NI 43-101 Technical Report on Resources and Reserves, Golden Star Resources, Bogoso/Prestea Gold Mine, Ghana

Effective Date: December 31, 2017

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Qualified Persons

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Abbreviations

ALS	ALS Minerals in Ghana-Kumasi
ANFO	Ammonium Nitrate Fuel Oil
Au	Gold
BGL	Bogoso Gold Limited
Bwi	Bond Ball Mill Work Index
C&M	Care and maintenance
CIL	Carbon-in-Leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CRO	Control room operator
DD	Diamond Drill
ECG	Electrical Company of Ghana
EIA	Environmental Impact Assessment
EIS	Environmental Impact Statement
EL	Elevation
EMP	Environmental Management Plan
EPA	Environmental Protection Agency
EPS	Enhanced Production Scheduler
FOS	Factor of Safety
FR	Footwall Reef
FS	Feasibility Study
G&A	General and Administrative
GEMS	Gemcom Software
GSR	Golden Star Resources Ltd.
GSBPL	Golden Star (Bogoso / Prestea) Limited
HARD	Half Absolute Relative Difference
HDPE	High-density polyethylene
ICOLD	International Commission on Large Dams
KP	Knight Piésold
L	Level
LoM	Life of Mine
MCF	Mechanized cut and fill
MDM	MDM Engineering Group Limited
MoA	Memorandum of agreement
MR	Main Reef
MRC	Murray & Roberts Cementation
MVS	Mine Ventilation Services Inc.
NaCN	Sodium Cyanide
NI 43-101	National Instrument 43-101
NPV	Net Present Value
PEA	Preliminary Economic Assessment
PGR	Prestea Gold Resources Limited
PLC	Programmable Logic Controller
PPE	Power Plant Electrical Technologies
PSGL	Prestea Sankofa Gold Limited

PUG	Prestea Underground
QA/QC	Quality Assurance Quality Control
RAB	Rotary Air Blast
QP	Qualified Person
RC	Reverse Circulation
RO	Reverse Osmosis
RoM	Run of Mine
RQD	Rock Quality Designation
SCADA	Supervisory control and data acquisition
SEDAR	System for Electronic Document Analysis and Retrieval
SFA	Screen fire assay
SG	Specific Gravity
SGMC	State Gold Mining Company Limited
SGS	SGS Laboratories in Tarkwa/Lakefield
SRK or SRK (Canada)	SRK Consulting (Canada) Inc.
SRK SA	SRK Consulting (South Africa) Pty Ltd.
SRK (UK)	SRK Consulting (UK) Limited
TSF	Tailings Storage Facility
TSF1	Tailings Storage Facility No. 1
TSF2	Tailings Storage Facility No. 2
TTG	Tonalite–Trondhjemite–Granodiorite
UCS	Uniaxial compressive strength
UG	Underground
VSD	Variable Speed Drive
WR	West Reef
WRP	West Reef Project

Units

cfm	Cubic feet per minute
deg.	Degrees Celsius
ft	Foot
g/t	Grams per tonne
kg	Kilogram
kg/a	Kilograms per annum
koz	Thousand troy ounces
kPa	Kilopascal
kV	Kilovolt
kW	Kilowatt
kWh	Kilowatt hour
kWh/t	Kilowatt hour per tonne
l/s	litres per second
m	Metre
m/d	Metres per day
m/s	Metres per second
m3	Cubic meter
Ma	Million years
ML	Million litres
mm	millimeter
Mt	Million tonnes
MPa	Megapascal
MVA	Mega-volt-ampere
oz	Troy ounce
oz/a	Troy ounces per annum
Pa	Pascal
psi	Pounds per square inch
t	Metric tonne
t/a	Tonnes per annual
t/d	Tonnes per day
$ or US$	US Dollars
V	Volt

1 Executive Summary

1.1 Introduction

The Bogoso/Prestea Gold Mine is located in western Ghana approximately 200 kilometres by road from the capital Accra. The Golden Star Bogoso/Prestea Ltd (GSBPL) company owns the rights to mine both the Bogoso and Prestea concessions. Golden Star Resources Ltd. (GSR), a Canadian federally-incorporated, international gold mining and exploration company producing gold in Ghana, owns a 90% interest in GSBPL.

Future production from the mine will be sourced primarily from the Prestea Underground Mine which is part of GSBPL.

This technical report summarizes the technical information that is relevant to support the disclosure of a Mineral Reserve Statement for mine pursuant to Canadian Securities Administrators' National Instrument 43-101. It presents the assumptions and designs at a level of accuracy that is required to demonstrate the economic viability of the mineral resources.

Summary:

- Proven and Probable mineral reserves estimated at $1,250/oz gold price as of December 31, 2017 are 1,920 kt at an average grade of 8.1 g/t containing 497 thousand ounces of gold
- Underground mining production of 1,165 kt at an average grade of 12.35 g/t containing 463 thousand ounces of gold
- Remaining open pit mining production and stockpiles containing 35 thousand ounces of gold
- A 5 year production period with a nominal production rate of 650 t/d
- A metallurgical process recovery of 94% yielding 468 thousand recovered ounces
- Revenue based on a gold price of US$1,300/ounce
- Total development capital cost estimate of $2.0 million
- Total sustaining capital cost estimate of $27.1 million
- $164 million Free cash flow
- $144 million NPV at 5% discount rate
- $624/oz life of mine cash operating cost
- $754/oz life of mine mine-site all in sustaining cost

1.2 Property Description and Ownership

The PUG mine is an underground gold mine located adjacent to the town of Prestea, and 15 km south of the Bogoso operations. The mine is currently undergoing infrastructure refurbishment. The mine has two serviceable access shafts and extensive underground workings and support facilities. Access to the mine site is via a combination of paved and unpaved roads from Tarkwa.

The PUG mine was operated from the 1870s 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 9 million ounces of gold, the second highest production of any mine in Ghana. The underground workings are extensive, reaching depths of approximately 1,450 metres (m) and extending along a strike length of 9 km.

GSR owns 90% interest of the PUG, inclusive of the WR project.

1.3 Geology and Mineralization

The Prestea-Bogoso mineralization occurs at the southern end of the Ashanti Greenstone Belt, where 11 gold deposits, mined or under exploration, are localized principally along three steep to subvertical major crustal structures. Rock assemblages from the southern area of the Ashanti Belt were formed during 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 Level [L]). The subordinate West Reef and East Reef, in the immediate hanging wall and footwall, respectively, of the former structure, are discontinuous. West Reef occurs some 200 m into the hanging wall 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 30 L.

1.4 Exploration Status

Prestea underground orebodies were based on a combination of GSR underground sampling from some 315 boreholes, 123 rock saw samples, and 160 chip channel samples from 13 crosscuts and 5 production alimak raises. The bulk of the drilling was conducted from 2003 and throughout 2006. GSR drilled an additional 14 underground boreholes into the West Reef (WR) orebody in 2012 and 2013 for geotechnical and metallurgical testing purposes. West Reef Reserve Infill drilling adding another 24 holes was completed in early 2017, upon resumption of operations, followed by the resumption of extensional drilling in the West Reef with a further 13 drill holes added. This drilling has been carried out using fan drilling from cubbies on the most accessible levels predominantly from the 12, 17, and 24 Ls.

1.5 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. In order to meet this requirement.

The Mineral Resource Statement has been prepared using a block cut-off grade of 5.54 g/t gold based on a US$1,450 per ounce gold price and appropriate costing data to produce a mineral resource that matches the requirements that the deposit should have “reasonable prospects for economic extraction” as defined by the CIM.

The statement was prepared by Mr. S. Mitchel Wasel who is a Qualified Person pursuant to National Instrument 43-101. Mr. Wasel is employed by GSR as Vice President of Exploration and is not independent of the company. The effective date of the Mineral Resource Statement is December 31 2017. The Mineral Resource Statement for Prestea Underground is given below in Table 1.

Table 1 Mineral Resource Statement

Mineral Resources	Dec 31, 2017 Measured Mineral Resource			Dec 31, 2017 Indicated Mineral Resource			Dec 31, 2017 Inferred Mineral Resource		
	tonnes	grade	ounces	tonnes	grade	ounces	tonnes	grade	ounces
	(000)	g/t Au	(000)	(000)	g/t Au	(000)	(000)	g/t Au	(000)
Bogoso/Prestea Refractory	-	-	-	17,809	2.84	1,625	922	2.60	77
Mampon	-	-	-	103	1.59	5	14	1.68	1
Prestea South	-	-	-	1,627	2.12	111	68	1.89	4
Prestea Underground	-	-	-	1,649	15.16	804	3,193	8.46	868
Bogoso/Prestea Other	-	-	-	2,414	1.65	128	470	1.50	23
TOTAL	-	-	-	**23,602**	**3.52**	**2,673**	**4,667**	**6.48**	**973**

Notes to Mineral Resource Estimate:

- The Mineral Resources for “Bogoso/Prestea Other” include Chujah, Dumasi, Bogoso North, Buesichem, Opon, and Ablifa.
- The open pit resources for Bogoso/Prestea Other has been estimated using a gold cut-off grade ranging from 0.72 to 1.04 g/t for oxide material; from 1.33 to 1.75 g/t for transition material and from 1.20 to 1.52 g/t for fresh material.
- Prestea Underground Mineral Resource has been estimated using a gold cut-off grade at 5.54 g/t Au.
- Open pit Mineral Resources were estimated using optimized pit shells at a gold price of $1,450 per ounce. Other than gold price, the same optimized pit shell and underground parameters and modifying factors used to determine the Mineral Reserves were used to determine the Mineral Resources.
- Mineral Resources are inclusive of Mineral Reserves.
- Numbers may not add correctly due to rounding.

1.6 Mineral Reserves

The Mineral Reserve statement at December 31, 2017 is presented in Table 2. The Reserve contains small quantities of open pit mining which is expected to be complete in Q1 2018. The details around the open pit mining will not be discussed due to the immaterial quantities remaining.

Table 2 Mineral Reserve Statement

Mineral Reserves	Dec 31, 2017 Proven Mineral Reserve			Dec 31, 2017 Probable Mineral Reserve			Dec 31, 2017 Proven and Probable Mineral Reserve		
	tonnes	grade	ounces	tonnes	grade	ounces	tonnes	grade	ounces
	(000)	g/t Au	(000)	(000)	g/t Au	(000)	(000)	g/t Au	(000)
Mampon	-	-	-	105	2.23	8	105	2.23	8
Prestea South	-	-	-	103	1.80	6	103	1.80	6
Prestea Underground	-	-	-	1,165	12.35	463	1,165	12.35	463
Stockpiles	547	1.21	21	-	-	-	547	1.21	21
TOTAL	***547***	***1.21***	***21***	***1,373***	***10.79***	***476***	***1,920***	***8.06***	***497***

Notes to Mineral Reserve Estimate:

- The stated mineral reserves have been prepared in accordance with the requirements of NI 43-101 and are classified in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum's "CIM Definition Standards – For Mineral Resources and Mineral Reserves". Mineral reserve estimates reflect the Company's reasonable expectation that all necessary permits and approvals will be obtained and maintained. Mining dilution and mining recovery vary by deposit and have been applied in estimating the mineral reserves.
- Mineral reserves are the economic portion of the measured and indicated mineral resources. Mineral reserve estimates include mining dilution at grades assumed to be zero.
- The 2017 mineral reserves were prepared under the supervision of Dr. Martin Raffield, Senior Vice President Project Development and Technical Services for the Company. Dr. Raffield is a QP as defined by Canada's NI 43-101.
- The mineral reserves at December 31, 2017 were estimated using a gold price assumption of $1,250 per ounce.
- The slope angles of all pit designs are based on geotechnical criteria as established by external consultants. The size and shape of the pit designs are guided by consideration of the results from a pit optimization program.
- Cut-off grades have been estimated based on operating cost projections, mining dilution and recovery, government royalty payment requirements and applicable metallurgical recovery. Marginal cut-off grade estimates for the open pits are as follows: Mampon 1.3 g/t; and Prestea South 1.1 g/t. Break-even cut-off grade estimates for Prestea Underground is 6.5 g/t;
- Numbers may not add due to rounding;
- Only non-refractory ore is included in mineral reserves.

1.7 Mining Method

The planned mining method is mechanized shrinkage mining. This method uses mechanical raise climber technology (Alimak) together with longholes drilled from the raise climber. The method is an advance in terms of safety and productivity from conventional hand-held shrinkage mining. Safety is greatly enhanced both in raise development and stoping operations by having miners protected from rockfalls and removed from working on the broken muck pile. The productivity (per miner) is enhanced by introducing short longholes for stope production.

A typical cycle for Alimak stoping is:

- Raise development in the centre of a stoping block;
- Raise hangingwall rock support;
- Drilling of longhole blast rings;
- Relocation of the raise climber infrastructure to the top of the stoping block, blasting of longhole rings and swell mucking; then,
- Final draw-down mucking and possible waste backfilling.

Small diesel and electric 2 m^3 LHD's will be used for rock haulage from the base of the stoping panel. Mine development will be by hand held drills with LHD mucking. Tramming of ore and waste along the two main levels (17 and 24) will be via an upgraded rail haulage system to Central Shaft.

The rib pillars between stopes are planned to be 3 m wide along the strike of the orebody. Stopes will be 20 m wide separated by these pillars. The pillars will run the entire length of each of the Alimak stopes and are not planned to be recovered.

Mine Ventilation Service's (MVS) engineers completed a detailed ventilation system design for the mechanized shrinkage mining. The emphasis of the study was to determine ventilation infrastructure and fan duties required for the West Reef development, using phased ventilation modeling.

In areas where diesel equipment is used a factor of 0.08 m^3/s per kW of motor power is used. This is not only for dilution of diesel particulate matter but also for dilution of heat and dust.

1.8 Recovery Methods

Three metallurgical testwork programs have been conducted on samples of mineralization from the Prestea underground West Reef deposit. A metallurgical testwork program was undertaken in 2008 in support of a study at that time investigating the potential for mining ore from both the West Reef and the Footwall Reef at Prestea. The testwork supporting the current operation commenced in 2013. That testwork was augmented with additional testwork conducted in 2015 in support of the current shrinkage mining method.

The metallurgical testwork programs have all indicated that the ore is free milling, with a relatively high gravity-recoverable component, and it demonstrates a degree, although relatively low, of preg-robbing behaviour. Processing using a carbon-in-leach (CIL) configured plant, rather than carbon-in-pulp (CIP), is indicated.

Optimization testwork of key operational parameters has indicated that:

- Grind size of up to 80% -110 µm is acceptable
- Leach residence time of 16 hours is sufficient at a grind size of 80% -75 µm (however for a coarser grind size of 80% -110 µm, a leach residence time of 24 hours is probably required)

Based on the testwork results, a processing recovery of 94% is predicted at the reserve average head grade of 12.4 g/t gold.

GSR intends to process the Prestea West Reef ore through a low throughput stream of the CIL plant within the confines of the existing Bogoso processing operation, a distance of approximately 15 km from the mine. The plant will consist of a single jaw crusher circuit, SAG and ball mill (1,5Mw each) , cyclone classification, gravity circuit comprising three Knelson Concentrators (incorporating the existing Acacia high intensity cyanidation reactor), six original leach tanks operating as surge and stock holding tanks and slow feeding CIL circuit, with associated services. The existing non-refractory plant 5 and 16 ton elution circuit will be used for gold recovery from the activated carbon, which will then be recovered as metal by electrowinning in the existing Bogoso gold room. Reagents such as cyanide, lime, flocculent, and oxygen will be supplied from the existing Bogoso plant services.

During the early phase of underground ore production the processing will take place in the currently operating Bogoso Oxide Plant in combination with ore produced at the Prestea South open pits.

When the pits are exhausted the oxide plant will be modified to a batch milling, continuous leach operation suitable for processing 650 t/d of high-grade underground ore.

1.9 Infrastructure

A number of infrastructure replacement and rehabilitation projects have been completed during 2016 and 2017.

Shaft rehabilitation – Central Shaft has been rehabilitated and Bondaye Shaft work is underway.

Winders – work is completed on the Central and Bondaye shaft winders including: gearbox refurbishment; motor replacement; brake replacement; installation of new control systems; replacement of liquid controllers and commissioning of dynamic braking systems (Central Shaft winders).

Development rehabilitation – work is complete on 24L, the first production level, and well advanced on 17L to rehabilitate ground support and replace rail and services.

Electrical infrastructure – the entire surface and underground electrical system has been refurbished including replacement of 90% of the system hardware and voltage standardization.

Compressed air – new compressors have been installed to replace the current units which date from the 1930's. Horizontal piping has been installed on 24L, existing piping on 17L will be replaced and vertical pipe in the shaft will be rehabilitated based on a non-destructive testing program.

Pumping – The mine wide pumping system will be replaced following commercial production.

Tailings

PUG West Reef tailings will be deposited in the existing tailings storage facilities (TSF) at Bogoso for which Knight Piésold (KP) is the engineer of record. GSBPL has two TSFs comprising four cells.

TSF 1 is a single cell, paddock style facility from which tailings was hydraulically re-mined from 2013 until August 2015 for reprocessing. In the period of reprocessing some 3 Mt tailings was removed from this facility. TSF 1 has been permitted by the EPA and engagement is underway with the Minerals Commission to enable the recommencement of tailings deposition into this TSF in the future. The void created by the tailings re-processing itself provides sufficient capacity for the LoM PUG West Reef tailings storage.

TSF 2 is a paddock style facility, consisting of three cells: a combined cell 1/2, 2A, and 3. A total of 12 embankments separate and border the cells. Cell 1/2 and Cell 2A are active and Cell 3 is presently subject to paddock deposition and revegetation ahead of closure. The remaining volume in TSF 2 is also sufficient to contain the LoM tailings although not all embankments are presently at their permitted extent.

Water Treatment

Impact assessment and technical studies for the PUG West Reef indicate that the mine will remain a dry mine for much of the mine life, with potential mine dewater volumes considered to be low (conservatively calculated to be less than 300 m^3/day). In the event that mine dewatering is required in the future, GSBPL has undertaken an assessment of treatment options to determine feasibility and comparative cost benefit analysis.

Specific West Reef geochemical studies found that host rocks of the West Reef are non-acid generating, and mine drainage from the area is predicted to be near neutral to alkaline drainage, with all parameters expected to be within EPA guidelines. Despite this, worst case predicted mine dewater qualities, that may result if the West Reef mine is contaminated by historic mine void drainage, have been used in the assessment of the feasibility of the various mine water treatment technologies available.

GSBPL has made sufficient allowance for the design and construction of a water treatment plant should conditions in the PUG West Reef mining area require dewatering in future.

1.10 Market Studies and Contracts

Gold is a freely traded commodity on the world market for which there is a steady demand from numerous buyers. GSR has a long-term sales contract currently in place with a South Africa gold refinery. The gold is shipped in the form of doré bars. The sale price is based on the London p.m. fix on the day of the shipment to the refinery. In addition, GSR has a number of contracts in place with local, national, and internal contractors on materials and services supply.

1.11 Social and Environmental Aspects

The PUG West Reef involves new underground development and infrastructure, connected to but isolated from previous underground workings, including surface waste disposal, underground raise bore development of ventilation shafts, mine dewatering, water treatment and discharge, and transportation of ore to existing approved processing facilities. It will extends roughly 9 km along strike in a north-south direction beneath the town of Prestea, to a current known extent of 1.4 km of depth.

Primary Environmental Approvals Required

The primary environmental approval that is required to proceed with the construction and operational phases of the PUG West Reef project is an Environmental Permit. An environmental impact assessment (EIA) has been completed for the project and the resulting Environmental Impact Statement (EIS) has been submitted to the Ghana Environmental Protection Agency (EPA). As the project utilizes previously disturbed areas, it is not expected to add to existing environmental impacts, and has wide community support, project permitting proceeded and on 29th January 2018 the EPA invoiced Golden Star for the EIA Permit.

The PUG West Reef project EIS contains a provisional environmental management plan (EMP) and conceptual closure plan. These components have subsequently been reflected in annual updates to the Mining Operating Plan submitted to the Mines Inspectorate Division of the Minerals Commission for approval. The provisional EMP and closure plan for the project will be integrated into wider Golden Star (Bogoso/ Prestea) Limited (GSBPL) EMP and closure plans for GSR's operations at the next revision of each of these. The most recent overall EMP remains in force for the period 2015-2018, with the next EMP due for submission by 2019.

Environmental and Social Management System

GSR has an environmental and social management system developed along the lines of an ISO 14001 management system. The management is carried out by dedicated and professional resources in the environmental and social/communities fields. GSBPL has also established a series of community mine consultative committees for on-going engagement of local communities. The company and the mine local communities have established corporate social responsibility agreements that specify protocols for engagement, grievance, local employment, Development Foundation, and other aspects.

Key Environmental and Social Issues

Land uses in the vicinity of GSR's Prestea operations include residential, commercial, agricultural, forestry, agroforestry (palm oil and rubber plantations), unauthorized small-scale mining operations, as well as authorized mining operations.

Some limited resettlement has been required for the PUG West Reef project and was carried out in accordance with the approved Prestea Projects Resettlement Action Plan that was developed in accordance with the International Finance Corporation Performance Standard 5 for Land Acquisition and Involuntary Resettlement, and approved by the Prestea Huni-Valley District Assembly. People from local communities express interest in the project for the potential in employment opportunities. There is a corporate responsibility agreement that promotes preferential employment for people from local stakeholder communities. The Golden Star Skills Training and Employability program offers training to young people in practical and technical skills in sectors unrelated to mining, contributing to the diversification of the local economy's employment base. These programs are complemented by an array of other alternative livelihoods initiatives.

The West Reef operations are not expected to have significant noise and blasting effects. The EIA studies and subsequent monitoring have confirmed the required controls for the operations.

Unauthorized small-scale mining operations have little potential to affect the project. The main Central shaft and Bondaye shaft sites are well secured. GSR has been active in assisting the government and local communities in the regularization of small-scale mining activities. In addition, GSR has ceded prospective areas of the GSR surface concession to the Ghana government for allocation to authorized small scale miners. Shafts potentially indirectly accessing the West Reef operational area have been backfilled in collaboration with the Government.

The PUG West Reef is not expected to add to existing impacts on biodiversity and ecology as it is located on previously disturbed land. The river ecosystems downstream of the project area have been degraded by anthropological influences, including historic mining operations, discharges of untreated sewage and unauthorized small-scale mining operations.

GSR has designed key elements of the PUG West Reef, like its completely dedicated mine dewatering system, so that it can isolate its underground mining activities from those of the predecessors and avoid posing any cumulative burden to the already heavily impacted systems. The dedicated mine dewatering system involves water removal via the Bondaye shaft to an appropriate treatment system to remove mine sediments, oil and grease, and trace metals. Treated waters that achieve the EPA effluent discharge guidelines will be released to the Anobaka stream.

Dedicated studies by external consultants for the PUG West Reef demonstrate low potential for acid drainage generation and the overall geochemical impact of mining the West Reef stopes is expected to be low.

1.12 Capital and Operating Costs

Total capital of $29.1 million is comprised of $2.0 million project capital and $27.1 million sustaining capital.

Mine operating costs include:

- $111/t underground mining operating cost
- $4.10/t open pit mining costs only in Q1 2018
- $25/t processing costs in H1 2018 when open pit material is being processed
- $75/t processing costs with underground feed of 650 tpd
- $45/t general and administration cost, covering both the Prestea and Bogoso sites

Life of mine cash operating cost was estimated at $624 per ounce and all-in sustaining cost at $754 per ounce during the production period.

1.13 Economic Analysis

The mine has been evaluated on a discounted cash flow basis. The cash flow analysis was prepared on a constant 2018 US dollar basis. No inflation or escalation of revenue or costs has been incorporated. With an applicable $593 million tax loss pool the pre-tax and post-tax present value of the net cash flow with a 5% discount rate ($NPV_{5\%}$) is $144 million using a base case gold price of $1,300/oz.

The $NPV_{5\%}$ is most sensitive to changes in gold price and plant head grade. The project is least sensitive to changes in capital costs.

2 Introduction

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”) achieved commercial production in February 2018. 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 a paved road from Tarkwa.

GSBPL is planning to process the Prestea West Reef Project material in the Bogoso Processing plant situated at the Bogoso operation.

This technical report describes a mechanized shrinkage mining (Alimak stoping) approach to mining the West Reef orebody between 17 and 27L. Diesel LHD’s will be used for development and stope mucking and rail haulage will move ore and waste along 24L to Central Shaft for skipping.

2.1 Scope of Technical Report

This technical report is intended to support the 2017 Mineral Resource and Reserve estimate for GSBPL.

The Technical Report has been prepared in accordance with the requirements of National Instrument 43-101 (“NI 43-101”) – ‘Standards of Disclosure for Mineral Projects’, of the Canadian Securities Administrators (“CSA”) for filing on CSA’s “System for Electronic Document Analysis and Retrieval” (“SEDAR”).

2.2 Qualified Persons

Dr. Martin Raffield is the QP responsible Sections 1 to 6; 13; and 15 to 27 of this report. He is based in Bogoso, Ghana and employed by GSR as Senior Vice President, Project Development and Technical Services.

S. Mitchel Wasel is the QP responsible for Sections 7 to 12 and Section 14 of this report. Mr. Wasel is based in Takoradi, Ghana and is employed by GSR as Vice President of Exploration.

2.3 Site Visits

Dr. Raffield was resident on site from Dec 31, 2017 to Feb 22, 2018.

Mr. Wasel last visited the site in March 2018.

3 Reliance on Other Experts

The preparation of this technical report 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:

- Geotechnical assessment prepared by SRK (UK) Ltd. – SRK (2016);
- Metallurgical assessment report prepared by SGS Canada Inc. in 2015
- Process design and cost estimates prepared by MDM Engineering (South Africa) in 2015
- Electrical design and cost estimate prepared by PPE Technologies (South Africa) in 2015
- Ventilation design prepared by MVS Ltd (USA) in 2015
- 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; and
- Environmental impact assessment studies undertaken by Golder Associates (Ghana and South Africa).

4 Property Description and Location

4.1 Location of Mineral Concessions

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 1), with the processing facilities situated approximately 10 km south of the town of Bogoso. GSR currently holds six mining leases (refer to Section 20.1.4) as well as several prospecting licenses to the southwest, northeast, and west of Bogoso.

Figure 2 shows the location of key mining licence boundaries in relation to the location of the main GSR mine workings at Bogoso, Prestea, and Pampe including:

- Bogoso mining lease: Chujah Main and Bogoso North are ceased open pit mines 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.
- Prestea mining lease (Land Registration No. 2799/2001): The Buesichem deposit and the Prestea underground lie to the north of the Prestea lease with the Beta Boundary South, Bondaye and Tuapim deposits (collectively, Prestea South) located southeast in the central part of the lease.

