6 million years (“Ma”) and 2,266 ± 2 Ma. Detrital zircons in the metasediments indicate the initiation of their deposition between 2,144 ± 24 Ma 2,154 ± 2 Ma. The Kumasi Group was intruded by the late sedimentary Suhuma granodiorite at 2,136 ± 19 Ma (U/Pb on zircon, Adadey et al., 2009).

The Tarkwa supergroup was first recognized by Kitson (1928) and consists of a succession of clastic sedimentary units, which have been divided in four groups by Whitelaw (1929) and Junner (1940). The Kawere Group located at the base of the Tarkwaian super group is composed of conglomerates and sandstones with a thickness varying between 250 and 700 m. The unit is stratigraphically overlain by the Banket Formation, which is characterized by sequences of conglomerates interbedded with cross-bedded sandstone layers, the maximum thickness of this group being 400 m. The conglomerates are principally composed of Birimian quartz pebbles (>90%) and volcanic clasts (Hirdes and Nunoo, 1994) that host the Tarkwa Placer deposits. The Banket formation is overlain by approximately 400 m of Tarkwa Phyllites. The uppermost unit of the Tarkwa super group is the Huni Sandstone, comprised of alternating beds of quartzite and
phyllite intruded by minor dolerite sills that form a package up to 1300 m thick (Pigois et al., 2003). U/Pb and Pb/Pb geochronology dating of detrital zircons provide a maximum depositional age of 2,132 ± 2.8 Ma for the Kawere formation and 2,132.6 ± 3.4 Ma for the Banket formation (Davis et al., 1994; Hirdes and Nunoo, 1994). These ages agree with the study by Pigois et al. (2003) that yielded maximum depositional age of 2,133 ± 4 Ma from 71 concordant zircons of the Banket formation. According to all concordant zircon histograms (161 grains) and their uncertainties, a reasonable estimation for the start of the Tarkwaian sedimentation could be as young as 2,107 Ma.

Abundant granites and granitoids intruded the Birimian and Tarkwaian units during the Paleoproterozoic. Eburnean plutonism in southwest Ghana can be divided into two phases between 2,180 to 2,150 Ma (Eoeburnean) and 2,130 to 2,070 Ma (Eburnean) that is supported by the current database of U/Pb and Pb/Pb zircon ages. Most of the granitoids intruded during both phases correspond to typical tonalite–trondhjemite–granodiorite (“TTG”) suites. However, in the southern part of the Ashanti Belt, intrusions within the Mpohor complex have granodioritic, dioritic and gabbroic compositions.

Dolerite dykes oriented north south and east northeast to west southwest which are typically less than 100 m in thickness are abundant across the West African craton where they cross-cut Archaean and Paleoproterozoic basement. In southwestern Ghana these dykes are well defined in magnetic data where they are characterised by strong magnetic susceptibility. Dolerite dykes are observed to cross-cut undeformed K-feldspar rich granites that formed during the late Eburnean, and are overlain by Volta basin sediments with a maximum depositional age of 950 Ma (Kalsbeek et al., 2008). These relationships constrain dyke emplacement to between 2,000 and 950 Ma. In contrast some older dolerite and gabbro dykes and sills were deformed during the Eburnean orogeny and are dated at 2,102 ± 13 Ma (U/Pb on zircon, Adadey et al., 2009).

With the exception of some late Eburnean granitoids, dolerite dykes and Phanerozoic sediments, all other lithologies have undergone metamorphism that generally does not exceed upper greenschist facies. Studies on amphibole/plagioclase assemblages suggest the peak temperature and pressure was 500 to 650°C and 5 to 6 kbar (John et al., 1999), dated at 2,092 ± 3 Ma (Oberthür et al., 1998).
Figure 6-1 Regional geology of the Ashanti Belt and location within the West African Craton (Source: Perrouty et al, 2012).
6.2 Local Geology and Mineralization

6.2.1 Local Geology

The Prestea concession lies within the southern portion of the Ashanti Greenstone Belt along the western margin of the belt. Rock assemblages from the southern area of the Ashanti belt were formed between a period spanning from 2,080 to 2,240 Ma as illustrated in Figure 6.2, with the Sefwi Group being the oldest rock package and the Tarkwa sediments being the youngest. The Ashanti belt is host to numerous gold occurrences, which are believed to be related to various stages of the Eoeburnean and Eburnean deformational events.

![Figure 6-2 Radiometric age data histograms](Source: Perrouty et al, 2012)

The geology of the Prestea concession is divided into three main litho-structural assemblages (as represented in Figure 6.3), which are fault bounded and steeply dipping to the west. This suggests that the contacts are structurally controlled and that the litho-structural assemblages are unconformable. From the eastern footwall to the western hanging wall, these packages are represented by the Tarkwaian litho-structural assemblage, the tectonic breccia assemblage, composed of sheared graphitic sediments and volcanic flows and the last assemblage is composed of undeformed sedimentary units of the Kumasi basin, which is located to the west of the Ashanti fault zone.
The Tarkwaian litho-structural assemblage to the east is mostly composed of sandstone, pebbly sandstone and narrow conglomerate units. Bedding and sedimentary textures have been observed sporadically, and in most cases they have been obliterated by hydrothermal alteration and deformation at the proximity of the Ashanti fault.

The litho-structural assemblage overlying the Tarkwaian sediments is a tectonic breccia bounded to the west by the Kumasi sedimentary basin. The tectonic breccia is a polygenic assemblage, composed of various rock types such as volcanic rocks, volcanioclastics, sediments of the Birimian Supergroup, and sparse Tarkwaian sedimentary slivers. Volcanic lenses have been divided into two units based on their alteration pattern, weakly altered mafic volcanic rocks are characterized by a distal chlorite/calcite alteration pattern while strongly altered mafic volcanic rocks are characterized by a proximal silica/sericite/Fe-Mg carbonates alteration pattern. These strongly altered mafic volcanic lenses are generally located at proximity to the Main Reef Fault or bounded by second order footwall faults. The tectonic breccia assemblage is believed to have been the focal point of the post thrusting Eburnean deformational events (syn-D3 to syn-D5), therefore, primary textures, whether syn-volcanic or syn-sedimentary, have only been locally preserved. Volcanic lenses are intercalated with sheared graphitic sedimentary horizons which represent strained and brecciated sequences of siltstones, mudstones and greywacke units affected by pervasive graphitic alteration. Primary textures are generally overprinted and obliterated by deformation, but bedding has locally been preserved.

The most western litho-structural assemblage underlying the Prestea concession consists of relatively undeformed to weakly strained sedimentary rocks of the Kumasi basin. The assemblage is composed of a series of flyschoid sequences where the most common units found are argillites, mudstones, siltstones and greywackes, which are all commonly referred to as phyllite in Ghana. Several syn-sedimentary textures have been observed such as bedding planes, graded bedding and cross-bedding. Chert horizons are locally intercalated within the flysch sequence, but appear to lack lateral continuity.
Figure 6-3 Simplified geological plan of the Prestea concession highlighting the main geological and structural features (Compiled from multiple sources, 2003)
The litho-structural assemblages are affected by six distinct Eoeburnean and syn-Eburnean ductile events and at least one post-Eburnean brittle event, resulting in late reactivation of the major thrust fault systems. The Eburnean deformational events have been described and observed by several authors throughout the West African shield, including Milesi, Alibone and Perrouty. The D1 deformational event affects the older volcano-sedimentary sequence of the Birimian supergroup and was generated during a North-South compressional phase. The second event of deformation is related to an extensional phase which generated the sedimentation of the Kumasi basin. The D3 deformational event is believed to be mainly a thrusting event, resulting in the thrusting of the volcano-sedimentary sequences and the Kumasi sedimentary basin over the younger Tarkwaian sedimentary group, which was unconformably deposited during the D3 event.

At Prestea, the principal structure is a mineralized fault-filled quartz vein known as the Main Reef which is relatively continuous and has been modelled and worked over a strike length of approximately 6 km and to a depth of approximately 1,450 m below surface (35 L). Several subsidiary structures such as the West Reef and East Reef have developed respectively in the immediate hangingwall and footwall of the Main reef structure. The West Reef is a second order structure where dilational zones occurs some 200 m into the hangingwall of the Main Reef structure and, at present is known to occur over a strike length of 800 m and has currently been defined by underground drilling between 550 to 1,150 m below topography as far as the 24 L. The major thrust faults such as the Main Reef fault and the West Reef fault, as well as the presence of an associated penetrative foliation, are the main syn-D3 structural features.

The D4 and D5 deformational events were associated with NW-SE shortening which resulted in the re-activation of the D3 thrust fault into sinistral shearing. These Eburnean phases are transpressional events with strike-slip movement along the major D3 faults, which resulted in the development of regional scale folds and of second and third penetrative foliations. The tectonic breccia assemblage and the graphitic shear zone were the focus of reactivation during these two deformational events resulting in a strong shear fabric whilst the sedimentary units in the hanging wall developed orthogonal steep penetrative fabrics at an angle to the main foliation. The last Eburnean phase, the D6 event, is defined as a minor transcurrent event, the effects of which are noticeable by the presence of a flat lying crenulation, observable locally on the main foliation plane within the Kumasi basin sedimentary package.

The structural complexity of the Prestea Mine site is mainly due to reactivation of the fault system during the later events of Eburnean deformation. Several studies along the Ashanti trend are suggesting a syn-D to Syn-D5 timing for gold emplacement. Earlier studies at Prestea also suggested a syn-D3 gold mineralized event, but it is still unclear whether syn-D1 gold events could have taken place and have been remobilized during later hydrothermal pulses. Field evidence and structural relationships are suggesting that certain quartz veins along the Prestea major fault systems were in place during the earlier deformational events and deformed by subsequent events.
Figure 6-4  Lithologies hosting the Prestea deposit

**Key to Figure 6-4**
- A: Weakly deformed Birimian sediments from Plant North pit.
- B: Graphitic Birimian sediments from Plant North pit.
- C: Brecciated graphitic Birimian sediments from hole UC1044 at a depth of 68.4 m.
- D: Contact between altered mafic volcanic and the footwall tectonic Breccia from 30 L at crosscut 204N.
- E: Hand sample from 24 L of mafic volcanic rocks.
- F: Footwall tectonic breccia from Plant North Pit.
6.2.2 Timing of Mineralization

The various authors interpret the Main and West Reef structures to represent reactivated Eoeburnean reverse faults. However, there is some debate about when quartz veining, i.e. emplacement of the lode deposits, took place. Undoubtedly, there are several generations of quartz veining. Davis and Allibone (2004) interpret a syn-D4 Eburnean quartz-veining event, but others believe a syn-D3 age is more likely. Gold within the West and Main Reefs is associated with smoky grey quartz veins which re-fracture the milky quartz veins that comprises the majority of the reef structures which are evidences of multiple fluid pulses along the major faults and most likely of Eburnean timing. To date, there have been no geochronological dating on the Prestea mineralization.

Over five hundred structural measurements were taken during the course of an underground geological compilation which was undertaken between 2003 and 2007. All measurements have been compiled per levels, distributions of S0 beddings and S1 foliations are very similar, suggesting a transposition of beddings along S1 foliation during the D1 event. Both distributions for S0 and S1 are consistent with increasing depths, showing identical patterns regardless of levels. A total of 241 measurements were taken for S1 foliations and 90 for S0 beddings. The average orientation for S0 beddings is 172/76 and 175/80 for S1 foliations, all measurements were taken at mine grid (40 degree eastern rotation to magnetic North). The average orientation for S4 foliations is 229/55, suggesting approximately a 55 degrees rotation in azimuth and 25 degrees in dip of the main stress fields from the D1 deformational event to the D4 event. No measurements of the S5 foliations were taken during the underground compilation as this feature is more discrete than the S4 foliation. A total of 8 measurements were taken for L51 lineations, very little attention was paid to syn-D5 structural features during the geological compilation as mineralization controls are believed to be syn-D1 to syn-D4. Nonetheless, the lineations’ average indicates a shallowly plunging feature (356/22).

The structural complexity of the Prestea Mine site is mainly due to reactivation of the fault system during the later events of Eburnean deformation. A more detailed look at S4 foliations recorded on 17 L from crosscut 307 to 308S shows that the tectonic breccia located in the footwall of the Main Reef fault has undergone intense shearing during the D4 and D5 deformational events. A comparison between the orientation of S1 foliations taken from the footwall and the hanging wall of the Main Reef fault was conducted and the average orientation for S4 foliations in the footwall of the Main Reef fault is 211/53, while the average for the hanging wall of the Main Reef fault is 252/49, suggesting syn-D4 reactivation in the footwall domain and transposition along the S1 foliation.

6.2.3 Mineralization Style

Davis and Allibone (2004) show the Main Reef to be deformed in a variety of styles, including being affected by folding and boudinage associated with sinistral deformation and a third subordinate folding event. Late structural studies conducted by SRK observed evidence for at least some of the lesser quartz veins in the walls to the West Reef to be strongly affected by a sinistral strike-slip deformation, but has not observed the mesoscopic folding documented by these authors.

The margins of the West Reef mineralised quartz vein are strongly sheared and comprise a brittle-ductile zone of deformation in the graphitic schists a few centimeters to up to 2 meters in width on both sides of the vein at any locality. The deformation along the margins of the vein is
interpreted to be due to post-mineralisation deformation nucleating on the margins of the vein, which represents a strong competency contrast with the graphitic wall rocks. The kinematics of this deformation appears to be dextral. Over the length of the vein exposed on the 17 L, several subsidiary shears were observed to cut through the vein which either caused the vein to be duplicated, causing local thickening of the mineralised vein over approximately 0 to 10 meters, or caused extensional offsets of the vein. One 10 meter gap in the continuity of the resource on 17 L could be attributed to one of these shears. Overall however, these are relatively minor disturbances which just cause local irregularities in the vein and there should be no overall material loss.

Critically for the vertical continuity of the mineralisation, the Ashanti Trend has not been affected by a major deformation event with a sub-horizontal fold axis which may have acted to truncate the mineralisation at depth. Moreover, the planarity of the mineralised trend is testament to the fact that, on the scale of the deposits, they have relatively simple sheet-like geometries, unaffected by major disruption.

Two distinct styles of mineralization are found on the Prestea Mine site. The more extensive of the two mineralization styles are laminated fault fill quartz veins (Reef style mineralization), bound to the Main Reef fault or to the second order faults found in the Kumasi sedimentary hanging wall. The second mineralization style, which has never been mined in past, consists of arsenopyrite rich, brecciated and altered volcanic lenses.

Fault fill quartz veins have been generated over an extensive period of time through multiple fluid pulses. They are bound to major and second order faults and characterized by laminated and stylolitic smoky to translucent quartz veins. Pictures A and B from Figure 6-5 are examples of two fault filled quartz veins coming from two different faults, the Spur Reef fault from Bondaye shaft and the West Reef fault from Main shaft. Late gauge is commonly associated to the mineralized quartz veins and is also generally mineralized, although in most cases at lower grades then the associated quartz veins. Quartz veins are typically one to two meters wide, but widths up to 5 m have been observed. Thicker fault fill quartz veins occur in dilation zones along the fault systems, it is still unclear what controls the emplacement of those dilation zones, but several mineralized quartz veins seem to have a spatial association with volcanic lenses found in the immediate footwall of the Main Reef fault.

An alteration assemblage of silica/sericite/Mg-Fe carbonates characterizes the arsenopyrite rich volcanic lenses. They are typically composed with 2 to 10 % acicular arsenopyrite crystals and 1 to 5 % euhedral to sub-euhedral pyrite grains. They are also characterized by presence of brecciated and stockwork-like smoky to translucent quartz veins that can account for up to 30 % of the volcanic lenses’ volume. Picture C from Figure 6-5 illustrates typical mineralized volcanic lenses with narrow, buckled smoky quartz veins. Those mineralized volcanic lenses are generally located in the immediate footwall of the Main Reef fault or faulted within the Main Reef fault system. Mineralized volcanic lenses are generally narrow (10 to 25 meters) and stretched along a North-South trend, sub-parallel to the main foliation. On average the lenses will be 50 to 100 m long and locally up to 300 m. This style of mineralization has never been in the past the object of economic mining activity.

Several subsidiary faults are developed within the Birimian sedimentary hanging wall, these faults are generally narrower in comparison to the Main Reef fault which is located at the contact between the tectonic breccia and the Birimian sedimentary package. The West Reef fault is
characterized as a hanging wall subsidiary fault, the fault is typically 1.5 to 2.0 m with fault filled quartz vein developed along a graphitic gauge rich fault. The quartz veins within the West Reef fault have laminated textures with smoky to translucent quartz.

Figure 6-5 Mineralization styles and structural features characterizing the Prestea deposit

Key to Figure 6-5
- A: Reef style mineralization (Fault fill quartz veins) from hole UC 1044 at a depth of 130 m.
- B: Reef style mineralization (Fault fill quartz veins) from the Spur Reef Fault, 7 L of the Bondaye shaft.
- C: Altered and brecciated mafic volcanic with hydrothermal breccia and arsenopyrite mineralization.
- D: Birimian sediments affected by F3 folding, both bedding (S0) and S1 foliation are tightly folded.
- E: Structural relationship between S0, S1, S4 and S6.
- F: Refraction of S4 foliations across Birimian mudstone and a more competent wacke sub unit.
Deposit Type (Item 8)

The Prestea deposit can be classified as a lode gold deposit or an orogenic mesothermal gold deposit, which are the most common gold systems found within Achaean and Paleoproterozoic terrains. In the West African shield, orogenic gold deposits are typically underlain by geology considered to be of Birimian age and are generally hosted by volcano-sedimentary sequences. The Ashanti belt, which hosts the Prestea deposit, is considered prospective for orogenic mesothermal gold deposits and hosts numerous other lode gold deposits such as at the Obuasi mine.

B. Dubé and P. Gosselin of the Geological Survey of Canada described these deposits as greenstone-hosted quartz-carbonate vein deposits in the 2007 special publication no.5 entitled Mineral Deposits of Canada. The authors described these deposits as typically occurring in deformed greenstone belts and distributed along major compressional crustal scale fault zones commonly marking the convergent margins between major lithological boundaries. The greenstone-hosted quartz-carbonate vein deposits correspond to structurally controlled complex deposits characterized by networks of gold-bearing, laminated quartz-carbonate fault-fill veins. These veins are hosted by moderately to steeply dipping, compressional brittle-ductile shear zones and faults with locally associated shallow-dipping extensional veins and hydrothermal breccias. In these deposits, gold is mainly confined to the quartz-carbonate veins, but can also occur within iron-rich sulphidized wall rocks or within silicified and arsenopyrite-rich replacement zones.

The Ashanti belt is considered prospective for orogenic mesothermal gold deposits and hosts numerous lode gold deposits and paleoplacer deposits. As illustrated by Figure 6-1, several major gold deposits are found within the Ashanti belt which can be classified into six different deposit types:

1. Sedimentary hosted shear zones
2. Fault fill quartz veins
3. Paleoplacer
4. Intrusive hosted
5. Late thrust fault quartz veins
6. Folded veins system

The sedimentary hosted shear zone deposits are localised principally along a steep to sub-vertical major crustal structures located along the western margin of the Ashanti belt referred to as the Ashanti trend. The Ashanti trend shows a range of mineralization styles associated with graphitic shear zones, which represents the principal displacement zone of a regional-scale shear zone that defines the mineral belt. These styles include highly deformed graphitic shear zones containing disseminations of arsenopyrite as the principal gold bearing phase and disseminations of sulphides in mafic volcanic rocks generally found in the footwall of the main shear zones. The sedimentary hosted shear zone deposits which occur along the Ashanti trend include Bogoso, Obuasi, Prestea and Nzema.

The second type of deposit found within the Ashanti belt are laminated quartz vein deposits containing free gold. Fault filled quartz vein deposits also occur along the Ashanti trend but are only present at Obuasi and Prestea. The third type of deposit are paleo placer deposits within the
Tarkwaian sedimentary basin which are hosted within narrow conglomerate horizons intercalated with sandstone units characterized by iron oxides cross beddings. Paleoplacer deposits occur in the southern portion of the Tarkwa basin and examples include Tarkwa, Teberebie and Iduaprim. The fourth type of deposit found within the Ashanti belt are intrusive hosted deposits which occur along second order structures such as the Akropong trend in the Kumasi basin and the Manso trend in the Southern portion of the Ashanti belt. These deposits can be hosted both within felsic and mafic intrusives and are characterized by a penetrative fabric where gold is associated with pyrite and arsenopyrite. Examples of such deposits include Edikan and Pampe along the Akropong trend and Benso and Hwini Butre along the Manso trend. The fifth type of deposit found within the Ashanti belt is late thrust fault associated quartz vein deposits. The Damang mine which is located in the Tarkwaian sedimentary basin is the only known thrust fault related deposit in the Ashanti belt. The deposit is characterized by low angle; undeformed extensional and tensional veins associated with low angle thrust Faults. This type of deposit contrasts with the last type of deposit found with the belt, the multi-phase folded vein deposit. The Wassa deposit which is located on the Eastern flank of the Ashanti belt can be classified as a Eoeburnean folded vein system and is the only such deposit recognized to date within the Ashanti belt. The Wassa mineralization consists of greenstone-hosted, low sulphide hydrothermal deposits where gold mineralization occurs within folded quartz-carbonate veins.

At Prestea, gold mineralization exhibits a strong relationship with major shear zones, fault zones and second order structures. Two types of mineralization have been identified at Prestea, which are both characterised as mesothermal gold mineralization:

- Fault-fill quartz veins along fault zones and second order structures, which typically contains non-refractory, free milling gold; and
- Disseminated mineralization associated with brecciated zones of iron-rich footwall volcanic lenses, which are characterized by finely disseminated arsenopyrite rich and silicified replacement zone. This type of mineralization is generally lower grade, refractory and locally termed ‘sulphide material’.

The weathering profile at Prestea is deep and typically results in extensive surface oxidation of bedrock, to a depth of up to 100 m. Generally, the weathering profile typically consists of a lateritic surface, a saprolitic horizon, a transitional zone and a deeper primary sulphide zone.
8 Exploration (Item 9)

Data validation and selected evaluation drilling from underground have helped to increase the confidence in the morphology and orientation of the mineralization at Prestea. Cross cut samples and JCI era drilling data (surface drilling) accounts for some 92% of the available data. The remainder is a mixture of surface (“RC”) and Diamond Drill (“DD”) holes drilled by GSR and underground channels and diamond drillholes acquired by GSR as part of their purchase of the Prestea Mining rights.

The cross cut and JCI data extends over a strike length of some 8 km with the majority lying between y=6000 to y=12000 in the GSR mine grid system. Sampling covers a depth extent of 1400m from surface. The GSR data is largely concentrated in the area underlying the Plant North open pit, Central Shaft and the northern extent of Beta Boundary.

The previous Mineral Resource estimate for the Prestea underground orebodies was based on a combination of GSR underground sampling from some 157 drillholes, 117 rock saw samples and channel sampling from 2 cross cuts. During late 2005 and throughout 2006 GSR drilled an additional 106 underground holes into the Main Reef (“MR”) and West Reef (“WR”) orebodies. This drilling has been carried out using fan drilling from cubbies on the most accessible levels but predominantly from the 12, 17 and 24 Levels.

The Prestea underground West Reef target was the last area to be mined by PGR in 2002. The subsequent exploration of the West Reef underground target has been planned and managed by Golden Star Resources Ltd and was initiated in 2004. The 17 L West Reef drive exposes the vein structure from 7618 N in the south to 8065 N in the north a distance of approximately 450 m. Along the West Reef drive the backs have been sampled approximately every 5 m with a 2 x 2 inch channel sample cut using an air driven diamond blade rock saw. The channel samples were cut orthogonal to the main structure. The channel samples and the reef drive have been surveyed and tied into the mine grid at surface. A total of 81 channel samples were collected on the 17 L reef drive averaging 2.4 m width with composite grades ranging from 0.1 to 127 g/t Au. The results are summarized in Table 8-1.

Table 8-1 Channel Sample results 17 L West Reef Drive

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The results from the 17 L channel sampling show that the mineralization along the reef is hosted in several higher grade pods. These high grade pods were drill tested at depth from cubbies on the 17 L and 24 L, drilled from the footwall to the hanging wall obliquely to the moderately west dipping foliation and reef. The details of this drilling will be discussed in Section 9.