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

Figure 1 Location of Bogoso/Prestea in the Regional Context of Ghana and West Africa

(Source: United Nations 2008)

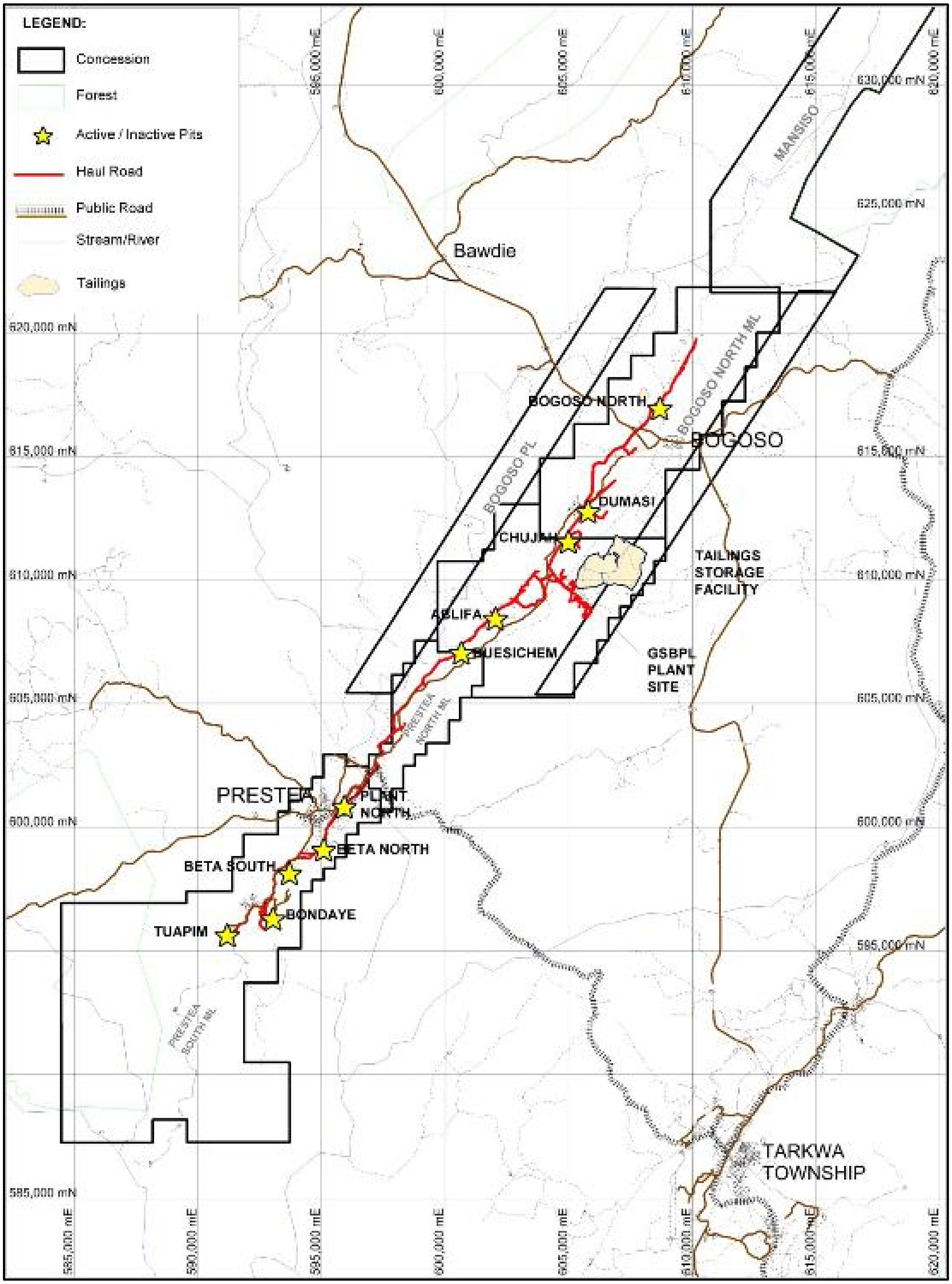
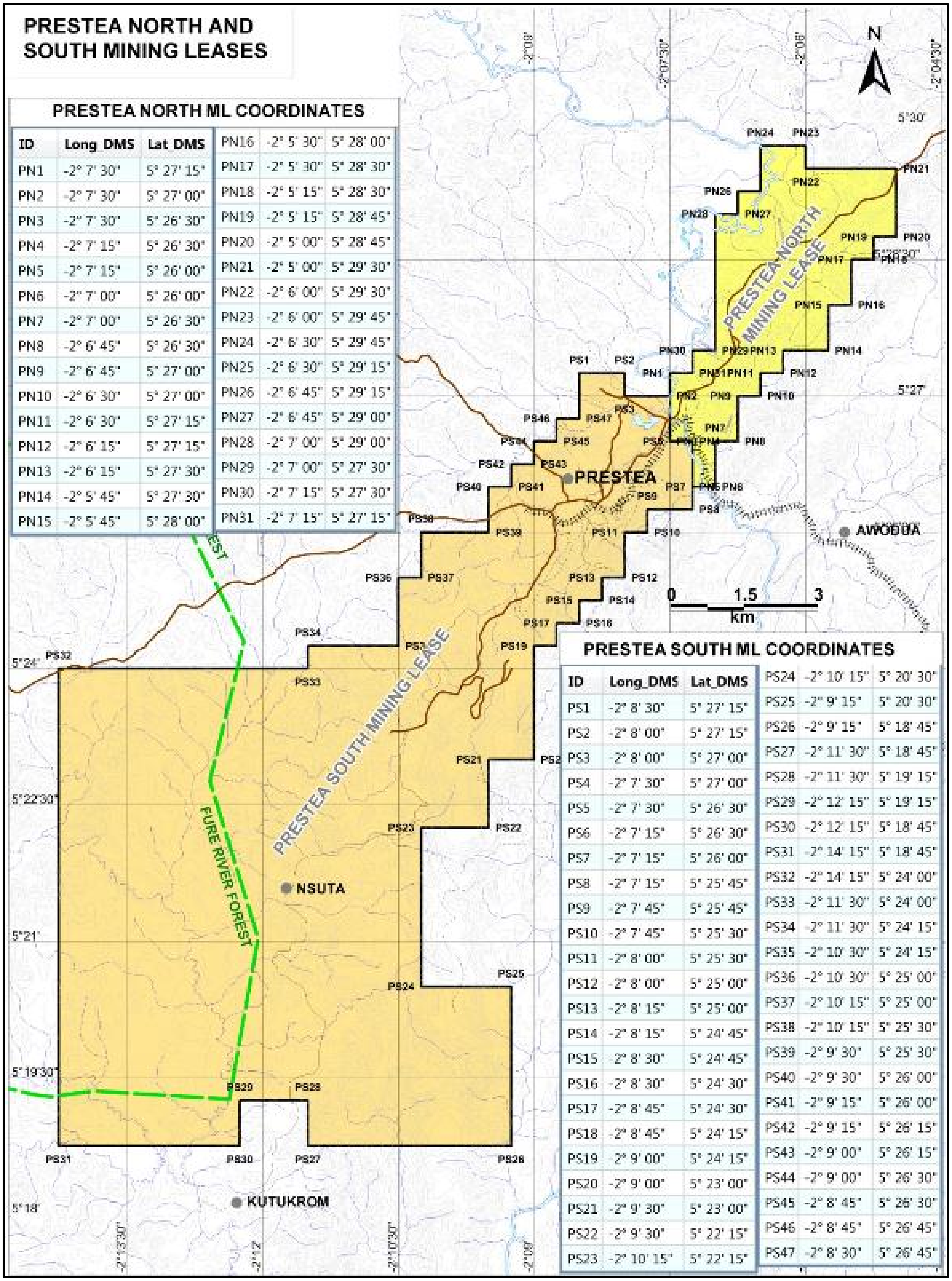


Figure 2 Location of Principal Operations in Relation to Mining Licence Boundaries

(Source: GSR, 2018)



PRESTEA NORTH ML COORDINATES

ID	Long_DMS	Lat_DMS
PN1	-2° 7' 30"	5° 27' 15"
PN2	-2° 7' 30"	5° 27' 00"
PN3	-2° 7' 30"	5° 26' 30"
PN4	-2° 7' 15"	5° 26' 30"
PN5	-2° 7' 15"	5° 26' 00"
PN6	-2° 7' 00"	5° 26' 00"
PN7	-2° 7' 00"	5° 26' 30"
PN8	-2° 6' 45"	5° 26' 30"
PN9	-2° 6' 45"	5° 27' 00"
PN10	-2° 6' 30"	5° 27' 00"
PN11	-2° 6' 30"	5° 27' 15"
PN12	-2° 6' 15"	5° 27' 15"
PN13	-2° 6' 15"	5° 27' 30"
PN14	-2° 5' 45"	5° 27' 30"
PN15	-2° 5' 45"	5° 28' 00"
PN16	-2° 5' 30"	5° 28' 00"
PN17	-2° 5' 30"	5° 28' 30"
PN18	-2° 5' 15"	5° 28' 30"
PN19	-2° 5' 15"	5° 28' 45"
PN20	-2° 5' 00"	5° 28' 45"
PN21	-2° 5' 00"	5° 29' 30"
PN22	-2° 6' 00"	5° 29' 30"
PN23	-2° 6' 00"	5° 29' 45"
PN24	-2° 6' 30"	5° 29' 45"
PN25	-2° 6' 30"	5° 29' 15"
PN26	-2° 6' 45"	5° 29' 15"
PN27	-2° 6' 45"	5° 29' 00"
PN28	-2° 7' 00"	5° 29' 00"
PN29	-2° 7' 00"	5° 27' 30"
PN30	-2° 7' 15"	5° 27' 30"
PN31	-2° 7' 15"	5° 27' 15"

PRESTEA SOUTH ML COORDINATES

ID	Long_DMS	Lat_DMS
PS1	-2° 8' 30"	5° 27' 15"
PS2	-2° 8' 00"	5° 27' 15"
PS3	-2° 8' 00"	5° 27' 00"
PS4	-2° 7' 30"	5° 27' 00"
PS5	-2° 7' 30"	5° 26' 30"
PS6	-2° 7' 15"	5° 26' 30"
PS7	-2° 7' 15"	5° 26' 00"
PS8	-2° 7' 15"	5° 25' 45"
PS9	-2° 7' 45"	5° 25' 45"
PS10	-2° 7' 45"	5° 25' 30"
PS11	-2° 8' 00"	5° 25' 30"
PS12	-2° 8' 00"	5° 25' 00"
PS13	-2° 8' 15"	5° 25' 00"
PS14	-2° 8' 15"	5° 24' 45"
PS15	-2° 8' 30"	5° 24' 45"
PS16	-2° 8' 30"	5° 24' 30"
PS17	-2° 8' 45"	5° 24' 30"
PS18	-2° 8' 45"	5° 24' 15"
PS19	-2° 9' 00"	5° 24' 15"
PS20	-2° 9' 00"	5° 23' 00"
PS21	-2° 9' 30"	5° 23' 00"
PS22	-2° 9' 30"	5° 22' 15"
PS23	-2° 10' 15"	5° 22' 15"
PS24	-2° 10' 15"	5° 20' 30"
PS25	-2° 9' 15"	5° 20' 30"
PS26	-2° 9' 15"	5° 18' 45"
PS27	-2° 11' 30"	5° 18' 45"
PS28	-2° 11' 30"	5° 19' 15"
PS29	-2° 12' 15"	5° 19' 15"
PS30	-2° 12' 15"	5° 18' 45"
PS31	-2° 14' 15"	5° 18' 45"
PS32	-2° 14' 15"	5° 24' 00"
PS33	-2° 11' 30"	5° 24' 00"
PS34	-2° 11' 30"	5° 24' 15"
PS35	-2° 10' 30"	5° 24' 15"
PS36	-2° 10' 30"	5° 25' 00"
PS37	-2° 10' 15"	5° 25' 00"
PS38	-2° 10' 15"	5° 25' 30"
PS39	-2° 9' 30"	5° 25' 30"
PS40	-2° 9' 30"	5° 26' 00"
PS41	-2° 9' 15"	5° 26' 00"
PS42	-2° 9' 15"	5° 26' 15"
PS43	-2° 9' 00"	5° 26' 15"
PS44	-2° 9' 00"	5° 26' 30"
PS45	-2° 8' 45"	5° 26' 30"
PS46	-2° 8' 45"	5° 26' 45"
PS47	-2° 8' 30"	5° 26' 45"

Figure 3 Prestea Mining Lease

(Source: GSR, 2018)

4.2 Mineral Titles and Agreements

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

Mineral rights in the Prestea concession have been granted to GSBPL under the Minerals and Mining Act, 2006 (Act 703). This Act is the governing legislation for Ghana's minerals and mining sector. As defined by Act 703, every mineral in its natural state in, under or upon land in Ghana, rivers, streams, water-courses throughout the country, the exclusive economic zone, and an area covered by the territorial sea or continental shelf, is the property of the Republic of Ghana and is vested in the president in trust for the people of Ghana. By means of the Act, land in the country may be made the subject of an application for a mineral right in respect of a mineral specified in the application.

The Prestea concession is a mining lease that was issued to Prestea Gold Resources Limited (PGR) (now part of GSR/GSBPL) on June 29, 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 150.37 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 BGL the right to develop and operate the PUG mine 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 March 14, 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 mine at the time and to consolidate the management of the underground mine. The agreement also defined the conditions for the PUG mine to be put under care and maintenance (C&M), which includes mine dewatering and shaft maintenance along with the number of employees required, and informs all parties of, and transfers the Environmental Indemnity Agreement established in December 2001 with the Government of Ghana, to the joint venture. The PUG mine is now on refurbishment. For more information, see Section 20.1.4.

4.3 Surface Rights

The Prestea surface mining lease was granted to Bogoso Gold Limited (now GSBPL) in November 16, 2001, giving GSBPL primary control over the Prestea mineral resources (Prestea Sankofa Gold Limited has a small tailings retreatment operation located on the Prestea surface concession).

Surface rights to land affected by the PUG West Reef project are traditionally in the ownership of the Himan and Nsuta Mbease Stools of the Wassa Fiase Traditional Area. The Paramount Chief of the Traditional Area exercises traditional control over the divisional and sub-chiefs of the settlements and hamlets within the project area.

Some limited resettlement may be required for the PUG West Reef project, and would be carried out in accordance with the approved Prestea Projects Resettlement Action Plan, developed in accordance with the International Finance Corporation Performance Standard 5 for Land Acquisition and Involuntary Resettlement, and approved by the Prestea Huni-Valley District Assembly in May 2014 (see Section 20.3.6).

Any physical or economic displacement for the PUG West Reef project is expected to be restricted to relocation of occupiers of SGMC structures. These structures are subject to a sublease and optional purchase agreement between GSBPL and SGMC, and the occupiers have never been given permission by GSBPL or SGMC to live at these locations. The project infrastructure will be largely located within existing operational areas that have been rented or compensated (see Section 20.3.6). Should any further compensation be required, e.g., for buffer areas, this will be undertaken in accordance with applicable laws and in accordance with the GSBPL Compensation and Land Acquisition procedures.

4.4 Royalties and Encumbrances

Royalties associated with the Prestea Mining Lease are defined under Section 21 of the Prestea Mining Lease that was issued to PGR on June 29, 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.

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$0.565 million per year.

In the news release dated May 7, 2015 GSR announced the securing of a $150 million financing with Royal Gold, Inc. (RGI) and its wholly-owned subsidiary RGLD Gold AG (RGLD). The $150 million financing consists of a $130 million stream transaction with RGLD and a further $20 million term loan from RGI.

4.5 Historic Environmental Liability and Indemnity

The pre-existing environmental liabilities associated with the Prestea underground operation were included in an indemnity granted to PGR (now part of GSBPL/GSR) by the Republic of Ghana. The indemnity document is dated December 21, 2001 and titled Prestea Gold Resources Indemnity Against Pre-Existing Environmental Liabilities. This indemnity states that the PUG West Reef project will be allowed to function independently of the rest of the Prestea underground, and that the Government will irrevocably and unconditionally indemnify PGR against all liabilities of whatsoever nature arising from or connection with the pre-existing environmental liabilities (refer to Section 20.1.5).

A parallel indemnity agreement was signed with Bogoso Gold Limited (now GSBPL), also on December 21, 2001, for the Prestea surface concession, and likewise protects GSBPL from pre-existing surface environmental liabilities.

The closure liabilities for the Prestea and Bogoso concessions, totaling $42 million, are included in the costs and economics in Sections 21 and 22.

4.6 Permits and Authorization

GSBPL currently holds the following major approvals related to PUG West Reef and Bogoso operations (see Section 20.1.4 for details):

- Prestea Underground Mining Lease (LVB 2799/01) (WR 3218/01).
- Prestea Surface Mining Lease (LVB 2876/01) (WR3218/01).
- Tailings Storage Facility II Extension (EPA/EIA/188).
- Prestea Underground Gold Mining Project Phase 1 (EPA/EIA/804).
- Water Use Permits (GSBPL ID 288/1/17 and GSBPL ID 288/2/17).

GSBPL currently also holds the following permits; subject to renewal:

- Mining Operating Permits for Prestea Mining Lease LVB/WR 2876/01.
- Mining Operating Permits for Bogoso Mining Lease Mo. WR/2762/278, WR 368/88 Schedule – An application for extension (renewal) of the Bogoso Mining Lease 1 and 2 was submitted to the Minerals Commission on 18 May 2017, and offer letters were subsequently received from the Minerals Commission on 30 August 2017. Fee payment has been actioned and Ministerial approval is now pending.
- Explosive Purchase for Mining Operations.

For the PUG West Reef, an environmental impact assessment (EIA) was completed and subsequent Environmental Impact Statement (EIS) was submitted to the Environmental Protection Agency (EPA). The EPA has since invoiced GSBPL for the EIA Permit and Environmental Permit issuance is now pending. The EIS contains a provisional environmental management plan (EMP) and a conceptual closure plan for the West Reef (Sections 20.1.1, 20.1.6, and 0). GSBPL has additionally obtained/updated the PUG Mining Operating Plan with the Mines Inspectorate to reflect the West Reef underground mining area.

Overall the environmental impact assessment found that the West Reef operations are in a disturbed area, unlikely to pose any residual or cumulative impacts of any significance, and have wide community support.

Since acquisition of the mineral concession in December 2001, GSBPL has undertaken a series of EIA studies on the Prestea concession to support the permitting of its various mining projects and therefore has considerable background data relating to the mining area and to support required environmental permitting processes (Section 20.1.4).

5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

5.1 Accessibility

Access to the property by road is a six-hour drive from Accra via the port city of Takoradi. The road is paved from Accra to Bogoso with the last thirty minutes to Prestea being unpaved. There are airports at Kumasi and Takoradi, which provide daily services to the international airport at Accra. Kumasi is situated approximately a 3.5-hour drive from Prestea. Road surfaces in the area vary from poor (on the section between Bogoso and Prestea) to good (Accra to Takoradi).

5.2 Physiography and Vegetation

The topography of the area within which the GSBPL assets are located generally slopes in a northern direction towards the Ankobra River. It can be described as gently rolling, punctuated by a number of low hills and rises. A series of northeast-southwest trending sub-parallel ridges, about 2 km wide, dominates the eastern part of the project area. These ridges range in height from 150 m to 195 m. The western part has lower hills generally ranging in height between 70 m and 110 m. The mineralization tends to occur on the western slopes of the ridges with the intervening valleys occupied by farming communities and seasonal streams.

The GSBPL assets are within the rainforest bioclimatic zone, but there is no primary forest left in the area of the assets as a result of logging, farming, historical mining, and unauthorized small scale mining activity. The vegetation types on the Prestea concession include: secondary forest, secondary thicket, farm re-growth, farmland, and marshes or freshwater swamp forest. Patches of secondary forest are only present in a few areas not accessible for farming. Biodiversity studies have not identified endangered or threatened species on the Prestea concession (AERC, 2006).

The PUG West Reef is not expected to add to existing impacts on biodiversity and ecology. No secondary forest will be disturbed for the PUG West Reef project and the project infrastructure will be in brownfields areas. The river ecosystems downstream of the project area have been heavily degraded by anthropological influences, including over a century of historic mining operations, discharges of untreated sewage, and unauthorized small-scale mining operations.

5.3 Land Use and Proximity to Local Population Centres

The PUG West Reef operations lie to the south of the town of Prestea in the Prestea Huni-Valley District of the Western Region of Ghana, some 22 km south of the district capital, Bogoso, and 40 km northwest of Takoradi. The PUG West Reef project workings will be between the Bondaye shaft on the south side and the Central shaft on the north side. The targeted WR mineralization is approximately 2 km south of the Central shaft, 800 m along strike and 550 – 1,150 m below surface, beneath the town of Prestea. The Central shaft complex and offices for the PUG mine are within the Prestea town limits.

The population data/estimates for the larger communities located within the Prestea concession boundaries are given in Table 3.

Table 3 Overview of Local Prestea Communities

Community	Divisional Area / Paramountcy	Population (GSS 2000)	Population (GSS 2010)	Estimated* Population (2011)
Prestea	Himan / Wassa Fiase	23,168	25,185	38,392
Himan		6,800	4,842	17,074
Bondaye	Mbease Nsuta / Wassa Fiase	3,182	4,775	1,923
Mbease Nsuta		NM	676	1,262

GSS = Ghana Statistical Service

* = as estimated by traditional leaders

Land uses in the vicinity of GSBPL operations include residential, commercial, agricultural, forestry, agroforestry (palm oil and rubber plantations), and unauthorized small-scale mining operations.

5.4 Local Resources and Infrastructure

The PUG mine is in an area where mining has occurred more or less continuously since the late 1800s, so local skilled underground workers are available. The following services and infrastructure support the PUG West Reef operations:

- Surface access to the PUG mine is via the public road network that extends to the project
- Electricity and water supply are available
- Processing and tailings storage is carried out at existing permitted facilities at Bogoso.

Geochemical studies show that the risk of acid generation from the West Reef will be low, regardless, any waste rock generated at the site will be managed to minimize the risks of acid generation.

5.5 Climate and Length of Operating Season

The climate is south western equatorial climate type with daily temperatures varying mostly between 20°C to 35°C. There are two rainy seasons, one from April to June and then a minor rainy season in October and November. This area has significant rainfall most months, with a short dry season in December and January.

Annual rainfall in the area averaged 1,641 millimetres (mm) between 2002 and 2014. The range in annual rainfall during this period was from minimum annual rainfall 1,197 mm to maximum annual rainfall 2,195 mm.

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

6 History

6.1 Prior Ownership

Recorded production for the Prestea mine began in 1912 under the British company Ariston Mining, which operated the mine until the 1950s. The company was responsible for the majority of the underground development including shaft sinking, ventilation, and level development. The mine was nationalized in the late 1950s, 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 State Gold Mining Company Limited (SGMC).

In the early 1990s, 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 a 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 three years until its closure in early 2002 due to depressed gold prices and financial difficulties. The mine had remained under care and maintenance (C&M) from the 2002 closure to 2014 when the refurbishment started.

Mampon, Abronye and Opon

Very little information is available for the area between 1940 and the early 1980s. In 1988, BHP Billiton Limited (“BHP”) obtained the prospecting licence for the Dunkwa concession and conducted regional scale geochemical and Very Low Frequency - Electromagnetic (“VLF-EM”) surveys which located the deposits at Mampon, Abronye and Adiokrom. Follow up detailed geochemical and VLF-EM surveys were then conducted and six diamond drill holes explored the extent of the Abronye deposit.

Bogoso North and Marlu

Marlu Gold Mining Areas Limited (“MGMAL”) explored for gold and operated a medium scale open pit and underground mining operation from 1935 to 1955. Surface gold mineralization was systematically explored utilizing trenching and shallow adits driven across strike. Deeper exploration, well below the depth of oxidation, was conducted on the Marlu deposits, where underground workings extended approximately 250 m below the surface. In 1935, MGMAL commenced mining oxide ore from a series of open pits extending from Bogoso North to Buesichem. During the period 1935 to 1955, MGMAL processed between 0.36 and 0.45 Mtpa of ore yielding 35,000 to 51,000 oz per year. During the 15 year period of mining (the mine was shut down for the duration of World War II), 6.9 Mt of ore with a recoverable grade of 4.1 g/t Au was processed through the plant generating about 10.9 Moz of gold. Marlu also mined a small amount of ore from underground at Bogoso North. The Marlu mining operation terminated in 1955.

The 30-year period between the closure of the Marlu mining operations in 1955, and the acquisition of the Bogoso concession by Denison Mines Limited, a Canadian company, in early 1986 only saw sporadic exploration activities. These activities included the sampling of old adits and two separate drilling programs, one by the SGMC and the other by the United Nations Development Program.

In 1986, Canadian Bogoso Resources Limited, a Ghanaian company, commenced exploration on the Bogoso concession. Exploration between 1986 and 1988 outlined potential for development of mining operations on the concession. Included as part of this work was drilling of the Marlu tailings, dewatering and sampling of the Marlu underground workings to a depth of about 100 m, DD beneath the old open pits, adit sampling and trenching.

Golden Star acquired the Bogoso concession in 1999, and since that time has operated a nominal 1.5 Mtpa CIL processing plant to process oxide and other non-refractory ores (termed the Bogoso non-refractory plant). In 2001, Golden Star acquired the Prestea property located adjacent to the Bogoso property and mined surface deposits at Prestea from late 2001 to late 2006. In July 2007, GSBPL completed construction and development of a nominal 3.5 Mtpa processing facility at Bogoso/Prestea that uses BIOX® technology to treat refractory sulphide ore.

Chujah, Dumasi, Ablifa and Buesichem

Billiton Plc ("Billiton"), now known as BHP Billiton Limited, then part of the Royal Dutch Shell Group, took control of the Bogoso/Prestea property in the late 1980s and its initial feasibility study established a "mineable reserve" of 5.96 Mt with a mean grade of 4.0 g/t Au, of which 461,000 t (or less than 8%) comprised oxidised material and the remainder fresh (sulphide) material. The reporting code and key assumptions and parameters used to report this mineral resource are not known and a QP has not done sufficient work to classify this historical estimate as a current mineral resource or mineral reserve. Hence, the Company is not treating the historical estimate as current mineral resources or mineral reserves. The construction of a mining and processing facility was completed in 1991, the latter comprising a conventional CIL circuit to treat the oxidised material at a rate of 1.36 Mtpa and a flotation, fluidized bed roasting, and CIL circuit with a design capacity of 0.9 Mtpa. However, Billiton encountered operation difficulties with the fluidized bed roaster, as a result of which the operation was then focussed solely on the oxide ore. The resulting standalone CIL plant had a capacity of approximately 2 Mtpa and on-going exploration was successful in delineating further ore thereby prolonging the mine life.

Mining and exploration at Prestea has been ongoing since 1873. During the majority of this period, the work was concentrated around the Prestea Village area with the development of the underground operation and a small open pit at Plant North in the north of the Prestea concession.

6.2 Past Exploration and Development

Ariston Mining established most of the current infrastructure and underground development prior to nationalization in the late 1950s. The Prestea Underground (PUG) workings extend over a distance of 6 km along strike and down to a maximum depth of about 1,450 m below surface. The two primary shafts of the Prestea mine are the Central and Bondaye shafts.

The Central shaft is the primary access to the underground mining levels and it extends to a depth of 1,238 m below surface to 30 Level (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. Run of mine (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 1,300 to 1,600 t/day.

The Bondaye shaft extends to a depth of 1,103 m, but unlike the Central shaft there is no dedicated rock handling system at Bondaye, and cars were loaded into the cages and raised to surface.

In addition to the Central and Bondaye shafts, there are several internal shafts. The No. 4 and No. 6 shafts are located to the south of the 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 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 boreholes were drilled that year and consisted of short boreholes at the Alpha shaft that targeted a subsidiary footwall structure. Exploration drilling ramped up in the 1960s after the nationalization of the mine. More than 350 boreholes were drilled during that decade mostly targeting subsidiary structures of the Main Reef.

The focus of this technical report 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 1970s and continued until the closure of the mine in 2002.

Mampon, Abronye and Opon

The first recorded work in the Mampon area was in 1929 when maps of the Dunkwa area were produced as part of a soil and stream sediment sampling campaign. In the mid-1930s, the Ghanaian Geological Survey mapped the area as part of an extensive investigation of the volcanic-sedimentary boundary between Prestea and Obuasi. Gold exploration is also recorded from this time, although no production records exist.

Chujah, Dumasi, Ablifa and Buesichem

Gold was first commercially mined at the Bogoso/Prestea property in the early 20th century. Notably, in 1935, MGMAL started commercial scale mining of high-grade oxide material from a series of open pits extending south from Bogoso North to Buesichem, just south of the Bogoso/Prestea property. MGMAL also mined a small amount of material from underground at Bogoso North, Marlu and Bogoso South and was still mining the Buesichem pit when it shut down its operations in 1955. According to BGL's records, during its 20 year period of operating from 1935 to 1955, MGMAL produced over 900,000 oz of gold at an average recovered grade of 3.73 g/t Au.