In addition to the exposures on 17 L the West Reef has been encountered on a small reef drive on 24 L, approximately 300 m below 17 L. On the 24 L West Reef drive the structure was encountered but the vein here has pinched. Drilling from 24 L has delineated wider vein widths to the north of this reef drive alluding to a steep northern plunge of the higher grade mineralized zone hosted in the West Reef structure. Prior to 30 L being flooded, two hangingwall crosscuts were excavated for drill access to test the Main Reef between 30 L and 35 L. In the 270S 30 L cross-cut the West Reef was sampled intersecting a horizontal width of 5.8 m grading 3.1 g/t Au and 5 m at a grade of 3.3 g/t Au on the north and south sides of the excavation respectively. Neither of these samples has been included in the current West Reef resource estimates and show that the structure continues at depth below 24 L.
9 Drilling (Item 10)

The samples used for the current Mineral Resource estimates at Prestea are based on a combination of surface DD drilling and underground core drilling. Fan drilling is carried out from drill cubbies in order to reduce the movement of the drill rigs. In addition to the drilling, rock saw channels have been cut on a number of levels to provide channel samples across the orebody and to investigate the grade distribution in the immediate contact zones adjacent to the orebody in order to better define the potential dilution. Drilling for the Main Reef orebody has been carried out at roughly 80-100 m spacing along strike from surface. The underground drilling has been largely concentrated on the West Reef orebody and has consisted of fan drilling from individual cubbies with up to 21 holes drilled from a single collar location. Underground collar locations are spaced approximately 80 m apart along strike on the 17, 24 and 30 Levels.

Data is currently available from some 675 surface and underground drillholes in the Prestea area broken down as follows:

- DD surface = 274 drillholes; 29 700 m
- RC surface = 137 drillholes; 14 000 m
- DD underground = 264 drillholes; 41 500 m

Drilling for the West Reef resource was conducted from underground drill stations, predominantly from 17 and 24 Levels. The drilling was conducted by Golden Star Resources and no historical data was used in the resource estimates. On 17 L, 10 drill stations were established along the MR footwall access where fan drilling was conducted dominantly horizontally and down dip (Figure 9-1). The up dip portion of the WR remains to be tested between 12 and 17 Levels and remains one of the priority drill targets. The 17 L drill stations are located on the following cross cuts, 274S, 277S, 280S, 285S, 287S, 290S, 293S, 297S, 302S and 308S testing the strike over approximately 775 meters.
On 24 L, which is approximately 300 meters below 17 L, drilling was conducted from three drill chambers, 274S, 284S and 287S (Figure 9-2). The drilling from the three drill stations enabled the West Reef to be tested approximately 550 meters along its strike length as well as up and down dip.
The underground drilling of the West Reef target was conducted in several campaigns from 2004 to 2006 with a total of 128 holes and 28,790 meters being completed during this time. All drilling was conducted with underground diamond drill core rigs using NQ2 (~50 mm) sized core. All drill hole collars were surveyed using the underground survey control brought down from surface using the mine grid. The holes were also surveyed nominally every 25 to 30 meters down hole using a Reflex single shot survey instrument.

Core recovery through the mineralized zone was optimized by using chrome core barrels, viscous muds and short drilling runs but in some holes some of the "graphitic fissures" (graphic rich fault gouge) were washed away. Areas of lost core were not sampled and in the database are
identified as insufficient sample or "IS" and were given a zero grade. Generally core recovery was good through the zone.

West Reef intersections in the areas where the resources have been classified as indicated are on a nominal 25 x 25 metre grid whereas inferred resources exceed the 25 metre drill hole spacing (Figure 9-3).

Figure 9-3  West Reef Long section looking west from footwall with drill hole pierce points and current drives

Several representative drill sections have been included below (Figure 9-4 and Figure 9-5) showing the attitude of the West Reef and the relatively consistent dip and gold tenures.
Figure 9-4 West Reef drill X Section 8000N showing reef and drill hole intersections
Figure 9-5  West Reef drill X Section 8200 N showing reef and drill hole intersections
10 Sample Preparation, Analyses and Security

(Item 11)

Sampling from Reverse Circulation (RC) drilling is carried out using a standard single cyclone with samples collected at 1 m intervals through the expected mineralized zone. In zones of waste rock the sample interval is occasionally increased to a 3 m composite. However all samples are assayed and if a 3 m sample returns a significant grade value the original 1 m samples will be assayed individually. All samples are riffled and bagged at the drill site and returned to the Bogoso Mine for reduction and sample preparation.

Diamond drilling (“DD”) core from surface drilling is collected using HQ size core barrels (63.5 mm). The core is logged and sawn in half at the Bogoso mine site and 1 m samples are prepared through the prospective mineralized zone. However, geological contacts are taken into account and samples will therefore vary slightly in length. In waste zones samples are collected at 1 m nominal intervals where alteration, sulfidation or quartz veins are observed. Underground drilling is carried out using NQ or HQ size core and the core is sawn in half and prepared for assay. The orebodies dip steeply to the west and depending if the drilling is from surface or underground is angled to try and intersect the mineralized zone orthogonally, however from underground drilling cubbies this is often not possible. Recoveries and SCR values are recorded in the database and 80 % of the diamond samples have a recovery greater than 95 % with 92 % showing a recovery greater than 80 %.

Samples used for the West Reef resource estimations were of two types, rock sawn channel samples on 17 L and 24 L reef drives and NQ sized diamond drill core.

Channel samples were collected using a double diamond blade Cheetah air driven rock saw. This saw produced a channel sample roughly 50 mm deep by 50 mm wide. Sample collection and dispatch to the laboratory was supervised by a geologist who ensured the samples were taken correctly, labelled and transported to the surface.

Core samples generated from the underground drilling were processed at either the core logging facilities at Prestea Central Shaft or at the main core storage facility near the Bogoso processing plant. Core boxes with lids were delivered to the logging facilities at the end of every shift by the drillers. The core logging process involved initial cleaning of the core and checking of the metre blocks and mark ups on the individual boxes, if there are any discrepancies they are addressed with the driller who was responsible for the core. All core was photographed prior to being logged and sampled. Two teams logged the core at surface one being responsible for recording geotechnical information and overall core recovery between drilling runs. Following the geotechnical drilling the core is logged by the geologist who pays particular attention to structure, lithology, alteration and mineralization. All of the core has been orientated with a spear orientation device and this has been used to take structural measurements while the core is being logged.

Sampling intervals are laid out by the geologist logging the core and are based on geological contacts with samples in mineralized zones generally not exceeding one metre. The physical sampling of the core was done with a diamond blade core cutting saw. The core was sawn in half along the line marked by the geologist to ensure a representative sample is taken. The half sawn core samples were deposited into individual plastic bags where the sample number was both written on the bag as well as on a piece of flagging which was inserted into the bag. The
remaining half core sample was returned to the core boxes and kept for future reference. During the sampling, standards and blanks are inserted in the sample numbering sequence and these are recorded on the lab dispatch sheets. Every 20 samples that are submitted to the laboratory are accompanied by a sample standard and a blank to check the precision of the analysis. Additional checks are done on samples once the results have been returned.

Samples are dispatched to either SGS laboratories or Transworld Laboratories (now Intertek Mineral Lab) in Tarkwa. Samples were organized in the core logging facilities where they were checked and put into numeric order. The transportation to the laboratory in Tarkwa is provided by the lab. Sample turnaround and dispatch are recorded either in a spreadsheet (earlier samples) or with the database software acQuire.

Sample rejects and pulps were returned to the Bogoso core logging facility where they are stored for up to a year and then disposed of. Approximately 10% of the coarse reject samples, above detection limit that are returned to site are renumbered and resubmitted to the laboratory for duplicate analysis and used for QA/QC evaluations. The processing, handling, analysis and storage of the samples for the Prestea Mine are considered to be within or exceed industry standards.
11 Data Verification (Item 12)

11.1 Introduction

Samples are obtained from various stages of the drilling and sampling procedure. Analysis is carried out using scatterplots to indicate bias between sets of sample pairs using correlation analysis. The other plot used is a HARD analysis (ranked Half Absolute Relative Deviation) which examines the relative precision of assay pairs representing the same sample interval within the drillholes. The relative error is obtained by dividing the absolute error in the quantity by the quantity itself. The relative error is usually more significant than the absolute error. When reporting relative errors it is usual to multiply the fractional error by 100 and report it as a percentage.

The following section discusses the results of the QAQC work carried out on the Prestea West Reef underground exploration and Prestea Main Reef (Plant North) samples obtained during 2004-2008.

11.2 Replicates

Two separate samples are collected at the drill site and bagged separately from which two individual samples will be produced. The results of these checks can be useful in highlighting natural variability of the grade distribution.

Replicates were produced from the surface diamond drilling at Plant North for those holes targeting the underground Main Reef and West Reef deposits. The results are summarised in the following HARD plot and indicate a relatively poor correlation between the two sets of results. Some 40% of the sample pairs have a HARD of >20%. However, the reason for this may be partly due to the use of a relatively low number of low grade intersections generally less than 1 g/t which are not representative of the underground orebody. Lower grade samples tend to be more likely to have larger relative differences in grade due to the nuggety nature of the orebody. Therefore the authors consider the replicate sampling to be inappropriate and recommend that the core be resampled with high grade intersections which more appropriately represent the orebody.
Duplicates

Duplicate samples are composed of coarse reject sample material which has been returned by the independent off site laboratories. Approximately 10% of the coarse rejects returned to site are selected for duplicate analysis. The sample selection concentrates on assays greater than 0.2 g/t. The duplicate samples are rebagged and given a new unique sample number and then resubmitted to the laboratory for a second analysis. It is these sample pairs that are used to determine the accuracy of the lab and repeatability of the sample results.

The results from these duplicates are useful in indicating problems with sample preparation and splitting and are also indicative of the inherent variability of grade within a sample size volume which has implications for the modelling of semi-variograms and estimation of nugget variance.

11.3.1 West Reef Underground Drilling 2006

This work has shown that reproducibility is relatively poor for the underground sampling data from Prestea as shown in Figure 11-2 and Figure 11-3. This is in part due to the difficulty in producing sufficient sample material from the small diameter core. The high grade of the deposit also contributes to the variability with only 50 % of the sample pairs exhibiting a ‘Half Absolute Relative Difference’ (HARD) value of 20 % or less.

However, there is a clear trend for the sample pairs with the highest variability to have an average grade of less than 5 g/t. Given that the current block COG for the Mineral Resource is 4.74 g/t the majority of the high variability pairs will be occurring in the lower grade areas of the deposit and it is likely that the effect of this variability will be to cause some local dilution issues.

Figure 11-1  Prestea Main Reef exploration replicates HARD plot
where low grade material is included within the mineralisation. However, the majority of the higher grade sample pairs show relatively good reproducibility.
11.3.2 Plant North area Main Reef and West Reef surface exploration

The results from the duplicate sampling generally show a positive trend with the majority of the sample pairs exhibiting HARD values of less than 20%. There is a correlation between lower grades and higher HARD values indicating greater variability in the lower grade areas of the orebody. However, a number of the sample pairs exhibit average grades below the current model cut off grade and care should be taken to make sure representative mineralised intersections are used for these studies in future.

Figure 11-3  Prestea West Reef underground exploration duplicates correlation plot
Figure 11-4 Prestea Main Reef RC duplicate HARD plot

Figure 11-5 Prestea Main Reef DD duplicate HARD plot
11.4 Blanks

Blanks were inserted in the samples sent for Screen Fire Assay (“SFA”) and used as a check on the efficiency of the laboratory. This method is useful for highlighting contamination problems and also cross labelling when samples are mislabelled in the laboratory. The blanks were prepared from RC chips known to be devoid of mineralization filtered to 0.01 ppm. A total of 35 Blank samples were inserted in different batch of samples sent as part of the SFA testwork. The laboratory values range from 0.01 to 0.03 ppm with two values being 0.04 and 0.05 ppm and the lab mean value is 0.02 ppm suggesting there is not a significant issue with contamination at the laboratory and the authors consider the results to be acceptable for use in Mineral Resource estimation.

![Blanks Results for 2008 sampling campaign](image)

**Figure 11-6** Results of blank analysis, January to March 2008

11.5 Gannet Standards

Standards are used for checking the precision and accuracy of the laboratory. A total of 161 gannet standard samples comprising 5 different grades were used as control samples for the Fire Assay standards. The performance accuracy of the lab is shown in Table 11-1 and Figure 11-7 to Figure 11-11.

The general results appear to be good with most results lying within 2 Standard Deviations of the certified value. There is a tendency for overestimation with only the lower grade standard being consistently underestimated. The overestimation by the laboratory is not considered significant being generally less than 2 %, and appears to be consistent suggesting an issue with the laboratories internal standards used for calibration.
Table 11-1 Standards used and summary results at Prestea WRP

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<th>Accuracy Performance (% Bias)</th>
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</tbody>
</table>

Figure 11-7 Standard analysis results for ST5359
Figure 11-8 Standard analysis results for ST05/2297

Figure 11-9 Standard analysis results for ST5343
11.6 Screen Fire Assay checks

Additional drilling was carried out on 24 L at Prestea on cross cuts 284 and 287 and a number of the original sample pulps were re-sent for SFA in 2008. The initial results indicated a slight overestimation by the SFA with respect to the original Fire Assay values (“FA”). However, the
sample size used for the initial SFA assay was generally lower than 250 g and in some cases was lower than the original FA sample size. As a result the second half of 54 selected core samples were prepared and sent as duplicate samples for SFA using a 1 000 g charge in order to better define the effect of coarse gold on the final estimates.

The results from the assay of the SFA duplicates produced using a sample charge of 1 000 g produces a significant difference in grade compared with the original sample pulp. The difference for those original pulps with values of 20 g/t or less are similar to those produced by the SFA of the original pulps. However, above 20 g/t the difference in grade between the original pulp and the duplicate core SFA (1 000 g sample) become significant. The average difference indicates a 200 % increase in grade compared with the original sample assay. However, this includes low grade samples where a small difference can exaggerate the percentage difference. For those original assays with a grade above 10 g/t the average increase in grade of the SFA assay is 160 % and for samples above 20 g/t the average increase is 155 %.

![Screen Fire Assay Analysis](image)

**Figure 11-12** Results from screen fire assay analysis based on 250 g (Pulp) and 1 000 g (2nd core) sample charges

The authors consider the difference between the FA and SFA (1 000 g) results to be expected given the high grade and nuggety nature of the West Reef and Main Reef deposits. However, the
extreme grades indicated by some of the SFA assays indicate that it is likely that the current Mineral Resource is being affected by a bias which could lead to an underestimation of the grade of the Prestea deposits based on the current sampling regime.

11.7 Conclusions

The QAQC results for the Prestea deposits show a high degree of reproducibility for those samples above the cut-off grade of the deposit. There is a higher degree of variability in the lower grade samples indicative of the high grade nature of the WR deposit and the nuggety nature of the mineralization. Gannet standard values indicate good quality control and a high level of accuracy in the laboratory. However the lack of high grade standards means that the accuracy of the high grade assay results is something which should be checked but is not considered material to the final Mineral Resource estimate for the WRP.

The Screen Fire Assay work demonstrates that, although coarse gold may be present in the higher grade areas of the Prestea deposit, it is unlikely to have a significant effect on the overall grade interpolation. However, the authors of this report recommend that future SFA work is carried out using a suitable sample size of at least 250 g in order to improve confidence in the estimates provided.

The authors also considers that the samples used for the standard analysis to date have average grades either at or lower than the COG used for modelling the deposit (and at which it will ultimately be mined), more suitable sample intersections be used for duplicate and replicate analysis in the future.
12 Mineral Processing and Metallurgical Testing

(Item 13)

12.1 Introduction

This section summarizes the test work program carried out between November 2012 and April 2013 by SGS, Lakefield, Canada (SGS), under the supervision of SRK Consulting (UK) Ltd. The test program was prepared to support the mechanized mining option used in the West Reef feasibility study published in June 2013. As such, the composite samples referred to in this test work assume an average dilution of 42% from footwall and hangingwall waste rock. This PEA is based on a shrinkage mining method with an expected dilution of less than 20%. There has been no additional testwork carried out with this lower dilution and all the results here relate to 42% dilution. It is expected that the reduced dilution will not negatively affect the results reported here.

The testwork was designed to study the response of the samples to:

- Gravity recovery;
- Preg-robbing potential;
- Cyanide leaching.

The samples used were 62 individual drill core samples. From these a further six composite samples were compiled to provide a better representation of the of the blend that would result from mining.

12.2 Geotechnical Testing of Cyanidation Residues

The average Specific Gravity of the Leach Tailings was determined as 2.83. The Particle sizing was determined to be 100% - 425 micron with an average $P_{80}$ of 75 micron. It is further defined as 24% Sand; 71% Silt and 5% Clay.

The Liquid Limit (LL) is defined as the moisture content (m/m) at which soil begins to behave as a liquid material and begins to flow. The LL was determined to average at 24%.

12.3 Results of Metallurgical Testwork

The metallurgical testwork program was conducted on the six composite samples and included preg-robbing bottle roll tests, gravity separation and cyanidation bottle roll tests of the gravity tailing with and without the addition of carbon.

12.3.1 Preg-robbing Testwork

Procedure

One kilogram samples of ground material (target 75 μm) from each composite were pulped to 40% solids with a 10 mg/L synthetic gold stock solution in glass bottles. The pH was adjusted to 10.5 – 11 with NaOH and the bottles placed on the rolls for a 24 hour period. Intermittent solution samples were taken at 1, 3, 6 and 24 hours for gold analysis.
Results
All the composites showed mild preg-robbing potential when the results are viewed holistically, with composites 5 and 6 indicating strong preg-robbing results.

12.3.2 Gravity Separation Testwork

Procedure
The potential for gold recovery by gravity was evaluated for all six composite samples at grind sizes of approximately 150 μm and 75 μm. The gravity separation tests were completed using 2 kg test charges. A Knelson MD-3 concentrator was used as the primary gravity gold recovery unit. The Knelson concentrates were recovered and further upgraded by treatment on a Mozley mineral separator.

Approximately 0.1% of the feed weight was recovered in the final Mozley concentrate. The Mozley concentrates were assayed to extinction for gold by standard fire assay methods. The Mozley and Knelson tailings were combined, blended and divided into representative tests charges for downstream cyanidation tests.

Results
Gold recovery for the six samples ranged from 52% to 90% and the gold grades ranged from 1,400 g/t to 61,128 g/t Au. These high recoveries indicate that a gravity separation circuit should be included in the process flowsheet.

12.3.3 Cyanidation Testwork

Procedure
Gravity tailing samples were reground, targeting a P_{80} of 75 μm. Each sample was pulped with water to 40% solids in a glass bottle. The pH was adjusted to 10.5 to 11 with hydrated lime and 0.5 g/L NaCN was added to each bottle. The pH and NaCN levels were maintained throughout the leaching period. The bottles were placed on the rolls for 48 hours.

Intermittent solutions samples were taken at 8 and 24 hours and submitted for gold assay to monitor the gold dissolution rate. After 48 hours the pulps were filtered and the pregnant solutions were collected. The residues were washed well with fresh water and the washes were discarded. The solutions were submitted for gold assay and the residues were submitted for duplicate gold assays.

CIL bottle roll tests were also completed on the gravity tailing samples with the addition of carbon. The same conditions as above were applied, with the exception of the addition of 10 g/L activated carbon. After 48 hours the carbon was screened from the pulp and the samples were filtered. All products were submitted for gold assay.

Results
Tests without the addition of carbon indicated the following:

- The gold extractions by cyanide leaching ranged from 18% to 90% with residues grades from 0.6 g/t to 1.6 g/t Au;
- Composites 4 and 6 had the lowest recoveries, 49% and 18%, respectively;
• Overall extraction/recovery of gold (gravity + gravity tailing cyanidation) ranged from 72% to 98%;
• Composite 6 had the lowest overall recovery at 72%;
• Reagent consumptions ranged from 0.14 – 0.90 kg NaCN/t of cyanide feed and 0.30 – 0.72 kg CaO/t of cyanide feed;
• Composites 1, 4 and 6 displayed strong preg-robbing characteristics as shown by the decrease in extraction over the 48 h leaching period. Hence the 18% minimum recovery achieved in this non-carbon leach test. This confirms the need for a CIL process over a CIP.

The results from the tests with the addition of carbon indicated the following:
• The gold extractions by cyanide leaching with carbon addition ranged from 70% to 96% with residue grades from 0.28 g/t – 0.54 g/t Au;
• Composite 6 had the lowest recovery of 70%;
• Overall extraction/recovery of gold (gravity + gravity tailing cyanidation) ranged from 89% – 99%;
• Reagent consumptions ranged from 0.25 – 0.83 kg NaCN/t of cyanide feed and 0.28 – 0.81 kg CaO/t of cyanide feed;
• The residual gold grade decreased with the addition of carbon in all cases, indicating that the samples contain preg-robbing carbonaceous matter. The gold will preferentially load onto the activated carbon during leaching.

12.4 Conclusions

The metallurgical testwork results indicate that the Prestea WRP material is non-refractory and not significantly preg-robbing, with high Au recoveries reported under both CIP and CIL operating conditions, following the removal of the gravity recoverable component.

The results of the testwork on the Prestea WRP samples indicate a relatively free-milling material, with a reasonably high gravity-recoverable component, and a relatively low preg-robbing potential. The proportion of gravity recoverable gold in the WRP material is relatively high, at approximately 80%. The cyanidation leach kinetics were also relatively rapid. The reagent consumptions in leaching – cyanide and lime – were reasonable and relatively low respectively.

A CIL leach circuit is recommended as process route and total recovery is expected to be 90%.
13 Mineral Resources (Item 14)


The GSR exploration team was responsible for the modelling portion of the resource estimate exercise which included all topographic surfaces, weathering surfaces, geological and grade wireframes. SRK Consulting (UK) Inc. was commissioned to construct a mineral resource model with estimated gold grades for the Prestea Underground deposit, the methodology of the estimate is described in this section and was authored by Dr John Arthur. The mineral resource statement was conducted by GSR under the supervision of Mitch Wasel, a Qualified Person pursuant to NI 43 101.

This section describes the Mineral Resource estimation methodology and summarises the key assumptions considered for the estimate. In the opinion of GSR, the Mineral Resource estimate reported herein is a reasonable representation of the global gold Mineral Resource found at the Prestea Underground deposit given the current level of sampling. The Mineral Resources have been estimated in conformity with generally accepted CIM “Estimation of Mineral Resource and Mineral Reserves Best Practices” guidelines and are reported in accordance with the Canadian Securities Administrators’ NI 43-101. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resource will be converted into Mineral Reserve.

13.1 Introduction

The Prestea Underground Mine consists of a number of separate deposits (reefs) some of which have been mined historically and some which are considered exploration targets. The following figure shows the currently defined outlines of the principal deposits in relation to the Plant North Pit. This open pit targeted the Main Reef (“MR”) and its subsidiary Footwall Reef (“FR”). Data validation and additional underground and surface drilling carried out during the period December 2005 to December 2007 led to the development of updated geological models for the MR, FR and the West Reef (“WR”) and those Mineral Resources contained within the Shaft Pillar (“SP”). The historic underground operations were concentrated on the MR and WR. GSR have conducted a detailed exploration campaign along the length of the MR deposit but this has largely concentrated on assessing the potential for further open pit operations (within 200 m from surface).

A portion of the MR, FR and WR Mineral Resource has been classified as Indicated based on the continuity of grade and the distribution of the recent drilling.

The West Reef deposit which lies to the south of the Central Shaft area has been worked historically (Figure 13-1). Production records and historic sampling identified this deposit as having good prospects for future exploitation as an underground resource.
13.2 Data Sources

SRK were provided with GEMS project directories containing the Prestea data. These directories contain the relevant drill hole databases, geological wireframes, oxidation and topographic surfaces and Block Model parameters. Additional information was provided as Excel spread sheets documenting QAQC data and results of density determinations.

13.3 Basic Statistical Tests

Statistics were produced for the various reef domains present at Prestea. The results presented here are based on the horizontal thickness reef composite values and therefore the individual composites lengths vary depending on the reef thickness. The MR, FR and HW statistics are mostly from surface diamond drilling. The WR statistics are derived from underground drill holes and channel sampling collared at locations along 17 and 24 Levels and targeting reef intersections as deep as the 30 L.

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<th>Max (g/t)</th>
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</tbody>
</table>

The above statistics highlight the relatively high grade of the shoots present in the West Reef when compared with the other domains present at Prestea. The following Vertical Longitudinal Projection (“VLP”) shows a representation of the grade distribution within the outlines of the WR domain for each of the reef thickness composites and highlights the relatively continuous nature of the high grade shoots over lateral and vertical distances in excess of 100 m.
Figure 13-3 shows QQ plot and histogram shows an approximate log-normal data distribution in the WR. The WR has a significantly higher grade than the MR and the drilling has confirmed the continuity of these high grades both along strike and down dip between levels.

13.4 Data Capping/filtering
The WR data has not been capped for grade interpolation as the production and use of reef thickness composites has implicitly applied a data cutting and smoothing to the original raw data statistics by combining individual high grades

Studies were conducted looking at capping at 100 g/t and 50 g/t but it was concluded that this would bias the resulting estimates given that the WR appears to be a high grade shoot and there are definite trends or shoots of high grade continuity within the domain (Figure 13-2).

The MR and FR deposits were cut to 30 g/t following studies which showed that high grade outliers are having a material effect on the variance of the data and can adversely affect the quality of the resulting semi-variograms and hence the confidence in the final kriged estimate.
Figure 13-2  West Reef VLP looking west showing the distribution of reef composite grades and preponderance of cross cut sampling on 17 L
D Variographic analysis

13.5.1 West Reef

The WR has been drilled out on relatively close spaced centres and this has resulted in reasonable quality semi-variograms being modelled in the four principle directions within the plane of the deposit which appear to be highlighting a plunge to the mineralization of approximately 42° to the north-west in the plane of the structure with a well-developed
anisotropy of up to 200 m along this plunge and approximately 50 m perpendicular to this to the south-west.

The following plot Figure 13-4 shows the experimental semi-variograms produced in the principle and secondary directions derived from analysis of the average dip and strike of the WR domain. The plane of the orebody was assumed to lie on a strike of 005° with a dip of 60° to the west. The best directions for continuity of grade were calculated towards 334° along a plunge of 42° to the north-west in the plane of the structure. The secondary direction to this is plunging 34° to the south-west (208°) in the plane of the structure.

Modelling of the experimental semi-variograms produced an anisotropic model with a significantly higher range in the principal direction of almost 200 m compared to only 50 m in the secondary direction. This is partly a result of the data distribution as shown in Figure 13-2 where the majority of the reef intersections occur in the northern upper area of the domain. Additional sampling in the 24 and 30 Levels in the south of the WR deposit may result in better quality semi-variograms in the secondary direction.

Figure 13-4 Directional semi-variograms produced in the principal directions in the plane of the West Reef deposit with associated model semi-variograms
Table 13-2  Semi-Variogram Modelling Results for the West Reef deposit

<table>
<thead>
<tr>
<th>Domain</th>
<th>Structure</th>
<th>Variance</th>
<th>Principle Direction plunge 42° to 334°</th>
<th>Secondary Direction plunge 34° to 208°</th>
</tr>
</thead>
<tbody>
<tr>
<td>WR</td>
<td>nugget (C₀)</td>
<td>0</td>
<td></td>
<td></td>
</tr>
<tr>
<td>WR</td>
<td>spherical (C₁)</td>
<td>195</td>
<td>100m</td>
<td>20 m</td>
</tr>
<tr>
<td>WR</td>
<td>spherical (C₂)</td>
<td>140</td>
<td>195m</td>
<td>50 m</td>
</tr>
</tbody>
</table>

13.5.2 Main Reef and Footwall

The MR semi-variograms produced good directional information and the maximum modeled range was 100 m using a lag of 15 m. The primary direction was modelled as a shallow plunge to the north roughly along strike of the orebody. The down dip direction also produced relatively good quality semi-variograms due to the increased amount of data available from the underground drilling. A maximum range of 55 m was modeled in this direction.

The footwall structure has sufficient sample intersections from the underground drilling to allow production of good quality semi-variograms for this structure. The models exhibited an almost isotropic nature with only a minor extension of the range in the along strike direction. Once again the principle direction was modeled with a range of approximately 100 m.

![Figure 13-5 MR semi-variogram along strike (N8) and down-dip (N112) directions. Gaussian anamorphosis transformation](image-url)
Figure 13-6 FR semi-variogram along strike (N3) and down-dip (N93) directions. Gaussian anamorphosis transformation

Table 13-3 Semi-Variogram Modelling Results for the Main Reef deposit

<table>
<thead>
<tr>
<th>Domain</th>
<th>Structure</th>
<th>Variance</th>
<th>Principle Direction plunge 00° to 005°</th>
<th>Secondary Direction plunge 60° to 275°</th>
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</thead>
<tbody>
<tr>
<td>MR</td>
<td>nugget (C₀)</td>
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<td></td>
</tr>
<tr>
<td>MR</td>
<td>spherical (C₁)</td>
<td>10.8</td>
<td>90m</td>
<td>44m</td>
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<td>MR</td>
<td>spherical (C₂)</td>
<td>21.3</td>
<td>105m</td>
<td>57m</td>
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Table 13-4 Semi-Variogram Modelling Results for the Footwall deposit

<table>
<thead>
<tr>
<th>Domain</th>
<th>Structure</th>
<th>Variance</th>
<th>Principle Direction plunge 00° to 005°</th>
<th>Secondary Direction plunge 60° to 275°</th>
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</thead>
<tbody>
<tr>
<td>FR</td>
<td>nugget (C₀)</td>
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<td>spherical (C₁)</td>
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<td>9.2m</td>
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<td>FR</td>
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<td>103m</td>
<td>52m</td>
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</table>
13.6 Block Model Grade Interpolation

Grade interpolation has been performed using ordinary kriging for all underground domains at Prestea. The MR, WR, FR and ST domains were interpolated directly, the HW and SP domains were filled using the geostatistical parameters derived from the MR analysis owing to lack of sufficient data points within these domains. The FW and HW are parallel structures to the MR structure and are assumed to form part of the MR mineralized domain and thus share its grade distribution characteristics. The WR is regarded as a separate structure from the MR to the north, given its location in the hangingwall of the main shear structure and the higher grades found within the WR deposit. Deposit contacts are interpreted as hard boundaries for the individual domains for the purpose of grade interpolation. As a result the composites used for the interpolation are clipped to the boundaries of the wireframe model.

Blocks were interpolated using an Ordinary Kriging algorithm in the Geovariance Isatis® software. Block size is 12.5 m x 25 m x 25 m (XYZ) and kriging was carried out in two phases. The first search was based on the ranges estimated from the experimental semi-variogram models and the second search was expanded to allow all remaining blocks in the model to be assigned a grade. Only those blocks filled in the first search pass were assigned an Indicated classification category, subject to additional kriging quality parameter results being met.

Table 13-5 WR Kriging Search Parameters

<table>
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<th>Radius (m)</th>
<th>Samples</th>
</tr>
</thead>
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<td></td>
<td>Strike</td>
<td>Dip</td>
</tr>
<tr>
<td>a</td>
<td>200</td>
<td>100</td>
</tr>
<tr>
<td>b</td>
<td>500</td>
<td>500</td>
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</table>

Table 13-6 MR Kriging Search Parameters

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</thead>
<tbody>
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<td>100m</td>
<td>60m</td>
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<tr>
<td>b</td>
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</table>

Table 13-7 FR Kriging Search Parameters

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</thead>
<tbody>
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<td>Strike</td>
<td>Dip</td>
</tr>
<tr>
<td>a</td>
<td>100m</td>
<td>60m</td>
</tr>
<tr>
<td>b</td>
<td>500m</td>
<td>500m</td>
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</table>
Table 13-8 Block Model Dimension Parameters (lower left corner)

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<th>Value</th>
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</thead>
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</tr>
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<td>y</td>
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<td>25</td>
</tr>
<tr>
<td>z</td>
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<td>56</td>
<td>25</td>
</tr>
<tr>
<td>Rotation</td>
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<td></td>
</tr>
</tbody>
</table>

Figure 13-7 shows a representation of the grade distribution in the Prestea deposits viewed from the Footwall. Figure 13-8 is a more detailed plot of the WR deposit showing the relative distribution of drill holes used for the interpolation. The plunging structure can be clearly seen as zones of high grade dipping to the north.

### 13.7 Resource Classification

Mineral Resource classification is typically a subjective concept, industry best practices suggest that resource classification should consider both the confidence in the geological continuity of the mineralized structures, the quality and quantity of exploration data supporting the estimates and the geostatistical confidence in the tonnage and grade estimates. Appropriate classification criteria should aim at integrating both concepts to delineate regular areas at similar resource classification.

GSR is satisfied that the geological modelling honours the current geological information and knowledge. The location of the samples and the assay data are sufficiently reliable to support resource evaluation. The sampling information utilised for the resource classification was acquired primarily by DD core, RC and Reverse Air Blast (“RAB”) drilling on sections spaced at 25 to 50 m.

Classification was initially based on calculating a slope of regression ($Z/Z^*$) value for individual blocks. All blocks filled using the wider search (search b) are assigned an Inferred classification category. Wireframes were constructed around blocks with a slope of regression (SL) value of generally greater than 0.8 and these blocks were assigned an Indicated category. However, the construction of the classification wireframe did not strictly adhere to the outline of the blocks with a SL value of >0.8, and there are areas where blocks of lower reliability are included in the Indicated wireframe for the purposes of continuity and where visual examination of the model in conjunction with the drill hole intercepts indicated that a high degree of confidence could be applied rather than relying solely on the statistical variable. At the WR, the large number of mineralisation intersections both along strike and down dip contribute to the increased confidence in the block grades. At MR, the number of blocks with a $Z/Z^*$ value of greater than 50 % was low and the classification here has been based largely on the visual confirmation of grade continuity from analysis of drillholes sections.
Figure 13-7  Long Section looking west from the footwall, showing schematically the block grade distribution of the Prestea deposits highlighting the relatively high grade of the West Reef
Figure 13-8 VLP view from the east (footwall) of the West Reef Block Model showing the distribution of Au grade values after kriging and the available drill hole traces
Figure 13-9 Histogram of Slope of Regression values for the West Reef after kriging using the short range search radius
Figure 13-10 VLP view from the east (footwall) of the West Reef Block Model showing the distribution of Slope of Regression values after kriging
Figure 13-11  VLP view from the east (footwall) of the West Reef Block Model showing the distribution of Block Classification categories.
13.8 Mineral Resource Statement

The “reasonable prospects for eventual economic extraction” requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the Mineral Resources are reported at an appropriate cut-off grade, taking into account extraction scenarios and processing recoveries. In order to meet this requirement, GSR considers that major portions of the Prestea deposits are amenable for underground extraction.

The Mineral Resource statement has been prepared using a block cut-off grade of 4.74 g/t based on a US$ 1,400/oz gold price and appropriate costing data to produce a Mineral Resource which matches the requirements that the deposit should have “reasonable prospects for economic extraction” as defined by the CIM.

The statement was prepared by Mr. Mitch Wasel who is a Qualified Person pursuant to National Instrument 43-101. Mr Wasel is employed by GSR as Vice President Exploration and is not independent of the company. The effective date of the Mineral Resource Statement is 31 December 2013. The Mineral Resource Statement for Prestea Underground, is given below in Table 13-9.

In declaring the Mineral Resources for the PUG deposits, GSR notes the following:

- The identified Mineral Resources in the block model are classified according to the CIM definitions for Measured, Indicated and Inferred categories and are constrained by a block cut off grade calculated using a gold price of US$ 1,400/oz and below the end of year topographic surface. The Mineral Resources are reported in-situ without modifying factors applied;

- The Mineral Resource models have been depleted using appropriate topographic surveys and underground stope data, to reflect mining until the 31 December 2013;

- The Mineral Resources were estimated using block models. The composite grades were capped where this was deemed necessary, after statistical analysis. Ordinary Kriging was used to estimate the block grades. The search ellipsoids were orientated to reflect the general strike and dip of the modelled mineralisation;

- Block model tonnage and grade estimates were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014). The basis of the Mineral Resource classification included confidence in the geological continuity of the mineralised structures, the quality and quantity of the exploration data supporting the estimates, and the geostatistical confidence in the tonnage and grade estimates. Three-dimensional solids were modelled reflecting areas with the highest confidence, which were classified as Indicated Mineral Resources;

- All figures are rounded to reflect the relative accuracy of the estimate. All composites have been capped where appropriate; and

- Mineral Resources are not Mineral Reserves and do not necessarily demonstrate economic viability.
**Table 13-9 Prestea Underground Mineral Resource Statement 31 December 2013**

<table>
<thead>
<tr>
<th>Orebody</th>
<th>MEASURED</th>
<th></th>
<th></th>
<th>INDICATED</th>
<th></th>
<th></th>
<th></th>
<th>INFERRED</th>
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<td>Grade</td>
<td>Content</td>
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<tr>
<td></td>
<td>(kt)</td>
<td>g/t Au</td>
<td>(koz Au)</td>
<td>(kt)</td>
<td>g/t Au</td>
<td>(koz Au)</td>
<td>(kt)</td>
<td>g/t Au</td>
<td>(koz Au)</td>
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</tr>
<tr>
<td>Main Reef</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>197</td>
<td>7.43</td>
<td>47</td>
<td>2,227</td>
<td>6.89</td>
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<td>Footwall</td>
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<td>-</td>
<td>363</td>
<td>9.19</td>
<td>107</td>
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<tr>
<td>Total</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>1,357</td>
<td>14.51</td>
<td>633</td>
<td>3,289</td>
<td>8.02</td>
<td>848</td>
<td></td>
</tr>
</tbody>
</table>

* The section 15 of this report, mining method, considers the West Reef resources only.
14 Mineral Reserve Estimation (Item 15)

There are no Proven or Probable Mineral Reserves estimated in this PEA.

Recent Mineral Reserve estimates have been prepared by:

- GSR – Year-end reserve estimate effective December 31, 2013.

Both of these recent reserve estimates relate to mechanized mining of the West Reef resource and are superseded by this PEA.
15 Mining Methods (Item 16)

15.1 Introduction

This section summarizes the mine design and planning work completed to support the preliminary economic assessment. The underground mine planning was a collaboration between GSR and SRK. This work was supervised by Dr. Martin Raffield, PEng of GSR, the Qualified Person taking professional responsibility, and prepared by Mr. Benny Zhang, MEng, PEng of SRK Consulting (Canada) Inc. and Dr. Martin Raffield. The shrinkage mining geotechnical assessment for the proposed underground mine was undertaken by Dr. Ben Westgate, CEng (MIMMM), of SRK Consulting (UK) Limited.

The objective of this PEA is to evaluate the potential economic viability of the PUG West Reef Project (WRP) at a scoping level when shrinkage mining is employed.

The WRP consists of an underground mine with a production mine life of five years (and a modified Bogoso processing plant). The maximum mill feed rate is set at 175,000 tonnes per annum (t/a), or a nominal 500 tonnes per day (t/d).

Mine grid elevations quoted are metres above sea level plus 5,000 metres. The terms of four digit levels, sublevels, and elevation are used interchangeably to describe underground mining levels. In some cases, historical two digit level names also used to describe mine level, such as 17L is the same as 4475L and 24L is the same as 4165L.

15.2 Mine Geotechnical

SRK UK has been commissioned to undertake a series of PUG mining geotechnical studies since 2007, namely a PEA level geotechnical investigation in 2007 and a feasibility study (FS) level geotechnical investigation for mechanized cut and fill (MCF) mining in 2013 (SRK UK 2013).

The current PUG shrinkage method PEA geotechnical assessment was based on previous field geotechnical investigation, laboratory testing results, and characterized rock geotechnical conditions. The detailed geotechnical investigation program can be found in the SRK UK 2013 technical report. The conclusion and recommendations from the shrinkage method PEA geotechnical assessment are summarized in the following sub sections.

On August 26, 2014, Dr. Martin Raffield visited a historical WR stope on the existing 17L, immediately above the proposed mining area. Dr. Raffield observed that:

- Stope raises are in good condition with no indication of spalling, progressive failure or squeezing.
- An open stope (15 m high x 25 m long) has been standing for 15 years following the drawdown of mined materials. It was clearly visible that only the quartz vein had been mined and that graphitic material was on the hangingwall and footwall. In some places the mining width was only 1 m. There was no indication of significant hangingwall or footwall failure of the graphitic material.

15.2.1 Rock Mass Classification

SRK UK conducted a feasibility level geotechnical study in support of the 2013 Prestea West Reef mechanized cut and fill mining study. Figure 15-1 shows the location of the West Reef
relative to the Prestea Main Reef, and a summary of rock mass clarification. It should be noted that a number of boreholes completed after the FS have not been included in the geotechnical interpretation presented herein and need to be interpreted in the next phase of study.

It was concluded from the 2013 study that the WR rock mass is generally strong and competent. However, there are a number of sheared and broken zones (interbedded greywacke and phyllites dipping at approximately 65 – 70° west) adjacent to and within the mineralization that will control the size and stability of the stopes as well as the plant feed dilution. The Q value of the mineralization and adjacent hangingwall and footwall materials, taking into account the presence of the weak zones, has a value of 6.0, making the rock mass of fair quality. This compares with a Q value in excess of 100 (extremely good rock mass) for the surrounding non-broken and non-sheared rock mass.

15.2.2 Shrinkage Stope Stability Assessment

SRK proposed preliminary shrinkage mining layouts (see details in Section 15.3 Proposed Mining Method) where the shrinkage stope total height would range from 35 to 40 m vertically from sill floor to sill floor, and 50 to 60 m along strike from access raise to access raise, with an open stope height of 25 to 30 m vertically and 42 to 52 m along strike. Rock bolting of the stope walls would be into the immediate hangingwall and footwall, in addition to wooden stulls used to prop the open void.

Based on the proposed shrinkage stope layouts, Table 15-1 and Table 15-2 show the stability graph analysis (Potvin 1988, Nickson 1992) results for the stope back and walls, respectively.
### Table 15-1: Stability Graph Results - Stope Back

<table>
<thead>
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<th>OB Thickness</th>
<th>Stope Span (m)</th>
<th>Hydraulic Radii (m)</th>
<th>N</th>
<th>Key</th>
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### Table 15-2: Stability Graph Results - Stope Walls

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<th>Hydraulic Radii (m)</th>
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<th>Unsupported Transitional</th>
<th>Stable with Support</th>
<th>Supported Transitional</th>
<th>Unstable</th>
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</tbody>
</table>
Figure 15-1: Location of West Reef from the Main Reef and 2013 Geotechnical Investigations
15.2.3 Dilution Assessment and Mining Sequence

Figure 15-2. For the stope design, the stope width equals the vein width, and a minimum mining width of 1.2 m and a maximum stope length of 60 m are applied. The stope heights (sill floor to sill floor) of the lower four levels are 35 m and the stope heights of the upper levels are 40 m. Sills will follow the vein. If vein widths are less than 2 m, a minimum drift width of 2 m was applied. The gap between the north part of the stopes and the south part of the stopes is a permanent central regional pillar. The current 24L (4165L) will be the main rock haulage level with material and supplies handled on 24L and 17L (4475L). Development will start from footwall development on both 24L and 17L as well as on-vein development on 24L (south side). Development will continue with the 17L decline downwards and on to the vein lateral development, with 24L on-vein lateral development (north side) and with the incline upwards. The central ventilation raise and rockpass will be given higher priority for a faster connection between 17L and 24L.

Figure 15-2: Shrinkage Stoping in West Reef
15.2.4 External Dilution Assessment

From a geotechnical perspective, external dilution could be in the range of 5 – 15% based on the vein thickness, ground conditions and stability number. This could extend to 20 – 25% where there are thicker graphitic schist units or sheared material. The current design and mining method permit the use of mechanical rock bolts or split set bolts and wooden stulls/props within the stopes to help minimize dilution.

15.2.5 Stope Sequencing

The global mining sequence for the shrinkage stopes will need to be considered in relation to the central rib pillar. A suggested mining direction would be a top down approach, while extending outwards from the central rib pillar (Figure 15-3). Advances in the stopes from top to bottom should be at approximately 45°. This is the most practical solution to minimize stress build up in the stopes.

A mining direction towards the rib pillar that will allow for partially recovered stope sill and rib pillars may cause localized stress increases, resulting in stability issues. The stresses involved should be understood in greater detail. This should be accomplished through 3D finite element modelling of the orebody and mining direction to investigate potential stress build up within the rib pillar and mined out shrinkage stopes.

Figure 15-3: Proposed Mining Direction at West Reef
15.3 Proposed Mining Methods

15.3.1 Mining Context

The relevant characteristics of the Prestea West Reef from a mining method selection perspective are provided below.

- It is a steeply dipping (65 – 70°) laminated quartz vein type gold deposit.
- The WR is within a graphitic shear zone running parallel to and in the hangingwall of the Prestea Main Reef.
- It consists of a single continuous vein without splays or pinch-out.
- The vein has an average in situ true thickness of 1.7 m, ranging from 0.8 to 4.5 m; while in the targeted mining area, the average true in situ thickness of the vein is approximately 1.95 m, ranging from 1.0 to 4.5 m.
- It is a relatively high grade deposit with good continuity at a 7 grams of gold per tonne (g/t gold) cut-off grade.
- The top of the mineralization modelled is at 4475L, approximately 575 m below surface (reference to Central Shaft collar), and the bottom of the mineralization in the block model is at 3880L, approximately 1170 m below surface.
- It is a past mining project, having more than 100 years of mining history, having employed a number of mining methods, essentially all non-mechanized mining methods with loading chutes such as shrinkage, and inclined rill and fill.
- Some of the existing development on 4475L (current 17L) and on 4165L (current 24L) has reached to the WR mineralization zones.
- Extensive historical development and infrastructure are in place throughout the mine site.
- There is a lack of backfill materials at the Prestea mine site.
- The mineralized materials at Prestea WR are non-cohesive, non-sulphide rich and non-pyrite mineralization, containing no or minimal radiation emitting minerals.

15.3.2 Mining Method

The proposed mining method is traditional shrinkage mining with the application of modern rock bolts and traditional wood stulls/props to support the stope walls in order to maintain stope stability and control waste dilution.

The proposed shrinkage mining method is an alternative to the 2013 FS proposed mechanized cut-and-fill mining method (MCF) for the Presteа underground mine re-opening. Compared with MCF, the shrinkage mining method is a lower capital investment method with minimal equipment and backfill requirements; the on-vein lateral and vertical development can serve for resource definition; and it can provide opportunity to install ground support to control stope stability and reduce dilution, partly due to the lower requirement for minimum mining width and partly due to installed ground support and blasted materials acting also as a temporary ground support.
The proposed typical shrinkage stope layout is shown in Figure 15-4. In the typical stope layout, the reef is divided into blocks approximately 60 m long along strike and 40 m high vertically for upper level stopes or 35 m high vertically for lower level stopes. Timbered access raises sized at 1.5 m long by stope width will be maintained at each end of the block from the level below to give dual access to the working area and provide a ventilation circuit through the 2 m high by stope width stope connection drifts spaced 6 m vertically along the raises from drift floor to drift floor.

Normally, these access raises are driven ahead of stoping to the level above, but they can be carried blind with the stope in certain circumstances but this is not preferred. The draw bells/raises are developed along the sill haulage way spaced 6 m center to center close to the footwall side, which has a direct influence on the effectiveness of the mucking operations and the in-stope muck pile flattening work.

A wooden chute will be installed under each draw bell raise to gravity-load tracked mine cars. In order to maintain a suitable in-stope working space and smoothly drawdown blasted materials, a minimum mining or stope width of 1.2 m will be required. To enhance mucking efficiency and safety, and to increase stope mining recovery, lightweight reusable vibration assisted chutes can be installed as an alternative to wooden chutes.

The back of the stope will be carried in a stepped configuration, with each stepped-section equivalent to the length of one blast, and will be drilled out using airleg mounted handheld drills. Upholes will be 1.8 m to 2.0 m long at between vertical and 65° to the horizontal, giving a vertical advance of 1.6 m. Ground support, including 0.9 – 1.2 m long mechanical rock bolts at approximately 5 t pretension and wooden stulls/props can be installed at the same time as the blasthole drilling activity preferably on a different section.

The stope activities in the production stage include water spraying, scaling, flattening, secondary in-stope breaking, ground support installation, blasthole drilling, explosive loading, production mucking (approximately one-third of the total materials blasted), and high pressure water spraying and blasting. One of the best shrinkage mining practices is to separate the drilling (and other in-stope activities) and mucking work into different shifts to reduce bottom mucking and in-stope activities coordination requirement and to minimize safety related issues.