Beta Boundary, Bondaye and Tuapim

Before 2001, little, if any work was carried out in the Bondaye and Tuapim areas. Exploration sampling was carried out over the Beta Boundary deposit immediately to the north. In June 2001, Golden Star was awarded the surface rights for the Prestea concession and commenced a program of detailed stream and outcrop geochemical sampling over the entire concession. The results from this work led to the recognition of potential exploration targets in the Bondaye and Tuapim areas and RAB drilling commenced in 2003. There are no historical (Pre-2004) mineral resource estimates for the Bondaye and Tuapim deposits.

6.3 Historic Mineral Resource and Reserve Estimates

Previous Prestea operators Barnex JCI Ltd. have historically classified a certain quantity of mineralized material as mineral resources. This historical mineral resource was reviewed by SRK, who reported this data in an NI 43-101 technical report to GSBPL in 2003. The historical mineral 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 mineral resource is classified as Inferred material under the category JCI Blocks, but is not included as a mining target in this technical report.

Previous mineral reserve estimates have been prepared by:

- SRK Consulting (UK) Limited: NI 43-101 Technical Report for the Prestea West Reef Feasibility Study, Effective Date May 1, 2013 and filed in June 2013
- GSR: Year-end reserve estimate effective December 31, 2013
- There was no reserve reported in December 2014: NI 43-101 Technical Report on Preliminary Economic Assessment, Shrinkage Mining of the West Reef Resource, Prestea Underground Mine, Ghana, effective Date December 18, 2014.

Both of these previous mineral reserve estimates relate to mechanized mining of the West Reef mineral resources and are superseded by this technical report.

Mampon, Abronye and Opon

BHP gave up its interest in the concession in the early 1990s and it was taken over by Sikaman Gold Resources Ltd, which subsequently sold its rights to Birim Goldfields Inc. ("BGI") in 1994. Bogoso Gold Limited ("BGL") entered into a joint venture with Hemlo Gold Mines Inc. ("HGM"). HGM gave up its joint venture rights in 1999, at which point some 4,500 m of trenching, 10,100 m of RC drilling and 8,500 m of DD had been carried out across the concession. During this period, the consultants Watts, Griffis and McOuat completed an independent technical review of the projects and produced an indicated and inferred mineral resource estimate totaling 1.6 Mt at 3.2 g/t Au of oxide material and a further 1.4 Mt at 1.4 g/t Au of fresh material at Mampon and Abronye combined. The reporting code and key assumptions and parameters used to report this mineral resource are not known and a QP has not done sufficient work to classify this historical estimate as a current mineral resource or mineral reserve. Hence, the Company is not treating the historical estimate as a current mineral resource or mineral reserve.

In 1999, Ashanti Goldfields became a joint venture partner and exploration continued for a further three years with 84 RC holes (5,300 m) and 26 DD (5,500 m) being drilled on the Mampon deposit. In 2002, Resource Services Group completed a technical review of an Ashanti pre-feasibility study and produced an inferred mineral resource estimate of 1.5 Mt at 4.75 g/t Au for oxide, transition and fresh horizons combined at Mampon. The reporting code and key assumptions and parameters used to report this mineral resource are not known and a QP has not undertaken sufficient work to classify this historical estimate as a current mineral resource or mineral reserve and the Company is not treating the historical estimate as current mineral resources or mineral reserves. In 2003, the properties were acquired from BGI by Golden Star.

6.4 Historic Mine Production

Mining in the Prestea area dates back several centuries. The first direct involvement by Europeans in the area occurred in the 1880s 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 southwest 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 northeast 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 4 summarizes 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 Ariston Mines and PGL was in the order of 16.8 million tonnes (Mt) of RoM material for the recovery of 5.95 million ounces (Moz) of gold. The average RoM grade is estimated to have been 11 g/t gold. In addition, the Brumasi mine is reported to have produced 0.3 Mt yielding 0.23 Moz gold for an average grade of 23.3 g/t gold. Prior to the amalgamation with Prestea, Ghana Main Reef produced about 2.0 Mt of RoM for approximately 1 Moz gold at an estimated RoM grade of 15 g/t gold. Total underground production from the area, excluding the Buesichem open pit is estimated to be in excess of 19 Mt of RoM and 7.18 Moz gold. 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 gold.

Recovered grade peaked much earlier in the life of mine with a grade of 20.4 g/t gold in 1927. Production endured a serious decline throughout the mid to late 1970s 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 until 2014 when the refurbishment started.

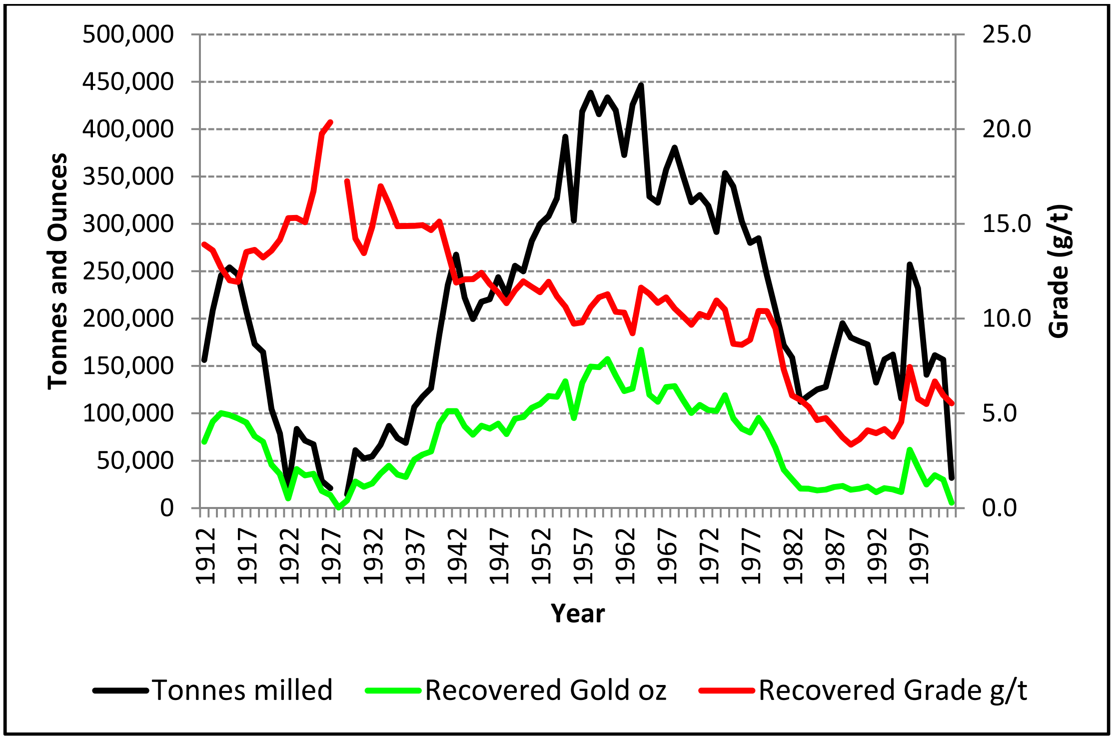


Figure 4 Historic Prestea Underground Production

(Source: GSR)

7 Geological Setting and Mineralization

7.1 Regional Geology

The regional geological setting of the Ashanti Greenstone 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 5) 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 who 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 (1928) based on outcrops located in the Birim River (approximately 80 km east of the Ashanti Greenstone Belt). Since this early interpretation, the Birimian stratigraphic column has been revised significantly. Before the application of geochronology, the Birimian Supergroup was divided into 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 2,142 ± 24 Ma and 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 into four groups by Whitelaw (1929) and Junner (1940). The Kawere Group located at the base of the Tarkwaian Supergroup 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 Supergroup 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 Greenstone Belt, intrusions within the Mpohor complex have granodioritic, dioritic, and gabbroic compositions.

Dolerite dikes 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 crosscut Archean and Paleoproterozoic basement. In southwestern Ghana these dikes are well defined in magnetic data where they are characterized by strong magnetic susceptibility. Dolerite dikes are observed to crosscut 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 dike emplacement to between 2,000 and 950 Ma. In contrast, some older dolerite and gabbro dikes 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 dikes 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).

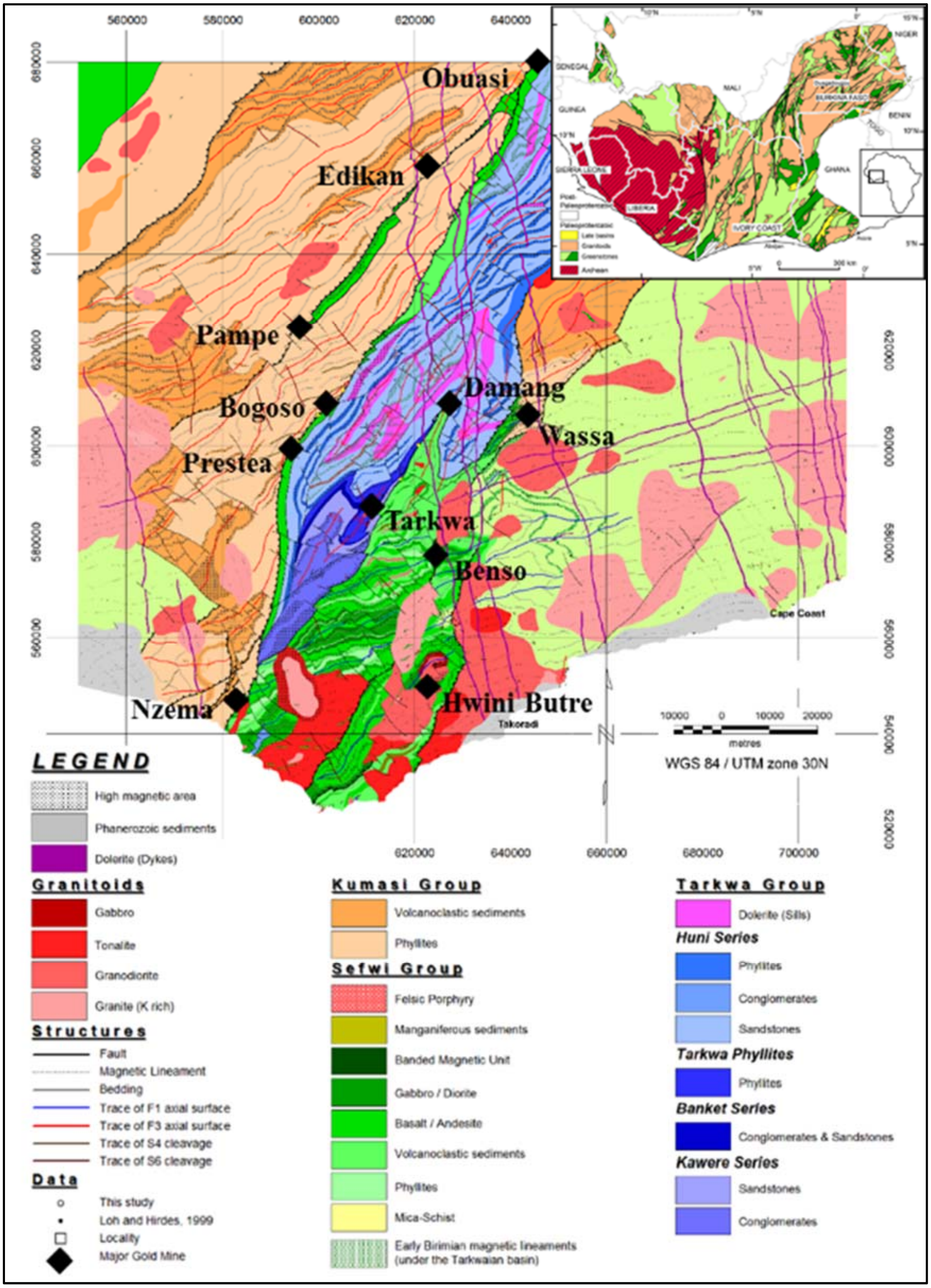


Figure 5 Regional Geology of the Ashanti Greenstone Belt and Location

(Source: Perrouty et al, 2012)

7.2 Local Geology and Mineralization

1.1.1 Introduction

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, 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.

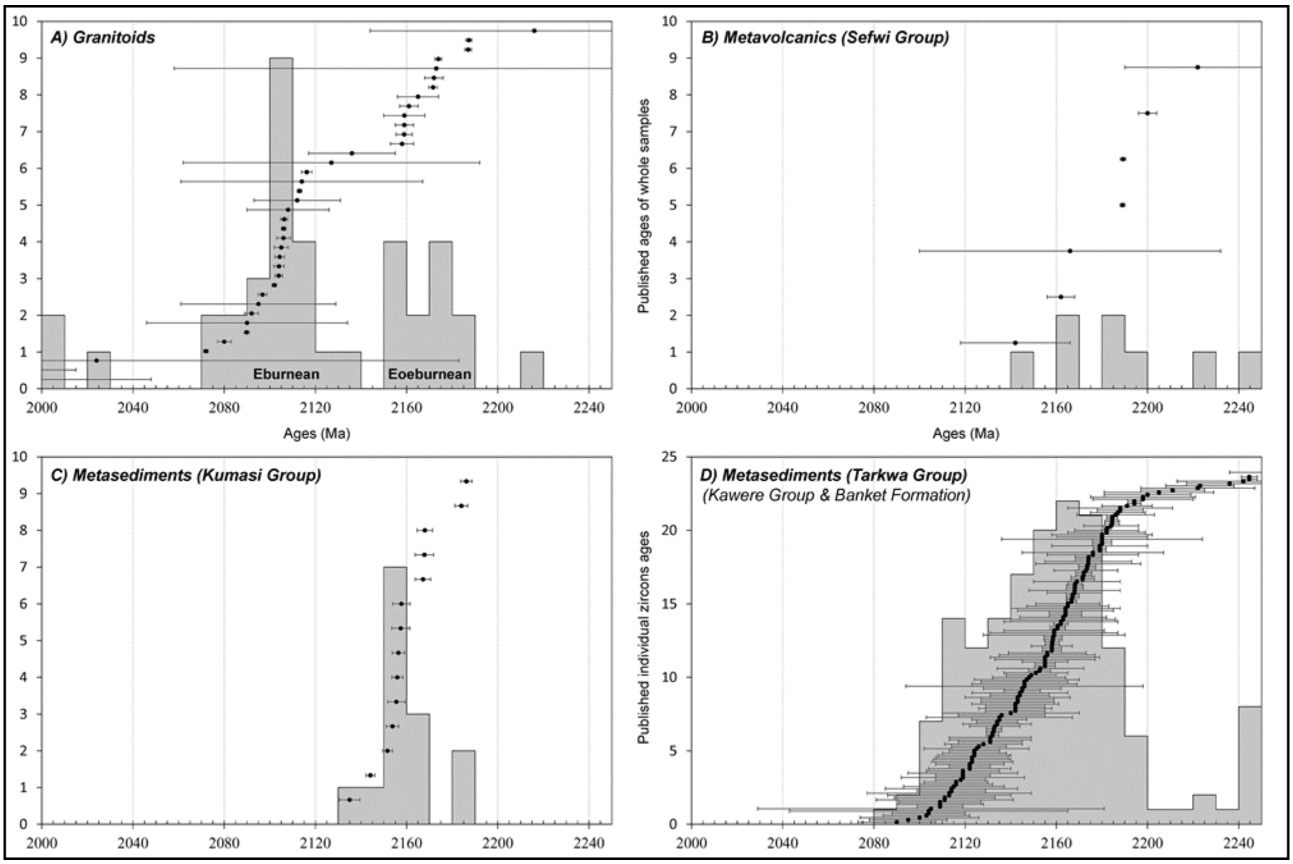


Figure 6 Radiometric Age Data Histograms

(Source: Perrouty et al, 2012)

The geology of the Prestea concession is divided into three main litho-structural assemblages (as represented in Figure 7), 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 various Birimian lithologies underlying the Prestea concession are illustrated in Figure 8.

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.

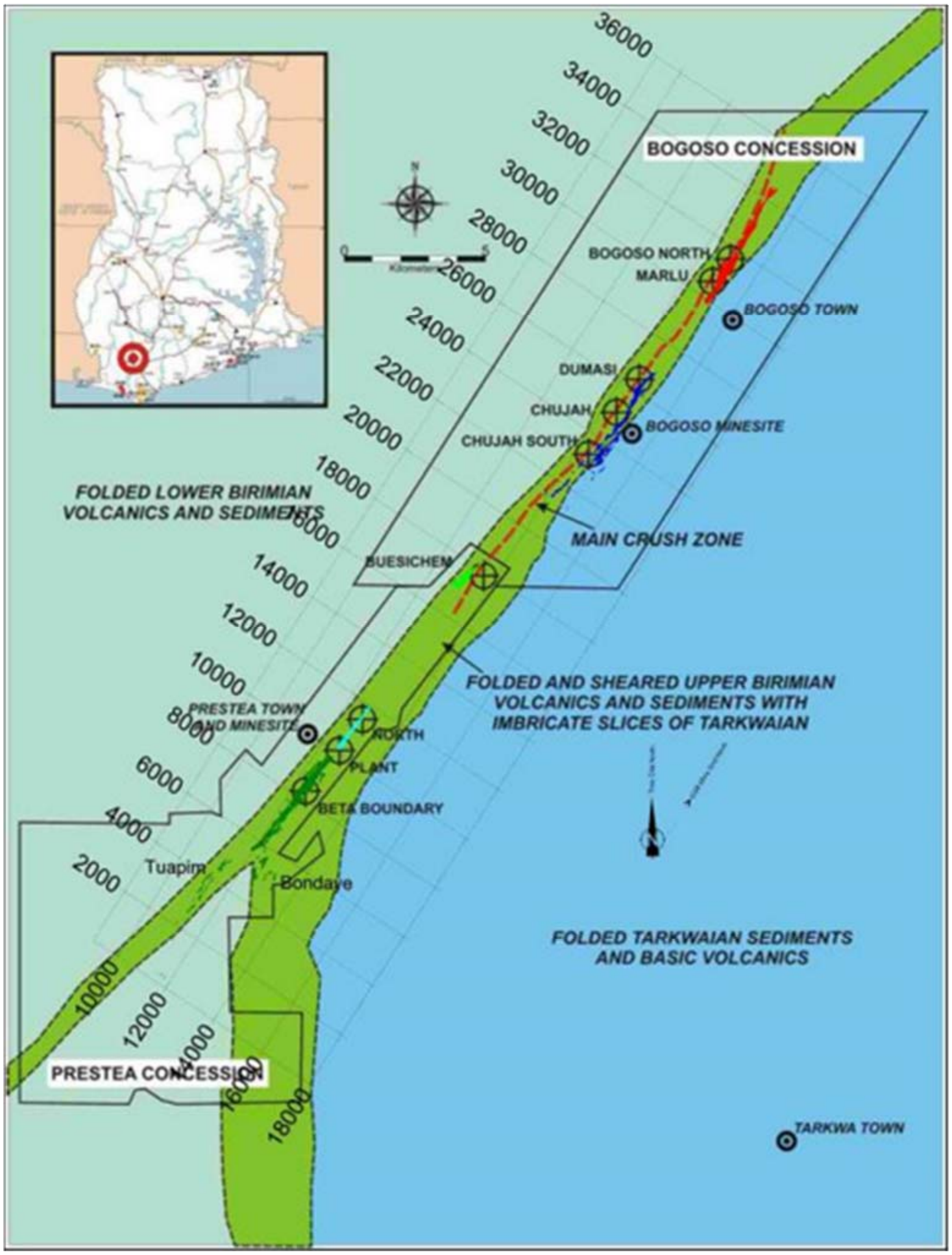


Figure 7 Simplified Geological Plan of the Prestea Concession

(Compiled from Multiple Sources, 2003)

7.2.1 Prestea Underground Mine

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, volcanoclastics, 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; all the major lithologies are illustrated in Figure 8.

Figure 8 Lithologies Hosting the Prestea Deposit

Key to Figure 8

- A: Birimian sedimentary rocks of the Kumasi Basin from the Buesichem pit.
- B: Footwall tectonic breccia (The Ashanti Shear Zone commonly referred to as a crush-zone) from Plant North pit.
- C: Mafic volcanic lens located within the footwall graphitic tectonic breccia on 17 Level of the Prestea Mine at crosscut 204N.
- D: Tarkwaian sandstone from the Beta Boundary area.

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 hanging wall and footwall of the Main Reef structure. The West Reef is a second order structure where dilational zones occurs some 200 m into the hanging wall 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 30 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 northwest-southeast shortening that 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 that 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; all major structural features are illustrated in Figure 9.

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.

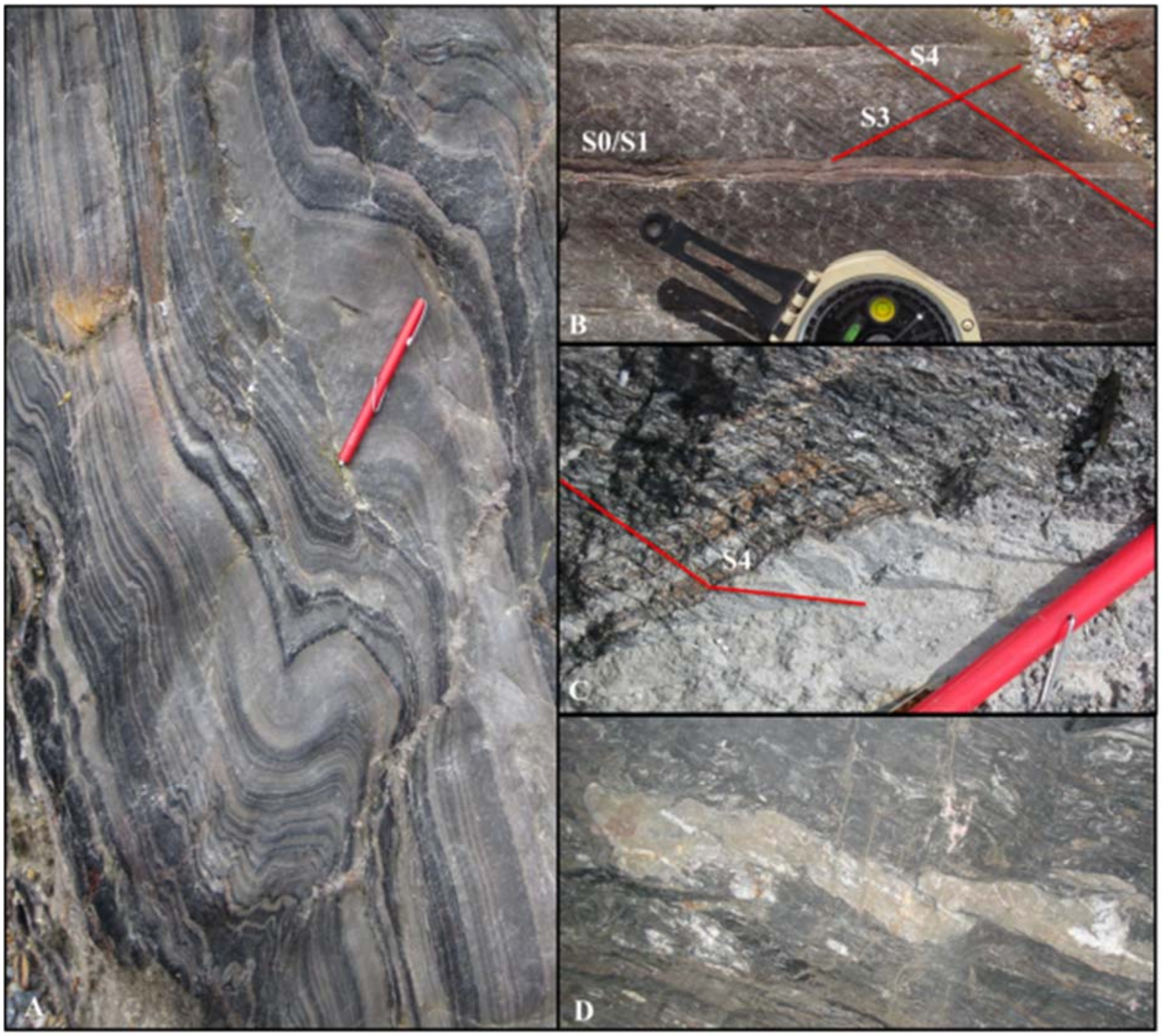


Figure 9 Structural Features Affecting the Prestea Deposit

Key to Figure 9

- A: Syn-D4 folding affecting the bedding and S1 foliation of the Birimian sedimentary units of the Kumasi basin located in the hanging wall of the Buesichem pit.
- B: Relationship between various structural features affecting the Birimian sedimentary units of the Kumasi basin located in the hanging wall of the Buesichem pit.
- C: Refraction of S4 foliation across a Birimian mudstone unit and a more competent wacke sub-unit in the Kumasi basin located in the hangingwall of the Plant-North pit.
- D: Boudinaged altered mafic volcanic lens located within the tectonic breccia in the footwall of the Main-Reef Structure, Prestea Mine, Level 30, cross-cut 219S.

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 spawning over an extended period of time, most likely from the Eoeburnean orogeny to the end of the Eburnean reactivation. Gold within the West and Main Reefs is associated with smoky grey quartz veins which re-fracture the milky quartz veins that comprise the majority of the reef structures that are evidences of multiple fluid pulses along the major faults and most likely of Eburnean timing. To date, there has been no geochronological dating on the Prestea mineralization, but gold emplacement has been interpreted to be of syn-D3 to syn-D4 Eburnean timing.

Over 500 structural measurements were taken during the course of an underground geological compilation that 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 eight 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 north 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.

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 sub-ordinate 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 mineralized quartz vein are strongly sheared and comprise a brittle-ductile zone of deformation in the graphitic schists a few centimetres to up to 2 m 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-mineralization 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 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 mineralized vein over approximately 0 to 10 m, or caused extensional offsets of the vein. One 10-metre 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 mineralization, the Ashanti trend has not been affected by a major deformation event with a sub-horizontal fold axis that may have acted to truncate the mineralization at depth. Moreover, the planarity of the mineralized 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 the past, consists of arsenopyrite rich, brecciated, and altered volcanic lenses, only the fault filled quartz vein mineralization style is illustrated in Figure 10.

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. 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 1 to 2 m 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 percent acicular arsenopyrite crystals and 1 to 5 percent 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 percent of the volcanic lenses' volume. 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 m) and stretched along a north-south trend, subparallel 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 of the Kumasi basin. The West Reef fault is characterized as a hanging wall subsidiary fault, and 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.