Uniform production mucking before blasting has a number of advantages, which include reduced possibility of choking and hang-up in blasted materials, easier ventilation after production blasting and reduced flattening work. Stope ventilation will mainly use mine wide ventilation circuit with assistance of axial type auxiliary fans when needed, especially after production blasting.

After the final lift of production blasting, top sill ground support will be installed to enhance sill pillar stability.
Figure 15-4: Typical Shrinkage Stope Layout
(SRK 2014)
Although broken materials retained in a shrinkage stope can be used as repository to balance daily production, rapid final mucking is preferred to reduce wall stability deterioration and sloughing which causes dilution and hang-ups during final mucking. After completion of the final mucking, a void management strategy will be implemented, such as sealing, manual wall caving, or backfilling using development waste.

The proposed shrinkage stoping sequence is a top-down and centre-out mining advance. It starts from the upper most level and proceeds downwards, and from the central regional pillar extends to the extremities of either side of the reef (Figure 15-3). An exception will be a few stopes on 4165L mined early in order to improve early production and to create stope voids that can be backfilled with development waste.

Based on local ground conditions and stability of the stope hangingwall and footwall, stope strike length can be adjusted to facilitate dilution control. With the currently proposed mining sequence, all sill and rib pillars will be left in place to support and maintain the sill drift in good condition.

15.4 Potential Plant Feed Estimate

15.4.1 Economic Cut-off Grade

The initial stope design was based on a cut-off grade defined at a gold price of US$1,200 per ounce, a royalty of 5%, a process recovery of 90%, and a site operating cost obtained from GSR’s recent in-house concept study work. An in situ cut-off grade of 7.0 g/t gold was used to define mining shapes in the resource block model. Mining shapes were interrogated with the mine planning software and checked against a cut-off grade of 7.0 g/t gold that includes an allowance for an initial estimated total dilution of 20% at zero grade.

Dilution is defined as waste/mineralization tonnes.

15.4.2 Stope Design

Prestea WR stope design method generally followed these steps:

- Geotechnical assessment to determine stable stope dimensions (refer to section 15.2)
- Selection of mining level spacing and elevations (refer to section 15.2.2)
- Selection of a central regional pillar location where gold grades are relatively low and near the center of the potential mining shapes
- Designing the practical sill shapes (sill wireframes) using 2 m height and vein width with a minimum sill width of 2 m in narrow areas
- Designing the practical mining shapes (stope wireframes) using selected cut-off grades as a guide, and by slicing a geological wireframe between the bottom level sill roof and top level sill floor
- Reporting of in situ quantities inside the mining shapes using block modelling software
- Calculating the in situ true thickness for each stope
- Applying a minimum in situ true thickness criteria of 1.2 m for the stopes in narrow areas
- Evaluating internal dilution
• Applying estimates for external dilution and mining losses
• Categorizing development sill material tonnes and stope material tonnes for scheduling

Shrinkage stopes are generally planned at a 60 m strike length and a height of 40 m for the upper levels (above 4235L) or a height of 35 m for the lower levels (below 4235L). There are some areas where this standard size has been varied including around the perimeter of the Indicated mineral resources.

The mining shapes followed the hangingwall and footwall of the mineralized vein structure as defined by geological wireframes without attempting to trim off any areas below the cut-off grade.

It should be noted that not all Inferred mineral resources that meet the economic cut-off grade are considered in the current mine plan, which is based on GSR’s guidance that the mine plan should not include more than 10% of Inferred mineral resources.

Figure 15-5 shows the PEA designed stopes overlapped with the resource block model by grade distributions where the pink color represents gold grades greater than or equal to the cut-off grade of 7.0 g/t and the yellow color represents gold grades below this cut-off grade.

Figure 15-5: PEA Designed Stopes versus Grade Distribution
(All Resource Categories, Looking East)
Figure 15-6 shows the PEA designed stopes overlapped with resource blocks by resource category where the purple color represents Indicated mineral resources and the green color represents Inferred mineral resources (no Measured mineral resources).

From Figure 15-5 and Figure 15-6, it can be seen the included Inferred mineral resources are near the edge of the Indicated mineral resources and can form regular shrinkage stopes together with the Indicated mineral resources that are included in the PEA mine design.

**Figure 15-6: PEA Designed Stopes versus Resource Categories**

(Looking East)

### 15.4.3 Dilution Assessment & Mining Recovery Parameters

As discussed in Section 15.2.3, the external dilution based on the width of the mineralization from a geotechnical perspective could be in the range of 5 – 15% based on the ground conditions and stability number.

The internal dilution estimate was based on sill and stope wireframes enclosed quantities with an adjustment of minimum mining widths of 2.0 m for sills and 1.2 m for stopes.

For the external dilution estimate, SRK assumed:
• A 0.25 m layer (sum of hangingwall and footwall) of stope wall rock with no grade would be mined with each stope, and an additional 3% of sloughing during the final mucking was added.

• 10% overbreak for sill development external dilution with no grade

Internal dilution on the total plant feed averages 1%. External dilution on the total plant feed averages 18%. SRK makes the conservative assumption that both internal dilution and external dilution carry no grades. Table 15-3 shows the details of the dilution assessment results. Dilution is defined as waste tonnes/mineralization tonnes.

### Table 15-3: Dilution Estimation

<table>
<thead>
<tr>
<th>Name</th>
<th>Internal Dilution</th>
<th>External Dilution</th>
<th>Subtotal</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sill</td>
<td>9.1%</td>
<td>25.7%</td>
<td>34.8%</td>
</tr>
<tr>
<td>Stope</td>
<td>0.4%</td>
<td>17.0%</td>
<td>17.4%</td>
</tr>
<tr>
<td>Total</td>
<td>0.9%</td>
<td>17.5%</td>
<td>18.4%</td>
</tr>
</tbody>
</table>

#### 15.4.4 Potential Plant Feed for the Mine Plan

Table 15-4 shows the conversions of in situ mineralized materials contained inside the planned mining shapes to plant feed. A gold price of US$1200 per ounce was used.

Table 15-5 shows the distribution of plant feed tonnes and grades by level. Initial production is scheduled from 4435L and 4165L. The bottom portion of the mine below the 4165L main level (current 24L) is composed of stope levels 4130L and 4095L, containing only 11.9% of the plant feed tonnes and 11.7% of the contained gold ounces.

The plant feed is distributed in the mining levels as shown in Figure 15-7.

![Figure 15-7: Prestea West Reef Mining Area Long Section Vertical Projection (Looking East)](image-url)
### Table 15-4: Estimation of Underground Plant Feed

<table>
<thead>
<tr>
<th>Name</th>
<th>Tonnes (000’s)</th>
<th>Grade (g/t)</th>
<th>Ounces (000’s)</th>
<th>Total Dilution %</th>
<th>Total Dil’d Tonnage (000’s)</th>
<th>Recovery (000’s)</th>
<th>Grade (g/t)</th>
<th>Ounces (000’s)</th>
<th>Indicated %</th>
<th>Inferred %</th>
<th>Indicated %</th>
<th>Inferred %</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sill</td>
<td>46</td>
<td>17.47</td>
<td>26</td>
<td>42</td>
<td>34.8%</td>
<td>57</td>
<td>100.0%</td>
<td>57</td>
<td>14.14</td>
<td>26</td>
<td>85%</td>
<td>15%</td>
</tr>
<tr>
<td>Stope</td>
<td>670</td>
<td>20.57</td>
<td>443</td>
<td>670</td>
<td>17.4%</td>
<td>616</td>
<td>96.0%</td>
<td>592</td>
<td>17.53</td>
<td>333</td>
<td>91%</td>
<td>9%</td>
</tr>
<tr>
<td>Total</td>
<td>716</td>
<td>20.37</td>
<td>469</td>
<td>713</td>
<td>18.4%</td>
<td>673</td>
<td>96.4%</td>
<td>649</td>
<td>17.23</td>
<td>359</td>
<td>91%</td>
<td>9%</td>
</tr>
</tbody>
</table>

* Figures have been rounded.

** The estimated plant feed is partly based on Inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment based on these mineral resources will be realized.

*** All pillars will be left in place, not counted in the PEA. This will be an upside potential for advanced stage of study or in mine operations.

### Table 15-5: Distribution of Underground Plant Feed by Level

<table>
<thead>
<tr>
<th>Name</th>
<th>Units</th>
<th>4435</th>
<th>4395</th>
<th>4355</th>
<th>4315</th>
<th>4275</th>
<th>4235</th>
<th>4200</th>
<th>4165</th>
<th>4130</th>
<th>4095</th>
<th>Total</th>
</tr>
</thead>
<tbody>
<tr>
<td>Total Tonnage</td>
<td>t</td>
<td>41,500</td>
<td>98,200</td>
<td>83,200</td>
<td>78,800</td>
<td>67,600</td>
<td>60,500</td>
<td>70,000</td>
<td>71,700</td>
<td>45,100</td>
<td>32,000</td>
<td>648,500</td>
</tr>
<tr>
<td>Grade (g/t Au)</td>
<td>g/t</td>
<td>20.77</td>
<td>13.80</td>
<td>16.68</td>
<td>17.42</td>
<td>18.87</td>
<td>20.15</td>
<td>17.61</td>
<td>16.14</td>
<td>18.64</td>
<td>14.77</td>
<td>17.23</td>
</tr>
<tr>
<td>Contained Au</td>
<td>oz</td>
<td>27,700</td>
<td>43,600</td>
<td>44,600</td>
<td>44,100</td>
<td>41,000</td>
<td>39,200</td>
<td>39,600</td>
<td>37,200</td>
<td>27,000</td>
<td>15,200</td>
<td>359,300</td>
</tr>
<tr>
<td>Sill tonnes</td>
<td>t</td>
<td>7,000</td>
<td>7,100</td>
<td>6,200</td>
<td>6,000</td>
<td>5,200</td>
<td>5,500</td>
<td>6,300</td>
<td>7,700</td>
<td>3,400</td>
<td>2,500</td>
<td>56,900</td>
</tr>
<tr>
<td>Stope tonnes</td>
<td>t</td>
<td>34,500</td>
<td>91,100</td>
<td>77,000</td>
<td>72,800</td>
<td>62,400</td>
<td>55,000</td>
<td>63,700</td>
<td>64,000</td>
<td>41,700</td>
<td>29,500</td>
<td>591,600</td>
</tr>
<tr>
<td>Grade (g/t Au)</td>
<td>g/t</td>
<td>22.50</td>
<td>13.84</td>
<td>16.84</td>
<td>17.65</td>
<td>19.17</td>
<td>20.53</td>
<td>17.90</td>
<td>16.53</td>
<td>18.75</td>
<td>15.19</td>
<td>17.53</td>
</tr>
</tbody>
</table>

* Figures have been rounded.
15.5 Underground Mine Model

15.5.1 Underground Mine Layout

Figure 15-8 shows the PEA target WR mineralization in relation to the existing Prestea underground infrastructure and historical mining voids. The PEA will take advantage of existing infrastructure. The Central Shaft will be the main access for hoisting plant feed and waste materials, personnel and supply materials, major fresh air entry, and other mine services. The existing 4165L (current 24L) will serve as the main haulage level. The existing 4475L (current 17L) will be another main level for personnel and supply materials, and temporary haulage level during the mine development stage.

Figure 15-9 shows the mine model of the WR mining area in plan view. In plan the extremities of the planned stoping (cyan colour) area covers a strike length of 635 m.
Figure 15-9: Prestea West Reef Mining Area 3D Mine Model Plan View
Figure 15-10 is an isometric view looking northwest from the footwall side of the WR deposit. The entire development infrastructure is on the footwall side. Colours in the view represent the following:

- Light green – decline/incline
- Dark green – level access drive
- Red – exhaust ventilation raises
- Light brown – rockpass
- Light grey – existing 4475L (current 17L)
- Dark grey – existing 4165L (current 24L)
- Magenta – sill
- Stope shapes are shown in various colours

The first area for stope production will be from the north end of 4165L (current 24L). There are a number of advantages related to starting in this area. First, it is the earliest fully accessed stope for early production; second, the two access raises of this stope, not visible in the figures, can be
used for facilitating waste mucking and ventilation during the second segment development of the incline (4200L to 4235L); third, the mined out area of the stope can be used as permanent waste storage or backfilled during the upper level waste development. For these reasons, there are four stopes planned on 4165L during the preproduction stage (two stopes on each side of the central regional pillar). After these four stopes, production will be stopped on 4165L and it will follow the normal stope production sequence.

The second starting area for stope production will be from the upper most level (4435L) near the central regional rib pillar. From there, it will retreat to out to the extremities and downward. This is the normal mining sequence in the PEA mine plan.

Access to the underground mining area is planned by decline from 4475L (current 17L) and incline/decline from 4165L (current 24L). The decline and incline will be connected on 4315L for the upper mine (above 4165L), and on 4165L for the lower mine (below 4165L). Winders will be installed on 4475L, 4315L, and 4165L for each leg of the decline/incline. The upper mine decline and incline will be used for services only during the production stage, while the decline of the lower mine will also be used to hoist plant feed and waste materials.

The decline/incline is designed at +/-28° gradient and has dimensions of 2.7 m width by 2.7 m height, these being selected based on the requirements of installing 600 mm gauge tracks and wooden stairways in the decline/incline.

The decline/incline layout was selected as the major access in the target mining area due to its services function being only for a non-mechanized mining method and no mobile equipment was to travel up and down. All the economic materials and waste will be mucked from stopes or development faces and then dumped to the rockpass down to 4165L, where materials will be loaded to mine cars and track hauled to the Central Shaft, then hoisted to surface. For the lower mine, materials will be hoisted to 4165L through a decline, then marshalled into the 4165L main level track haulage fleet.

The exhaust ventilation raise and the rockpass are located on the footwall side and near the central regional rib pillar area. The ventilation raise will be an Alimak raise and sized at 2.4 m by 2.0 m where a ladderway will be installed as a second egress. The Alimak raise rockpass is sized at 1.5 m by 1.5 m.

The exhaust raise will connect to the 4475L (current 17L) by a newly developed ventilation access drift, and then will connect to the existing mine-wide ventilation network.

15.5.2 Lateral Development

Table 15-6 is a summary of the LoM lateral development requirements. Most of the waste development was modelled as centerline advance and can be seen in the figures presented in the preceding report section. Major waste development line advance was marked up by an average of 15%. This was done to account for development not modelled in the 3D mine design such as safety bays, level sumps, electrical cut outs, gear storage areas, explosive magazines, etc.

With lateral development totalling 7,308 m, the project achieves a development ratio of 89 t/m. Waste development tonnages (including vertical development) is estimated at 54,000 t yielding a waste/plant feed ratio of 0.08.
Table 15-6: Summary of LoM Lateral Development Requirements

<table>
<thead>
<tr>
<th>Heading</th>
<th>Type</th>
<th>Size (m x m)</th>
<th>LoM Length (m)</th>
<th>Preprod (m)</th>
<th>Production (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Capitalized</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Decline/incline</td>
<td>Waste</td>
<td>2.7 x 2.7</td>
<td>926</td>
<td>641</td>
<td>284</td>
</tr>
<tr>
<td>Footwall drive</td>
<td>Waste</td>
<td>2.4 x 2.7</td>
<td>1,249</td>
<td>936</td>
<td>313</td>
</tr>
<tr>
<td>Vent/rockpass access</td>
<td>Waste</td>
<td>2.0 x 2.0</td>
<td>342</td>
<td>182</td>
<td>160</td>
</tr>
<tr>
<td>Sill (equiv. meters)</td>
<td>On-vein</td>
<td>2.0 x 2.0</td>
<td>2,461</td>
<td>2,461</td>
<td></td>
</tr>
<tr>
<td><strong>Subtotal Capitalized</strong></td>
<td></td>
<td></td>
<td>4,977</td>
<td>4,220</td>
<td>757</td>
</tr>
<tr>
<td>Expensed</td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Sill (equiv. meters)</td>
<td>On-vein</td>
<td>2.0 x 2.0</td>
<td>2,331</td>
<td></td>
<td>2,331</td>
</tr>
<tr>
<td><strong>Subtotal Expensed</strong></td>
<td></td>
<td></td>
<td>2,331</td>
<td></td>
<td>2,331</td>
</tr>
<tr>
<td><strong>Total Lateral Development</strong></td>
<td></td>
<td></td>
<td>7,308</td>
<td>3,088</td>
<td></td>
</tr>
</tbody>
</table>

Note: Decline/incline and footwall drive waste development meters include a 15% markup.

15.5.3 Raise Requirements

Table 15-7 is a summary of the LoM raising requirements. The ventilation raise includes a three-level height of traditional raising segments. The shrinkage stope access raise is excluded from the table.

Table 15-7: Summary of LoM Raise Development Requirements

<table>
<thead>
<tr>
<th>Heading</th>
<th>Type</th>
<th>Size (m x m)</th>
<th>Length (m)</th>
<th>Manway</th>
</tr>
</thead>
<tbody>
<tr>
<td>Ventilation raise</td>
<td>Waste</td>
<td>2.4 x 2.0</td>
<td>290</td>
<td>Y</td>
</tr>
<tr>
<td>Alimak raised</td>
<td>Waste</td>
<td>2.4 x 2.0</td>
<td>135</td>
<td>Y</td>
</tr>
<tr>
<td>Traditional raised</td>
<td>Waste</td>
<td>2.4 x 2.0</td>
<td></td>
<td></td>
</tr>
<tr>
<td><strong>Subtotal of vent raise</strong></td>
<td>Waste</td>
<td>2.4 x 2.0</td>
<td>425</td>
<td></td>
</tr>
<tr>
<td>Rockpass</td>
<td>Waste</td>
<td>1.5 x 1.5</td>
<td>298</td>
<td>N</td>
</tr>
<tr>
<td><strong>Total Meters</strong></td>
<td>Waste</td>
<td></td>
<td>723</td>
<td></td>
</tr>
</tbody>
</table>

15.6 Production Schedule

The production rate analysis was derived from first principles, using the reduced rated capacity of the equipment, reduced similar mines’ benchmark performance, and available mining fronts. SRK notes that rules-of-thumb predict a mining rate that varies from 420 t/d to 550 t/d.

The Prestea underground mine rehabilitation and preproduction period is defined as a 24-month period from January 2015 (start of underground rehabilitation which requires a period of 12 months) to December 31, 2016. This is dependent on GSR securing the necessary financing and approvals to develop the project. In H1 2017, an average production rate of 404 t/d is planned, which is more than 80% of the designed underground mine capacity. The full production period extends from January 1, 2017 to December 2020 for a period of four years. At full production, the planned mining rate is 500 t/d (175 kt per year).

The planned LOM production is 649 kt at 17.23 g/t, containing 359,000 ounces of gold.

The underground lateral development advance rates were scheduled at no more than 1.8 meters per day (m/d) for the decline/incline and no more than 2.7 m/d for horizontal lateral development (footwall waste drive and on-vein sill), with consideration being given to typical potential
bottlenecks such as capacity to semi-manually or manually move muck from the face, available ventilation, and tramming capacities on the 4475L main level (current 17L).

The underground vertical development advance rates were scheduled at 3.2 m/d for the Alimak raising and 1.6 m/d for the traditional raising.

Table 15-8 shows the LoM development-production plan.

SRK prepared a detailed Gantt chart schedule using EPS (Enhanced Production Scheduler, mining Gantt chart software) and created a detailed daily scheduling animation. There are five existing faces available to initiate development headings from the beginning. The early scheduling priorities during the start-up period include:

- Advancing the decline down from the 4475L (current 17L) north side footwall drive to the lower levels on the 4435L, 4395L, 4355L, and 4315L and the vein on 4435L and 4395L, the levels needed for preparing early stoping and forming the top levels ventilation circuit.

- Accessing the bottom of the main return air raise from the middle footwall drive on 4165L (current 24L) and commencing work on the Alimak raise from 4165L. The return air ventilation raise will be developed by Alimak to 4435L, while the top segment from 4435L to 4475L will be raised by a traditional raising method concurrently with the Alimak in order to speed up the ventilation network connection from 4165L to 4475L.

- Advancing the incline up from the 4165L north side footwall drive to 4200L and the 4200L footwall drive as well as the north most two stope on-vein sills in order to connect 4200L to 4165L by the stope access raises to create the local ventilation network and mucking the upper segment incline (4200L to 4235L) development waste down to 4165L through the stope access raise rockpass compartment.

- Advancing sill development from the 4165L south end existing footwall drive which has been developed to the location of the WR vein. This is an advantage for the PUG project which allows generating plant feed from Day 1 of mine development.

- Advancing the sill development from the 4165L north end newly developed footwall drive to make connections with the south end sill development and middle footwall drive to form the 4165L main level ventilation and haulage networks. When the 4165L sill development reaches the strategic points, initiate the stope access raise and other stope preparation development work. Planned early development of stope access raises will form the local ventilation networks between levels and serve as muck transfer raises for the upper level incline and horizontal development.

- Completing the 4435L (4475L has been developed at the planned stope locations), 4165L, and 4200L sill development to prepare for stoping.

- Advancing the decline to 4315L and connecting to the ventilation raise and the rockpass, then developing the incline/decline down to 4235L to make decline/incline connections. From 4315L to 4235L, downward developing the incline/decline is based on the following major considerations: 1) take advantage of established major ventilation circuit on 4315L (close to the development face); 2) easier to move development muck down to 4165L main level through the rockpass; 3) downward decline development is safer than upward incline development.
• Commencing production activities on 4435L and 4165L, where 4165L has only four planned stopes of production in the early stage in order to get early production, and to use the mined stope voids for waste storage to reduce waste hauling and hoisting to surface while not affecting global stability.

• Underground installation/construction of the rockpass loading chute.
Table 15-8: Underground Mine LOM Development-Production Plan

<table>
<thead>
<tr>
<th>Name</th>
<th>Unit</th>
<th>2015</th>
<th>2016</th>
<th>2017</th>
<th>2018</th>
<th>2019</th>
<th>2020</th>
<th>LoM Total</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Qtr 1</td>
<td>Qtr 2</td>
<td>Qtr 3</td>
<td>Qtr 4</td>
<td>Qtr 1</td>
<td>Qtr 2</td>
<td>Qtr 3</td>
</tr>
<tr>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td><strong>Tonnage per Day</strong></td>
<td>t/d</td>
<td>24</td>
<td>127</td>
<td>212</td>
<td>320</td>
<td>352</td>
<td>456</td>
<td>429</td>
</tr>
<tr>
<td><strong>Total Production</strong></td>
<td>kt</td>
<td>2</td>
<td>11</td>
<td>19</td>
<td>28</td>
<td>31</td>
<td>40</td>
<td>39</td>
</tr>
<tr>
<td><strong>Gold Grade</strong></td>
<td>g/t</td>
<td>7.16</td>
<td>12.49</td>
<td>16.59</td>
<td>16.62</td>
<td>17.69</td>
<td>19.00</td>
<td>20.40</td>
</tr>
<tr>
<td><strong>Capitalized Gold</strong></td>
<td>000oz</td>
<td>0.5</td>
<td>4</td>
<td>10</td>
<td>15</td>
<td>17</td>
<td>25</td>
<td>25</td>
</tr>
<tr>
<td><strong>Sill Development</strong></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td><strong>Stoping grade</strong></td>
<td>g/t</td>
<td>7.16</td>
<td>10.46</td>
<td>17.64</td>
<td>13.60</td>
<td>16.68</td>
<td>15.91</td>
<td>9.78</td>
</tr>
<tr>
<td><strong>Sill dev't production</strong></td>
<td>kt</td>
<td>2</td>
<td>7</td>
<td>9</td>
<td>11</td>
<td>7</td>
<td>7</td>
<td>4</td>
</tr>
<tr>
<td><strong>Rehabilitation</strong></td>
<td></td>
<td>104</td>
<td>231</td>
<td>151</td>
<td>72</td>
<td>98</td>
<td>75</td>
<td>9</td>
</tr>
<tr>
<td><strong>Footwall drive</strong></td>
<td>m</td>
<td>358</td>
<td>200</td>
<td>156</td>
<td>100</td>
<td>48</td>
<td>24</td>
<td>58</td>
</tr>
<tr>
<td><strong>Vent/rockpass access</strong></td>
<td>m</td>
<td>136</td>
<td>2</td>
<td>16</td>
<td>28</td>
<td>6</td>
<td>7</td>
<td>8</td>
</tr>
<tr>
<td><strong>Vent raise</strong></td>
<td>m</td>
<td>249</td>
<td>88</td>
<td></td>
<td></td>
<td>4</td>
<td>3</td>
<td>7</td>
</tr>
<tr>
<td><strong>Rockpass</strong></td>
<td>m</td>
<td>178</td>
<td>122</td>
<td></td>
<td></td>
<td>44</td>
<td>44</td>
<td></td>
</tr>
<tr>
<td><strong>Total Capitalized Dev't</strong></td>
<td>m</td>
<td>597</td>
<td>683</td>
<td>587</td>
<td>352</td>
<td>147</td>
<td>30</td>
<td>58</td>
</tr>
<tr>
<td><strong>Expensed Development</strong></td>
<td>m</td>
<td>176</td>
<td>631</td>
<td>747</td>
<td>907</td>
<td>604</td>
<td>596</td>
<td>344</td>
</tr>
<tr>
<td><strong>Total Lateral Dev't</strong></td>
<td>m</td>
<td>773</td>
<td>1,064</td>
<td>1,071</td>
<td>1,106</td>
<td>751</td>
<td>626</td>
<td>402</td>
</tr>
<tr>
<td><strong>Meters per day</strong></td>
<td>m/d</td>
<td>8.9</td>
<td>12.1</td>
<td>12.0</td>
<td>12.9</td>
<td>8.6</td>
<td>7.1</td>
<td>4.5</td>
</tr>
<tr>
<td><strong>Waste Quantity</strong></td>
<td>kt</td>
<td>11</td>
<td>12</td>
<td>11</td>
<td>6</td>
<td>3</td>
<td>1</td>
<td>1</td>
</tr>
<tr>
<td><strong>Total Materials Quantity</strong></td>
<td>kt</td>
<td>13</td>
<td>24</td>
<td>30</td>
<td>34</td>
<td>34</td>
<td>41</td>
<td>39</td>
</tr>
</tbody>
</table>

Note: Numbers are rounded.