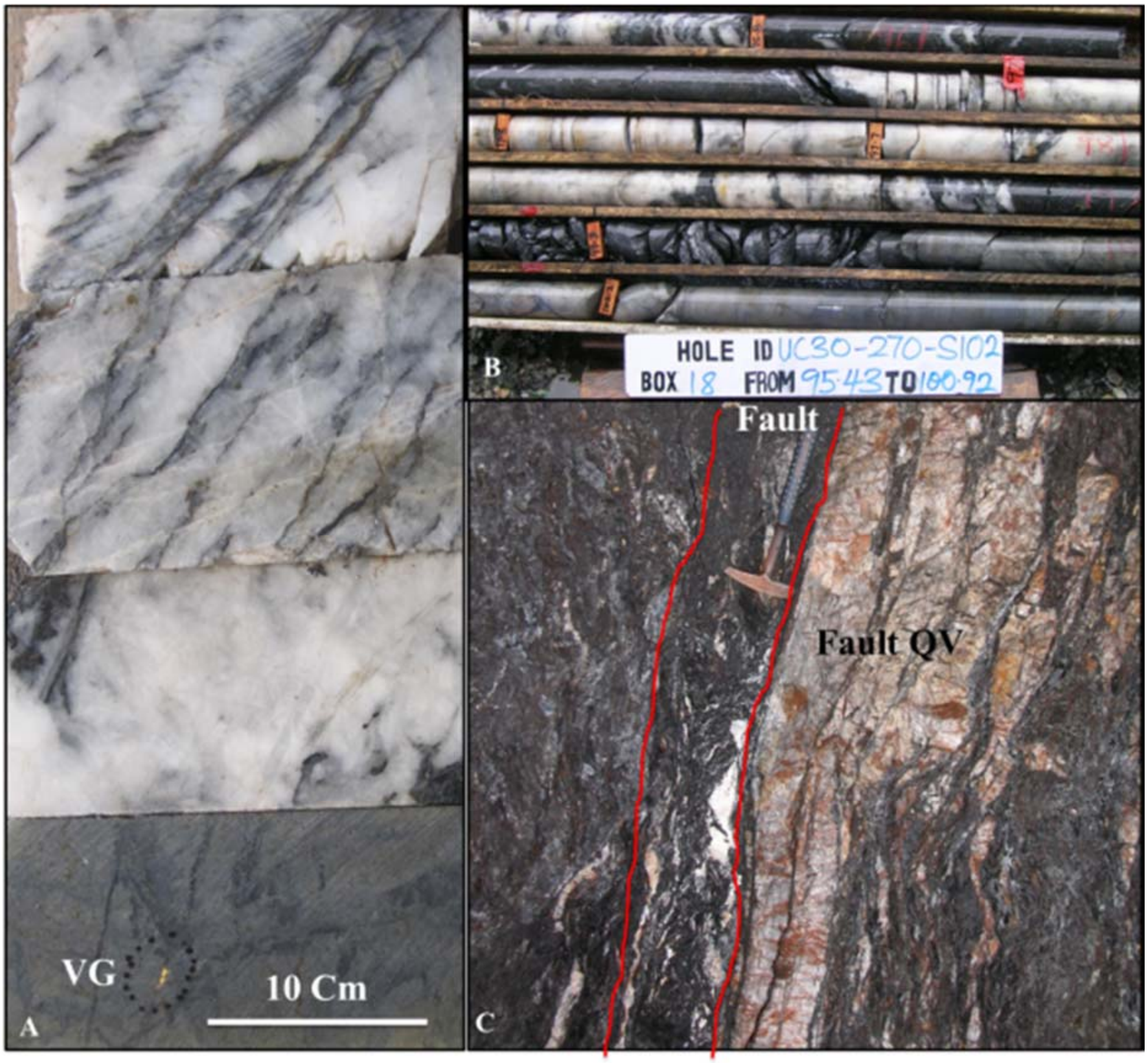


Figure 10 Mineralization Styles Characterizing the Prestea Deposit

Key to Figure 10

- A: Mineralized core intervals from four different Boreholes, from top to bottom, reef style mineralization (fault fill quartz veins) from Borehole GT-280S2 at a depth of 74.7 m. intersecting the West-Reef Structure, Borehole GT-287S1A at a depth of 130.7 m. intersecting the West-Reef Structure as well, Borehole UC30-270S-108 at a depth of 78.0m intersecting the Main-Reef Structure, and at the bottom, a mineralized volcanic lens interval with visible gold and characterized by stockwork-like smoky quartz veins intersected in the footwall of the Main-Reef Structure by Borehole UC-270S-108 at a depth of 94.8m.
- B: Main-Reef fault-filled quartz vein and fault intersected by Borehole UC30-270-S102.
- C: Reef style mineralization (fault fill quartz veins) from the Main-Reef fault at Level 17, laminated quartz veins in the footwall of the structure with graphitic fault gauge on the hangingwall side.

7.2.2 Mampon/Abronye and Opon

The Asikuma and Mansiso licenses host the Opon, Mampom and Aboronye deposits; structural setting contolling the style of mineralization is similar for all three deposits. Both concessionss are underlain by north-northeast trending metasedimentary rocks of the Kumasi basin, including coarse-grained wackes, mudstones and argillites, interpreted to represent turbiditic sedimentary sequences. Discontinuous mafic to intermediate metavolcanic rocks occur in the footwall of the main shear zones. These lithologies have been subjected to intense compressional deformation and lower-greenschist facies metamorphism.

Typically the main foliation within the sedimentary sequence dips steeply to subvertically at Mampon and Abronye while Opon is relatively flat. A persistent graphitic shear zone dips approximately 70 to 75º towards the west at Mampon. This structure has been consistently intersected within the drilling of the Mampon deposit. Carbonate-chlorite-pyrite alteration is present within the barren footwall of the shear zone. Within the shear-zone hangingwall, quartz-sericite-arsenopyrite±gold alteration is present. To the west of the shear-zone, road-cuts created during drilling expose some shallowly-west dipping quartz-veins. In addition, shallowly west-dipping quartz-veins within the western half of the Mampon deposit were identified using oriented core. Gold mineralization is developed within the graphitic metasedimentary rocks in association with quartz veining located in the hanging wall of the main shear zone, which is barren. This type of structural controls differs to other deposit along the Ashanti trend where mineralization is mostly associated with the shear zones.

Veins range from narrow stringers to robust quartz bodies up to 4 m in width. This veining reportedly is both early and late suggesting that deformation of early veins and cross-cutting later veins may be recognized. Additionally, the veining is suggested to be associated with north-northwest striking splays or oblique shears close to their intersection with a major north-south trending shear zone. Mineralization at Mampon, Abronye and Opon is associated with pyrite and arsenopyrite dissemination within the wallrock surrounding the quartz veins and within the quartz veins themselves.

The Opon deposit has a strike length of approximately 400 m and mineralization is constrained along six sub-parallel structures. The main Opon zone has a maximum thickness of 40 m while the hanging wall and footwall structures are generally narrower with an average thickness of approximately 10 m. The Opon deposit has been modelled to a depth of approximately 125 m and is open at depth.

The Mampon deposit is modeled over a 1000 meters strike length and is oriented North-South. Mineralization is constrained mainly to a continuous main structure whose thickness averages 10 to 20 m and the deposit is characterized by several discontinuous hanging wall zones. The Aboronye deposit is modeled over 700 m strike length along a South-East, North-West oriented structure. The Aboronye structure has a maximum thickness of 35 meters in the centre of the deposit, with an average width of 10 m. Both deposits have been modelled to a depth of approximately 500 m and are open at depth.

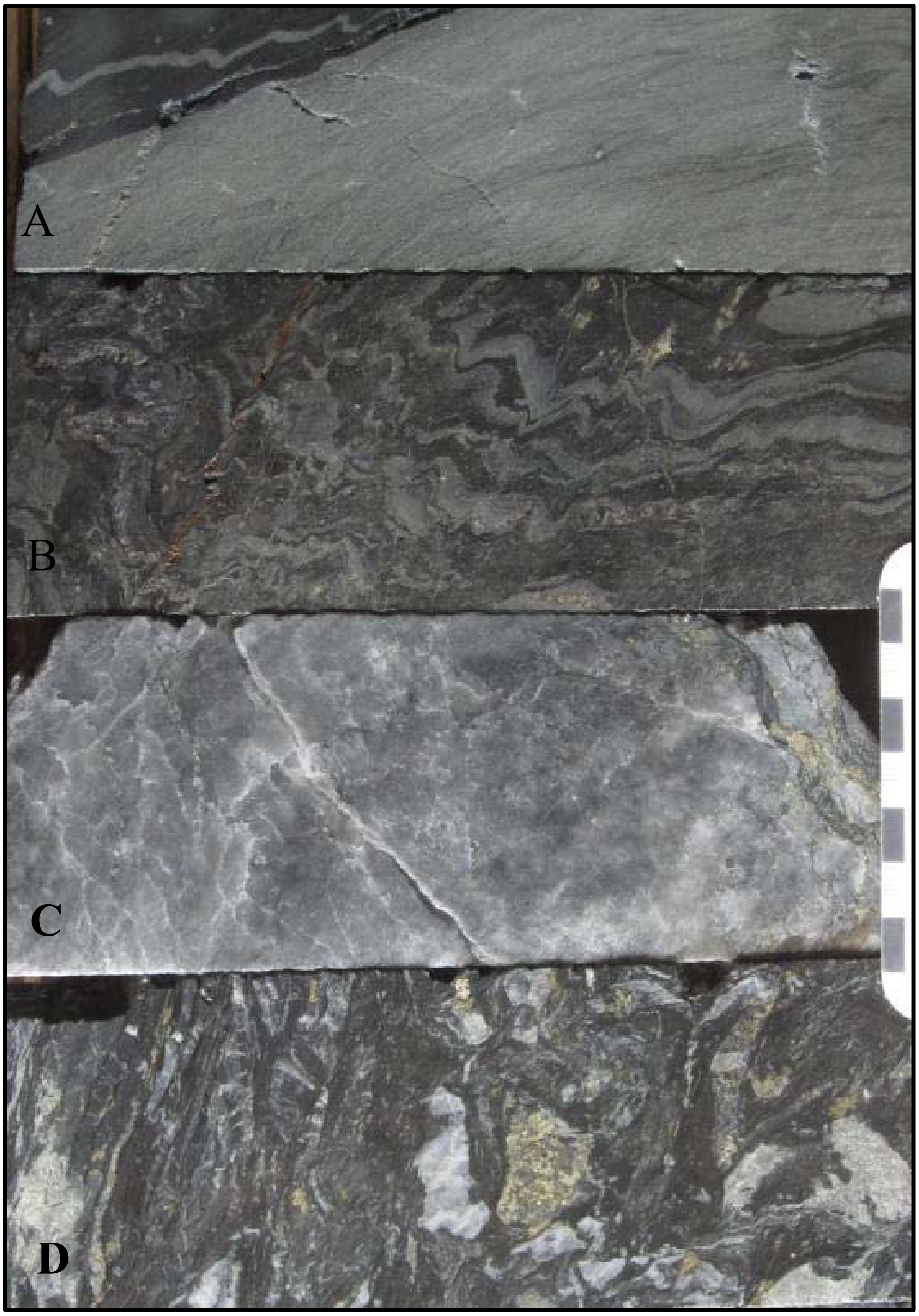


Figure 11 Various mineralization style occurring at Mampon

Key to Figure 11

- Mineralized sedimentary host rocks and associated shear zones underlying the Mampon deposit in drill hole MPMET020.
- A: Sedimentary contact between an argillitic horizon and a mineralized wacke at 52.3m.
- B: Mineralized and folded argilitic horizon at 60.9m.
- C: Mineralized quart vein and silica flooding alteration at 70.7m. D: Unmineralized shear zone located in the footwall of the Mampon deposit at 89.9m characterized by a strong shear fabric and quartz boudins. (Source: Golden Star Resources, 2013)

7.2.3 Bogoso North/Marlu

Gold mineralization between Marlu and Bogoso North is restricted to a narrow graphitic fault zone located in the sedimentary hanging wall of the deposits with the footwall composed of volcanic lenses and sheared sedimentary rocks. The Bogoso North deposit consists of two splays of the main shear zones, a quartz vein dominated hanging wall splay, and a highly graphitic footwall structure. The two splays of the main shear zone at Bogoso North extend for approximately 500 m along strike and range in true width from 5 to 15 m. Gold mineralization at Bogoso North dips moderately to the northwest at 40 to 50º. The mineralisation is modelled over a 2 km strike at Bogoso North and an additional 2.7 km northwards to a depth of some 300 m. Bogoso North gold mineralization is associated with either quartz veins or graphitic cataclasites.

Quartz vein hosted gold mineralization generally is coincident with relatively high gold tenors, typically between 5 to 15 g/t Au. Vein gold is associated with abundant disseminated arsenopyrite in and around slices of graphitic mylonitic rock within the quartz veins. Although visible gold is occasionally observed in the veins at Bogoso North there are not abundant amounts of free gold as at Prestea and Ashanti. The quartz veins mark dilatant fractures, which focussed gold-bearing fluids, although gold deposition appears to have been promoted by interaction with adjacent wall rock fragments (Allibone et al 1998). The dilatant zones enabling quartz vein emplacement are localised in a left-stepping fault segment in the hanging wall fault splays of the main shear zone. Other veins are developed at the intersection of hanging wall spays with the main shear zone footwall structure.

The footwall graphitic shear mineralization at Bogoso North generally is associated with lower gold values than that of the quartz vein hosted hanging wall structure. Gold mineralization is associated with finely disseminated pyrite and arsenopyrite.

Key to Figure 12

Volcanic lense in the Bogoso North Pit steeply dipping to the West with sheared graphitic sedimentary package in the hanging wall. The lower picture illustrates the deformed nature of the quartz veins within the graphitic shear zone. (Source: Golden Star Resources, 2013).

Figure 12 Various mineralization style occurring at Bogoso North

7.2.4 Chujah/Dumasi

Both Birimian and Tarkwaian lithologies are situated within the Chujah/Dumasi area. Birimian and Tarkwaian rocks are separated by a deformation corridor referred to as the central structural corridor or tectonic breccia. The tectonic breccia is hosted both in Birimian and Tarkwaian rocks and is characterized by an anastomosing network of faults and imbricated fault slices. These northeast-striking faults represent the principal thrust planes which were developed during the third event of deformation at the beginning of the Eburnean orogeny. The largest fault zone, located close to the western boundary of the deposits is referred to as the main shear zone and is characterized by fault gouge, mylonites, breccias and cataclasites.

The attitude of the main shear zone varies locally but generally strikes between N025º to N040º, dipping between 35 to 75º towards the northwest. The thickness of the main shear zone ranges from a few metres to over 50 m in true width. The combined length of Chujah and Dumasi is some 3 km along strike and the deposit has been modelled to a vertical depth of 500 m. The main shear zone also hosts multiple generations of internal non-cylindrical folding, pervasive foliation development, quartz-ankerite veining and graphite ± ankeritic, carbonate ± sericite alteration assemblages. Reversals in stratigraphic younging direction across the main shear zone indicate that it forms the hinge of a km-scale shallow-plunging isoclinal syncline. The widespread development of at least sub-economic gold mineralization along essentially the full strike length of the main shear zone and numerous subsidiary structures implies that movement on them was widespread during mineralization.

The eastern edge of the tectonic breccia zone is marked by another major fault zone; the Tarkwaian Boundary Fault zone. The Tarkwaian Boundary Fault zone separates the basal Tarkwaian gritty immature sandstones, pebble conglomerates and interbedded tuffs from the Tarkwaian, Kawere quartzites and conglomerates. The attitude of the Tarkwaian Boundary Fault is slightly steeper than the main shear zone with dips in the range of 60 to 80º towards the northwest. Numerous fault networks interconnect the two bounding fault structures where two dominant interconnected fault orientations have been observed one striking east-northeast and the other striking north to north-northeast.

Figure 13 Various mineralization style occurring at Chujah

Key to Figure 13

Chujah pit lithologies. The top picture represents sheared graphitic sedimentary rocks of the Kumasi basin in contact with altered mafic volcanic lenses. The lower picture is a mineralized graphitic shear zone characterized by quartz vein boudins and ankerite alteration. (Source: Golden Star Resources, 2013).

7.2.5 Ablifa

The Ablifa deposits are also situated along the Ashanti Trend south of Chujah and North of Buesichem. The geology is therefore similar to the host lithologies found at both deposits, with the hanging wall composed of the Kumasi sedimentary basin and the footwall of tectonically imbricated Birimian and Tarkrwaian rocks, including volcanic lenses and sandstones.. Mineralisation occurs within a narrow north-east striking corridor, in which mineralization dips predominantly to the northeast at angles ranging between 50 and 70º. The deposit is modelled over a strike length of 4.3 km and to a maximum vertical depth of 250 m, with mineralization constrained to narrow mineralized structures which averages approximately 5 to 20 meters.

The area is situated between the Buesichem North Pit, on the southern boundary of the Bogoso lease area and the Mackenzie/Big Hill pits immediately south of Chujah South Pit. The area includes the Deinsu Pits (South, 2 and Central) as well as the Ablifa West, Ablifa 2 West, Polipoli Pataye and Akosu Pits. Five distinct mineralization styles have been described including the following:

- graphitic shear zone hosted mineralization;
- carbonate alteration associated mineralization;
- wall rock hosted mineralization;
- siliceous alteration hosted mineralization; and
- quartz vein hosted gold mineralization.

Within the first four styles of mineralization, gold is predominantly occluded within sulphides except where the original mineralogy has been altered by weathering.

7.2.6 Buesichem

The Buesichem deposit is the most northerly deposit located on the Prestea concession. In the vicinity of the Buesichem deposit the main shear zone appears to encroach on the Birimian – Tarkwaian contact, whereas this contact is typically 250 to 300 m east of the main shear zone on the Bogoso concession, however this contact might be the result of a Tarkwaian sliver thrusted along the main shear zone. As the contact is approached, the frequency of imbricated fault slices of Tarkwaian increases. In the Buesichem pit, the eastern wall is composed of discontinuous mafic volcanic lenses and a phyllite unit which, has been interpreted as Tarkwaian. The deposit is modelled over a 1.3 km strike length and to a maximum depth of 500 m.

The Buesichem mineralization is hosted within a sequence of greywackes, turbidites and phyllites of the Kumasi sedimentary basin with local narrow horizons of metavolcanics located in the footwall of the main shear zone. The entire package has been extensively deformed forming a shear zone which is up to 100 m wide and comprises splays of anastomosing protomylonitic to cataclastic bands as defined by phyllosilicate bands and discrete, but discontinuous, re-crystallized quartz ribbons and porphyroblasts.

Shear zones within the Buesichem pit indicate steeply dipping gold mineralization (>70º) with a plunge towards the south-southwest. Gold mineralization is constrained by splays and bifurcation shear planes composing the main fault zone.

Mineralogical studies have consistently shown the presence of undeformed sulphide minerals indicating that the mineralization was post-deformational in age. The fact that up to 50% of the gold is occluded in arsenopyrite and is predominantly fine-grained (1-30 µm) accounts for the refractory nature of the ore.

Owing to the unique structural setting, gold mineralization of the Buesichem area is distinctly different from the rest of the Prestea deposits with respect to the texture. Ore paragenesis is nevertheless similar. Buesichem sulphide gold mineralization is similar to the deposits located along the 18 km strike length of the Ashanti trend north on the Bogoso Concession.

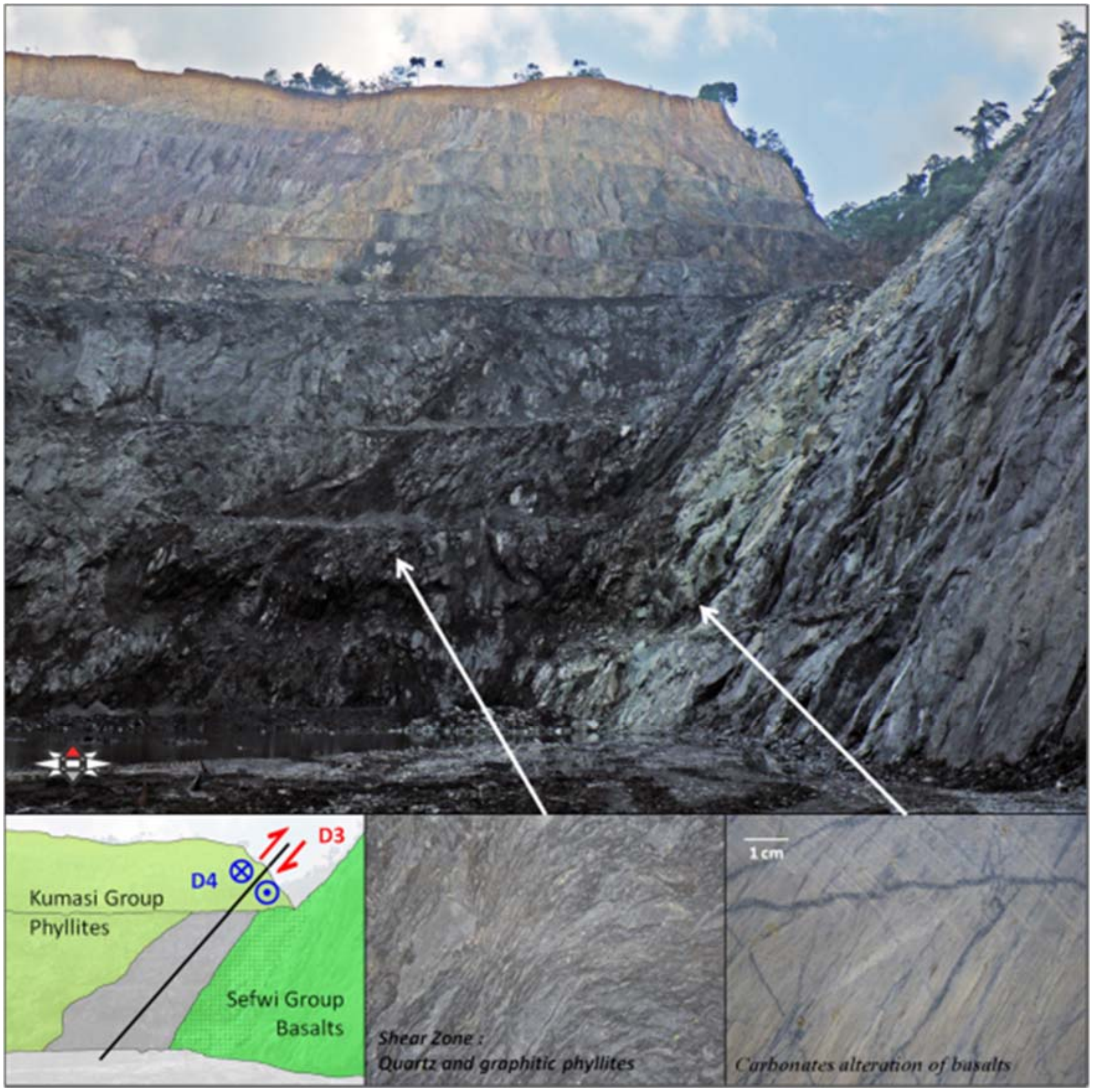


Figure 14 Various mineralization styles occurring at Buesichem

Key to Figure 14

Host rock sequence on the Buesischem North wall illustrating the mineralization styles of the Buesichem deposit with strongly shear graphitic sediments in the Hanging wall of the main shear zone and altered volcanic lenses in the footwall (Source: Perrouty et al., 2012).

7.2.7 Beta Boundary, Bondaye and Tuapim

The Beta Boundary, Bondaye and Tuapim deposits are collectively referred to as the Prestea South project and are located approximately 10 km south of the Buesichem deposit along the Ashanti trend. Locally, mineralization at Beta Boundary, Bondaye and Tuapim is characterised by lode gold mineralization which typically contains non-refractory, free milling gold associated with arsenopyrite and hosted within laminated fault filled quartz veins.

Oxidation of the upper layers of the deposits is extensive and in places can reach tens of metres in depth. Generally oxidation is deeper in areas of higher topography and shallower in the valleys and in areas of previous mining activity. There is a thin (generally <10 m) transition zone between the oxide and fresh material. This transition zone has been modelled as a separate unit in all three of the deposits in order to more effectively characterise the statistics of the individual layers.

Beta Boundary is modelled over a 4 km strike length and to a depth of 450 m while Bondaye is modelled over a 1.3 km strike and Tuapim over 2 km. Bondaye and Tuapim are modelled to a maximum vertical depth of 150 m. Mineralization along the Prestea South deposits is constrained along narrow structures with width varying between 5 to 20 meters.

The Prestea south deposits are hosted by three main litho-structural assemblages which are fault bounded and steeply dipping to the west. From the eastern footwall to the western hanging wall these packages are represented by the Tarkwaian litho-structural assemblage and the tectonic breccia assemblage, composed of sheared graphitic sediments and volcanic flows. The mineralized fault fill quartz veins tend to develop along brittle structures within this litho-structural package. 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 west is composed of sandstone, pebbly sandstone and narrow conglomerate units. 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, volcaniclastics, 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 deformational event (Syn-D4 to syn-D5). The most western litho-structural assemblage within the Bogoso Prestea Mine site 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.

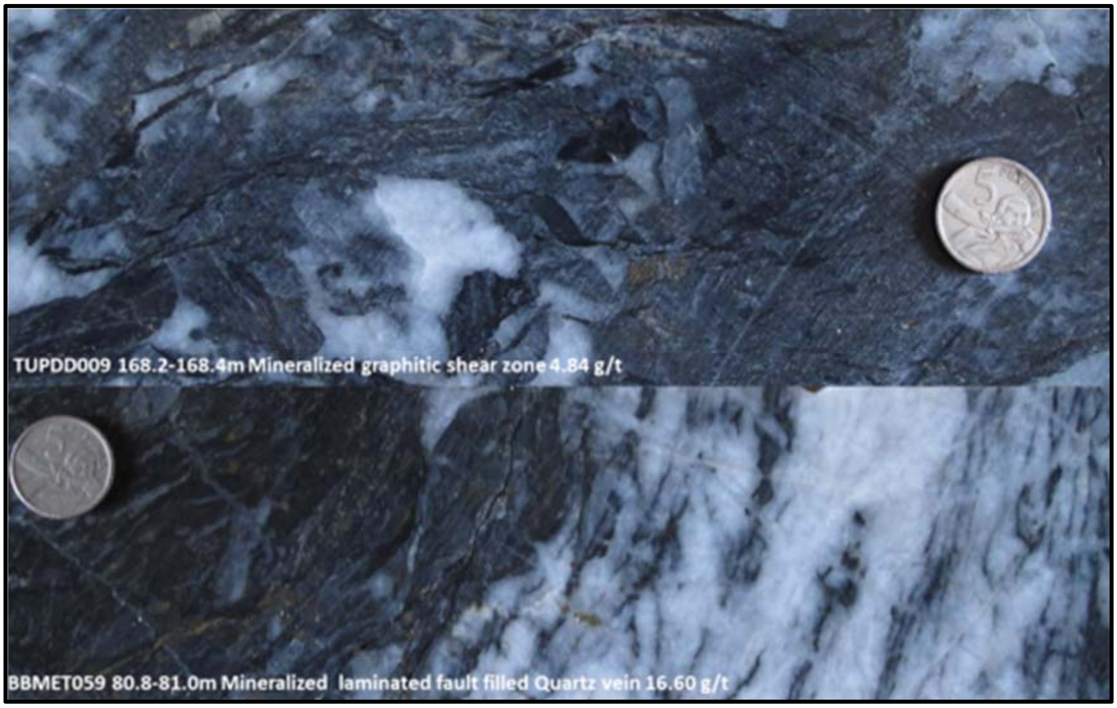


Figure 15 Various mineralization styles at Prestea South

8 Deposit Types

The Golden Star Resources deposits are located on the 250 km long northeast-southwest trending, Ashanti Belt, a Paleoproterozoic granitoid-greenstone assemblage of southwest Ghana. These greenstone belts and dividing sedimentary basins were formed and deformed during the Eoeburnean and Eburnean orogeny. The Prestea-Bogoso area occurs at the southern termination of the Ashanti Belt, where the gold deposits, mined or under exploration, are localised principally along two steep to subvertical major crustal structures referred to as the Ashanti trend and the Akropong trend as illustrated in Figure 16. The principal structures are graphitic shear zones and mineralised fault filled quartz veins.

The Bogoso Prestea section of the Ashanti Trend shows a range of mineralisation styles associated with graphitic shear zone, which represents the principal displacement zone of a regional-scale shear zone that defines the mineral belt. These styles include laminated quartz vein deposits containing free gold, highly deformed graphitic shear zones containing disseminations of arsenopyrite as the principal gold bearing phase (e.g. Buesichem, Chujah-Dumasi and Bogoso North) and disseminations of sulphides in mafic/intermediate volcanic rocks generally found in the footwall of the main shear zone.