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Table 15-8 shows the total lateral development advance rate reaches a peak of 12.9 m/d in Q4 2016. This will be achieved by five development crews, with each crew breaking slightly less than one half round per shift.

The schedule also shows the total material being broken each period, as the sum of the on-vein sill and waste development tonnes and stope production tonnes, with the peak of 47 kt (528 t/d) occurring in Q3 2018.

The LOM waste development quantity amounts to 54 kt. Part of the waste material can be dumped into stope voids as backfill. SRK adopted a conservative cost estimation approach by treating all waste materials as being hauled and hoisted to surface.

15.7 Mining Equipment

Table 15-9 shows SRK’s estimate of the mining equipment to execute the mine plan, including the surface units required to support the mine plan. The maximum number of units is shown for each equipment type, and these numbers vary throughout the mine life.

Underground equipment requirements include spare units such as locomotives and mine cars, but replacement equipment required during the mine life is not included.

Table 15-9: Planned Mining Equipment

<table>
<thead>
<tr>
<th>Name</th>
<th>Description</th>
<th>Units</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Underground</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Mine car - side dump</td>
<td>1 m³</td>
<td>60</td>
</tr>
<tr>
<td>Mine car - still dump</td>
<td>2 m³</td>
<td>15</td>
</tr>
<tr>
<td>Material car</td>
<td>1 t</td>
<td>10</td>
</tr>
<tr>
<td>Air leg &amp; drill</td>
<td>17 kg &amp; 29 kg</td>
<td>20</td>
</tr>
<tr>
<td>Stoper drill &amp; leg</td>
<td>39 kg &amp; 17 kg</td>
<td>20</td>
</tr>
<tr>
<td>Rail-bound loader</td>
<td>0.14 m³</td>
<td>7</td>
</tr>
<tr>
<td>Rail-bound loader</td>
<td>0.25 m³</td>
<td>2</td>
</tr>
<tr>
<td>Scraper loading machine</td>
<td>0.15 m³/11 kW</td>
<td>1</td>
</tr>
<tr>
<td>Loco incl. battery, charger</td>
<td>2 t</td>
<td>7</td>
</tr>
<tr>
<td>Loco incl. battery, charger</td>
<td>10 t</td>
<td>2</td>
</tr>
<tr>
<td>Decline hoists</td>
<td>1.5 m diameter</td>
<td>3</td>
</tr>
<tr>
<td>Winder/scrap hoist</td>
<td>5 t double drum</td>
<td>2</td>
</tr>
<tr>
<td>Slusher &amp; bucket</td>
<td>11 t &amp; 762 mm</td>
<td>2</td>
</tr>
<tr>
<td>Tugger (air powered)</td>
<td>150 kg</td>
<td>5</td>
</tr>
<tr>
<td>Auxiliary fan</td>
<td>21,000 cfm/11 kW</td>
<td>8</td>
</tr>
<tr>
<td>Portable refuge station</td>
<td>MineARC PERM20</td>
<td>3</td>
</tr>
<tr>
<td>Cap lamp set</td>
<td>MineARC 200 Kit</td>
<td>2</td>
</tr>
<tr>
<td>Stench gas system</td>
<td>MineARC</td>
<td>1</td>
</tr>
<tr>
<td><strong>Surface</strong></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Toyota tractor</td>
<td>95 kW</td>
<td>2</td>
</tr>
<tr>
<td>Surface forklift</td>
<td>Miller 56 kW</td>
<td>2</td>
</tr>
</tbody>
</table>

15.8 Manpower

Table 15-10 shows the composition of the Prestea mine workforce on the owner’s payroll during the full production period. Currently, Prestea is in care and maintenance, employing 177 experienced staff and employees. During the predevelopment rehabilitation and preproduction development periods, GSR will gradually increase hiring according to the mine plan.
requirements. It is not expected that there will be hiring difficulties due to the plentiful underground mining experience in local communities.

Staff positions include mine management, human resources, community relations, security, finance, mine supervision, safety and training, maintenance supervision, and technical services. Operations and maintenance employees include 20% leave loading for vacation/spare employees.

There are three expatriate positions included in the payroll, and the rest of workforce will be recruited locally.

Table 15-10: Prestea Mine Manpower – Full Production Period

<table>
<thead>
<tr>
<th>Function</th>
<th>Payroll</th>
</tr>
</thead>
<tbody>
<tr>
<td>General &amp; Administration</td>
<td></td>
</tr>
<tr>
<td>Mine management</td>
<td>20</td>
</tr>
<tr>
<td>Human resources</td>
<td>7</td>
</tr>
<tr>
<td>Community</td>
<td>3</td>
</tr>
<tr>
<td>Security</td>
<td>18</td>
</tr>
<tr>
<td>Finance &amp; stores</td>
<td>5</td>
</tr>
<tr>
<td><strong>Subtotal of G&amp;A</strong></td>
<td><strong>53</strong></td>
</tr>
<tr>
<td>Mining</td>
<td></td>
</tr>
<tr>
<td>Mine supervision</td>
<td>16</td>
</tr>
<tr>
<td>Safety &amp; training</td>
<td>10</td>
</tr>
<tr>
<td>Development and stoping</td>
<td>151</td>
</tr>
<tr>
<td>Materials handlerings - main level and surface</td>
<td>15</td>
</tr>
<tr>
<td>Shaft operations</td>
<td>18</td>
</tr>
<tr>
<td>Decline operations</td>
<td>11</td>
</tr>
<tr>
<td>Mine rehabilitation</td>
<td>3</td>
</tr>
<tr>
<td>Construction &amp; utility</td>
<td>14</td>
</tr>
<tr>
<td>Mine services</td>
<td>19</td>
</tr>
<tr>
<td>Maintenance supervision</td>
<td>5</td>
</tr>
<tr>
<td>Maintenance</td>
<td>15</td>
</tr>
<tr>
<td>Technical services</td>
<td>18</td>
</tr>
<tr>
<td><strong>Subtotal of Mining</strong></td>
<td><strong>295</strong></td>
</tr>
<tr>
<td><strong>Total Mine Employees</strong></td>
<td><strong>348</strong></td>
</tr>
</tbody>
</table>

15.9 Mine Infrastructure and Services

15.9.1 Materials Handling

Mining will utilize rail bound equipment. Track haulage and shaft hoisting were selected over truck haulage based on the following considerations:

- Reduced capital expenditure and short preproduction development period
- Utilization of the existing infrastructures, such as shafts, main levels, ventilation and dewatering systems, and off-shaft infrastructure
- Less demanding maintenance skills and effort
- Less ventilation requirement
• Familiarity of the mining systems by local mining communities
• Operations reliability

Primary access for workers, equipment, and consumable materials will be via the existing Central Shaft, 4475L and 4165L main levels (existing 17L and 24L) to the newly developed declines and incline. The declines and incline will connect on 4165L and 4315L. The declines and incline are designed at +/-28° gradient with a cross-section of 2.7 m wide by 2.7 m in height. All plant feed above 4165L will use chute loading for stope plant feed or manual/semi-manual loading for sill plant feed to the mine cars, then it will be dumped to the rockpass down to 4165L. On 4165L, plant feed will be chute loaded into the main haulage mine cars and transported to the Central Shaft for hoisting to surface. For plant feed on 4165L, loaded mine cars will be directly hauled to the Central Shaft for hoisting to surface. For plant feed below 4165L, loaded mine cars will be hoisted to 4165L through the decline, then transported to the Central Shaft for hoisting to surface.

Waste materials will generally be hoisted to the two main levels through decline or incline, and then transported to the Central Shaft for hoisting although some of the waste rock can be dumped into mined stope voids as backfill materials.

15.9.2 Shaft and Hoist Upgrades

Central Shaft is a rectangular set shaft 7.5 m x 3.0 m that was constructed around 1935. It serves the mine down to the 30 L at a depth of 1204.5 m below collar level, with skip loading pockets at just below 20L and 25L.

The shaft is generally unlined having been excavated through competent rock, although there are some sections of concrete repair where there have been areas of loose ground. The shaft guides and services are carried on horizontal steel sets at 1.8 m apart with vertical support members forming a tower structure that supports the skip and cage timber guides, a ladder way and shaft services. At every station level, there are 5 horizontal steel main bearers that support the tower structure from one level to the next.

The shaft has four compartments where compartments, 1 and 2 are designated for the skip hoisting with a maximum skip capacity of 7 t. Compartments 3 and 4 house the double deck man and materials cages, each with a capacity of 15 men per deck and rails at 18” gauge for small tubs and material trollies. The shaft services and ladder way compartments are to the south side of the skipping compartments, running the full depth of the shaft. Owing to the current condition of the Central Shaft steelwork the permissible speed of hoisting and payload has been limited to reduce slamming loads on the structure. In its current state hoisting capacity is approximately 224 tpd using half full skips at 1 m/s winding speed.

The Central Shaft steelwork upgrade is intended to allow hoisting capacity to increase to 600 tpd over a 16 hour operational period to align with WRP LoM production and also serve as the main egress and services shaft (e.g.: dewatering, compressed air).

Technical reports have identified the rehabilitation work scope and the rehabilitation work is related mostly to replacing heavily corroded sett steelwork and bearers with new throughout the shaft structure. For scheduling purposes the work has been split into priority works and maintenance works. The priority works include section sets that are heavily corroded and all the bearer sets. This work will be addressed with almost exclusive access to the shaft and once
completed the remaining maintenance works to complete the shaft rehabilitation will continue at a slower pace owing to reduced shaft access time.

The Central Shaft winder requires electrical upgrades to their instrumentation and controls for safe operation. The work involves replacing the electrical control with a modern and reliable system, replacing the relay panel with a modern PLC panel for the control, fault indication and operator interface and installing new resistor banks to replace the liquid rheostat controller.

Bondaye Shaft is rectangular in construction and has been sunk to a depth of 1,012 m below collar. The shaft is currently equipped with a steel frame arrangement of outer cross sectional dimensions of approximately 3.9 m x 1.6 m. The shaft is currently equipped with a similar steel frame arrangement as mentioned above for Central Shaft. The shaft has three compartments: two service conveyances and a ladder way.

The Bondaye Shaft current acts as the secondary means of egress and needs to continue in operation for this purpose as well as for dewatering. The current steelwork condition of the Bondaye Shaft from the collar to 9L is poor and in need of immediate attention. Beyond 9L conditions improve, however there are localized areas that require rehabilitation.

An upgrade of steel work and guides is required and will be split into priority work and maintenance work, with the priority works generally capturing the section between the collar and 9L. As the priority works will require additional personnel to complete in parallel with the Central Shaft it is envisaged that the ongoing maintenance works to complete the rehabilitation will be undertaken by the permanent team at a slower pace owing to resource and shaft availability.

15.9.3 Horizontal Development Rehabilitation

The 24L drive links the Central Shaft to the West Reef mineralization, some two kilometers to the south, and is integral to the development of the West Reef and LoM production. The drive is generally at 2.7m high x 2.4m wide excavation with an 18” gauge rail track.

The current condition of the 24L drive itself is fairly good however localized ground support and slashing to the design profile is required. The track is generally in poor condition and services along the drive require realignment and support.

In order to hoist at 600 tpd from the Central Shaft, 24 L rehabilitation consists of:

- Installation of new track at a wider gauge to facilitate faster tramming, including new ballast and sleepers (materials will be recycled if in suitable condition);
- Drive clean-up and removal of old ventilation door frames and steel work restrictions and re-routing of pipework along 24 L;
- Localised rock support and rehabilitation in areas where current rock support measures are reducing the drift cross section; and
- Additional equipment to facilitate tramming on 24 L, i.e. locos, batteries and wagons.

Refuge chambers and crib rooms, located to allow blasting whilst personnel are underground, will be constructed to support the West Reef.

The 24L rock pass and the 25 L loading pocket will undergo significant rehabilitation.
The 17L drive connects the Central Shaft to the top of the West Reef mineralization and will primarily be used during development for waste tramming and in production for egress and ventilation. The current condition of 17L is poor and requires some rehabilitation work commensurate with its purpose.

In order to support the development activities and ongoing mining the rehabilitation work required consists of:

- Additional rock support and rehabilitation in areas where current rock support measures are reducing the drift cross section;
- Drive clean-up and removal of old ventilation door frames and steel work restrictions and re-routing of pipework;
- Repair, and where required, replacement of track to facilitate safe tramming.

As with the 24L, refuge chambers will be constructed to support the development and egress on the 17L.

### 15.9.4 Mine Ventilation

Tables 15-11 and 15-12 show the calculated total ventilation requirements for the WR area based on number of personnel in the area and minimum airflow requirements, respectively.

An airflow of 50 m$^3$/s has been targeted based on these two analyses.

In October 2014, Mine Ventilation Services, Inc was contracted to look at several methods by which the West Reef area could be ventilated through the existing mine workings. The goal of the study was to determine if it is feasible to draw 50 m$^3$/s of airflow from both the Central Shaft and Bondaye Shaft to the West Reef mining area and exhaust the air back to the South Waste Shaft. Four different design scenarios were developed to determine the infrastructure and fan sizes to support this ventilation system. The current overall ventilation system layout is shown in Figure 15-11.

The base case ventilation layout provides fresh air from the Central Shaft on both the 17 Level and the 24 Level, and from the Bondaye Shaft on the 24 Level. Air is exhausted from the mining areas back up to the 17 Level through a ventilation raise (assumed to be 2.5m diameter). Between the 17 Level and the 12 Level there is a series of ramps, and raises through which the exhaust air will travel.
Table 15-11 Ventilation requirements based on number of personnel

<table>
<thead>
<tr>
<th>No. Stopes</th>
<th>Workers/Stope</th>
<th>Total Workers</th>
<th>Workers on Peak Shift</th>
</tr>
</thead>
<tbody>
<tr>
<td>Active Mining Stope</td>
<td>11</td>
<td>8</td>
<td>88</td>
</tr>
<tr>
<td>Active Final Mucking Stope</td>
<td>2</td>
<td>9</td>
<td>18</td>
</tr>
<tr>
<td>24L Haulage</td>
<td>1</td>
<td>9</td>
<td>9</td>
</tr>
<tr>
<td>Subtotal</td>
<td></td>
<td></td>
<td>115</td>
</tr>
<tr>
<td>Services Workers (30%)</td>
<td></td>
<td></td>
<td>35</td>
</tr>
<tr>
<td>Supervision and Technicals</td>
<td></td>
<td></td>
<td>15</td>
</tr>
<tr>
<td>Total</td>
<td></td>
<td></td>
<td>165</td>
</tr>
</tbody>
</table>

Reg Min Requirement cu.m/s-worker 0.25

Table 15-12 Ventilation requirements based on minimum airflow requirements

<table>
<thead>
<tr>
<th>No. Drifts</th>
<th>Section Areas (Sq.m)</th>
<th>Air Velocity (Min m/s)</th>
<th>Air Flow m³/s</th>
</tr>
</thead>
<tbody>
<tr>
<td>Active Normal Level Segments</td>
<td>14</td>
<td>4.84</td>
<td>0.25</td>
</tr>
<tr>
<td>Haulage Level</td>
<td>2</td>
<td>7.84</td>
<td>0.25</td>
</tr>
<tr>
<td>Infrastructure</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Subtotal</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Leakage (25%)</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Normal Contingency (30%)</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Hot Weather Contingency (30%)</td>
<td></td>
<td></td>
<td></td>
</tr>
<tr>
<td>Total</td>
<td></td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Figure 15-11 Current ventilation layout
Several models were developed with various size raises; raises sized based on the original ventilation model (1.5m x 1.8m), raises sized at 1.5m x 1.5m, and raises sized at 1.25m x 1.25m. The range of raise sizes provides a best/worst case design for the sizing of the 17 Level booster fan. Additional models were developed to represent the extension of the West Reef exhaust raise to the 12 Level, and the development of a new exhaust ramp to the 12 Level from the 17 Level. Each of these scenarios require that the 11 Level and the 12 Level be configured together to operate in parallel back to the South Waste Shaft. Using a single surface exhaust fan was also investigated, however, this arrangement only works if the West Reef exhaust raise is extended to surface. The system was also investigated with only the 17 Level booster fan installed, however, this arrangement promotes significant recirculation and as such was dismissed. The results of these evaluations are summarized on Table 15-13.

**Table 15-13 Comparison of fan operating points for different system configurations**

<table>
<thead>
<tr>
<th>Option</th>
<th>Description</th>
<th>Surface Exhaust Fan</th>
<th>17 Level Booster Fan</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td></td>
<td>Airflow (m³/s)</td>
<td>Pressure (kPa)</td>
</tr>
<tr>
<td>A</td>
<td>Split Fan System – no additional raise from 17 Level to 12 Level</td>
<td>130</td>
<td>2.26</td>
</tr>
<tr>
<td>B</td>
<td>Split Fan System – no additional raise from 17 Level to 12 Level – worst case existing raises</td>
<td>130</td>
<td>2.26</td>
</tr>
<tr>
<td>C</td>
<td>Split Fan System – new ramp between 17 Level and 12 Level (2.3m x 2.2m)</td>
<td>130</td>
<td>2.26</td>
</tr>
<tr>
<td>D</td>
<td>Split Fan System – new raise between 17 Level and 12 Level (2.5m)</td>
<td>130</td>
<td>2.26</td>
</tr>
<tr>
<td>E</td>
<td>Single Surface Exhaust Fan – new raise between 17 Level and 12 Level (2.5m)</td>
<td>175</td>
<td>4.51</td>
</tr>
</tbody>
</table>

**Figure 15-12 Primary exhaust air routing (Option A)**
Figure 15-13  Original modeled raise/rock pass sizes (Option B)

Figure 15-14  Modified raise/rock pass sizes (Option C)

Figure 15-15  Worst case raise/rock pass sizes (Option D)
The installation of a single exhaust fan (Option E) on surface requires a significantly higher airflow through the exhaust system due to the increased leakage resulting from the high fan pressure (and therefore high differential pressures across the general level infrastructure). Splitting the applied fan pressure between the surface installation and a new installation on 17 Level provides for a more efficient ventilation system provided the bulkheads and doors between the fresh air and exhaust air routes above 17 Level can be maintained. The addition of the 17 Level booster fan will help to reduce the differential pressures which in turn will help to manage leakage. For this study the surface fan (single fan installation) operating duty did not significantly change when comparing the different options that included the 17 Level booster fan. For the 17 Level Booster fan the cost associated with the worst case raise sizes are included in the design budget.

The ventilation system may be further optimized by incorporating 9 Level into the exhaust for the West Reef. This arrangement was not examined during this study as there may be an impact on the upper escapeway. However, without the incorporation of the 9 Level as a parallel exhaust route, the air velocity through the 11 Level and 12 Level will approach 4.5 to 5 m/s which will represent a significant departure from the current mining conditions. In order to decrease this air velocity an additional parallel path will be required.

It should also be noted that the air velocity through the single exhaust drift between the West Reef exhaust raise and the proposed ramp, current exhaust raises, or proposed exhaust raise will be in the order of 10 m/s to 11 m/s. An air velocity of this magnitude may create issues with respect to maintaining this route as an escapeway.

15.9.5 Mine Dewatering

The two principal shafts, Central Shaft and Bondaye Shaft, situated about 4 km apart, are used as conduits to dewater Prestea underground mine.

Each shaft has a number of pump stations and associated collection sumps. The pump stations pump water from the sumps to pump stations at higher levels and then to surface before the water is discharged to the receiving environment. The system is manually operated with sump levels checked regularly; pumps are shut down and re-started depending on sump levels.

Figure 15-16 shows the pump station configuration for Central Shaft. At Central Shaft there are four operating pump stations located on the following levels:

- 6L – two pumps stationed at 6L discharge to the surface at a head of around 175 m;
- 17L North Shaft – pump station pumps water horizontally to the main pump station;
- 17L – pump station discharges directly to surface at a head of 600 m and to the 6L sump at a head of around 422 m; and,
- 24L – pump station discharges to 17L at a head of around 298 m;

The current water level in Central Shaft is just below 25L and a shaft bottom submersible pump dewaterers to the 24L sump. There are 2 further pump stations that are currently flooded at 28L and 30L, which may be reinstated as required for operational needs as mining in WR extends below 24L.

Mine water pumped from Central Shaft is currently discharged in to an open collector tank, then gravity flows to constructed marsh and then to the receiving environment.

Figure 15-17 shows the pump station configuration for Bondaye Shaft. At Bondaye Shaft there are two pump stations in operation, located on the following levels:
- 3 L – pump station discharging to the surface at a head of 114 m; and
- 9 L – one pump discharges water directly to the surface at a head of 359 m and the other pumps to 3 L at a head of 245 m.

Mine water pumped from Bondaye Shaft is discharged directly to the receiving environment via a natural surface water channel.

Figure 15-16: Central Shaft pumping system
The current underground pumping system availability at both the Central Shaft and Bondaye Shaft is low due to the following:

- Age and condition of the pumps. Several of the pumps installed are more than 40 years old and are no longer supported by vendors, resulting in sourcing of spares and repair work difficult, not to mention affecting pump performance;
- Lack of duty / standby arrangements. The lack of redundancy affects overall system availability; and
- Lack of spares: Of the vendor spares obtainable, there are limited amounts available onsite resulting in delayed repair time.

It is proposed to replace aged and non-supported pumps and re-establish duty/standby operating arrangements at each pump station. A total of ten pumps have been identified (Table 15-14).
Table 15-14 New pump requirements

<table>
<thead>
<tr>
<th>Shaft</th>
<th>Level</th>
<th>Size</th>
<th>Qty</th>
</tr>
</thead>
<tbody>
<tr>
<td>Bondaye</td>
<td>3</td>
<td>90kw</td>
<td>1</td>
</tr>
<tr>
<td>Bondaye</td>
<td>9</td>
<td>275kW</td>
<td>1</td>
</tr>
<tr>
<td>Bondaye</td>
<td>Shaft bottom</td>
<td>18kW</td>
<td>1</td>
</tr>
<tr>
<td>Central</td>
<td>6</td>
<td>185kW</td>
<td>1</td>
</tr>
<tr>
<td>Central</td>
<td>17 North</td>
<td>75kW</td>
<td>2</td>
</tr>
<tr>
<td>Central</td>
<td>17</td>
<td>650kW</td>
<td>2</td>
</tr>
<tr>
<td>Central</td>
<td>24</td>
<td>375kW</td>
<td>1</td>
</tr>
<tr>
<td>Central</td>
<td>Shaft bottom</td>
<td>37kW</td>
<td>1</td>
</tr>
<tr>
<td><strong>Subtotal</strong></td>
<td></td>
<td></td>
<td><strong>10</strong></td>
</tr>
</tbody>
</table>

The Central Shaft 24L pump station is located on a sub-level with poor access for personnel and parts. Delivering parts to the pump station will prevent rail use as a winch is used across the track to lower the parts to the sub-level. As a result the current pump station will be relocated and installed on 24L.

West Reef water will be treated as required to meet the Ghana EPA Effluent Discharge Guidelines and all other mine water extracted will be covered by the environmental indemnity. The water from the West Reef will be captured and undergo settling underground before being pumped to surface through a dedicated and standalone pumping system to the water treatment plant if required to meet the discharge guidelines. Figure 15-18 shows the proposed general water management system.
If required, the water treatment plant will be based on microfiltration followed by reverse osmosis (MF/RO) to produce a permeate stream suitable for environmental discharge. The brine from the MF/RO will then pass through a sulfate precipitation plant using lime; clarified; and the solids dewatered through a filter press to form a cake suitable for placement at the Bogoso TSF or underground.

The water treatment plant shown in Figure 15-19 will be sized for 0.8 ML/d comprising an extraction rate from the West Reef of 0.6 ML/day and a recycle within the plant of 0.2 ML/day.
For the existing underground workings, it is assumed that treatment via a marsh system for all water removed from the Central Shaft will be required to remove suspended solids.

### 15.9.6 Compressed Air Supply

Compressed air is a critical service for the use of hand-held compressed air powered rock drills that are proposed for the non-mechanized mining method. This equipment will be the main consumer with each drill requiring around 200 cfm (cubic feet per minute) to operate. It has been estimated that total compressed air requirement for hand held mining will be 12,000 cfm at 90 psi (pounds per square inch) pressure.

The current installed compressed air capacity at Central Shaft is 5,600 cfm at 100 psi, a shortfall of 6,400 cfm required for the West Reef. The two operating air compressors are over 40 years old, are not supported with spares by the original equipment manufacturer and are unreliable in operation.

The compressed air distribution system from the compressors at the Central Shaft to the point of use underground has not undergone a technical review but on visual inspection and through discussion with the Prestea engineers, it requires replacement.

It is proposed to install a complete compressed air system including a compressor and dryer arrangement at the surface, coupled with a distribution stem to points of use, primarily at 24L and 17L. To ensure continuity of air supply underground the concept is to install three air-compressors each providing 6-7,000 cfm for a combined 18-21,000 cfm capacity; however at any one time there will be two duty compressors and one standby.

The distribution network will be installed new and will include air receivers close to the West Reef to provide greater assurance of pressure at the point of use. The main delivery pipe will be installed in the Central Shaft services compartment with legs branching out to the main West
Reef drives and other levels as required to support the existing Prestea underground infrastructure. A steel pipeline around 250-300 mm in diameter is expected to be required as the main header to the West Reef and will reduce down as the network distributes throughout the shrinkage stoping area to finally 50 mm HDPE lines in each stope.

15.9.7 Electrical Power Distribution

The Prestea mine is supplied with electrical power from the local utility ECG (Electrical Company of Ghana) at a voltage level of 34.5 kV (typically referred to as 33 kV). The local ECG substation feeds the Prestea Slimes substation, which was the primary intake for the Prestea operations in previous times. Presently the Slimes substation has one 34.5 / 3.3 kV step down transformer, feeding some local load, and has a through switch, which feeds the Job 600 substation by means of an overhead line. The Prestea mine management office is also fed from the Slimes substation.

The Job 600 substation receives 34.5 kV supply from the Slimes substation, and transforms this to 3.3 kV, through three x 34.5 / 3.3 kV transformers, one of which is not on load. This substation also steps up 3.3 kV supply to 55 kV through a 6 MVA step up transformer, to feed an overhead line supply to the remote Bondaye substation.

The 3.3 kV switchboard at Job 600 substation is the primary feed into the Prestea central shaft surface and underground operations. This substation feeds the compressor house, the winder house, the Bondai step up transformer, the Ridge and Borehole area, the South Waste Vent fan area and several levels underground, through a radial feed network. A single 1550 kW standby diesel generator is available to provide emergency power in the event of a power black-out.

The Bondai shaft receives power from the Job 600 substation, by means of a 55 kV overhead line and a local 3 MVA step down transformer, reducing the voltage to 3.3 kV. This is fed into a substation, located well away from the winder house, where the 3.3 kV is distributed through a radial network to the compressor house, winder house and several levels underground.

The underground electrical infrastructure provides motor drives and small power at various voltage levels including 3.3 kV, 550 volt, 440 volt, 415, 400 and 230 volts. The underground equipment at each level has a combination of ring switches, stand-alone motor starters and low voltage distribution boards. A manner of cable supply redundancy is provided through manual cable jointing, to allow some form of contingency, should the supply fail to a specific level.

PPE Technologies was commissioned to complete an inspection of the primary electrical infrastructure at the GSR Prestea operation. A site inspection was initiated to identify the immediate risks, suitability of equipment for re-commissioning and to detail a work scope and budget for this infrastructure, to form part of the West Reef project budget. The Prestea mine electrical infrastructure dated back to the 1950s and 1960s, and has not worked at effective loads for the past 15 years, as the mine has been in a condition of care and maintenance, with no production throughput. With the West Reef development, this infrastructure will be revived and modernized in the most effective way possible.

15.9.8 Backfill

The mining voids are planned to be left open and without backfill, however development waste will be dumped into empty stope voids in order to reduce waste haulage and hoisting costs and to provide additional ground support.
15.9.9 Definition Drilling
A definition drilling unit cost of $0.30/t plant feed is included in the PEA. A detailed drilling plan has not been prepared. Considering that approximately 90% of the targeted mineralization is in the Indicated mineral resources category and the characteristics of the deposit, which will require less definition drilling, it is SRK’s opinion that the planned unit cost for definition drilling should be sufficient for the Prestea project.
16 Recovery Methods (Item 17)

MDM Engineering have been contracted to undertake process design for the West Reef material in the existing Bogoso Plant which is about 15km from the Prestea Mine and accessible by road. The existing Bogoso processing plant consists of separate processing routes and equipment for oxide and refractory ores, of which the refractory processing route would effectively be closed-down by the time the new plant is required. This presents an opportunity to utilise some of the existing equipment and achieve an overall cost effective capital solution. The principal applied is to utilize certain selected equipment from the Bogoso refractory plant (which will be on care and maintenance), and supply and install new equipment where no suitable existing equipment can be sourced from site or refurbished.

The plant is designed to treat 500 tpd of non-refractory material. The processes involved are crushing, milling, CIL, elution and electrowinning.

For the purposes of the PEA, it has been assumed that:

- The Oxide Plant will keep operating at current production levels, and that most of the required services and reagents will be supplied by the existing systems used for the Oxide Plant;
- Areas in the existing plant affected by excessive corrosion will be refurbished as part of the ongoing operation;
- Sufficient space is available to fit a 196 m² RoM stockpile (24 hr) inside the existing security fence, with sufficient space for the haul trucks to turn around requiring additional space.
- Both the Oxide and Sulphide Crushing and stockpile areas are available for re-use; however, most of the existing equipment in these areas is oversized for the 500 tpd feed from Prestea Underground. A new, small high-grade RoM storage area would be prepared inside the plant compound (inside the security fence) where haul trucks would dump RoM material directly onto the existing open area just inside the current access gate.
- The RoM area will be managed by means of a front end loader (FEL), supplied by the client. The operating philosophy of the RoM pad is to crush the delivered material as soon as possible and then store crushed RoM material in a new mill feed storage bin because there is limited available space. An area for 24 hours of RoM will be allowed as stockpile when the crusher plant experiences downtime.
- A conventional 2-stage crushing circuit will accept a feed of F₁₀₀ of 300mm and deliver a crushed product at a P₁₀₀ of 15mm. A Pilot Crushtec modular crushing and screening plant, including conveyors, will be used. The crushing plant will consist of a 55 kW CJ408 primary jaw crusher, 90 kW CH420 secondary cone crusher and 1500 mm X 3700 mm triple deck screen. The final crushed product will be stored in a new 105 ton, 60 m³ mill feed bin. The bin will overflow into a designated stockpile area when the mill is not operating. The FEL will feed overflow material back onto the bin feed conveyor.
- The correct mill size required is a minimum of a 10’Ø x 17’L mill with a 600 kW motor. The available mills were reviewed to ascertain their viability for the 500 tpd throughput however they were significantly larger in duty and not suitable for use. The smaller refractory plant regrind mill was analysed for integration into the circuit, however it is unfortunately too small and could
not meet the required duty. Therefore a new, suitably sized 3mØ x 5.2mL ball mill with 600 kW will be procured and installed.

A classification cyclone, cluster arrangement of 4 cyclones (3xduty and 1 stand-by), will be installed in closed circuit with the mill. This will accommodate the requirement to feed the gravity circuit from a split portion of the cyclone underflow. The cyclone will be fed by a 6/4-D-AH-R centrifugal pump with a 45 kW motor. The existing Rougher concentrate pump no’s 3 and 4 are identical pumps and have been identified for this application.

The gravity circuit will be fed from a bleed stream from the cyclone underflow and will report to a gravity scalping screen to remove any +2 mm material. The balance of the underflow will constitute 250% circulating load and report back to the mill feed.

The cyclone overflow will be screened to remove any plastic, wood chips or other oversize material still present. The screen underflow will be pumped to the pre-leach thickener feedbox by a 6/4-D-AH-R centrifugal pump with a 15 kW motor. The existing rougher concentrate pump no’s 1 and 2 have identical pump wet-ends with 22 kW motors and are proposed for this duty.

It is proposed to use the existing gravity circuit that consists of 2 x KC-XD30 Knelson concentrators. Only one unit is required for the design feedrate, but the opportunity exists to include the second unit in a stand-by application. It is assumed that the client will undertake this refurbishment as part of ongoing operation of the plant. The Knelson accepts the slurry from the ball mill via cyclone underflow and scalping screen and separates the expected 50% gravity gold content from the RoM material. The existing 4/3-C-AH-R pumps with 30 kW motors from the regrind mill discharge are proposed for this duty.

The Knelson overflow returns to the mill discharge sump under gravity and the concentrate will follow the existing route to a concentrate tank, from where it will be further upgraded over the existing Gemini table.

The existing 15 mØ concentrate thickener was designed to accept 34.2 tph of reground sulphide flotation (D₈₀ of 45 microns) concentrates, and de-water it to an underflow solids concentration of 48.58%. The new requirement is for a feedrate of 26 tph mill discharge material with a D₈₀ of 75 microns. Based on assumed settling rates for similar material, and in the absence of settling testwork data, the existing 15 mØ concentrate thickener should be suitably sized for the requirements. This thickener appears to be in good structural condition and the drive and rake mechanism is assumed to be in similar good working condition. Any required refurbishment to this thickener will be undertaken as part of ongoing operation of the plant. The client is required to refurbish the thickener feedbox.

Leach testwork was done at 40% solids only. The new design uses 44% solids as a worst case assumption. At this underflow condition, the current underflow pumps (2/1.5 B-AH-R centrifugal pumps) are too small and we recommend that 2 x new 3/2-AH-R WARMAN pumps with a 9.2 kW motor be used. An alternative is to use the existing 4/3-C-AH-R pumps with 30 kW motors from the flotation concentrate transfer circuit be used.

The opportunity exists to achieve a higher underflow density, which will enable us to utilise the existing pumps without any required changes. During a later study phase it is recommended to complete settling testwork to verify this opportunity.

The original BIOX feed was ground finer than that of the new application and it is therefore assumed that the flocculent consumption will be lower than that used for BIOX concentrates and
hence the existing flocculent system will be sufficiently sized. The existing lime plant was also sized to feed lime to the sulphide circuit where the lime requirements would have been much higher than that of the new circuit because of the amount of sulphides. It is therefore assumed that the existing lime plant will suffice for pH adjustment requirements prior to the new leach.

Leach testwork was done in excess conditions for a period of 48 hours. Standard CIL application would be to target a residence time of 24 hours. The provided leach curves do indicate very minimal benefit in recovery beyond the 24 hours timeline.

It is proposed that the existing sulphide CIL circuit, with minimum changes, be utilised for the new leach requirements. These tanks are larger than the requirement for new tanks in a greenfields application, but it does present a significant opportunity. It is proposed to use the first tank as a conditioning tank, with 5 subsequent CIL leach tanks. In comparison to the testwork, the residence time will be 18.8 hrs in the conditioning tank alone, and a further 85.7 hours in the 5 subsequent leach tanks. The carbon handling philosophy due to this extended residence time will be to reduce the carbon concentration per tank and hence the overall inventory of carbon. Standard CIL carbon concentrations vary between 10 and 20 g/l. It is recommended to apply a concentration of 12 g/l in the first CIL tank, and 8 g/l in the 4 tanks thereafter.

Thickened feed slurry will be pumped from the pre-leach thickener to the conditioning tank via a new two stage vezin slurry sampler. There may be an opportunity to utilise the existing samplers on the flash flotation concentrate circuit for this duty.

The existing tower crane, interstage screens, reagent addition, oxygen injection and carbon transfer pumps will be used. The existing loaded carbon pump from the BOIX CIL currently feeds a screen sized for the pump duty. The existing carbon transfer pumps will be utilised and therefore the loaded carbon will still report to the existing sulphide circuit’s loaded carbon hopper. The amount of gold in the leach circuit is much lower than that of the original BIOX CIL giving lower loaded carbon values, therefore only 1 ton of loaded carbon will be removed from the circuit daily. This used to be 15 tpd previously. The original sulphide 15 ton acid wash and elution circuit is oversized for the new requirement and will result in a huge carbon inventory if used. It is propose to transfer the loaded carbon from the loaded carbon hopper to the existing 5 ton oxide acid/elution circuit.

The leach tailings will follow the existing route via existing samplers to the existing oxide circuit carbon safety screen and oxide CIL tailings tank.

Loaded carbon from the existing sulphide loaded carbon hopper currently feeds a 15 ton acid/elution circuit. This circuit is significantly oversized for the 1 tpd of carbon required and therefore it is proposed to re-route the PUG loaded carbon to the existing 5 ton acid/elution circuit and utilising the existing electrowinning and smelting system thereafter.

Depending on the gold accounting philosophy applied by the client, the 5 ton acid wash and elution batches could be done as either:

1 ton loaded carbon per day form the PUG circuit, to mix with 4 tons from the oxide circuit, or

A full 5 ton loaded carbon batch could be eluted from the PUG circuit separately every 5 days.

The combined use of this circuit will affect a slower required loaded carbon transfer rate from the existing oxide circuit, and oxide carbon loadings of circa 700 g/t should be achieved. Currently the oxide plant feed averages 1 g/t Au and achieves only carbon loadings of 600 g/t.
MDM further recommends the consideration to install a tertiary heat exchanger in the 5 ton Zadra system. This will cool the feed to the electrowinning circuit to below 95°C, thus improving single electrowinning efficiency. A further advantage is a reduction in elution time, constituting a power saving and operating flexibility in this circuit. Cooling water could be supplied from the existing transfer water tank.

All the equipment in the goldroom is sized to accommodate the gold production from both the oxide and sulphide sections. The gold loading from the original sulphide section is much higher than that of the new circuit and it is assumed that no changes are required to this section. Any refurbishment will be undertaken as part of ongoing operation of the plant.

The existing oxide regeneration circuit is designed to regenerate 275 kg/h of carbon feed. This effectively treats the 5 ton batch in 20 hours. As the new plant will make use of this 5 ton elution circuit, the existing regeneration kiln will remain in use in this circuit. The only proposed modification will be to route 1 ton per day of regenerated carbon back to the existing sieve-bend sizing screen on top of the existing sulphide leach tanks.

Fresh virgin carbon will be added here as well and it is proposed that the option to split some of the virgin carbon to the front of the leach (leach tank 1) be considered. The reasoning is the concern regarding the carbon activity due to the extended residence time in the leach.

The existing oxide tailings system will be used. The existing system is shared between the oxide and sulphide plants and the existing tailings pumps are sized to cope with this load. It is proposed that the issue of planned maintenance downtime from either of the plants, and subsequent loss in processed volume to tailings, be addressed by means of process water addition to the existing tailings transfer tank. There is however an added risk of the coarse material settling in the tailings line should the transfer velocity not be maintained above the critical settling velocity of the coarsest particles. Currently none of the existing pumps are on a variable speed drive. The system is currently in use, and should therefore be suitable for the new PUG material as well. It is recommended that the pump duties in this section be confirmed for the expected life of mine.

The existing in-plant services were designed for the current oxides and sulphides which is a significantly larger demand than that of the oxide and new PUG treatment together. It is proposed that tie-ins to all existing services be made at the closest possible location. The new requirements would be lower than that of the original sulphide circuit and hence the current supply capacity is assumed to be able to accommodate the PUG treatment without negatively affecting the oxide circuit.
17 **Project Infrastructure** *(Item 18)*

17.1 **Introduction**

The WRP does not require substantial infrastructure in terms of roads and other facilities and it is in long-established mining town that is generally well served with roads and power infrastructure.

17.2 **Roads**

The Prestea Mine is located in a developed part of Ghana close to Prestea town with well-established road infrastructure. Two roads are required to service the operation as described below.

The haul to the Bogoso processing plant is 16 km and is part national highway and part private GSR haul road. The road alignment has been planned to service GSR’s Prestea South Mbease Nsuta Project (“PSMN”) that will also truck RoM material to the Bogoso plant. The capital cost of this road is covered by the PSMN although some operating costs for maintenance have been applied in the financial model for the WRP. The operating costs consist of a fixed monthly operating cost of US$ 40k for road repair and a trucking charge of 22.5 USc/kt.

17.3 **Waste Storage Facilities**

In general, waste rock produced from the WRP will be kept underground and used for backfill. During the early years of the WRP, there are insufficient voids available to allow back-filling to commence. In this case, the waste rock will be transported to surface.

In the first year of project development, about 40 kt will be hoisted up the Central Shaft. If the Prestea South Mine is operating, the waste will be moved to one of the waste dumps for storage. If the Prestea South Mine is not operating, the waste will be moved to the Buesichem Pit where it will be placed within the open pit void. Any additional waste that cannot be stored underground as backfill will be treated in the same manner.

17.4 **RoM Stockpiles**

No surface RoM stockpiles are planned at the WRP as all the material will be trucked directly from the shaft bin to the Bogoso plant ROM pad.

17.5 **Tailings Storage Facilities**

17.5.1 **Introduction**

The RoM from PUG will be processed in the facilities at Bogoso and the tailings will be delivered to the present Tailings Storage Facility (TSF), termed TSF2.

TSF2 is a paddock type facility consisting of three cells and 17 embankments (see Table 17-1). The WRP will generate approximately 0.65 Mt of tailings, which will be deposited into TSF2 cells 2/1 and 2A. Although the testing on the new Prestea tailings is not fully completed, it is anticipated that the tailings will be coarser than presently deposited in the TSF2.

17.5.2 **Site Description of TSF2**

The tailings from the existing Bogoso operation are stored in the TSF2. TSF2 is a paddock type facility consisting of three cells (2/1, 2A, and 3).
The tailings deposition management at Bogoso can be described as follows:

- TSF2 is a paddock type facility consisting of three cells:
  - Cell 2/1 and 2A receive CIL tailings from both the BIOX® and non-refractory CIL plants
  - Cell 3 receives BIOX® flotation tailings.
- Tailings deposition is generally sub-aerially via spigots around the perimeter of the facility.
- As required, tailings are placed sub-aquously to fill voids identified within the centre of certain cells.

Based on the memorandum by KP dated January 2013, Table 17-1 below shows the allowable elevations for the each embankment around cells 2/1, 2A and 3.

**Table 17-1 Permitted elevations for TSF2 as of June 13, 2012 (Knight Piésold, 2013)**

<table>
<thead>
<tr>
<th>Cell</th>
<th>Embankment</th>
<th>Maximum Elevation (mRL)</th>
</tr>
</thead>
<tbody>
<tr>
<td>2/1</td>
<td>1, 2, 3, 4, 5, 6, 7, 8, 9</td>
<td>90.8, 91.8</td>
</tr>
<tr>
<td>2A</td>
<td>11, 12</td>
<td>83.3</td>
</tr>
<tr>
<td>3</td>
<td>13, 16, 17, 18, 19, 20</td>
<td>86.0</td>
</tr>
</tbody>
</table>

As the refractory processing line will be suspended in late 2015, there is adequate tailings storage capacity for the tailings that would be generated from the WRP.

**17.5.3 Water Management**

There is a lot of surplus of water in TSF2 cell 2/1 and 2A that needs to be managed. The water treatment plant was built to treat the excess water. The plant was commissioned in Q4 2012 and has continued to treat excess water from the TSF 2, as well as water from the Buesichem process water storage facility.

The concept of storing the tailings from the WRP relies on the use of the currently-approved tailings storage facilities including the existing TSF2 (Cells 1, 2 and 2A), which is presently storing tailings from the Bogoso mine. Supernatant pond levels need to be maintained at operational levels through water management and treatment.

The volumetric analysis should be regularly undertaken to ensure that the adequate storage volume is achieved at the permitted elevations for TSF2. In addition, the densities achieved should be verified for each cell for the remaining Bogoso and WRP LoM plan.
18 Market Studies and Contracts (Item 19)

All gold production will be shipped to a South African gold refinery in accordance with a long-term sales contract currently in place for GSR’s Bogoso Mine. The gold is shipped in the form of dore bars which average approximately 90% gold by weight with the remaining portion being silver and other metals. The sales price is based on the London P.M. fix on the day of the shipment to the refinery.
Environmental Studies, Permitting and Social or Community Impacts (Item 20)

19.1 Introduction

The Prestea area has a long gold mining history; starting in the 18th Century, with official mining underway by about 1887. By 1929, the Ariston and Gold Coast Main Reef Mines were operational on the area covered by the current Prestea concession and by 1965, the gold mines at Prestea had been acquired by the Government of Ghana through the SGMC and the local mines were held by Prestea Goldfields Limited. Prestea Goldfields Limited continued operation until the 1990s, when the operations were privatized and eventually awarded to PGR in 2000. PGR operated for a short period and ceased operations in 2002.

Immediately PGR’s operations ceased, New Century Mines (“NCM”), a three-partner joint venture, consisting of the Ghana Government, GSBPL, and PGR was formed to take over the rights held by PGR on the Prestea concession. The underground mine was placed on care and maintenance by NCM, while GSBPL commenced studies on the potential for the re-opening of the underground operations.

Since March 2002, PUG has been under care and maintenance and GSBPL wishes to commence production. In mid-2012, GSBPL commenced the environmental and socioeconomic permitting for the Prestea WRP, which is Phase 2 of the operations at the Prestea underground mine.

19.1.1 Historic Environmental Liability and Indemnity

In June 2001, BGL acquired the surface and mineral rights to gold and other associated mineral substances within the Prestea Mining Lease. This area has been subjected to over 100 years of mining activity and BGL’s acquisition agreement provides indemnity from the Government of Ghana for the environmental liabilities emanating from these activities. Golden Star Resources later renamed BGL to GSBPL.

In order to quantify the nature of the existing liability at the time of the agreement of indemnity with the Government of Ghana, audits were completed and documented as registers of environmental liabilities.

19.1.2 Environmental and Socioeconomic Permitting

Since acquisition of the mineral concession in December 2001, GSBPL has undertaken a series of impact assessments on the Prestea concession to support the permitting of its various mining projects.

GSBPL completed the ‘Prestea Underground Mine Environmental Scoping Report and Terms of Reference’ in August 2012 for submission to the EPA as part of the WRP permitting process.

Golder Associates (Ghana) Limited (“Golder”) has been commissioned by GSBPL to undertake an EIA and produce an EIS for the environmental permitting of the proposed expansion of the Prestea WRP. The EIA is being produced in accordance with Environmental Assessment Regulations, LI1652, which is the principal enactment under Ghana’s Environmental Protection Agency Act 1994.
It is anticipated that there will be both potential environmental and social benefits and impacts associated with the project implementation, which will be considered and, where appropriate, enhanced or mitigated.

19.2 Environmental and Social Setting

19.2.1 Climate

A south-western equatorial climate type characterizes the Prestea area. The area is affected by moist, southwest monsoon winds and the northeast trade winds (Harmattan) that blow over the Sahara desert from the northern sub-tropical high-pressure zone. The annual rainfall pattern includes major and minor rainy seasons occurring from April to June and from October to November, respectively. The mean rainfall is in the region of 1,790 mm, with the monthly rainfall ranging from a minimum of 33 mm in January to a maximum of 253 mm in June.