The Bogoso Prestea deposits can be classified as a lode gold deposits or orogenic mesothermal gold deposits, which are the most common gold systems found within Archean and Paleoproterozoic terrains. In the West African shield, orogenic gold deposits are typically underlain by geology considered to be of Eburnean age and are generally hosted by volcano-sedimentary sequences. The Ashanti belt is considered prospective for orogenic mesothermal gold deposits and hosts numerous other lode gold deposits such 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 sulphidised wall rocks or within silicified and arsenopyrite-rich replacement zones.

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

- Arsenopyrite-pyrite rich graphitic shear zones;
- 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-pyrite rich and silicified replacement zone.

The graphite rich shear-hosted and volcanic hosted mineralization types are refractory and generally lower grade in comparison to fault-filled quartz vein hosted mineralization type.

The weathering profile at Bogoso Prestea is deep and typically results in extensive surface oxidation of bedrock, to a depth of up to one hundred metres. Generally, the weathering profile typically consists of a lateritic surface, a saprolitic horizon, a transitional zone and a deeper primary sulphide zone.

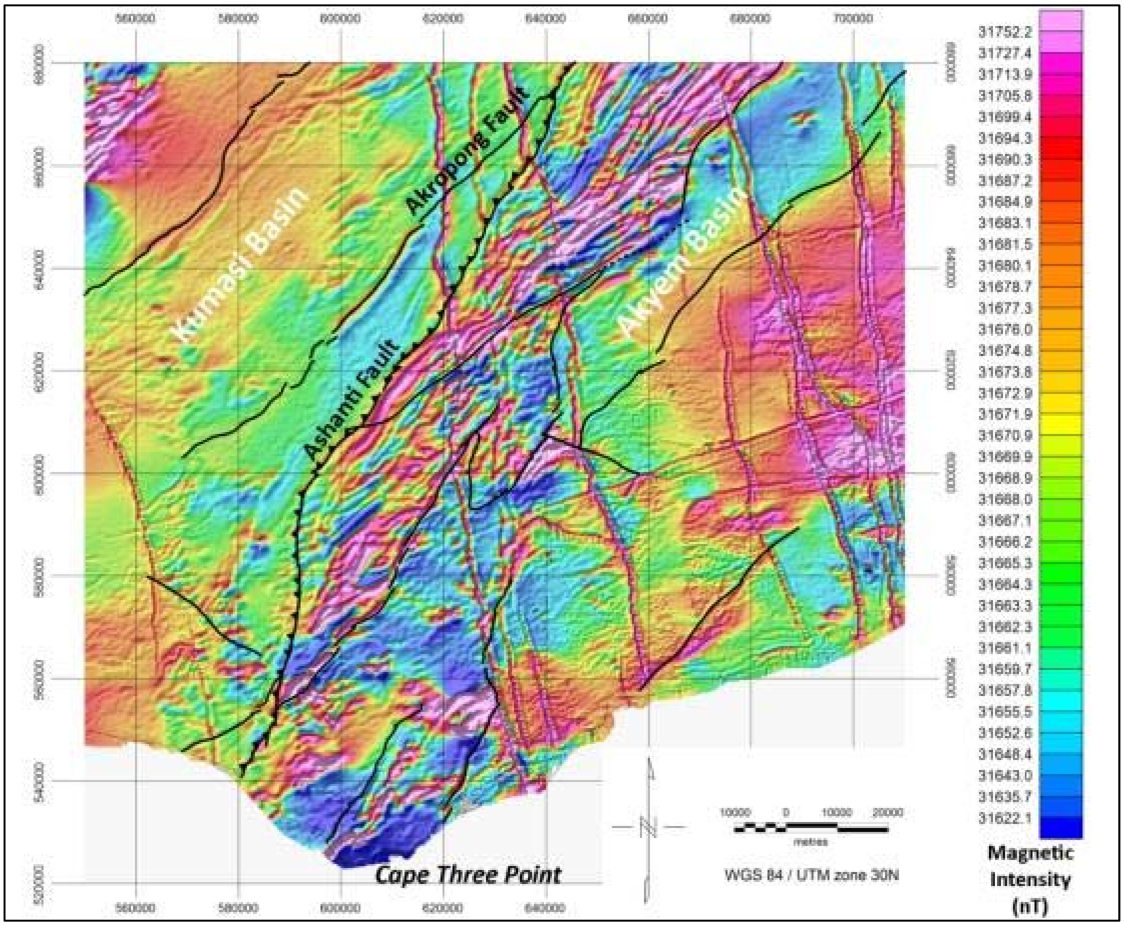


Figure 16 Total magnetic intensity - Ashanti Belt

(Source: Perrouty et al., 2012)

The Ashanti Greenstone Belt is considered prospective for orogenic mesothermal gold deposits and hosts numerous lode gold deposits and paleoplacer deposits. As illustrated by Figure 5, several major gold deposits are found within the Ashanti Belt that can be classified into six different deposit types:

- Sedimentary-hosted shear zones
- Fault-fill quartz veins
- Paleoplacer
- Intrusive-hosted
- Late thrust fault quartz veins
- Folded veins system

The sedimentary hosted shear zone deposits are localized principally along a steep to subvertical 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 that 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 paleoplacer 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 that 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 are late thrust fault associated quartz vein deposits. The Damang mine, which is located in the Takwaian 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 within 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 an 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 characterized as mesothermal gold mineralization:

- Fault-fill quartz veins along fault zones and second order structures, which typically contain non-refractory, free milling gold.
- 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.

9 Exploration

Exploration was historically carried out by drilling and is summarised in the following section. Prior to drilling, soil geochemistry sampling was carried out over the main deposits as an aid to targeting of initial drilling programs. Extensive geophysical surveys were also conducted over the concessions which include aero-magnetic surveys, Versatile Time-Domain Electromagnetic ("VTEM") surveys and induced-polarization surveys.

9.1 Bogoso Prestea

The earliest reported mining activity on the Bogoso concession was at the Bogoso South deposit, where in 1905 and 1906, approximately 7,500t of material was excavated from two shafts. Processing of the mineralisation yielded 5,300 oz of gold. Bogoso South underground workings were pumped out in 1933 and a third shaft was sunk 30 m north of the old workings. The third shaft intersected a 1.2 m quartz reef that was sampled and returned reported grades of 6.2 and 18.6 g/t Au.

Historical exploration mining activities from the 1930s to mid-1950s are discussed in Section 5.3. The first drilling program, carried out by SGMC, was centred on the old Buesichem lease, located at the southern boundary of the Bogoso concession. The 36 holes drilled outlined a broad, low-grade zone of gold mineralization. In a United Nations report from 1983, tonnage was estimated at about 3 Mt grading 5 g/t Au and was located in two lenses to a depth of about 200 m.

The UNDP drilling program was carried out in 1976 to 1978 in the vicinity of Chujah and Dumasi. Several holes were also drilled at Bogoso North in an effort to trace possible extensions of the large oxidized deposit previously mined by MGMAL. Between Chujah and Dumasi, the UNDP drilled 21 holes spaced 122 m apart, covering a strike length of approximately 1,000 m. Results of the drilling done at Bogoso North were discouraging, but at Chujah and Dumasi, a broad zone of mineralization was outlined.

In 1986, CBRL commenced exploration on the Bogoso concession. Exploration between 1986 and 1988 outlined potential for development of mining operations on the concession. Included as part of this work was drilling of the Marlu tailings, dewatering and sampling of the Marlu underground workings to a depth of about 100 m, drilling beneath the old open pits, adit sampling and trenching at Chujah, Dumasi, Dumasi North, and Boppo3.

Exploration activities carried out by CBRL by mid-1988 included:

- 10,835 m over 146 holes of drilling at Chujah, Dumasi, Nankafa, Marlu, Bogoso North and Boppo2;
- 3,738 m of trench sampling at Chujah, Dumasi, Nankafa, Bogoso North and Boppo2; and
- 1,666 m of channel samples from old MGMAL exploration adits at Chujah, Dumasi, Nankafa and Boppo2.

In 1989, JCI Limited acquired the Prestea concession and collared 38 holes totalling about 6,000 m at the Buesichem deposit. JCI also tested the southern extensions of the Buesichem deposit along the Beposo oxide pits which were drilled further by Bogoso Gold Limited ("BGL") and have since been mined and processed.

In late 1994, some 2,526 m of DD was carried out in Chujah and Bogoso North pits to provide HQ core for bio-oxidation testwork using the BIOX® marketed by Biomin Limited, a subsidiary of Gold Fields.

Billiton took control of the Bogoso property in the late 1980's. The initial feasibility study established a total of approximately 6 Mt of material grading 4.0 g/t Au, of which approximately 0.5 Mt comprised of oxide mineralisation. The feasibility study forecast gold recoveries of 83% from sulphide mineralisation and 78% from oxide material and estimated a stripping ratio (t_{waste}:t_{ore}) of 5.6:1. Construction of a mining and processing facility was completed in 1991. The facility was designed to process oxides by using conventional CIL technology at a design capacity of 1.36 Mtpa and to process sulphides by using flotation, fluid bed roasting and CIL technology at a design capacity of 0.9 Mtpa.

Billiton conducted only limited exploration programs between 1991 and 1998. GSR acquired the rights to the Bogoso operations in 1999 and acquired the surface rights for Prestea in 2001. Following these acquisitions, GSR conducted exploration programs along the entire Ashanti trend covered by the Bogoso and Prestea concessions which included mapping programs, pit sampling, soil geochemistry and drilling programs.

In 2007, GSR contracted Geotech Airborne Geophysical Surveys ("'GEOTECH") to run a VTEM survey which they flew over the entire project area. The total drill production up to end of 2012 stands at 89,216 m of RAB, 141,115 m of Reverse Circulation ("RC") and 195,133 m of DD drilling. A number of targets were generated from the VTEM survey and a drilling program was embarked upon in 2009 ending in 2011 resulting in some 22,475 m of RC and 93,425 m of DD drilled over a total of 230 RC and 364 diamond holes over the various deposits.

9.2 Mampon-Abronye

The first recorded work on the Asikuma Concession was in 1929. Maps with geological data and the location of soil and stream samples covering part of the concession, formerly referred to as the Jappa Mansi and Asikuma Options, are on file at the Minerals Commission in Accra.

In 1934 and 1937, the Ghana Geological Survey mapped the western boundary of the Tarkwaian basin and the Birimian metavolcanic-metasedimentary contact from Bogoso to Dunkwa. Exploration for gold was carried out at the same time as this mapping, indicating the presence of gold in stream gravels draining westward across the metavolcanic-metasedimentary contact from the village of Mampon northward to the village of Nsuaem, a distance of about 8 km.

Abandoned gold prospects and small-scale workings of unknown age are found at Chirbra on the Asikuma Concession. Chirbra is located 6 km south of Dunkwa and includes several shafts up to 20 m in depth. These are recorded on an old survey map compiled by Gold Coast Selection Trust in 1937. Exploratory drifting was made from these shafts and also from several adits in the prospect area. No production is recorded. Gold Coast Selection Trust also prospected towards the north end of the Asikuma Concession and also undertook trenching and sampling in the immediate area of the Chirbra Prospect.

From 1988 to 1992, BHP - Utah conducted a regional stream and soil geochemical and VLF-EM surveys covering what constituted Birim Goldfields' Asikuma and Mansiso Concessions. They located three new prospects, Mampon, Abronye and Adukrom within the Asikuma Concession. These prospects are located at the south end of the Mampon-Nsuaem zone of placer gold occurrences identified by the Ghana Geological Survey in the mid-1930s.

Birim Goldfields and Battle Mountain Gold ("BMG") took over the property when BHP abandoned it in 1992 and undertook the following exploration activities, induced polarization surveys, ground VLF-EM, airborne multi-frequency electromagnetic and aeromagnetic survey, processing of satellite imagery data, soil geochemical sampling, mapping, trench sampling, RAB, RC, DD and metallurgical test work covering the Mampon deposit.

GSR entered into an agreement with Birim Goldfields in 2003 to acquire the Asikuma and Mansiso concessions. GSR took over ownership of the concession following the agreement and undertook exploration activities which included auger sampling, regolith mapping, RAB, RC and DD.

In 2006, a baseline environmental monitoring study as part of the environmental impact assessment for the Mampon project was conducted. Community consultations for the Mampon project were conducted throughout the various exploration programs, a public hearing was held on 9 November 2006.

In 2007, GSR contracted GEOTECH to run a VTEM survey over both concessions. A total of 17 RC holes, 30 DD holes and four geotechnical holes were drilled at Mampon over the years by Birim Goldfields and GSR for a total of 7,454 m of drilling.

9.3 Prestea Underground

Data validation and selected evaluation drilling from underground have helped to increase the confidence in the morphology and orientation of the mineralization at Prestea. Crosscut samples and Barnex 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) boreholes drilled by GSR and underground channels and diamond boreholes acquired by GSR as part of their purchase of the Prestea mining rights.

The crosscut 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 1,400 m from surface. The GSR data is largely concentrated in the area underlying the Plant North open pit, Central shaft and the northern extent of the Beta boundary.

The mineral resource estimate for the Prestea underground orebodies was based on a combination of GSR underground sampling from some 315 boreholes, 123 rock saw samples, and 160 chip channel samples from 13 crosscuts and 5 production alimak raises. The bulk of the drilling was conducted from 2003 and throughout 2006. GSR drilled an additional 14 underground boreholes into the West Reef (WR) orebody in 2012 and 2013 for geotechnical and metallurgical testing purposes. West Reef Reserve Infill drilling adding another 24 holes was completed in early 2017, upon resumption of operations, followed by the resumption of extensional drilling in the West Reef with a further 13 drill holes added. This drilling has been carried out using fan drilling from cubbies on the most accessible levels predominantly from the 12, 17, and 24 Ls.

The Prestea underground West Reef target was the last area to be mined by Prestea Gold Resources Limited (PGR) in 2002. The subsequent exploration of the West Reef underground target has been planned and managed by GSR and was initiated in 2003. 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 gold. The results are summarized in Table 4.

Table 4 Channel Sample Results for 17 L West Reef Drive

Channel #	X (m)	Y (m)	Z (m)	Level	From (m)	To (m)	Interval (m)	Grade g/t Au
17WR_01A	11949.0	8041.3	4464.9	17	0.0	2.6	2.6	0.6
17WR_02	11946.0	8034.2	4466.3	17	0.0	1.9	1.9	6.2
17WR_02A	11946.3	8034.5	4467.1	17	0.0	1.9	1.9	6.5
17WR_03	11945.0	8029.1	4466.6	17	0.0	1.3	1.3	127.1
17WR_03A	11944.9	8028.8	4466.8	17	0.0	1.3	1.3	1.7
17WR_03B	11945.0	8029.7	4466.5	17	0.0	1.5	1.5	1.1
17WR_04	11943.9	8022.1	4466.2	17	0.0	2.0	2.0	20.2
17WR_04A	11944.2	8022.8	4467.0	17	0.0	2.1	2.1	3.9
17WR_05	11943.3	8018.0	4466.3	17	0.0	2.3	2.3	5.3
17WR_05A	11943.4	8018.2	4467.0	17	0.0	2.3	2.3	9.0
17WR_06	11943.0	8015.0	4466.6	17	0.0	2.4	2.4	64.9
17WR_06A	11943.2	8015.2	4466.6	17	0.0	2.0	2.0	1.7
17WR_06B	11943.1	8014.0	4466.3	17	0.0	2.8	2.8	22.2
17WR_07	11937.5	7992.6	4466.2	17	0.0	3.0	3.0	37.4
17WR_07A	11937.5	7993.1	4466.5	17	0.0	2.8	2.8	63.4
17WR_07B	11937.3	7992.3	4466.1	17	0.0	3.1	3.1	13.9
17WR_08	11936.1	7987.9	4466.0	17	0.0	3.1	3.1	74.9
17WR_08A	11936.3	7988.2	4466.0	17	0.0	3.2	3.2	69.5
17WR_08B	11936.3	7988.3	4466.0	17	0.0	3.0	3.0	28.7
17WR_09	11934.7	7977.8	4466.0	17	0.0	2.5	2.5	31.4
17WR_09A	11934.7	7978.2	4466.0	17	0.0	2.3	2.3	53.2
17WR_10	11933.1	7967.1	4465.6	17	0.0	3.0	3.0	42.1
17WR_10A	11933.1	7967.4	4465.5	17	0.0	2.9	2.9	1.8
17WR_10B	11933.1	7967.6	4465.6	17	0.0	2.8	2.8	18.4
17WR_11	11928.6	7941.2	4464.9	17	0.0	2.2	2.2	116.9
17WR_11A	11928.4	7940.7	4465.0	17	0.0	1.6	1.6	6.2
17WR_11B	11929.1	7942.0	4464.8	17	0.0	2.8	2.8	57.8
17WR_12	11926.1	7923.3	4464.9	17	0.0	2.1	2.1	108.0
17WR_12A	11925.9	7922.8	4464.7	17	0.0	2.1	2.1	6.4
17WR_12B	11925.8	7922.2	4464.6	17	0.0	2.5	2.5	34.9
17WR_13	11925.2	7913.9	4464.7	17	0.0	2.4	2.4	51.3
17WR_13A	11925.2	7913.5	4464.7	17	0.0	2.3	2.3	4.0
17WR_13B	11925.0	7914.4	4464.9	17	0.0	2.2	2.2	18.8
17WR_14	11924.3	7906.1	4464.9	17	0.0	2.1	2.1	5.7
17WR_14A	11924.3	7906.4	4464.9	17	0.0	2.4	2.4	36.9
17WR_15	11923.2	7898.7	4464.7	17	0.0	2.1	2.1	42.8
17WR_15A	11923.1	7898.1	4464.8	17	0.0	1.8	1.8	22.3
17WR_15B	11923.2	7898.9	4464.7	17	0.0	2.3	2.3	17.5
17WR_16	11923.6	7887.5	4465.2	17	0.0	2.7	2.7	12.4
17WR_16A	11923.6	7887.9	4465.2	17	0.0	2.8	2.8	4.7
17WR_18	11921.8	7869.7	4465.1	17	0.0	1.4	1.4	8.9
17WR_19	11921.4	7865.1	4464.9	17	0.0	2.7	2.7	3.2
17WR_20	11923.4	7861.5	4464.6	17	0.0	3.0	3.0	20.3
17WR_21	11922.0	7853.3	4464.3	17	0.0	3.3	3.3	1.4
17WR_22	11922.0	7848.3	4464.0	17	0.0	2.9	2.9	1.5
17WR_23	11922.2	7843.4	4464.2	17	0.0	3.1	3.1	1.8
17WR_24	11922.7	7838.8	4464.3	17	0.0	2.7	2.7	0.6
17WR_25	11924.4	7830.9	4464.7	17	0.0	3.2	3.2	0.9
17WR_26	11923.9	7826.3	4464.8	17	0.0	2.5	2.5	2.4
17WR_27	11923.7	7821.5	4464.6	17	0.0	2.3	2.3	0.3
17WR_28	11922.6	7817.3	4464.9	17	0.4	3.2	2.8	0.3
17WR_29	11919.9	7811.9	4464.5	17	0.0	2.5	2.5	0.2

Channel #	X (m)	Y (m)	Z (m)	Level	From (m)	To (m)	Interval (m)	Grade g/t Au
17WR_30	11916.9	7807.9	4465.0	17	0.0	1.8	1.8	1.2
17WR_31	11916.1	7804.1	4464.9	17	0.0	3.0	3.0	5.9
17WR_32	11915.9	7782.9	4464.6	17	0.0	2.6	2.6	6.0
17WR_33	11916.1	7776.9	4464.9	17	0.0	2.3	2.3	12.9
17WR_34	11917.3	7767.0	4465.3	17	0.0	2.3	2.3	12.2
17WR_35	11917.6	7762.5	4465.7	17	0.0	1.9	1.9	14.4
17WR_36	11918.7	7757.2	4465.9	17	0.0	1.8	1.8	34.2
17WR_37	11919.1	7752.2	4465.2	17	0.0	2.3	2.3	20.2
17WR_38	11918.7	7745.9	4465.5	17	0.0	2.1	2.1	7.9
17WR_39	11917.0	7741.3	4465.7	17	0.0	2.3	2.3	4.4
17WR_40	11916.5	7736.4	4465.7	17	0.0	3.0	3.0	7.1
17WR_41	11915.2	7732.2	4465.7	17	0.0	2.6	2.6	1.9
17WR_42	11915.6	7726.8	4465.6	17	0.0	2.4	2.4	5.4
17WR_43	11914.2	7721.9	4466.3	17	0.0	2.5	2.5	2.2
17WR_44	11914.2	7716.7	4465.4	17	0.0	2.5	2.5	1.3
17WR_45	11912.2	7712.2	4465.9	17	0.0	2.9	2.9	1.0
17WR_46	11912.0	7706.6	4466.1	17	0.0	2.4	2.4	5.1
17WR_47	11911.3	7701.8	4466.8	17	0.0	1.7	1.7	0.4
17WR_48	11910.6	7697.1	4466.2	17	0.0	2.1	2.1	1.4
17WR_50	11909.7	7687.1	4466.4	17	0.0	2.4	2.4	0.3
17WR_51	11907.2	7679.5	4466.4	17	0.0	2.9	2.9	0.6
17WR_52	11906.3	7674.6	4466.8	17	0.0	2.9	2.9	0.1
17WR_53	11902.5	7654.1	4465.9	17	0.0	2.2	2.2	2.7
17WR_54	11901.0	7644.7	4466.9	17	0.0	2.2	2.2	2.0
17WR_55	11898.4	7633.9	4466.4	17	0.0	2.4	2.4	3.0
17WR_56	11897.7	7628.1	4467.9	17	0.0	1.9	1.9	1.2
17WR_57	11896.1	7618.5	4466.6	17	0.0	2.3	2.3	2.5
*	(borehole_id)A: Samples assayed at Bogoso Laboratory except for 17WR_14A, 15A and 16A							
*	(borehole_id): Samples assayed at SGS Tarkwa							
*	(borehole_id)B: Resamples of high grade channels incorrectly sampled. Assayed at SGS Tarkwa							
*	Not true widths							

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 10.

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 293S 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.

Upon re-commencement of operations at PUG in 2017, development of the West Reef to the north of the previous exposures in the 293S reef drive, starting from the 270S cross cut. A footwall drive was developed, and the reef was cross cut every 23 m, as accesses to the alimak raises to be stoped as the new mining method in the west reef. Chip channel samples were taken from each of these accesses, chipping sample from within a 20cm channel of the access to acquire a 5kg chip sample per interval of between 0.5-1m. Several more were taken in the reef drives between each of these access and in the alimak raises themselves, up to 150m vertical above the 24 level, once established. In all, 142 channel samples taken on the 24 level and S1-S5 (8280N-2375N), raises initiated from the 24 level were included in the December 2017 Mineral Resource Estimate update.

Prior to 30 L being flooded, two hanging wall crosscuts were excavated for drill access to test the Main Reef between 30 L and 35 L. In the 270S 30 L crosscut the West Reef was sampled intersecting a horizontal width of 5.8 m grading 3.1 g/t gold and 5 m at a grade of 3.3 g/t gold 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.

Table 6A Channel Sample Results for 24 level and Alimak raises.

Channel #	X(m)	Y(m)	Z(m)	Level	From	To	Interval	Grade g/t Au
CH17-24-270-001	8395.1	11801.8	4167.035	6.9025	0.8	3.2	2.4	6.9025
CH17-24-270-002	8398.2	11802.5	4166.817	8.21	0	2	2	8.21
CH24-270S1-001	8373.593	11803.725	4166	3.4849	0.6	3.75	3.15	3.4849
CH24-270S1-002	8379.396	11803.2	4166	16.809	1.85	4.95	3.1	16.809
CH24-270S2-001	8355.223	11802.433	4167.173	4.9191	2.8	5.1	2.3	4.9191
CH24-270S2-002	8349.83	11802.066	4167.202	3.9333	1.4	2.6	1.2	3.9333
CH24-270S5-003	8284.827	11801.3	4167	15.4904	1	3.2	2.2	15.4904
CH24-270S5-004	8281.176	11801.15	4167	21.4666	1	2.9	1.9	21.4666
CP24-270S1-003	8375.7	11821.5	4195	12.3966	0	3.8	3.8	12.3966
CP24-270S1-004	8376	11841.35	4224.5	16.1969	0	2.9	2.9	16.1969
CP24-270S1-005	8378.9	11842.28	4224.5	9.0994	0	1.8	1.8	9.0994
CP24-270S1-006	8376.1	11865.75	4257.5	15.57	0	1.5	1.5	15.57
CP24-270S1-007	8378.785	11875.172	4272.75	17.4843	0	2.8	2.8	17.4843
CP24-270S1-008	8375.886	11874.545	4272.75	19.6014	0.6	3.4	2.8	19.6014
CP24-270S1-009	8378.2	11883.555	4285.5	10.5544	0	3.09	3.09	10.5544
CP24-270S1-010	8375.151	11887.7	4292.2	13.1708	0	2.4	2.4	13.1708
CP24-270S1-011	8377.9	11887.65	4292.2	26.832	0	2.5	2.5	26.832
CP24-270S1-012	8375.4	11882.455	4285.399	31.4958	1.5	3.99	2.49	31.4958
CP24-270S1-013	8374.9	11894.2	4302.556	7.4112	1	2.6	1.6	7.4112
CP24-270S1-014	8375.4	11811.05	4180.011	4.479	0.5	4.4	3.9	4.479
CP24-270S1-015	8375.7	11831.338	4209.367	5.849	0	3	3	5.849
CP24-270S1-016	8376.1	11849.5	4234.218	3.9	0	1	1	3.9
CP24-270S1-017	8376.1	11856.9	4244.072	0.615	0	2	2	0.615
CP24-270S11-001	8147.458	11799.358	4167.25	4.2214	1.04	3.06	2.02	4.2214
CP24-270S11-002	8159.122	11800.482	4167.25	16.19	0.9	1.9	1	16.19
CP24-270S12-001	8114.257	11798.291	4166.5	3.5197	1.2	3.24	2.04	3.5197
CP24-270S12-002	8130.757	11798.416	4167.25	4.8024	1.52	3.22	1.7	4.8024
CP24-270S12-004	8088.288	11793.34	4167.6	1.12	0.7	2.7	2	1.12
CP24-270S12-005	8070.273	11791.906	4167.7	0.4867	0	2.4	2.4	0.4867
CP24-270S12-006	8053.433	11788.337	4167.969	2.677	0	1.91	1.91	2.677
CP24-270S12-007	8048.872	11787.969	4168.013	0.6308	2.1	3.3	1.2	0.6308
CP24-270S2-004	8343.485	11802.011	4166.919	14.22	2.5	3.3	0.8	14.22
CP24-270S2-005	8365.586	11803.869	4166.924	12.475	1.69	3.65	1.96	12.475