19.2.2 Topography, soils and land use

The Prestea concession is within the Ankobra River drainage basin. The local elevation ranges from 30 m above sea level at the Ankobra River to over 200 m on the hills to the north of Bogoso. The reach of the Ankobra River near Prestea (and the WRP) is highly polluted and affected by extensive unauthorized small-scale mining. Marshlands are associated with several streams; many have been created or extended by damming and the diversion of streams by ‘unauthorized small scale mining’ and illegal medium scale mining activity, Figure 19-1.

![Unauthorized mining on the Ankobra River](image_url)
The land use capability of the area is limited as the soils are nutrient-poor due to natural leaching. The majority of the area in the project footprint has been disturbed, so reducing the agricultural land use capability. Within the study site, most of the natural soils have either been removed or affected by previous mining activity, forest timber removal, or rotational subsistence farming, thus contributing to soil erosion and nutrient depletion (Scott Wilson, 2003).

Common land uses in the area include residential, commercial, agricultural, forestry, agroforestry (palm oil and rubber plantations), and unauthorized small-scale mining operations.

19.2.3 Geochemistry

As noted above, the Prestea deposits lie within the West African Pre-Cambrian shield, a geological formation that hosts the Birimian and the Tarkwaian sequences. Gold mineralization is associated in some cases with reactive sulphide minerals, with the potential for acid rock drainage (“ARD”). Potential ARD generation from existing or future operation of PUG has been identified as a key area of environmental concern for GSR.

GSR conducted investigations to expand knowledge of the country rock geochemistry, which found that the geology and ARD potential of rock is complicated by extensive in-situ weathering and by the degree of structural and chemical alteration that has occurred within the shear zone hosting the gold mineralisation.

GSR’s approach is acid drainage management is to: conduct standard static geochemical test work to confirm the conceptual model developed by SENES in 1999; and conduct test work on the immediate hanging wall and footwall of the proposed West Reef mining operations.

The findings of the geochemical testing program will inform on the approach selected for rock management that will be approved by the regulatory agencies, via the submission of the environmental impact statement and issuance of a permit for the project.

19.2.4 Air Quality, Noise and Vibration

Existing Air Environment

The air quality within the project area is degraded to some extent and is affected by the seasonal Harmattan winds, the use of wood and charcoal for cooking, burning of domestic refuse, mining, logging, and agricultural practices. The impacts of the anthropogenic activities on air quality are generally localized and do not contribute substantially to a regional degradation of air quality.

Golden Star has been monitoring dust deposition across the Bogoso and Prestea concessions since early 2004, with monitoring points selected to represent receptor groups across the concessions (e.g. schools, communities, employees). To complement the dust depositional data, GSR also collects monthly air quality data for contaminants (NOx, SOx, CO, CO2), Total Suspended Particulate (“TSP”), and particulate matter in the sub 10 µm size range (inspirable) (PM$_{10}$).

Monitoring results from 2004 to present reflect the wet and dry climatic cycles, with local minor variation related to discrete sources such as roads and local ground disturbing activities. Expanded monitoring since 2011 continues to reflect the seasonal Harmattan effects on air quality. Throughout the baseline monitoring, NOx, primarily sourced from vehicle emissions, continued to be non-existent (fully dispersed) at the monitoring locations.
Predicted Air Quality, Noise and Vibration

The operations will use the existing infrastructure and air circulation options (upgraded). The mining operations will be similar to those used before the mine was placed on care and maintenance. Initial modeling of mechanical mining indicated that the operations should achieve the required limits for air quality, noise and vibration. Therefore, using hand held methods, the contaminant levels should be lower in all categories.

Predicted Vibration Impacts and Mitigation

Data for previous blasting on the concession (Agbeno, 2006) were used for ground vibration predictions. The impact assessment was based on a ‘worst case’ scenario assuming maximum explosive weights per delay period, and minimum distances between the source and receptor. Near surface blasting will not be required for mining. As drift and fill blasting will occur at depths of some 650 m or more, it is expected that all such blasting will achieve the required ground vibration limits. Should mitigations be required in the future for remnant mining closer to the surface, ground vibration effects can be reduced by reducing the maximum charge weight per delay.

19.2.5 Drainage and Water Resources

Surface Water Resources

The Prestea concession lies within the Ankobra River drainage system, which has an approximate catchment area of 3,300 km². Illegal medium-scale, and unauthorized small-scale mining, activities (Knight Piésold 2002) have environmentally degraded much of the project area.

Within the Prestea community, surface drainage and sewage control is almost non-existent. Therefore, extensive organic loadings and silt enter the surface drainage channels, further degrading the local surface water quality.

Groundwater Resources

Groundwater in the Prestea and Bondaye areas occurs primarily in weathered regolith aquifers that contain and transmit groundwater along zones of secondary permeability, which are often discrete and irregular, and occur as fractures, faults, lithological contacts, and zones of deep weathering. Groundwater bores in the Prestea concession demonstrate that aquifer horizons are associated with weathered and fractured argillaceous rocks at the base of the weathered layer. The Tarkwaian system has shallower aquifer horizons than the Birimian system; yields are higher in the Birimian than in the Tarkwaian.

Groundwater gradients are general towards the Ankobra River. This is influenced by the orientation of the water bearing strata, and the joints, faults and other structural features that provide the secondary porosity of the aquifer. Recharge of the aquifer strongly correlates with rainfall.

Underground Mining and Ground Water

After the cessation of the underground mining by PGR, the workings have been dewatered into the surface streams through pumping at two locations; the Prestea Central Shaft and the Bondaye Main Shaft. GSBPL retains data on dewatering volumes and quality dating back to January 2003.
In spite of the extensive dewatering associated with the underground mine workings that can be traced back about a century, nearby water supply boreholes continue to yield suitable volumes of water for local domestic water supply.

It can, thus, be concluded that while the effect of the underground mine workings is felt in groundwater levels above the workings, groundwater levels in wells less than 1 km to the west, supplying water to the Prestea area, remain at more or less constant levels, a reflection of the discontinuous nature of the groundwater aquifers in the area.

**Water Quality**

GSBPL commenced water quality monitoring in the Prestea concession in 2003 as part of the baseline data collection for the Plant North project; regional data earlier than this time are also available. Over 20 sites have been monitored routinely since this time. They demonstrate the following typical surface water characteristics:

- Extensive anthropogenic influences associated with sewage and small miner activity reducing the environmental values of surface water;
- Elevated arsenic concentrations in groundwater that comes into contact with the Prestea mineralization;
- Naturally elevated iron concentrations in ground water; and
- Poor surface water quality in the major surface water courses as affected by unauthorized small miner activity that is exacerbated in the dry season when flows approach or reach zero in many ephemeral streams.

The historical monitoring data will be summarised within the project EIS.

**19.2.6 Biodiversity**

Prestea falls within the rainforest bioclimatic zone and the moist evergreen forest type of Hall and Swaine (1981); there is none of this forest left in the project area as it has all been previously disturbed by logging, farming, and unauthorized small scale mining activity. Patches of secondary forest are present in some limited areas that are not generally accessible for farming. The vegetation cover has been seriously altered within the concession. The vegetation types typically encountered are: secondary forest, secondary thicket, farm re-growth, farmland, and marshes or freshwater swamp forest. No secondary forest will be disturbed for the Project, with proposed infrastructure to be sited at existing mine compounds or in ‘brownfields’ areas.

The closure phase of the project will provide GSBPL with the opportunity to conduct rehabilitation activities that may enhance and return an ecosystem to an area or complete the transformation to a land use that has been selected by the local stakeholder community.

**19.2.7 Social Setting**

The communities and organisations identified through the socioeconomic studies are located in the Prestea Huni-Valley District of the Western Region of Ghana. The 2010 population census (GSS, 2012) reported the population of the Prestea Huni-Valley District as 159 304 people, with 37% of this population residing in the urban centres of Bogoso and Prestea. More recent data, collated in 2011 indicate a population in the Prestea catchment of 38 390 in the Himan catchment of 17 074, and in the Bondaye catchment of 1 923.
19.2.8 Administrative structures and demography
The Prestea-Huni Valley District (“PHVD”), which hosts a number of important mining operations, was created in 2008, with Bogoso the district administrative capital. The Prestea-Huni Valley District Assembly is headed by the District Chief Executive (“DCE”), who is appointed by the Government. The key civil administrative bodies in the project area are the Bondaye Area Council and the Prestea Urban Council (PUC). The PUC has the responsibility for providing basic services like water, electricity, schools, and ablution facilities to the community. Its main source of revenue comes from a common fund from the DCE. Some funding for the Prestea Huni-Valley District Assembly is generated from royalties paid by mining companies in the District.

The WRP is within the Wassa Fiasse Traditional Area, for the Prestea area, is ruled by the Himan and Nsuta Mbease Divisional Chiefs.

The PHVD has had a long history of mining and its economy is dominated by mining and related activities. The other main economic activities are forestry and agriculture. The people outside the formal sector engage in cash crop and subsistence farming, and unauthorized mining. Other sources of employment include government, financial services, trading, and artisanal activities. Unauthorized mining is a source of livelihood for many of the local population and migrants.

The health services in the Prestea Huni-Valley District are offered at three main levels; hospitals, sub-district clinics and community clinics. The district hospital is the Prestea Government Hospital and provides services including; out-and in-patient care, maternity, dental, laboratory, X-ray and surgery, in addition to an expanded programme on immunization, management of epidemics, and public information and education. There are several private health delivery facilities at Prestea. GSBPL has constructed clinics in Bondaye and Bogoso. Communities are also serviced by a variety of traditional healers.

Interviews with a cross section of community members and data from the Ghana Health Service (Prestea Government Hospital) revealed the prevalence of the following diseases; malaria, acute respiratory condition, oral conditions, skin conditions, hypertension, diarrhoea, typhoid, intestinal worms, and anaemia, with malaria typically accounting for over 40% of the total cases (2008 through 2010).

Kumasi Ventilated Improved Pit (“KVIP”) latrine, tipping bucket latrines, and pan latrines are the main sanitation facilities available. Where latrines are not available, people use nearby bushes.

The PAP surveys found most people had some education, with 10 % having received no schooling. 56 % had junior high school or higher education levels.

19.3 Legal, Regulatory and Policy Framework
19.3.1 Environmental and Social Legislation Pertaining to Mining Projects
The Mining Act (Act 703 of 2006) requires that mines obtain environmental approvals from relevant environmental agencies as outlined in Table 19-1.

Ghanaian environmental legislation is well developed and is enforced by the EPA. The overarching Act that regulates the environmental regime of Ghana is the EPA Act (Act 490 of 1994). The main legal framework used by the EPA for regulating and monitoring mineral
operations is the Environmental Assessment Regulations, Legal Instrument 1652 of 1999 (LI 1652). The EPA grants environmental approvals for projects, in the form of an Environmental Permit, based on the findings of an environmental impact assessment, which also covers social aspects, as documented in an EIS report. For a mine, an EIS report must include a reclamation plan (Regulation 14 of LI 1652) and a provisional EMP. The EIS is subject to a public exhibition period, public hearing and review by the EPA before a permit can be granted. An EMP must be submitted within 18 months of commencement of operations and must be approved by the EPA.

Table 19-1 Primary Environmental Approvals Needed for Mining Operations

<table>
<thead>
<tr>
<th>Regulatory institution</th>
<th>Approvals that have to be obtained</th>
<th>Reporting, inspections and enforcement</th>
</tr>
</thead>
<tbody>
<tr>
<td>The Environmental Protection Agency (EPA) Established under the Environmental Protection Agency Act, 1994 (Act 490), the EPA is responsible for among other things, the enforcement of environmental regulations.</td>
<td>Environmental Permit In accordance with Section 18 of the Mining Act (Act 703 of 2006) and the Environmental Assessment Regulations, 1999 (LI 1652) of the EPA, a holder of a mineral right requires an environmental permit from the EPA in order to undertake any mineral operations. In most cases, an EIS is to be submitted to the EPA for the approval of a mining project. Environmental management plan (EMP) An EMP must be submitted within 18 months of commencement of operations and updated every three years (Regulation 24 of LI 1652). Environmental Certificate This must be obtained from the EPA within 24 months of commencement of an approved undertaking (Regulation 22 of LI 1652). Approved reclamation plan Mine closure and decommissioning plans have to be prepared and approved by the EPA; Provisional plans are submitted in the EIS (Regulation 14 of LI 1652). Reclamation bond Mines must post a reclamation bond based on an approved reclamation plan (Regulation 22 of LI 1652). The reclamation bond is often part of the environmental permit.</td>
<td>Annual reports Mines must submit annual environmental reports to the EPA. Inspections The EPA undertakes regular inspections to ensure that mineral right holders are compliant with permit conditions and the environmental laws generally. Enforcement The EPA is empowered to suspend, cancel or revoke an Environmental Permit or certificate and/or even prosecute offenders when there is a breach.</td>
</tr>
<tr>
<td>Water Resources Commission (WRC) Established under the Water Resources Commission Act, 1996 (Act 522), the WRC is responsible for the regulation and management of the use of water resources.</td>
<td>Approvals for water usage Under Section 17 of the Mining Act (Act 703 of 2006), a holder of a mineral right may obtain, divert, impound, convey and use water from a watercourse or underground reservoir on the land of the subject of the mineral right, subject to obtaining the requisite approvals under Act 522. The Water Use Regulations, 2001 (LI 1692) regulate and monitor the use of water.</td>
<td>Inspection The WRC has power to inspect works and ascertain the amount of water abstracted. Enforcement Both Act 522 and L.I. 1692 prescribe sanctions for breaches.</td>
</tr>
<tr>
<td>Forestry Commission</td>
<td>In accordance with Section 18 of the Mining Act (Act 703 of 2006), a holder of a mining right must obtain necessary approvals from the Forestry Commission.</td>
<td></td>
</tr>
</tbody>
</table>
19.3.2 Stakeholder Engagement

Key stages of consultation that remain for the Prestea Underground Phase 2 operations include the following:

- Feedback to key stakeholders on the finalisation of project designs and environmental and social controls;
- Submission of the EIS to regulatory agencies;
- EIS public exhibition period; and
- EPA public hearing following EIS public exhibition.

19.3.3 Resettlement and Land Acquisition

The preparation of the Resettlement Action Plan ("RAP") for the Prestea projects is in accordance with the relevant legal and regulatory framework operating in Ghana and the International Finance Corporation Performance Standard 5. No resettlement is required for the WRP.

19.3.4 Status of Project Permitting

West Reef Project

On 30 May 2012, GSBPL submitted a project registration form EA2 to the EPA in respect of the WRP. In August 2012, the EPA advised that an environmental impact assessment should be conducted as the basis for consideration for an environmental approval.

In July 2012, GSBPL engaged Golder to conduct an environmental and socioeconomic impact assessment for the PUG Project Phase 2 operations (the WRP under another name). These impact assessments are well advanced and are presently being updated to reflect the modified scope of work.

The Prestea Projects Resettlement Action Plan was submitted to the Prestea Huni-Valley District Assembly in November 2012 and approved by the PHDA on 1 May 2014. The final RAP submitted to the PHDA on 29 September 2014.

GSBPL General Operations

Regulatory approvals held or being permitted by GSBPL and pertaining to the Prestea Underground Project are listed in Table 19-2.

In accordance with LI 1652, GSBPL submitted an updated EMP for its operations to the EPA in December 2011, ahead of the expected expiry of the previous EMP. GSBPL was subsequently invoiced by the EPA and payment was made for the Environmental Certificate in March, 2013. GSBPL is yet to be issued with the updated Environmental Certificate for the operation of the overall project.

Table 19-2 Existing and Pending Regulatory Approvals Pertaining to Prestea Mine
### Status

<table>
<thead>
<tr>
<th>Status</th>
<th>Type of approval</th>
<th>Approval</th>
</tr>
</thead>
</table>
| Existing     | Mining leases    | - Mining Lease WR 348B/87 (21st August 1987) and WR 368/88 (16th August 1988) (Bogoso Lease Area). Site of Bogoso processing plant.  
These stipulate conditions for the encroachment of mining activities on community infrastructure, the disturbance of vegetation, the conservation of resources, reclamation of land and prevention of water pollution. |
| Environmental |                  | - Environmental permit EPA/EIA/147 (Sulphide Project). Operation of the Bogoso Processing Plant.  
- Environmental permit EPA/EIA/188 (Tailings Storage Facility II Extension)  
- Environmental permit EPA/EIA/804 (Prestea underground gold mining project Phase 1) |
| Water use     |                  | - EPA Approval letter CM 8/7 of September 2011 to use the Buesichem Pit for TSF water storage.  
- Approval letter CM 81/8 of October 2012 for the storage of process water in the Buesichem Pit.  
- Water Resources Commission permit for water use and abstraction from ground water. Most of the water used in the process is recycled from the tailings storage facility.  
- Water Use Permit WRC (GSBPL ID 228/11) issued on 1-Jan-11 and valid to 31-Dec-13 was received from WRC on 4-Apr-11. Permit renewal in process. |
| Pending       | Environmental    | - Prestea South Mbease Nsuta Project: Updated EIS submitted to EPA in March 2013, and public exhibition period commenced in April 2013. Comments were received from the EPA and submission of the final EIS is anticipated no later than Q1 2015.  
- Prestea Projects Resettlement Action Plan (RAP): The RAP was approved by the Prestea Huni Valley District Assembly on 1 May 2014. |
| Water use     |                  | - A Water Treatment Plant letter report was submitted to EPA on 22nd June 2011. Approval from EPA was received (verbal) in November, 2012 and process water treatment plant is operating |
| Documents to be submitted | Environmental | - Environmental Impact Statement for the Prestea Underground (West Reef Project) Phase 2 to the EPA for review and approval. |

### 19.3.5 Voluntary Codes

Golden Star Resources has adopted a number of voluntary international codes and standards of practice pertaining to corporate responsibility:

- Tailings storage facilities - International Committee on Large Dams (“ICOLD”);  
- Gold mining and processing - World Gold Council Responsible Gold Standard; and  
As GSR has adopted these voluntary standards and codes, a key component of GSR’s corporate assurance includes independent review, audit, and/or validation of conformance to the principles ascribed herein.

19.3.6 Golden Star Corporate Commitment

Golden Star Resources has policies pertaining to the environment, community relations and human rights, and health, safety, and wellbeing. These policies enunciate the commitment of the company to appropriate corporate governance, are reviewed annually, and are endorsed by the company President / CEO.

19.4 Impact Assessment Approach

The environmental and socioeconomic impact assessment process for the PUG Phase 2 (WRP) operations is advancing as scheduled, with the majority of the baseline studies now completed. The impact assessment team is now undertaking iterations of the various impact assessment models to refine and define the mitigations to be implemented for the project and to reflect the project engineering.

Upon conclusion of these design parameters, the EIS will be documented for submission to regulatory agencies. Given GSR’s experience in environmental and social impact assessments for other projects, it is reasonable to assume that the EIA report will meet the Ghanaian environmental requirements.

19.5 Land Acquisition

No additional land acquisition is required for the project.

19.6 Identification of Project Affected Persons

The project will be developed entirely within existing areas. The movement of the RoM material to the Bogoso processing plant, and any waste rock to the PSP waste dumps or the Buesichem pit, will be at a low level and will use existing infrastructure.

19.7 Security

For the WRP, security is provided by dedicated security personnel. GSBPL, through their security provider, ensures these personnel are appropriately trained.

19.8 Stakeholder Engagement

19.8.1 Overview of Consultation Approach - Stakeholders

Consultations on the need for possible resettlement of identified PAP’s commenced in October 2006. Various concerns about both the Prestea Projects and the resettlement of affected hamlets and communities were raised. Consultations continued throughout 2007 to 2009 with numerous efforts to hold a public hearing on the PSMN project. Consultations continued in 2010 and in 2012 as updated socioeconomic and environmental impact assessments were progressed. From 2012, GSBPL developed a number of project designs, started the collection of further socioeconomic data, and undertook further consultation to communicate these aspects to stakeholders.
The design assessments enabled GSBPL to understand the relative importance of the Valued Socioeconomic Components (“VSEC”) potentially affected by the project, and identify project designs to avoid or minimise impacts to the VSEC’s whilst maximising benefits.

19.9 Summary of key stakeholder issues

The most frequently discussed and critical stakeholder issues have been with respect to the following:

- Local employment and contracting;
- Potential project impacts (dust, noise, vibration, land acquisition) and mitigation measures;
- Land acquisition, resettlement and compensation;
- Mediation and agreement for building mutual trust between GSBPL and the community;
- Alternative livelihoods and community development interventions; and
- Negotiations with small-scale miners and ceding parts of Prestea concession to relocate small-scale miners.

19.9.1 Grievance Mechanism

GSBPL maintains a grievance mechanism enabling catchment communities to document concerns and grievances for investigation / action. The mechanism is well publicised by GSBPL and used actively by the community and other stakeholders. Details of registered grievances and their resolution are recorded and reported internally and to the regulators.

19.10 Substantive Issues

Substantive environmental and social issues identified for the project are as follows:

- Potential acid generating materials and their management;
- Avoidance of cumulative impacts to aspects subject to the Environmental Indemnity Agreement;
- Interactions with activities of illegal and unauthorized small scale miners; and
- Safety of community members adjacent to mine workings.

19.10.1 Acid Rock Generation

It has been identified by historic studies that transitional rock and fresh rock in the geological region are Potentially Acid Generating (PAG). Mine water for the existing Prestea underground mine show evidence of ARD, neutral leaching, as well as saline drainage. To mitigate for these potential impacts, the proposed project design incorporates the storage of waste rock such that acid generation is prevented. GSBPL also has the opportunity to preferentially backfill these materials within the underground mine voids (as allowed by mine scheduling).

19.10.2 Cumulative Impacts to Areas of Historic Liability

The Government of Ghana is accountable for the rectification of the historic environmental degradation of the Prestea Concession. Should the proposed GSBPL underground mine activities contribute to a further deterioration of environmental conditions, then it would be expected that
the Government of Ghana would seek to transfer its liabilities to GSBPL. This could have substantive cost implications for the project.

**19.10.3 Interactions with Activities of Illegal Miners**

Interactions with the activities of unauthorized small-scale miners and illegal medium scale miners could lead to the following:

- Physical intersection between un-surveyed historic workings and the associated risk of inrush of water and flooding of the WRP;
- Actions of the Government of Ghana, including the National Security Council and perceived involvement of GSBPL in the actions taken by these groups to regulate the activities of unauthorised miners; and
- Ability of unauthorized small-scale miners and their supporters to jeopardise the project through political means by making spurious claims of impacts to jeopardise WRP activities.

**Safety of community members adjacent to mine workings**

Accidents involving local communities due to trucks on the haulage road, leading to community uprising, potential loss of social license to operate and the temporary inability to deliver RoM material to the processing plant.

**19.11 Closure Requirements and Costs**

The closure requirement will be outlined in the reclamation bonding document that is expected to be similar to other such documents currently held by GSBPL. As the current infrastructure is must be returned to SGMC at the end of the mine life, the rehabilitation and closure costs will be minimal. The main structure outside the existing NCM mining area will be the waste rock storage area, which will be covered under either the Buesichem Pit closure or the Prestea South Project closure.

**19.12 Risks**

In the development of the project, risk assessment workshops were conducted by Golder Associates (EIS consultant), and SRK Consulting (FS consultant) in conjunction with Golden Star Resources and GSBPL team members, to identify risks with the potential to affect the project or the company. Risks were recorded in a risk register, and ranked in inherent and residual conditions. Only material environmental and social risks (i.e. those that may stop the project or have significant cost implications) have been discussed in this section.

**19.12.1 Permitting Schedule cannot match the Construction Schedule**

There is a risk of project construction activities being delayed due to the environmental permit not having been obtained prior to the construction commencement date. GSR average for obtaining mining permits is longer than 365 days on recent projects. It is generally accepted that the West Reef Project is desired by the authorities but the exact timing of the permitting is unclear. There are various environmental approvals needed for the mining operation and the refusal of any of these permits by government could delay the commencement of the mine and have serious cost implications. Broad community support for this development project means that such delays are considered unlikely.
19.12.2 Regulatory Evolution

Sovereign risk that is manifested as changing and un-formalised expectations of regulators ‘enforced’ through project permitting, e.g. EPA have recently retracted Environmental Permits (of a peer company) to modify operating conditions to best practices from a previous position of industry standard practice.

As part of the environmental management for the site, adaptive management is included. This allows the site personnel to adjust to ongoing changes in regulations and EPA requirements. Therefore, the site is well-placed to predict these changes and stay ahead of the advancing legislation.

19.12.3 Loss of Community Support

This could be caused by a number of different issues as follows:

- Lack of transparency with respect to flow of project revenues to the local authority - failure by Government to redirect royalties and government payments to entitled parties, including Traditional Authorities and Stool Lands, leads to a stakeholder agitation / campaign against GSBPL to motivate for action, which could threaten the company’s community support.
- Unmet expectations: negotiations – through various iterations, the Prestea Goldfields International School was thought to require resettlement. However, this is not the case with the current project. Therefore expectations need to be managed.
- Unmet expectations: job creation - The community expects that the expansion of the underground operation would result in high levels of job creation.
- Obstruction from unauthorized small scale miners - perception or allegation that GSBPL is involved in, or associated with, the actions of Government officers or security forces in removing / regulating unauthorised miners, i.e. small scale miners could be promoted.

19.12.4 Entanglement of WRP with other Prestea Projects

The reliance of the PUG on other GSBPL infrastructure (e.g. tailings storage facility and processing plant) places it at risk if there are environmental and/or social risks associated with this existing infrastructure that have not been identified.

In addition, communities do not differentiate between the WRP from other Prestea Projects. Risks to community support on any of the Prestea Projects will ultimately affect the proposed underground project. GSBPL would manage this risk by ensuring that similar environmental and social standards are maintained across all its operations.

19.12.5 Transfer of Historic Liabilities

In order to reduce its liability or influence for environmental improvement, the Government of Ghana / regulators place pressure on GSBPL to remediate historically disturbed lands, manage PCBs and other environmental contaminants, encapsulate / backfill historic waste and tailings piles, treat underground mine water discharges (e.g. from Bondaye or Central shafts) that are each subject to the environmental indemnity components of the lease acquisition agreement. There could also be a move to enforce strict water quality requirements on water bodies that are outside the mines’ control (i.e. where unauthorized small scale mining activities are taking place).

19.12.6 Full Water Treatment at Prestea
Ghana EPA specifies full water treatment at Prestea. This would add around US$ 6 million to capital costs and several million dollars annually to operating costs.

**19.12.7 Failure to Develop Prestea Underground Project**

Failure by the company to develop the Prestea Underground Mine results in major reputational risk to Golden Star and GSBPL including loss of Government support, loss of trust by the community and resulting impact on future project development and permitting.

**19.12.8 Acid Rock Drainage**

Whilst management of PAG rock can affect the environment and have increased operating costs, the controls that have been incorporated into the project design reduce the risk levels.

**19.12.9 Flooding of the Mine from Historic Workings**

There is a risk of the historical Ankobra and Beta shafts, which are located in a depression close to a stream, possibly flooding the main mine workings through the in-rush of water from these historic surface and underground workings. This may also take place through tunnels dug by artisanal miners.

**19.13 Recommendations and Way Forward**

**19.13.1 Summary**

The corporate responsibility (environmental and social) planning for the Prestea WRP has been extensive. The corporate responsibility team was an integral part of the design engineering team, so allowing for a seamless approach to corporate responsibility planning within the Project. This interaction allowed for a complete assessment of options for the project components and their interaction with the environment and socioeconomic fabric of the Project area.

**19.13.2 Additional Studies**

With the commissioning of the PUG Project Phase 1 recovering some remnant mineralized material from the mine, expectations within the stakeholder communities are high. Therefore, regular meetings with key stakeholder groups (CMCC and other) will be held to update the broader stakeholder community on developments within the Project and allow the population to understand the processes of approval and project development. Some additional studies for the EIS are required including the following:

- Additional air quality modelling to reflect the changes to the project parameters;
- Risk assessment for development and safety management, more specifically with the interaction with the un-surveyed unauthorized small mining workings;
- Evaluation of the acid generation potential for the waste rock that will be moved to surface; and

Opportunities for synergies with other GSBPL projects.
20 Capital and Operating Costs (Item 21)

20.1 Capital Costs

Table 20-1 presents the capital cost schedule for the West Reef project.

### Table 20-1 Capital cost summary

<table>
<thead>
<tr>
<th>Category</th>
<th>Description</th>
<th>Unit</th>
<th>Contingency</th>
<th>TOTAL US$m</th>
<th>2015</th>
<th>2016</th>
<th>2017</th>
<th>2018</th>
<th>2019</th>
<th>2020</th>
</tr>
</thead>
<tbody>
<tr>
<td><strong>Infrastructure upgrade projects</strong></td>
<td>Central Shaft Upgrade</td>
<td>$m</td>
<td>25%</td>
<td>0.2</td>
<td>0.2</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
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<tr>
<td></td>
<td>Bondaie Shaft Upgrade</td>
<td>$m</td>
<td>11%</td>
<td>0.7</td>
<td>0.7</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
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<tr>
<td></td>
<td>CS Hoist Upgrade</td>
<td>$m</td>
<td>16%</td>
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<td>1.5</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
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<tr>
<td></td>
<td>Bondaie Hoist Upgrade</td>
<td>$m</td>
<td>10%</td>
<td>0.0</td>
<td>0.0</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
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<tr>
<td></td>
<td>Electrical Upgrade</td>
<td>$m</td>
<td>12%</td>
<td>8.7</td>
<td>8.7</td>
<td>-</td>
<td>-</td>
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<td>-</td>
<td>-</td>
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<tr>
<td></td>
<td>17L Re-tracking</td>
<td>$m</td>
<td>25%</td>
<td>0.5</td>
<td>0.5</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
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</tr>
<tr>
<td></td>
<td>24L Re-tracking</td>
<td>$m</td>
<td>25%</td>
<td>1.0</td>
<td>1.0</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
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<tr>
<td></td>
<td>Ventilation System Upgrade</td>
<td>$m</td>
<td>10%</td>
<td>1.0</td>
<td>1.0</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td></td>
<td>Env and Soc Permitting</td>
<td>$m</td>
<td>0%</td>
<td>0.7</td>
<td>0.6</td>
<td>0.0</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td></td>
<td>25L Loading pocket</td>
<td>$m</td>
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<td>-</td>
<td>-</td>
<td>-</td>
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<tr>
<td></td>
<td>Compressed air</td>
<td>$m</td>
<td>13%</td>
<td>3.5</td>
<td>3.5</td>
<td>-</td>
<td>-</td>
<td>-</td>
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<tr>
<td></td>
<td>Dewatering Upgrade</td>
<td>$m</td>
<td>13%</td>
<td>1.9</td>
<td>1.6</td>
<td>0.3</td>
<td>-</td>
<td>-</td>
<td>-</td>
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<tr>
<td></td>
<td>Water Treatment</td>
<td>$m</td>
<td>30%</td>
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<td>0.1</td>
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<td>-</td>
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</tr>
<tr>
<td></td>
<td>Surface infrastructure</td>
<td>$m</td>
<td>11%</td>
<td>1.4</td>
<td>1.2</td>
<td>0.3</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td></td>
<td>Accommodation</td>
<td>$m</td>
<td>20%</td>
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<td>0.6</td>
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<td>-</td>
<td>-</td>
<td>-</td>
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<tr>
<td></td>
<td>Bogoos plant modifications</td>
<td>$m</td>
<td>20%</td>
<td>7.9</td>
<td>7.3</td>
<td>0.5</td>
<td>-</td>
<td>-</td>
<td>-</td>
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<tr>
<td><strong>Indirect</strong></td>
<td>Engineering</td>
<td>$m</td>
<td>14%</td>
<td>1.8</td>
<td>1.4</td>
<td>0.4</td>
<td>-</td>
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</tr>
<tr>
<td></td>
<td>Owners costs</td>
<td>$m</td>
<td>23%</td>
<td>0.4</td>
<td>0.3</td>
<td>0.1</td>
<td>-</td>
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<td>-</td>
</tr>
<tr>
<td><strong>Care &amp; Maint.</strong></td>
<td>Power</td>
<td>$m</td>
<td>0%</td>
<td>6.5</td>
<td>2.6</td>
<td>3.9</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
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<tr>
<td></td>
<td>Labor</td>
<td>$m</td>
<td>0%</td>
<td>4.9</td>
<td>1.9</td>
<td>2.9</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
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<tr>
<td></td>
<td>Hoist/headframe rental</td>
<td>$m</td>
<td>0%</td>
<td>6.8</td>
<td>1.1</td>
<td>1.1</td>
<td>1.1</td>
<td>1.1</td>
<td>1.1</td>
<td>1.1</td>
</tr>
<tr>
<td></td>
<td>Services and consumables</td>
<td>$m</td>
<td>0%</td>
<td>3.1</td>
<td>1.6</td>
<td>1.6</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td></td>
<td>Equip. and Maint. cost</td>
<td>$m</td>
<td>0%</td>
<td>2.5</td>
<td>0.3</td>
<td>2.2</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td><strong>West Reef Development</strong></td>
<td>Incline/Decline</td>
<td>$m</td>
<td>15%</td>
<td>0.9</td>
<td>-</td>
<td>0.6</td>
<td>0.1</td>
<td>0.2</td>
<td>-</td>
<td>-</td>
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<tr>
<td></td>
<td>FW drive</td>
<td>$m</td>
<td>15%</td>
<td>1.0</td>
<td>-</td>
<td>0.8</td>
<td>0.1</td>
<td>0.1</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td></td>
<td>Vent/O/P access</td>
<td>$m</td>
<td>0%</td>
<td>0.2</td>
<td>-</td>
<td>0.1</td>
<td>0.0</td>
<td>0.1</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td></td>
<td>Vent raise</td>
<td>$m</td>
<td>0%</td>
<td>0.8</td>
<td>-</td>
<td>0.6</td>
<td>-</td>
<td>0.2</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td></td>
<td>O/P raise</td>
<td>$m</td>
<td>0%</td>
<td>0.2</td>
<td>-</td>
<td>0.2</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td></td>
<td>Stope development</td>
<td>$m</td>
<td>35%</td>
<td>2.2</td>
<td>-</td>
<td>2.2</td>
<td>-</td>
<td>-</td>
<td>-</td>
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</tr>
<tr>
<td></td>
<td>Stopping</td>
<td>$m</td>
<td>35%</td>
<td>1.5</td>
<td>-</td>
<td>1.5</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td></td>
<td>West Reef Equipment</td>
<td>$m</td>
<td>25%</td>
<td>6.3</td>
<td>1.4</td>
<td>4.5</td>
<td>-</td>
<td>0.4</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td></td>
<td>Closure</td>
<td>$m</td>
<td>0%</td>
<td>2.0</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>2.0</td>
</tr>
<tr>
<td></td>
<td>Sustaining capital</td>
<td>$m</td>
<td>0%</td>
<td>22.3</td>
<td>-</td>
<td>-</td>
<td>6.9</td>
<td>6.3</td>
<td>6.6</td>
<td>2.5</td>
</tr>
<tr>
<td><strong>Total Capital</strong></td>
<td></td>
<td>$m</td>
<td></td>
<td>98.7</td>
<td>39.6</td>
<td>24.0</td>
<td>8.3</td>
<td>8.4</td>
<td>7.7</td>
<td>5.6</td>
</tr>
</tbody>
</table>

The capital costs presented are to a PEA level of accuracy with contingency applied by item depending on the level of confidence and the estimation methodology.

20.2 Operating Costs

PEA level operating costs were estimated as follows:

- Mining costs – zero based personnel, equipment and maintenance, supplies and power estimate, including 35% contingency;
• Haulage costs – based on current haulage contracts in place for the Bogoso Mine;
• Processing costs – based on GSR and MDM estimate for operating the plant as described in Section 16;
• Haulage costs – based on current haulage contracts at Bogoso Mine;
• G&A – zero based personnel, equipment and maintenance, supplies and power estimate.

Table 20-2 summarizes the operating costs incorporated in the economic analysis.

**Table 20-2 Operating cost summary**

<table>
<thead>
<tr>
<th>Item</th>
<th>Cost</th>
<th>Unit</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mining cost</td>
<td>134</td>
<td>$/t-mined</td>
</tr>
<tr>
<td>Processing cost</td>
<td>25</td>
<td>$/t-processed</td>
</tr>
<tr>
<td>Haulage cost</td>
<td>3.60</td>
<td>$/t-processed</td>
</tr>
<tr>
<td>G&amp;A cost</td>
<td>27</td>
<td>$/t-processed</td>
</tr>
</tbody>
</table>
21 Economic Analysis (Item 22)

21.1 Model Inputs

The assumptions used in the preparation of the economic analysis are presented in Table 21-1.

Table 21-1 Economic model parameters

<table>
<thead>
<tr>
<th>Description</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Mine Life</td>
<td>6 years</td>
</tr>
<tr>
<td>Gold Recovered</td>
<td>323 k ounces</td>
</tr>
<tr>
<td>Gold Price</td>
<td>$1,200 per ounce</td>
</tr>
<tr>
<td>Mill Recovery</td>
<td>90%</td>
</tr>
<tr>
<td>Royalty</td>
<td>5%</td>
</tr>
<tr>
<td>Income Tax</td>
<td>0%</td>
</tr>
<tr>
<td>Discount Rate</td>
<td>5%</td>
</tr>
</tbody>
</table>

The total mine life for the West Reef resource is 6 years, including a one year pre-development phase. The mine will have a nominal production rate of 480t/day.

The projected revenues are based on a market gold price of $1,200/oz.

Mill recovery through the Bogoso Oxide Plant is expected based on test results to be 90%.

The economic analysis present post-tax results with a 0% tax rate. Prestea Mine is a subsidiary of Golden Star Bogoso/Prestea Ltd. for which a tax loss of US$ 400 million has been assessed. This figure represents tax pools available at both Prestea and Bogoso operations. This results in no income tax being payable for the WRP.

A government royalty of 5% of gross revenue is also included.

21.2 Project Economic Results

Table 21-2 presents the after-tax economic model results.

Project net present value at a discount rate of 5% is US$121m with an internal rate of return of 72%. The payback period is 2.5 years from the time of initial investment. Total operating cost is estimated to be US$429 per ounce produced.

The cash profile for the project is summarized in Figure 21.1.
### Table 21-2 Economic Model

<table>
<thead>
<tr>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
<th></th>
</tr>
</thead>
<tbody>
<tr>
<td>ROM tonnes mined</td>
<td>kt</td>
<td>649</td>
<td>-</td>
<td>60</td>
<td>148</td>
<td>171</td>
<td>176</td>
</tr>
<tr>
<td>Grade mined</td>
<td>g/t</td>
<td>17.2</td>
<td>-</td>
<td>15.5</td>
<td>19.6</td>
<td>16.0</td>
<td>16.0</td>
</tr>
<tr>
<td>Contained ounces</td>
<td>koz</td>
<td>359</td>
<td>-</td>
<td>30</td>
<td>93</td>
<td>88</td>
<td>90</td>
</tr>
<tr>
<td>Horiz/incline dev - Capital</td>
<td>m</td>
<td>4,977</td>
<td>-</td>
<td>4,220</td>
<td>301</td>
<td>457</td>
<td>-</td>
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<tr>
<td>Raise dev - Alimak</td>
<td>m</td>
<td>723</td>
<td>-</td>
<td>635</td>
<td>-</td>
<td>88</td>
<td>-</td>
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<tr>
<td>Horiz dev - Operating</td>
<td>m</td>
<td>2,331</td>
<td>-</td>
<td>-</td>
<td>1,819</td>
<td>512</td>
<td>-</td>
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<tr>
<td>Processed</td>
<td>Ounces recovered</td>
<td>%</td>
<td>96</td>
<td>323</td>
<td>27</td>
<td>84</td>
<td>79</td>
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<tr>
<td>Revenue</td>
<td>$/1,200/oz</td>
<td>Sm</td>
<td>388</td>
<td>-</td>
<td>32</td>
<td>101</td>
<td>95</td>
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<tr>
<td>Royalty</td>
<td>%</td>
<td>$m</td>
<td>19</td>
<td>-</td>
<td>2</td>
<td>5</td>
<td>5</td>
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<tr>
<td>Net revenue</td>
<td>$m</td>
<td>369</td>
<td>31</td>
<td>96</td>
<td>90</td>
<td>93</td>
<td>59</td>
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<tr>
<td>Operating Cost</td>
<td>$/333/t</td>
<td>Sm</td>
<td>83</td>
<td>-</td>
<td>23</td>
<td>23</td>
<td>22</td>
</tr>
<tr>
<td>Mining</td>
<td>$/29/t</td>
<td>Sm</td>
<td>19</td>
<td>-</td>
<td>2</td>
<td>4</td>
<td>5</td>
</tr>
<tr>
<td>Haul and Process</td>
<td>$/20/t</td>
<td>Sm</td>
<td>18</td>
<td>-</td>
<td>5</td>
<td>5</td>
<td>5</td>
</tr>
<tr>
<td>G&amp;A</td>
<td>$/180/t</td>
<td>Sm</td>
<td>119</td>
<td>2</td>
<td>31</td>
<td>32</td>
<td>32</td>
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<tr>
<td>Total Operating Cost</td>
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<td>269</td>
<td>30</td>
<td>69</td>
<td>63</td>
<td>66</td>
<td>40</td>
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<tr>
<td>Operating Cash Flow</td>
<td>$m</td>
<td>269</td>
<td>30</td>
<td>69</td>
<td>63</td>
<td>66</td>
<td>40</td>
</tr>
<tr>
<td>Capital</td>
<td>$m</td>
<td>5.3</td>
<td>4.5</td>
<td>0.3</td>
<td>0.5</td>
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<td>-</td>
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<tr>
<td>Development</td>
<td>$m</td>
<td>1.5</td>
<td>1.5</td>
<td>-</td>
<td>-</td>
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<td>-</td>
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<td>Initial stopping</td>
<td>$m</td>
<td>6.3</td>
<td>1.4</td>
<td>4.5</td>
<td>-</td>
<td>0.4</td>
<td>-</td>
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<td>Mining equipment</td>
<td>$m</td>
<td>30.2</td>
<td>29.0</td>
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<td>-</td>
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<td>Infrastructure rehab</td>
<td>$m</td>
<td>2.2</td>
<td>1.7</td>
<td>0.5</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Owners cost</td>
<td>$m</td>
<td>23.8</td>
<td>7.5</td>
<td>11.7</td>
<td>1.1</td>
<td>1.1</td>
<td>1.1</td>
</tr>
<tr>
<td>Care &amp; Maint.</td>
<td>$m</td>
<td>2.0</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
<td>-</td>
</tr>
<tr>
<td>Closure</td>
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<td>22.3</td>
<td>-</td>
<td>-</td>
<td>6.9</td>
<td>6.3</td>
<td>6.6</td>
</tr>
<tr>
<td>Sustaining Capital</td>
<td>$m</td>
<td>93.7</td>
<td>39.6</td>
<td>24.0</td>
<td>8.3</td>
<td>8.4</td>
<td>7.7</td>
</tr>
<tr>
<td>Total Capital</td>
<td>$m</td>
<td>156</td>
<td>(40)</td>
<td>5</td>
<td>56</td>
<td>50</td>
<td>53</td>
</tr>
<tr>
<td>Net Cash Flow</td>
<td>$m</td>
<td>156</td>
<td>(40)</td>
<td>5</td>
<td>56</td>
<td>50</td>
<td>53</td>
</tr>
<tr>
<td>Cum. NCF</td>
<td>$m</td>
<td>(40)</td>
<td>(35)</td>
<td>21</td>
<td>71</td>
<td>124</td>
<td>156</td>
</tr>
<tr>
<td>DOC (opex excl. royalty and capex)</td>
<td>$/oz</td>
<td>369</td>
<td>64</td>
<td>374</td>
<td>406</td>
<td>389</td>
<td>431</td>
</tr>
<tr>
<td>TOC (DOC + royalty)</td>
<td>$/oz</td>
<td>429</td>
<td>124</td>
<td>434</td>
<td>466</td>
<td>449</td>
<td>491</td>
</tr>
<tr>
<td>ASC (TOC + sustaining capex + C&amp;M)</td>
<td>$/oz</td>
<td>512</td>
<td>502</td>
<td>470</td>
<td>500</td>
<td>484</td>
<td>501</td>
</tr>
<tr>
<td>Total Cost</td>
<td>$/oz</td>
<td>659</td>
<td>958</td>
<td>473</td>
<td>512</td>
<td>484</td>
<td>501</td>
</tr>
</tbody>
</table>

| NPV ($m) | 0% | 156 | 5% | 121 | 10% | 95  | 15% | 75  |
| IRR | 72% |
| Payback (yr) | 2.5 |
Table 21-3 presents the project sensitivity to gold price, grade, operating cost and capital cost. The project is most sensitive to grade/gold price. Since operating cost and capital cost are approximately equal in total value the sensitivity is the same for each.

Table 21-3 Project sensitivity NPV₅% (US$m)

<table>
<thead>
<tr>
<th>Gold price sensitivity</th>
<th>1000</th>
<th>1100</th>
<th>1200</th>
<th>1300</th>
<th>1400</th>
</tr>
</thead>
<tbody>
<tr>
<td></td>
<td>75</td>
<td>98</td>
<td>121</td>
<td>144</td>
<td>167</td>
</tr>
</tbody>
</table>

<table>
<thead>
<tr>
<th>Grade and cost sensitivity (% change)</th>
<th>-20%</th>
<th>-10%</th>
<th>-</th>
<th>10%</th>
<th>20%</th>
</tr>
</thead>
<tbody>
<tr>
<td>Grade</td>
<td>66</td>
<td>94</td>
<td>121</td>
<td>149</td>
<td>176</td>
</tr>
<tr>
<td>Operating Cost</td>
<td>138</td>
<td>129</td>
<td>121</td>
<td>113</td>
<td>104</td>
</tr>
<tr>
<td>Capital Cost</td>
<td>138</td>
<td>129</td>
<td>121</td>
<td>113</td>
<td>104</td>
</tr>
</tbody>
</table>
22 Adjacent Properties (Item 23)

GSR’s Bogoso Mine is currently mining or preparing to mine open pit deposits adjacent to the Prestea Underground Mine along the mineralization corridor to the north and south.
23 Other Relevant Data and Information (Item 24)
There is no further additional data and information to report.
24 Conclusions and Recommendations (Item 25 and 26)

The findings of this PEA provide compelling arguments to move the study to the FS design stage. The following are the recommendations for advancing the project in the individual areas:

**Resource**
- Further drilling along strike and at depth could expand the current resource and upgrade the inferred material to indicated. This drilling will be conducted once project construction commences.

**Underground Mining**
- Advance the geotechnical evaluation of the shrinkage mining to a FS level
- Advance the mine design and scheduling to a FS level
- Undertake FS level capital and operating cost estimates

**Process and Tailings**
- Develop processing strategy considering processing through the Bogoso CIL Plant and the Wassa CIL Plant in addition to the option considered in this PEA

**Environment and Infrastructure**
- Initiate community consultation for the development of the underground operation and determine the methods for including community concerns within the project design and operation
- Develop the environmental and socioeconomic baselines for the operation including a community health assessment
- Complete the environmental permitting process with the appropriate involvement of stakeholders and regulators such that stakeholder concerns are addressed in the design and environmental management plan for the project.

The cost of the recommendations are estimated to be as follows:

- **Feasibility Study**
  - Geotechnical $70,000
  - Mine design $150,000
  - Cost estimation $30,000
  - Metallurgical testing $100,000
  - Process design $100,000
  - Electrical design $30,000
  - Compressed air $60,000
  - Pumping $60,000
  - **Total** $600,000
Effective Date of Technical Report – December 18, 2014

__________________________  ________________________
Martin P. Raffield                      S. Mitchel Wasel
Signed this 18 day of December, 2014   Signed this 18 day of December, 2014
25 References (Item 27)


Junner N.R., (1935), Gold in the Gold Coast, Gold Coast Geological Survey, Memoir No. 4.

Junner N.R., (1940), Geology of the Gold Coast and Western Togoland, Gold Coast Geological Survey, Memoir No. 11.

Kitson, A.E., (1928), Provisional geological map of the Gold Coast and Western Togoland, with brief descriptive notes thereon, Gold Coast Geological Survey, Bulletin No. 2.