Channel #	X(m)	Y(m)	Z(m)	Level	From	To	Interval	Grade g/t Au
CP24-270S2-006	8352.5	11809.403	4177.356	1.5933	0	0.9	0.9	1.5933
CP24-270S2-007	8349.6	11808.3	4177.451	2.8178	0.9	2.7	1.8	2.8178
CP24-270S2-008	8349.886	11811.999	4181.751	66.45	1	1.5	0.5	66.45
CP24-270S2-009	8353.212	11816.501	4189.379	24.1019	0	2.1	2.1	24.1019
CP24-270S2-010	8350.076	11815.977	4189.388	3.22	1.1	2.6	1.5	3.22
CP24-270S2-011	8350.111	11819.727	4195.565	39.2035	0.9	2.6	1.7	39.2035
CP24-270S2-012	8352.993	11819.489	4195.565	21.9827	1.2	2.98	1.78	21.9827
CP24-270S2-013	8350.209	11820.485	4198.386	13.9659	2.4	3.33	0.93	13.9659
CP24-270S2-014	8352.95	11821	4198.382	13.1211	1.5	3.3	1.8	13.1211
CP24-270S2-015	8350.335	11824.235	4202.927	14.1067	1.2	3	1.8	14.1067
CP24-270S2-016	8352.945	11824.054	4202.871	6.5248	1.2	3.3	2.1	6.5248
CP24-270S2-017	8350.368	11827.42	4207.357	16.3906	0	3.3	3.3	16.3906
CP24-270S2-018	8352.771	11827.425	4207.386	7.2609	0	3.2	3.2	7.2609
CP24-270S2-019	8350.624	11832.363	4215.842	30.7019	0.67	3.2	2.53	30.7019
CP24-270S2-020	8352.633	11832.678	4216.005	37.3111	0.46	2.66	2.2	37.3111
CP24-270S2-021	8350.299	11835.692	4220.314	12.116	0	2.86	2.86	12.116
CP24-270S2-022	8352.356	11835.632	4220.315	22.282	0	3	3	22.282
CP24-270S2-023	8350.1	11840.08	4228.354	30.0692	0.47	3.37	2.9	30.0692
CP24-270S2-024	8352.9	11840.519	4228.4	27.5843	0	2.8	2.8	27.5843
CP24-270S2-025	8350.177	11842.059	4230.914	31.5213	0	3.1	3.1	31.5213
CP24-270S2-026	8352.836	11841.819	4230.9	13.4414	0	3.5	3.5	13.4414
CP24-270S2-027	8350.28	11845.071	4235.6	58.3133	0	2.7	2.7	58.3133
CP24-270S2-028	8352.913	11845.262	4235.6	33.36	0	1.8	1.8	33.36
CP24-270S2-029	8350.5	11850.194	4242.1	17.0249	0	2.05	2.05	17.0249
CP24-270S2-030	8352.902	11850.073	4242.1	10.5663	0	2.05	2.05	10.5663
CP24-270S2-031	8350.277	11853.269	4246.7	15.1105	0	1.85	1.85	15.1105
CP24-270S2-032	8352.811	11853.368	4246.7	22.9349	0	1.75	1.75	22.9349
CP24-270S2-033	8350.4	11858.242	4254.2	21.8815	0	2	2	21.8815
CP24-270S2-034	8353.052	11858.247	4254.2	4.8278	0	1.85	1.85	4.8278
CP24-270S2-035	8350.55	11861.42	4258.74	14.6889	0	1.9	1.9	14.6889
CP24-270S2-036	8352.74	11861.7	4258.76	10.5448	0	1.32	1.32	10.5448
CP24-270S2-037	8350.75	11865.55	4263.39	17.638	0	1.2	1.2	17.638
CP24-270S2-038	8353.35	11865.7	4263.4	19.43	0	0.9	0.9	19.43
CP24-270S2-039	8351.39	11869.26	4269.08	42.4757	0	1.4	1.4	42.4757
CP24-270S2-040	8353.51	11869.115	4269.08	56.7942	0	1.37	1.37	56.7942
CP24-270S2-041	8351.098	11875.9	4278.9	14.5	0.8	1.8	1	14.5
CP24-270S2-042	8353.874	11876.55	4278.9	11.3438	0	1.3	1.3	11.3438
CP24-270S2-043	8351.15	11881.6	4288.2	12.8691	1.14	2.86	1.72	12.8691
CP24-270S2-044	8353.873	11882.02	4288.2	18.7573	0.4	2.56	2.16	18.7573
CP24-270S2-045	8351.116	11887.65	4297	16.9605	1	2.9	1.9	16.9605

Channel #	X(m)	Y(m)	Z(m)	Level	From	To	Interval	Grade g/t Au
CP24-270S2-046	8353.75	11887.95	4297	15.255	0.8	2.4	1.6	15.255
CP24-270S2-047	8351.12	11896.8	4312.4	9.9969	2.3	3.6	1.3	9.9969
CP24-270S2-048	8353.826	11897	4312.4	6.914	2	3.5	1.5	6.914
CP24-270S3-001	8327.239	11801.655	4167	2.95	1	1.49	0.49	2.95
CP24-270S3-002	8330.325	11801.825	4167	2.06	1	1.35	0.35	2.06
CP24-270S3-003	8334.5	11802.67	4167.8	47.61	0.9	1.36	0.46	47.61
CP24-270S3-005	8327.331	11804.97	4172.914	15.3794	0	1.36	1.36	15.3794
CP24-270S3-006	8330.495	11805.3	4172.914	5.88	0.5	0.9	0.4	5.88
CP24-270S3-007	8330.7	11808.102	4178.361	5.3754	1.3	1.95	0.65	5.3754
CP24-270S3-008	8327.753	11808.219	4178.361	16.57	1.6	1.9	0.3	16.57
CP24-270S3-009	8327.579	11817.917	4194.114	31.09	1.85	3.25	1.4	31.09
CP24-270S3-010	8330.588	11818.05	4194.114	6.26	1.4	2.8	1.4	6.26
CP24-270S3-011	8327.45	11828.25	4210.237	59.03	1.9	3.4	1.5	59.03
CP24-270S3-012	8330.572	11828.275	4210.259	42.5952	1.6	3.15	1.55	42.5952
CP24-270S3-013	8327.7	11833.15	4218.171	21.0601	1.6	3.1	1.5	21.0601
CP24-270S3-014	8331.1	11832.925	4218.171	31.6188	1.6	3.35	1.75	31.6188
CP24-270S3-015	8328.265	11839.944	4230.66	24.0019	0.7	3.3	2.6	24.0019
CP24-270S3-016	8330.596	11840.009	4230.66	32.8439	0.4	3.5	3.1	32.8439
CP24-270S3-017	8327.666	11847.84	4241.708	40.0298	0	1	1	40.0298
CP24-270S3-018	8330.754	11847.728	4241.793	8.7054	0	1.9	1.9	8.7054
CP24-270S3-019	8327.589	11854.231	4250.945	37.9	0	0.65	0.65	37.9
CP24-270S3-020	8329.985	11854.609	4250.87	24.6	0	0.7	0.7	24.6
CP24-270S3-021	8327.226	11860.008	4259.205	21.74	0	0.5	0.5	21.74
CP24-270S3-022	8330.399	11859.844	4259.073	32.77	0	0.8	0.8	32.77
CP24-270S3-023	8327.111	11863.902	4266.257	1.4725	1.1	2.3	1.2	1.4725
CP24-270S3-024	8330.297	11864.036	4266.329	8.246	0.8	2.3	1.5	8.246
CP24-270S3-025	8326.894	11870.238	4276.005	16.0605	1.4	3.5	2.1	16.0605
CP24-270S3-026	8330.268	11870.702	4275.911	31.5963	1	2.7	1.7	31.5963
CP24-270S3-027	8326.734	11876.5	4285.949	20.145	2.3	3.3	1	20.145
CP24-270S3-028	8329.515	11877.588	4286.039	16.6412	1	2.7	1.7	16.6412
CP24-270S3-029	8326.482	11884.2	4297	8.3158	0.9	2.8	1.9	8.3158
CP24-270S3-030	8329.73	11884.454	4297	3.2412	0	3.4	3.4	3.2412
CP24-270S3-031	8326.882	11890.1	4305.55	5.8238	0	3.2	3.2	5.8238
CP24-270S3-032	8329.599	11889.8	4305.578	5.92	1.2	3	1.8	5.92
CP24-270S3-033	8339.5	11890.35	4303.5	15.335	1.4	2.3	0.9	15.335
CP24-270S4-001	8304.06	11799.197	4167.387	48.255	1	3	2	48.255
CP24-270S4-002	8307.93	11799.863	4167.154	24.99	1	3	2	24.99
CP24-270S4-003	8304.586	11807.764	4177.874	14.996	0.94	2.88	1.94	14.996
CP24-270S4-004	8307.671	11808.404	4177.876	29.0303	0	2.54	2.54	29.0303
CP24-270S4-005	8305.055	11809.44	4179.694	8.8314	0	2.8	2.8	8.8314

Channel #	X(m)	Y(m)	Z(m)	Level	From	To	Interval	Grade g/t Au
CP24-270S4-006	8307.692	11809.457	4179.694	37.1164	0.5	2.15	1.65	37.1164
CP24-270S4-007	8305.502	11815.829	4190.268	6.0394	1.5	3.1	1.6	6.0394
CP24-270S4-008	8308.181	11815.966	4190.268	6.488	1	3.04	2.04	6.488
CP24-270S4-009	8305.725	11822.257	4200.546	11.7963	1.3	3.05	1.75	11.7963
CP24-270S4-010	8308.527	11821.9	4200.546	27.512	2	3	1	27.512
CP24-270S4-011	8306.05	11828.683	4211.253	8.2832	1	3.03	2.03	8.2832
CP24-270S4-012	8308.864	11828.514	4211.25	5.5571	1.1	3.2	2.1	5.5571
CP24-270S4-015	8306.35	11843.499	4238.018	7.9775	1.2	3.2	2	7.9775
CP24-270S4-016	8309.289	11844.019	4238.018	10.84	0.85	1.95	1.1	10.84
CP24-270S4-017	8305.981	11851.135	4249.028	9.8467	0	2.1	2.1	9.8467
CP24-270S4-018	8308.938	11851.14	4249.028	12.6447	0	1.9	1.9	12.6447
CP24-270S4-019	8304.1	11859.4	4263.702	2.2996	0	1.2	1.2	2.2996
CP24-270S4-020	8306.9	11859.35	4263.702	21.898	0	1.3	1.3	21.898
CP24-270S4-021	8303.75	11866.825	4275.117	1.2173	0	3.2	3.2	1.2173
CP24-270S4-022	8306.5	11865.575	4275.117	3.6069	2.3	3.15	0.85	3.6069
CP24-270S5-001	8281.176	11801.35	4167	23.1133	1	2.5	1.5	23.1133
CP24-270S5-002	8284.821	11801.3	4167	25.3264	1	3.2	2.2	25.3264
CP24-270S5-005	8249.14	11801.39	4167.75	8.4814	0.9	2.3	1.4	8.4814
CP24-270S5-005a	8284.424	11810.924	4180.897	8.9118	0	2.2	2.2	8.9118
CP24-270S5-006	8281.552	11810.294	4180.897	29.9057	1	2.4	1.4	29.9057
CP24-270S5-007	8284.398	11813.531	4187.223	19.6816	1.7	3.6	1.9	19.6816
CP24-270S5-008	8281.438	11813.758	4187.223	3.625	1.5	3.5	2	3.625
CP24-270S7-001	8234.963	11801.89	4167.75	12.5489	1	2.9	1.9	12.5489
CP24-270S8-001	8212.381	11801.073	4167.75	16.31	0	3.4	3.4	16.31
CP24-270S8-002	8215.356	11800.925	4167.75	22.648	0	4	4	22.648
CP24-270S9-001	8189.542	11800.553	4167.249	30.4309	1	3.3	2.3	30.4309
CP24-270S9-002	8192.706	11800.224	4167.251	20.5838	2	3.6	1.6	20.5838
CP24-281-001	8120.067	11799.108	4167	12.1047	0.37	1.34	0.97	12.1047
CP24-281-002	8124.495	11799.235	4167	21.0099	0.57	1.59	1.02	21.0099

10 Drilling

10.1 Open Pit

10.1.1 Drilling

Drilling is carried out by a combination of DD, RC and RAB techniques at the GSBPL operations. In general, the RAB method is used at early stages as a follow up to soil geochemical sampling and during production for testing contacts and deposit extensions around the production areas and has a maximum drilling depth of around 30 m. RC drilling is used as the main method for obtaining suitable samples for Mineral Resource estimation and is carried out along drill lines spaced between 25 and 50 m apart along prospective structures and anomalies defined from soil geochemistry and RAB drilling results. RC drilling is typically extended to depths of in the order of 150 m. The DD method is used to provide more detailed geological data and in those areas where more structural and geotechnical information is required. Generally, the deeper intersections are also drilled using DD and, as a result, most section lines contain a combination of RC and DD drilling.

With over 5,000 holes drilled and over 400,000 m of drilling conducted on the various deposits it is not considered appropriate to discuss all relevant drilling results. Suffice to say, the continued production and grade control drilling is providing appropriate reconciliation with the original drilling. The interpretation of the relevant results is directly related to the wireframe modelling used for the purpose of defining the volume of material used for the Mineral Resource volume.

All drill hole data is verified by GSBPL staff and independently by their consultants and there are no recovery or survey factors which are considered to materially impact the accuracy and reliability of the results.

Exploration data for the GSBPL projects including Mampon are based on a general strike orientated at N32˚E with magnetic declination of easterly deviation of 7˚ culminating in the Bogoso Mine Grid of N39˚E. In the case of the Pampe deposit, the established Mine Grid used for the data is N47˚E. All drill hole collars and local topographic features are reported in the respective mine grid location coordinates and are done by Golden Star surveying team employees. Topographic surface surveying over the various areas are also completed by staff surveyors.

Drill collar azimuth and dip for both RC and DD holes are established prior to drilling by a surveyor using a total station whereas RAB holes are done using a hand-held magnetic compass. All drill holes locations including RAB holes are surveyed using a total station once the hole is completed and the drill rig is moved away from the drill pad. Downhole surveys of RC and DD boreholes are conducted using Reflex survey instruments or EZ-Shot instruments. Downhole measurements are conducted at intervals of 30 m for both RC and DD drill holes, RAB holes on the other hand are too short with maximum depth of 30 m to conduct downhole surveys.

Initial drilling phases are generally conducted to investigate soil anomalies and are mostly drilled on 800 m line spacing and later in-filled at 400 to 200 m spacing depending on results. The first pass drilling programs are generally RAB type drilling which is initially drilled and oriented perpendicular to the interpreted strike of the anomaly. RAB drilling is generally followed by RC drilling along defined sections and drilled at fence spacing of 50 m and later in-filled at 25 m spacing pending positive assay results. The drilling strategy aims at sampling the gold bearing structures orthogonal to the interpreted strike on sections regularly spaced at 25 m.

A summary of the exploration data used in the Mineral Resource models is given in Table 5.

Table 5 Summary exploration data used for the Mineral Resource

Location	Type	Number of Holes	Meterage (m)
Pampe	RC	92	8,971
	DD	74	15,019
	Geotech	-	-
	GC (RC)	409	12,020
Mampon	RC	17	1,659
	DD	30	5,255
	Geotech	4	540
	GC (RC)	-	-
Bogoso North (Marlu)	RC	225	12,230
	DD	140	28,212
	Geotech	-	-
	GC (RC)	66	3280
Chujah/Dumasi	RC	618	40,247
	DD	315	56,546
	Geotech	6	1,110
	GC (RC)	1,076	29,267
Ablifa	RC	153	10,568
	DD	30	5,642
	Geotech	-	-
	GC (RC)	36	1889
Buesichem	RC	207	19,046
	DD	295	63,742
	Geotech	4	776
	GC (RC)	601	15,415
Beta Boundary South	RC	226	24,025
	DD	46	11,843
	Geotech	4	706
	GC (RC)	-	-
Bondaye	RC	196	18,879
	DD	15	1,948
	Geotech	-	-
	GC (RC)	-	-
Tuapim	RC	187	18,116
	DD	10	15,20
	Geotech	-	-
	GC (RC)	-	-

10.1.2 Sampling

All exploration data from the GSBPL leases and concessions are collected by GSR personnel, with the exception of auger sampling programs where contractors are occasionally hired. Contractors are supplied with comprehensive field procedures designed to ensure reliability of the exploration data and minimise voluntary and inadvertent contamination and supervised by GSR employees.

Shallow auger samples are collected below the organic layer and overburden material from half a meter below the surface to one meter. The augers are manually operated metal tools which dig holes of approximately 10 cm in diameter. Deep auger holes generally go to a depth of approximately 3 to 4 m and are normally conducted to follow anomalous results from the shallow auger programs.

For all drilling programs, GSR follows a standardised approach to drilling and sampling on all its Ghanaian projects. Sampling is typically carried out along the entire drilled length. For RC drilling, samples are collected every 1 m. Where DD holes have been pre-collared using RC, the individual 1 m RC samples are combined to produce 3 m composites which are then sent for analysis. Should any 3 m composite sample return a significant gold grade assay, the individual 1 m samples are then sent separately along with those from the immediately adjacent samples.

DD samples are collected, logged and split with a diamond rock saw in maximum 1 m lengths. Detailed logging of the core is done by an appropriate qualified geologist at the core shed where the entire length of the hole is laid on core tables in sequential order. The geologist takes elaborate information on descriptions of colour, lithology, alteration, weathering, structure and mineralisation. The core is cut according to mineralisation, alteration or lithology. The core is split into two equal parts along a median to the foliation plane using a core cutter. The sampling concept is to ensure a representative sample of the core is assayed. The remaining half core is retained in the core tray, for reference and additional sampling if required.

RC sampling protocols were established in 2003. The composite length of 3 m has been specified to allow a minimum of at least two composites per drill hole intersection based on experience from exploration drilling and mining. The hangingwall and footwall intersections can generally be easily recognised in core from changes in pyrite content and style of quartz mineralisation.

- The 3 m composite sampling methodology is as follows:
- A sample of each drilled meter is collected by fitting a plastic bag on the lower rim of the cyclone to prevent leakage of material;
- The bag is removed once the “blow-back” for the meter has been completed and prior to the commencement of drilling the subsequent meter;
- Both the large plastic sample bags and the smaller bags are clearly and accurately labelled with indelible ink marker prior to the commencement of drilling. This is to limit error and confusion of drilling depth while drilling is proceeding;
- 3 m composite samples are taken by shaking each of the 1 m samples (approximately 20 kg) and taking equal portions of the 3 consecutive samples into a single plastic bag to form one composite sample (approximately 3 kg);
- Composite samples are taken using tube sampling, which uses a 50 mm diameter PVC tube cut at a low oblique angle at one end to produce a spear of approximately 600 mm length;

- The technique assumes that a sample from the cyclone is stratified in reverse order to the drilled interval. A representative section through the entire length of the collected sample is considered to be representative of the entire drilled interval;
- The PVC tube is shuffled from the top to bottom of the sample, collecting material on the way, to ensure sample accumulated in the tube does not just push the remaining sample away; and
- The material in the tube is emptied into the appropriately labelled sample bag and, in the case of 3 m composite samples, stored separately from the 1 m samples.

The 1 m sample collection methodology is as follows:

- The 1 m re-sampling of selected mineralised composite zones using the 20 kg field samples is undertaken with a single stage riffle splitter;
- The splitter is clean, dry, free of rust, and damage is used to reduce the 20 kg sample weight to a 3 kg fraction for analysis;
- Care is taken to ensure that the sample is not split when it is transferred to the splitter, and is evenly spread across the riffles;
- When considered necessary, the sample is assisted through the splitter by tapping the sides with a rubber mallet;
- Excessively damp or wet samples are not put through the splitter, but tube-sampled or grab-sampled in an appropriate manner. Alternatively, the sample is dried before splitting. A common sense approach to wet sampling is adopted on a case by case basis;
- Similarly, clods of samples are not forced through the splitter, but apportioned manually in a representative manner; and
- The splitter is thoroughly cleaned between each sample using a brush. Where possible, the splitter is cleaned using an air gun attached to the drill rig compressor.

RAB samples are collected and bagged at 1 m intervals. As the samples are generally smaller in size than the RC samples, 3 m composites are prepared by shaking the samples thoroughly to homogenise the sample, before using the PVC tube to collect a portion of the three individual 1 m samples. After positive results from the 3 m composites, the individual 1 m samples are split to approximately 2 to 3 kg using the Jones riffle splitter and then submitted to the laboratory for analysis.

Logging data for all drill hole types is recorded directly into acQuire software using GETAC notebook computers for all logging. The logs are validated before they are integrated into the online live acQuire database. In the case of RC sampling where field conditions will not permit the use of the notebooks, the log data is recorded on paper and transferred to data entry staff for input into the database. All exploration drillhole data are ultimately transferred into Gemcom database by database staff and validated at the Bogoso site exploration office. The master exploration acQuire and Gemcom databases are stored separately and access is controlled and limited to maintain database integrity. All exploration data are first validated before being accepted into the master database.

10.2 Prestea Underground

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 boreholes drilled from a single collar location. Underground collar locations are spaced approximately 80 m apart along strike on the 17, 24, and 30 Ls.

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

- DD surface = 274 boreholes; 29,700 m
- RC surface = 137 boreholes; 14,000 m
- DD underground = 315 boreholes; 53,500 m

Drilling on the West Reef deposit was conducted from underground drill stations, predominantly from 17 and 24 Ls. The drilling was conducted by GSR and no historical data was used in the mineral 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 17). The up dip portion of the WR remains to be tested between 12 and 17 Ls and remains one of the priority drill targets. The 17 L drill stations are located on the following crosscuts, 274S, 277S, 280S, 285S, 287S, 290S, 293S, 297S, 302S, and 308S, testing the strike over approximately 775 m.

On 24 L, which is approximately 300 m below 17 L, drilling was conducted from four drill chambers, 266S, 274S, 284S, and 287S (Figure 19). The drilling from the four drill stations enabled the West Reef to be tested approximately 775m along its strike length as well as up and down dip.

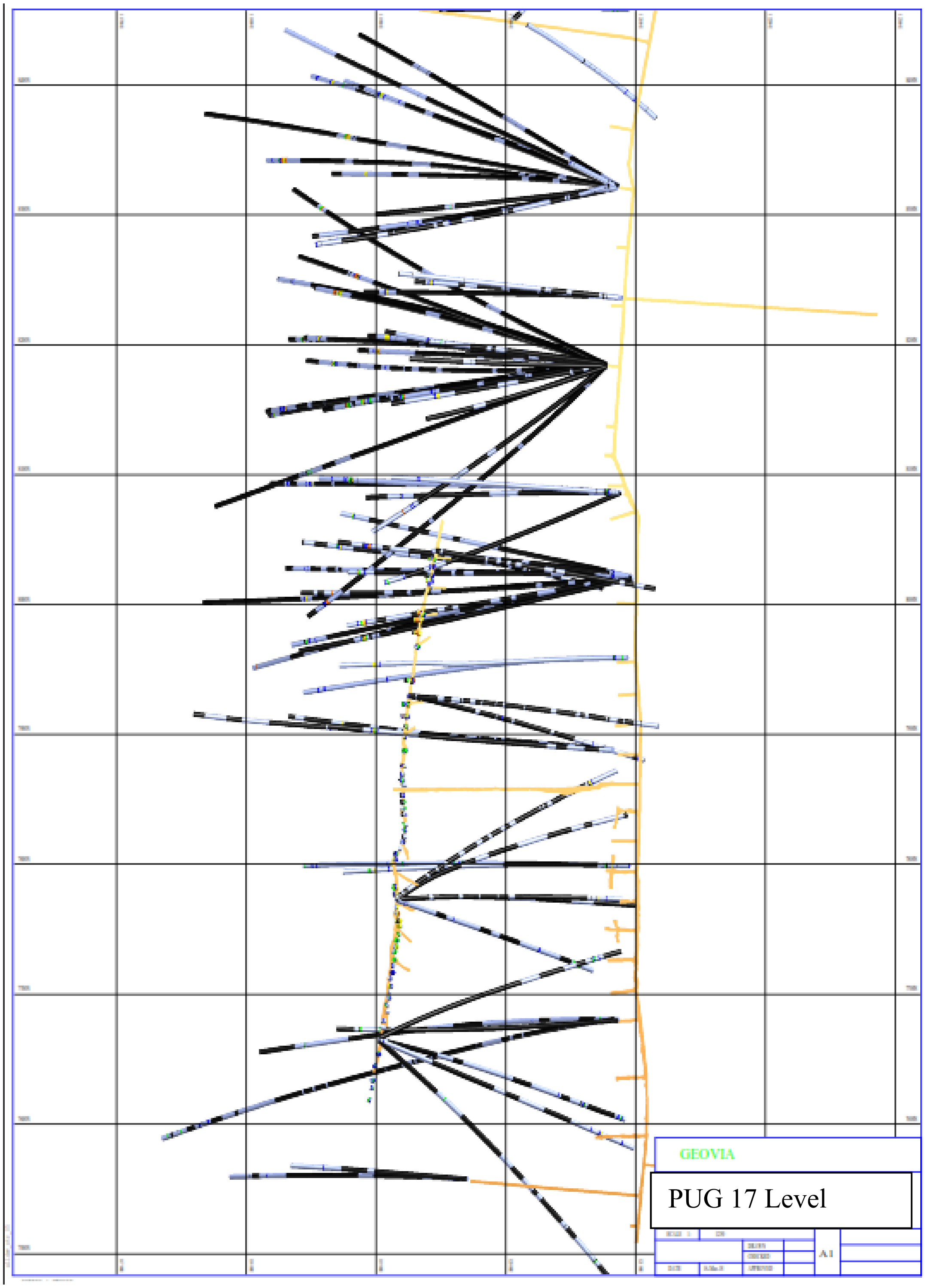


Figure 17 17 Level drilling

Showing Reef Intersections Including Channel Sampling Locations

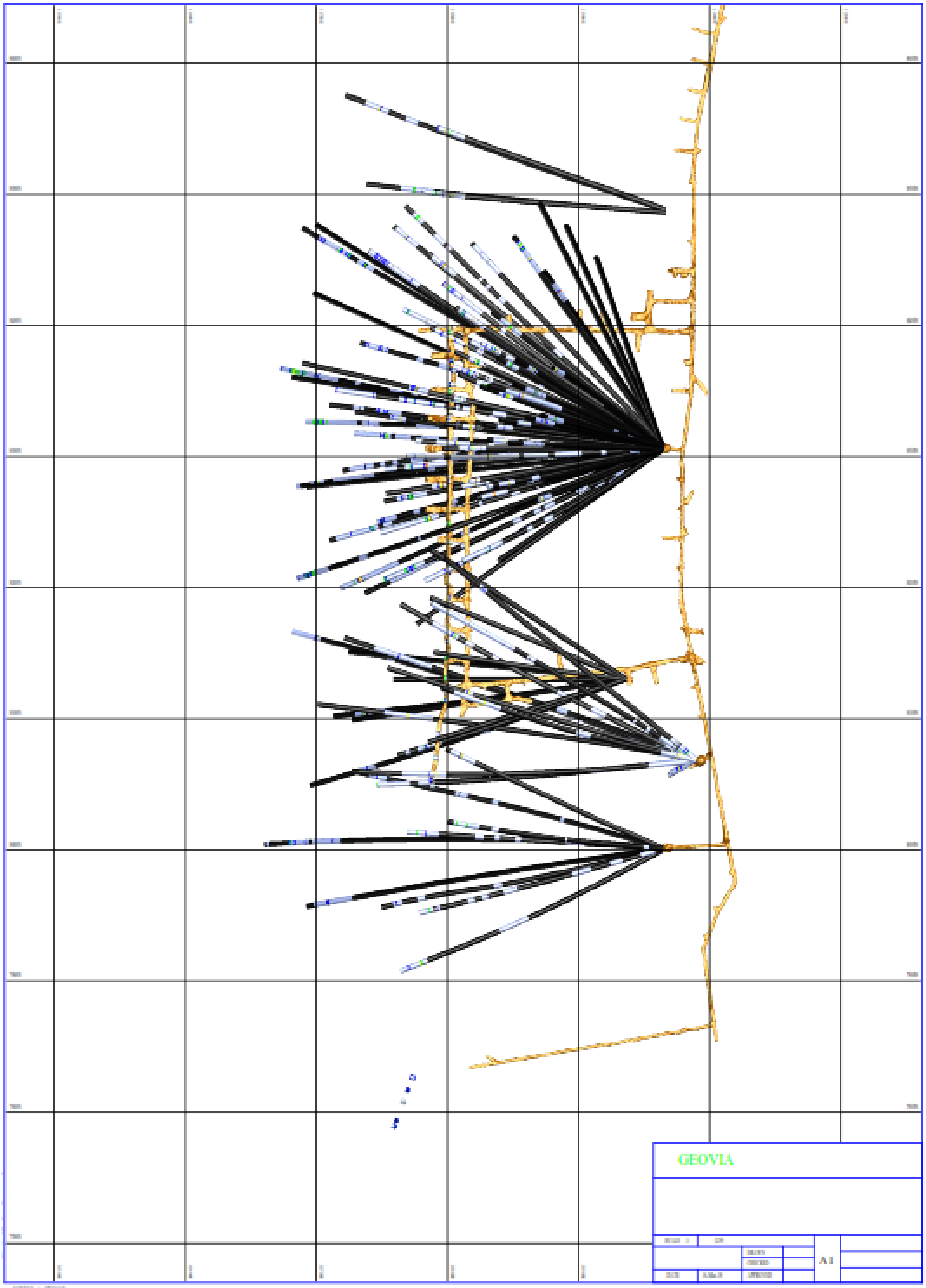


Figure 18 24 Level drilling

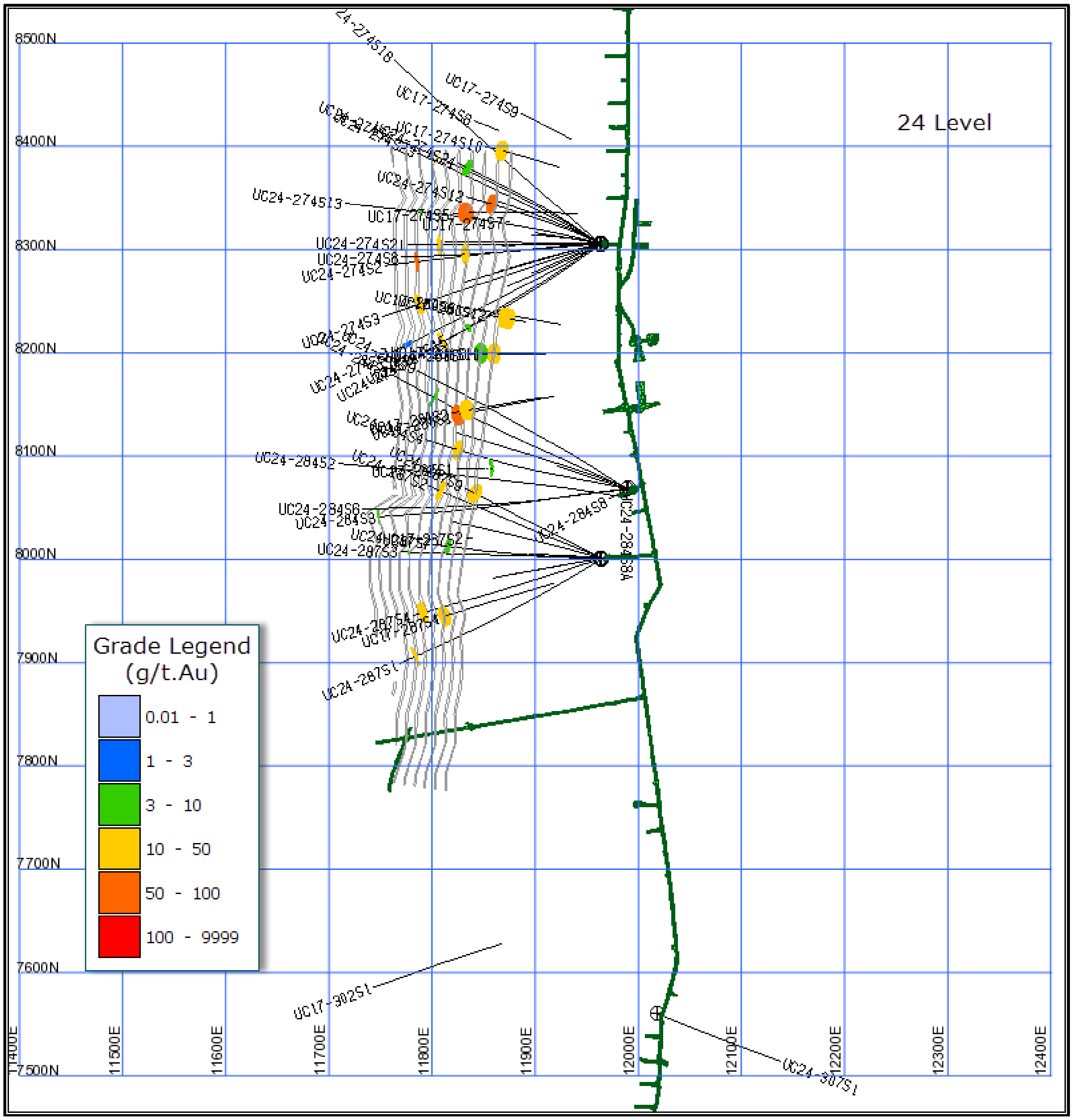


Figure 19 24 Level Drill Plan and West Reef Contours

Showing Reef Intersections

The underground drilling of the West Reef target was conducted in several campaigns from 2004 to 2017 with a total of 151 boreholes and 36,500 m being completed during this time. All drilling was conducted with underground diamond drill core rigs using a mixture of NQ2 (~ 50 mm) and HQ (~64mm), sized core. All borehole collars were surveyed using the underground survey control brought down from surface using the mine grid. The boreholes were also surveyed nominally every 25 to 30 m down-hole using a Reflex single shot survey instrument. The most recent 2017 drilling was surveyed at 6m intervals with a reflex multishot survey instrument.

Core recovery through the mineralized zone was optimized by using chrome core barrels, viscous muds, and short drilling runs and split core tubes but in some boreholes 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 mineral resources have been classified as Indicated are on a nominal 25x25-metre grid whereas Inferred mineral resources exceed the 25-metre borehole spacing (Figure 20).

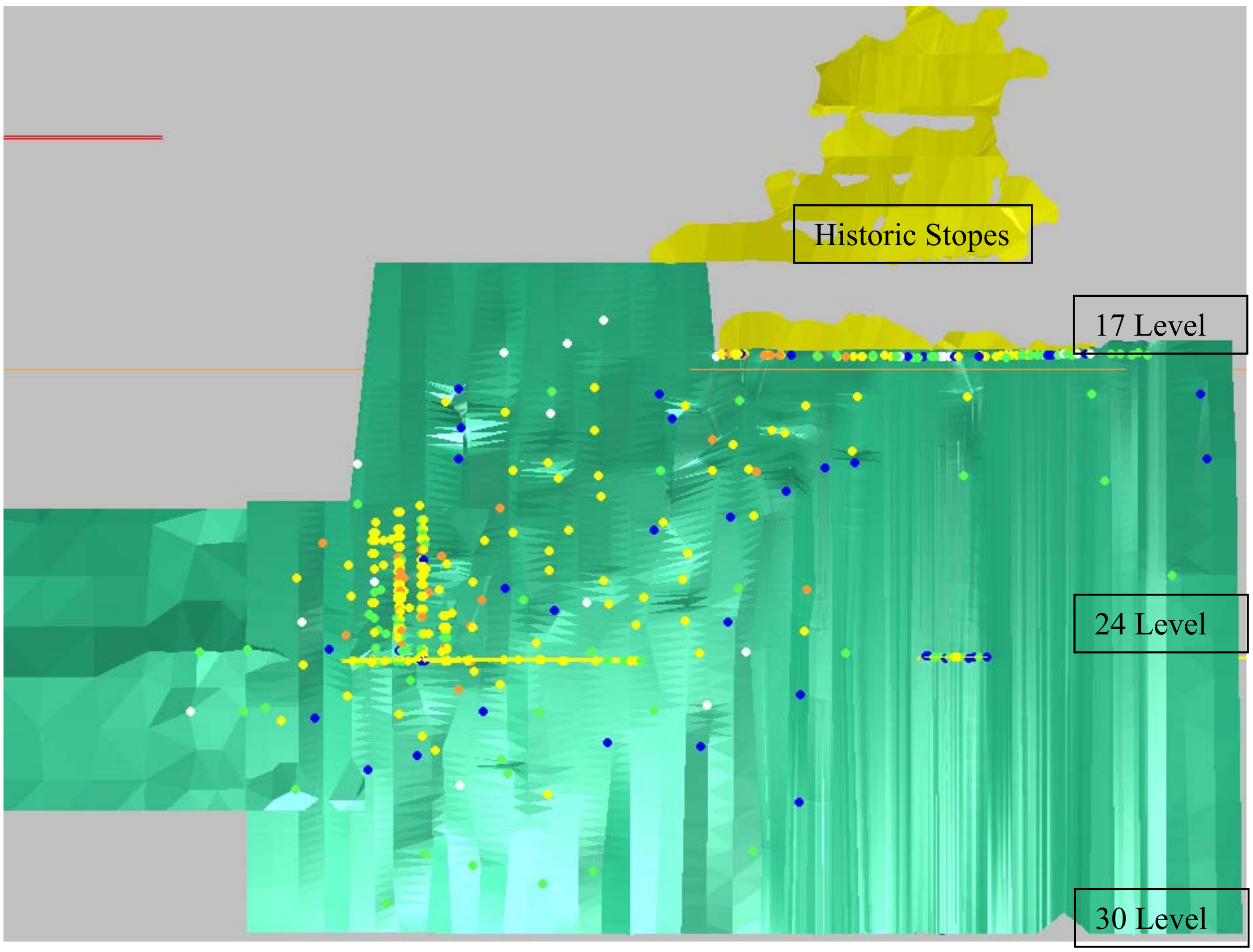


Figure 20 West Reef Long-Section Looking West from Footwall

Several representative drill sections have been included below (Figure 21 and Figure 22) showing the attitude of the West Reef and the relatively consistent dip and gold tenures.

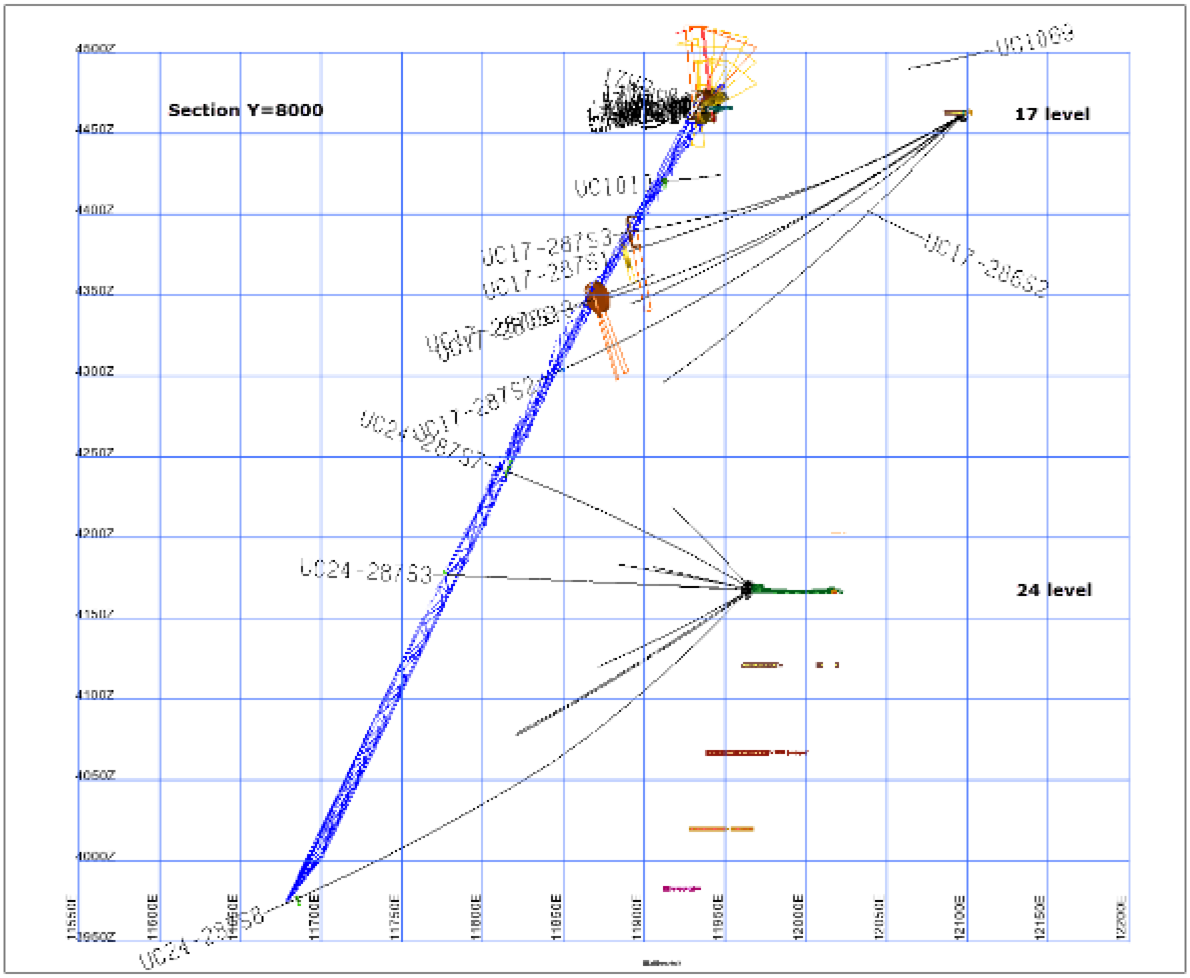


Figure 21 West Reef drill cross-Section 8000N showing ore and drill holes

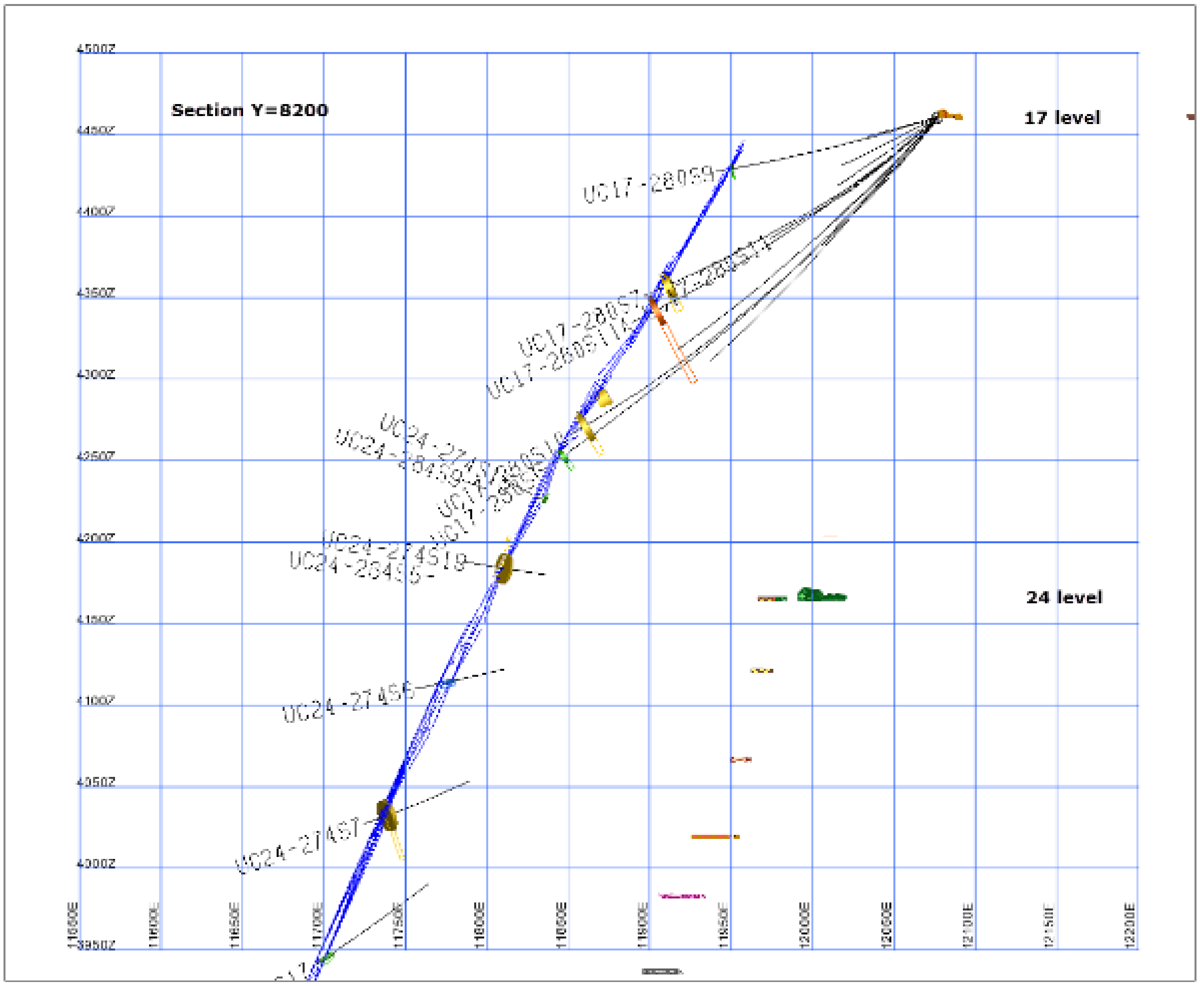


Figure 22 West Reef drill Cross-Section 8200N showing ore and drill holes

11 Sample Preparation, Analyses and Security

Sampling from reverse circulation (RC) drilling is carried out using a standard single cyclone with samples collected at 1-metre intervals through the expected mineralized zone. In zones of waste rock, the sample interval is occasionally increased to a 3-metre composite. However, all samples are assayed and if a 3-metre sample returns a significant grade value the original 1-metre 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.

Core from surface diamond drilling (DD) 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-metre 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-metre nominal intervals where alteration, sulphidation, 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, it 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 drill core samples have a recovery greater than 95% with 92% showing a recovery greater than 80%.

Samples used for the West Reef mineral resource estimations were of two types, rock sawn channel samples on 17 L and 24 L reef drives and NQ size 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 the 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 were any discrepancies, they were 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 was logged by the geologist who paid particular attention to structure, lithology, alteration, and mineralization. All of the core was orientated with a spear orientation device and this was used to take structural measurements while the core was being logged.

Sampling intervals were laid out by the geologist logging the core and were based on geological contacts with samples in mineralized zones generally not exceeding 1 m. 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 was 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 were inserted in the sample numbering sequence and these were recorded on the lab dispatch sheets. Every 20 samples that were submitted to the laboratory were accompanied by a sample standard and a blank to check the precision of the analysis. Additional checks were done on samples once the results were returned.

Samples were 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 was provided by the lab. Sample turnaround and dispatch were 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 were returned to the site were 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 by the author to be within or exceed industry standards.

12 Data Verification

12.1 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 boreholes. 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 QA/QC work carried out on the Prestea West Reef underground exploration and Prestea Main Reef (Plant North) samples obtained during 2004 – 2017.

12.2 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 boreholes targeting the underground Main Reef and West Reef deposits. The results are summarized in the Figure 23 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 gold, 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 that more appropriately represent the orebody.

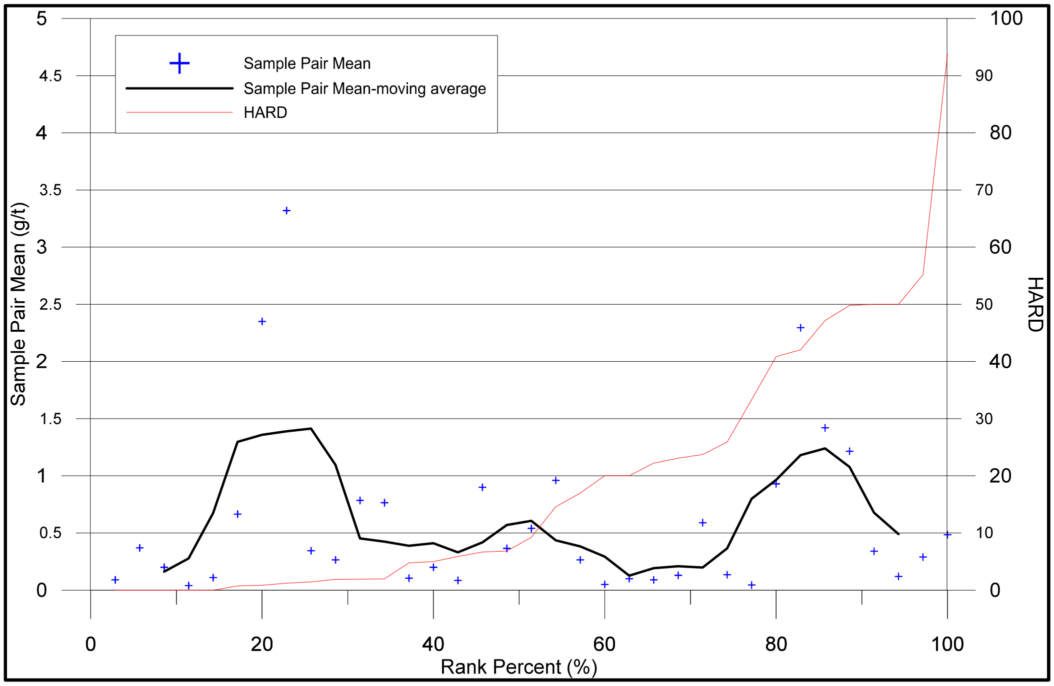


Figure 23 Main Reef Exploration Replicates HARD Plot

12.3 Duplicates

Duplicate samples are composed of coarse reject sample material that 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 gold. The duplicate samples are re-bagged 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.

12.3.1 West Reef Drilling 2006

This work has shown that reproducibility is relatively poor for the underground sampling data from Prestea as shown in Figure 24 and Figure 25. 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 gold. Given that the current block cut-off grade for the mineral resource is 5.5 g/t gold, 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 where low grade material is included within the mineralization. However, the majority of the higher grade sample pairs show relatively good reproducibility.

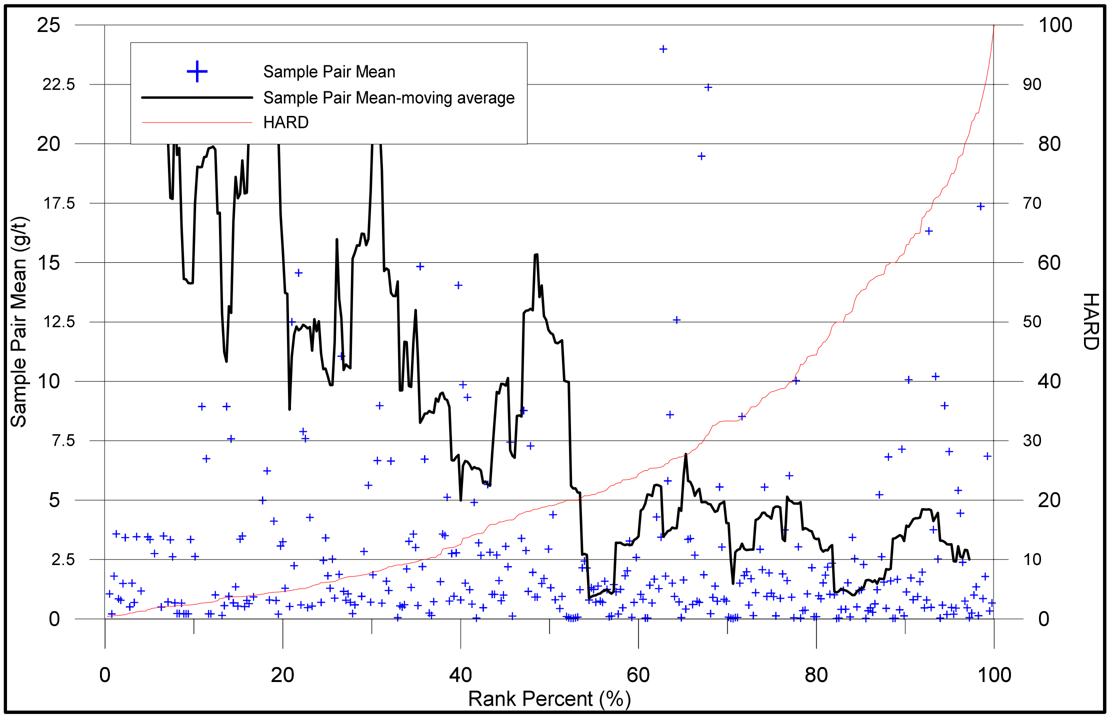


Figure 24 West Reef Underground DD Duplicates HARD Plot

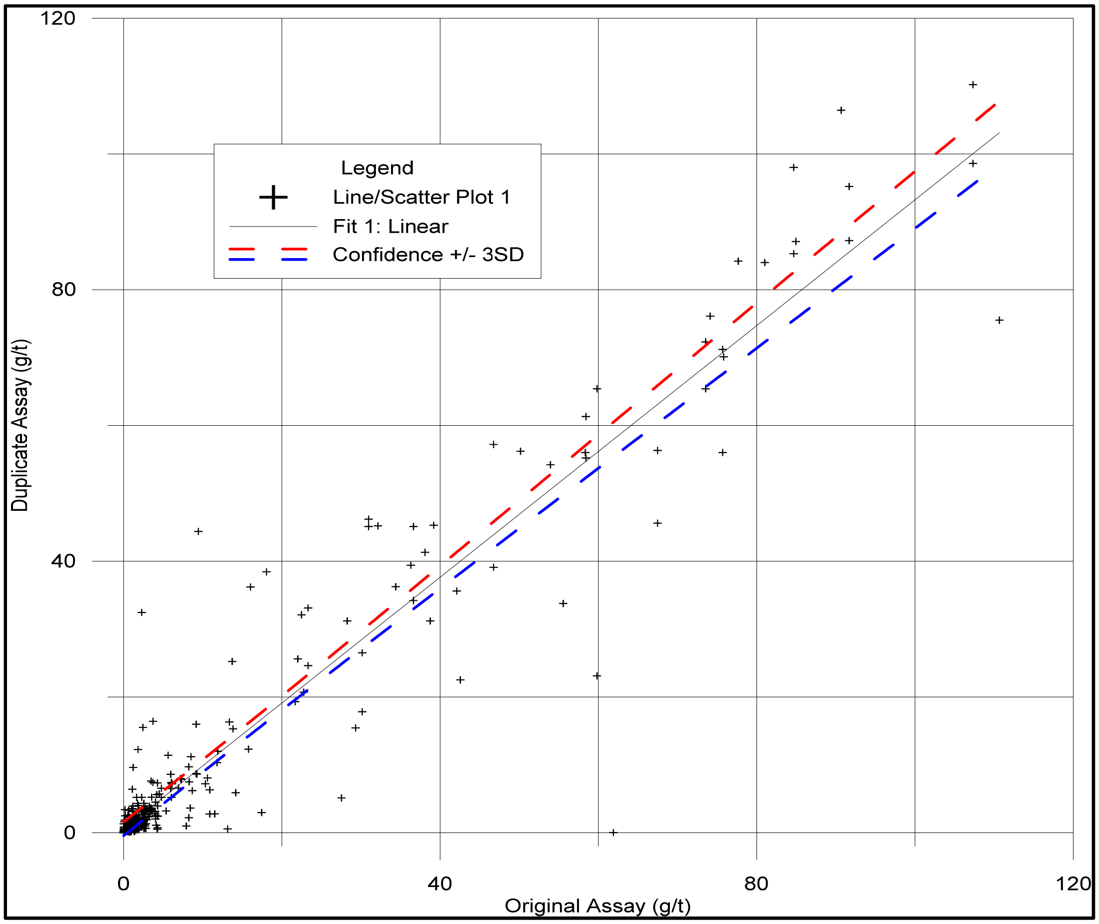


Figure 25 West Reef Underground Exploration Duplicates Correlation Plot

12.3.2 West Reef Underground Drilling 2017

The pulp duplicate samples are re-bagged 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 analysis confirmed the high variability associated with the West Reef deposit with 74% of sample pairs exhibiting a Half Absolute Relative Difference (HARD) value of 20% or less.

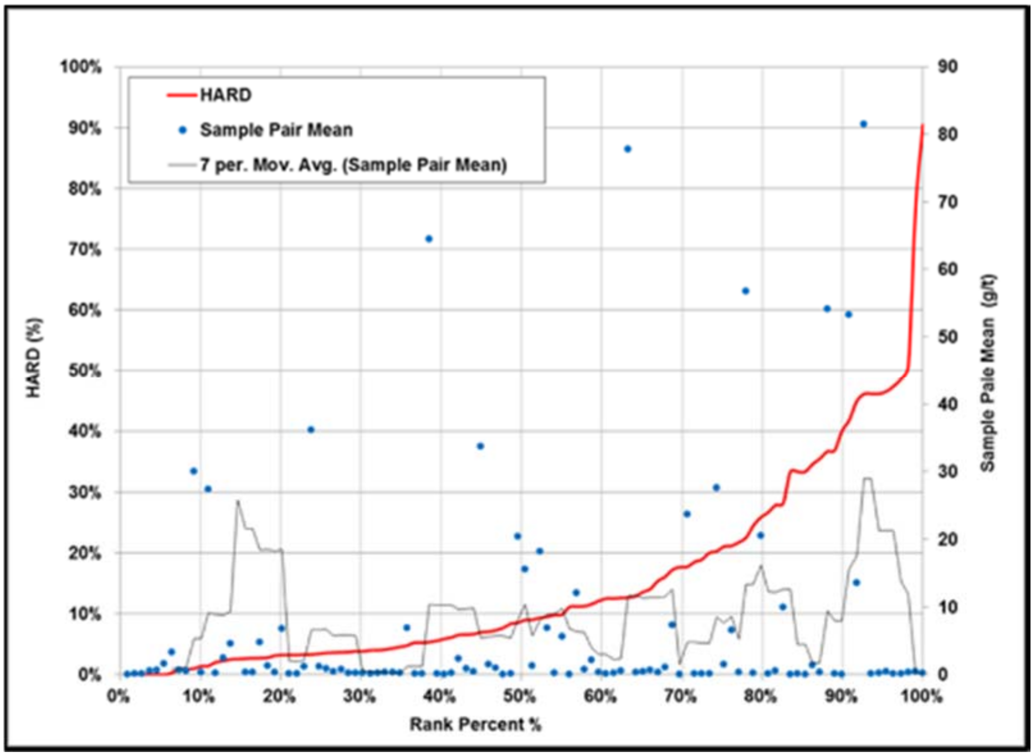


Figure 17 West Reef Underground 2017 Pulp Duplicates HARD Plot

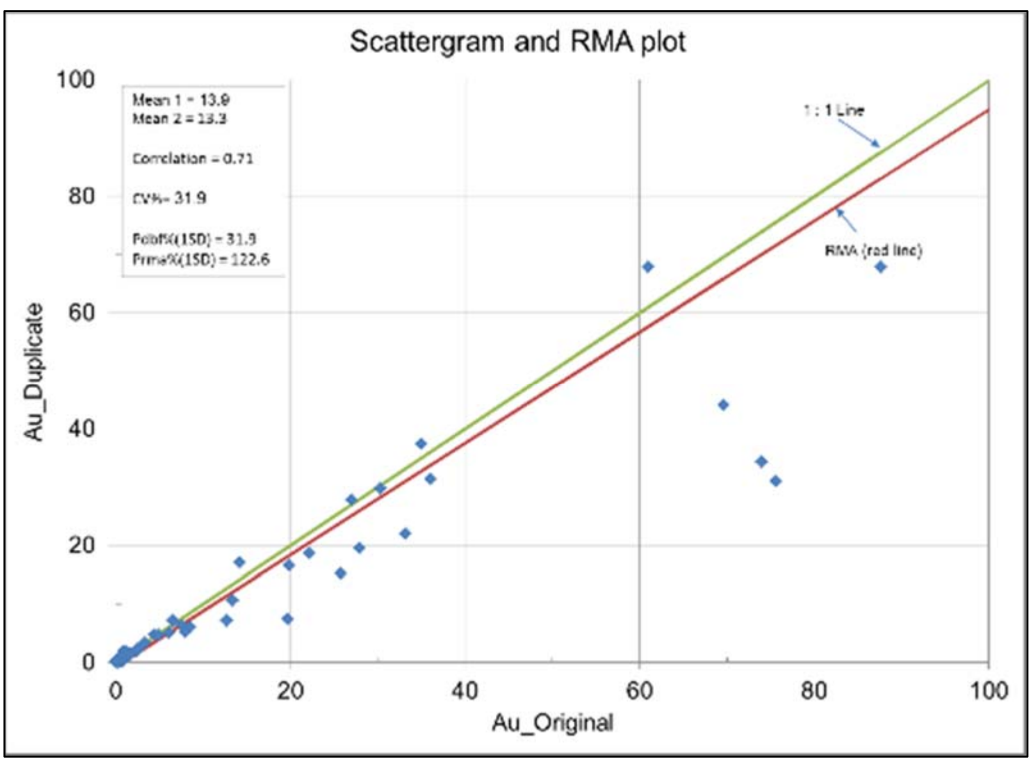


Figure 18 West Reef Underground 2017 Pulp Duplicates Correlation Plot

12.3.3 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% (Figure 26 and Figure 27). 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 mineralized intersections are used for these studies in future.

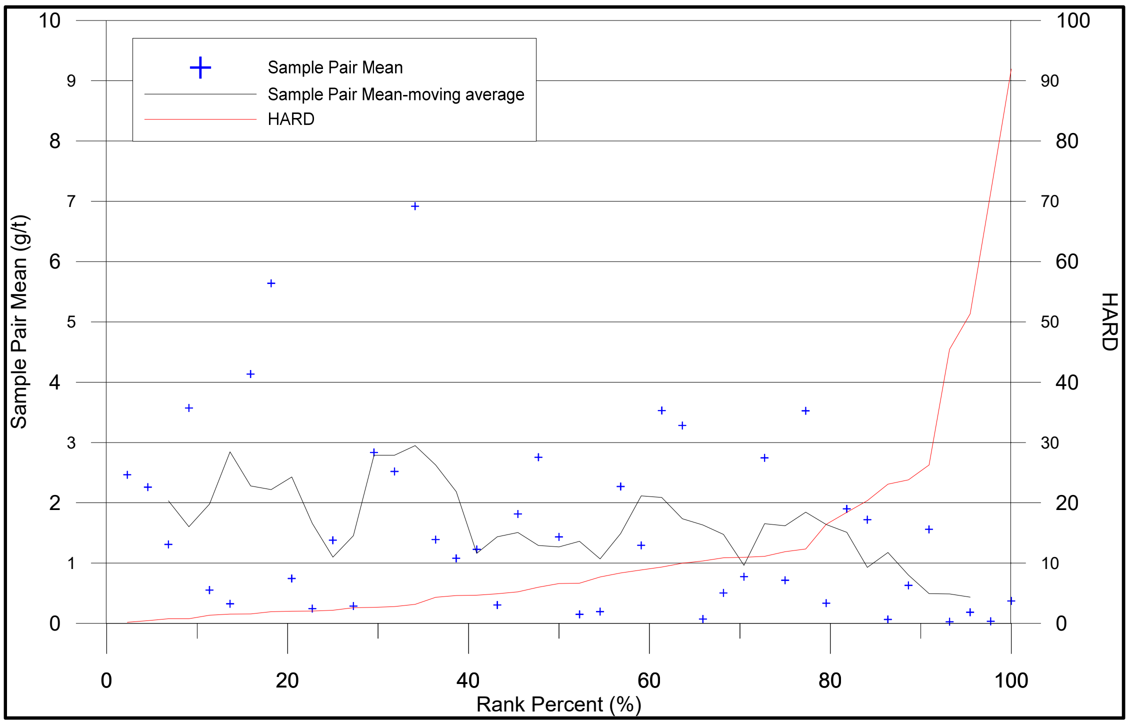


Figure 26 Main Reef RC Duplicate HARD Plot

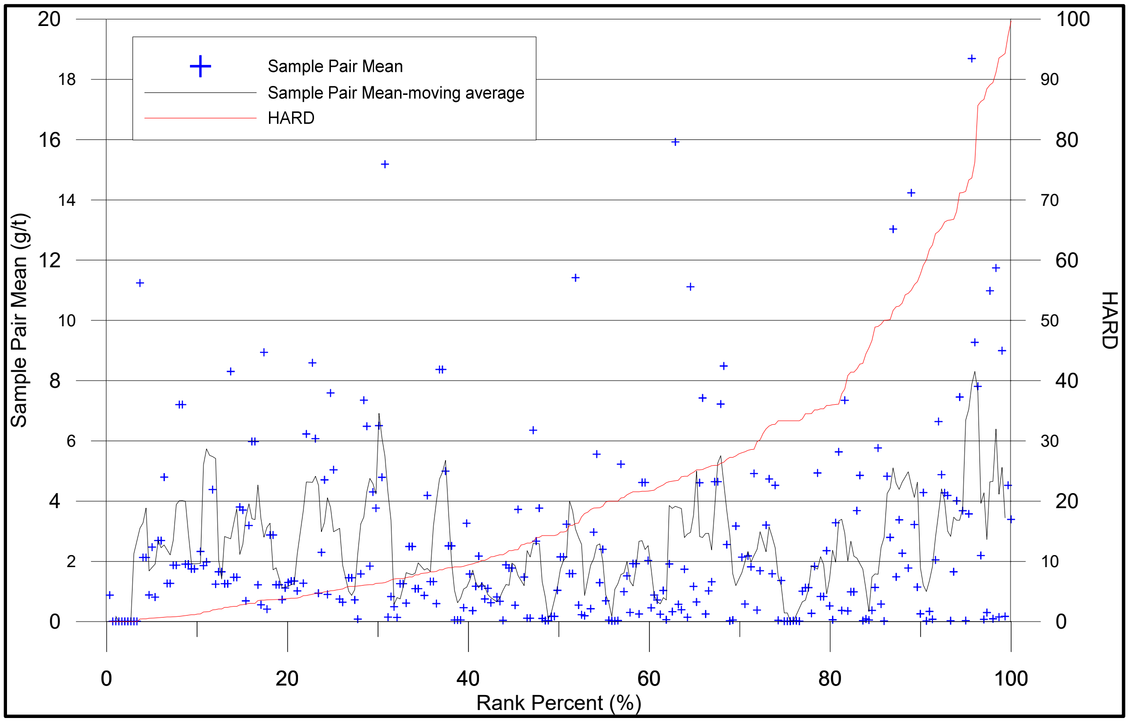


Figure 27 Main Reef DD Duplicate HARD Plot

12.4 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 parts per million (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 (Figure 28) 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.

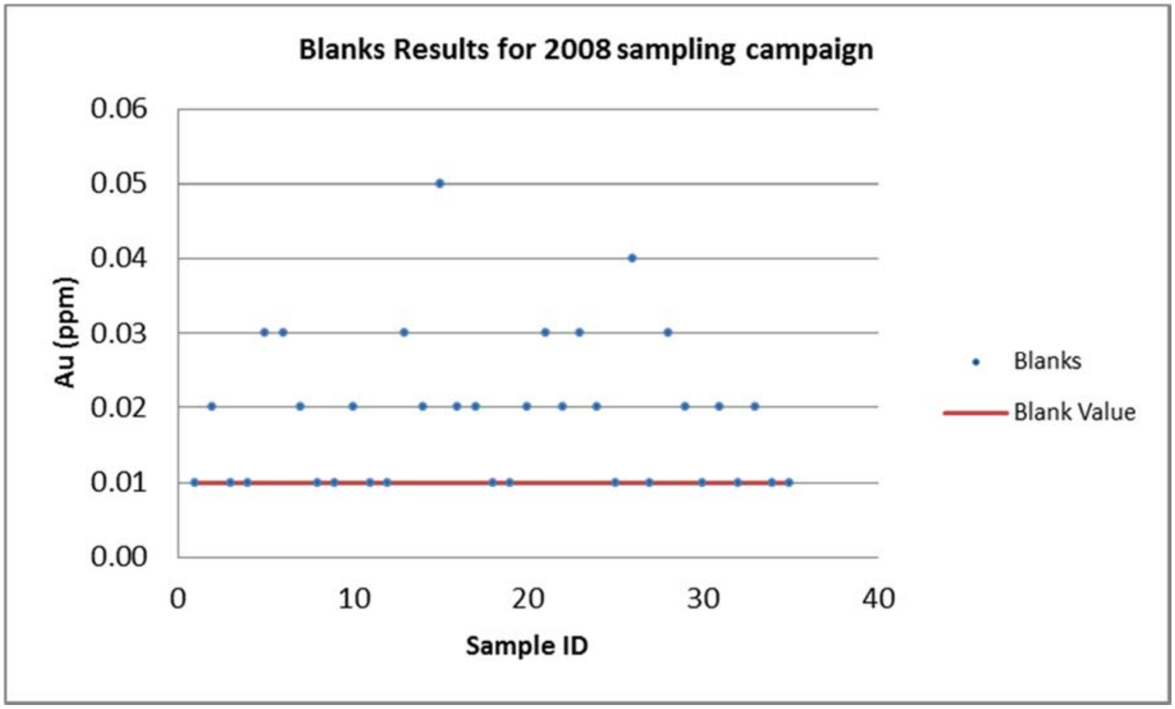


Figure 28 Results of Blank Analysis, January to March 2008

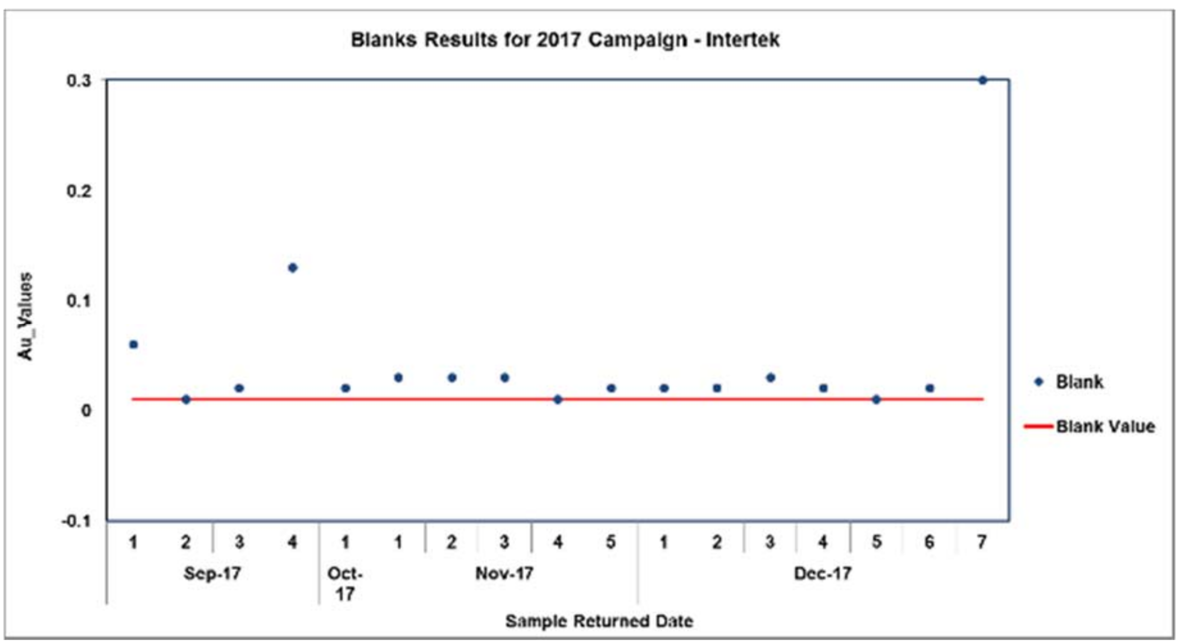


Figure 21a Results of Blank Analysis, February to December 2017 from Intertek Laboratory

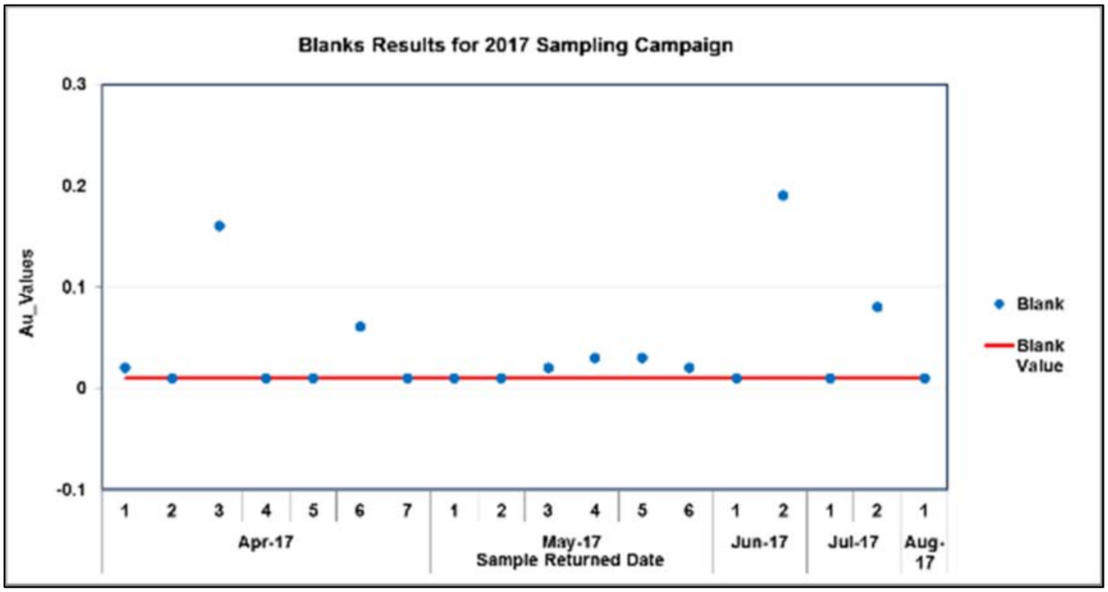


Figure 21b Results of Blank Analysis, Jan to Dec 2017 from SGS Laboratory

12.5 Gannet Standards

Standards are used for checking the precision and accuracy of the laboratory. A total of 161 gannet standard samples comprising five different grades were used as control samples for the fire assay standards. The performance accuracy of the lab is shown in Table 6 and from Figure 29 to Figure 33.

Table 6 Standards Used and Summary Results at Prestea WRP

Standard ID	Number	Certified Value	Lab mean	Accuracy Performance (% Bias)	Comment
ST5343	35	0.208	0.207	-0.48	Negative Bias
ST322	37	1.04	1.06	1.92	Positive Bias
ST5355	34	2.37	2.41	1.69	Positive Bias
ST05/2297	21	2.56	2.59	1.17	Positive Bias
ST5359	34	3.91	3.94	0.77	Positive Bias

Table 6a: Standards Used and Summary Results at Prestea WRP – 2017

Standard ID	Number	Certified Value	Lab mean	Accuracy Performance (% Bias)	Comment
G315-8	9	9.93	10.22	3	Positive Bias
G914-9	8	16.77	16.69	0	No Bias
G915-10	7	48.68	48.78	0	No Bias
ST06/7384	2	1.08	1.12	4	Positive Bias
ST442	2	2.5	2.65	6	Positive Bias

Table 6b Standards Used and Summary Results at Prestea WRP – 2013 and 2017

Standard ID	Number	Certified Value	Lab mean	Accuracy Performance (% Bias)	Comment
ST05/6372	1	2.46	2.43	-1	Negative Bias
ST06/7384	17	1.08	1.06	-2	Negative Bias
ST10/9298	5	3.22	2.87	-11	Negative Bias
ST138	3	0.022	0.02	-9	Negative Bias
ST252	8	0.054	0.08	-48	Negative Bias
ST359	11	4.03	4.07	1	Positive Bias
ST442	26	2.5	2.55	2	Positive Bias
ST481	14	1.02	1	-2	Negative Bias
ST485	3	2.63	2.76	5	Positive Bias
ST486	18	0.54	0.55	-3	Negative Bias
ST5355	3	2.37	2.45	3	Positive Bias
ST5357	7	0.52	0.5	-4	Negative Bias

The general results appear to be good with most results lying within two 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 significantly being generally less than 2%, and appears to be consistent suggesting an issue with the laboratories internal standards used for calibration.

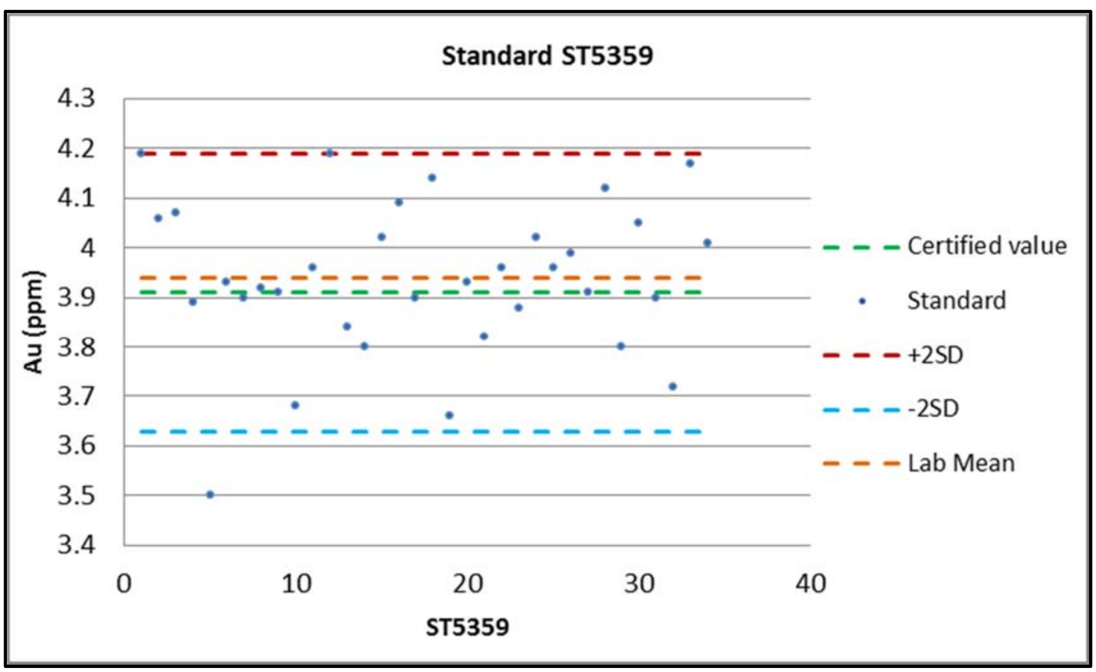


Figure 29 Standard Analysis Results for ST5359

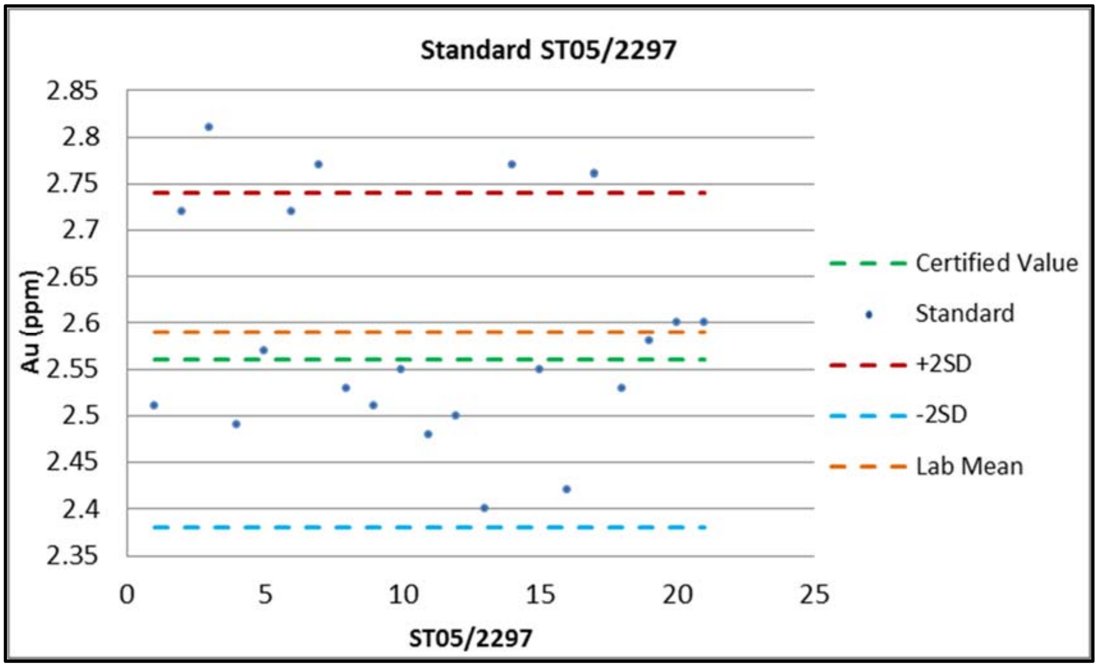


Figure 30 Standard Analysis Results for ST05/2297

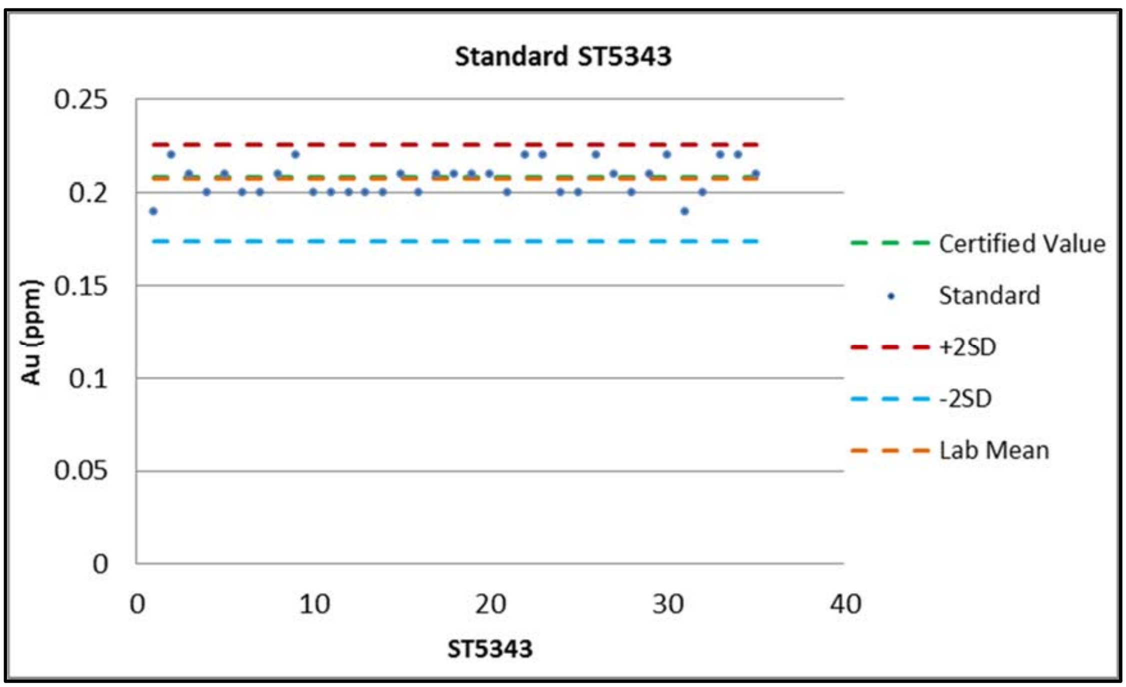


Figure 31 Standard Analysis Results for ST5343

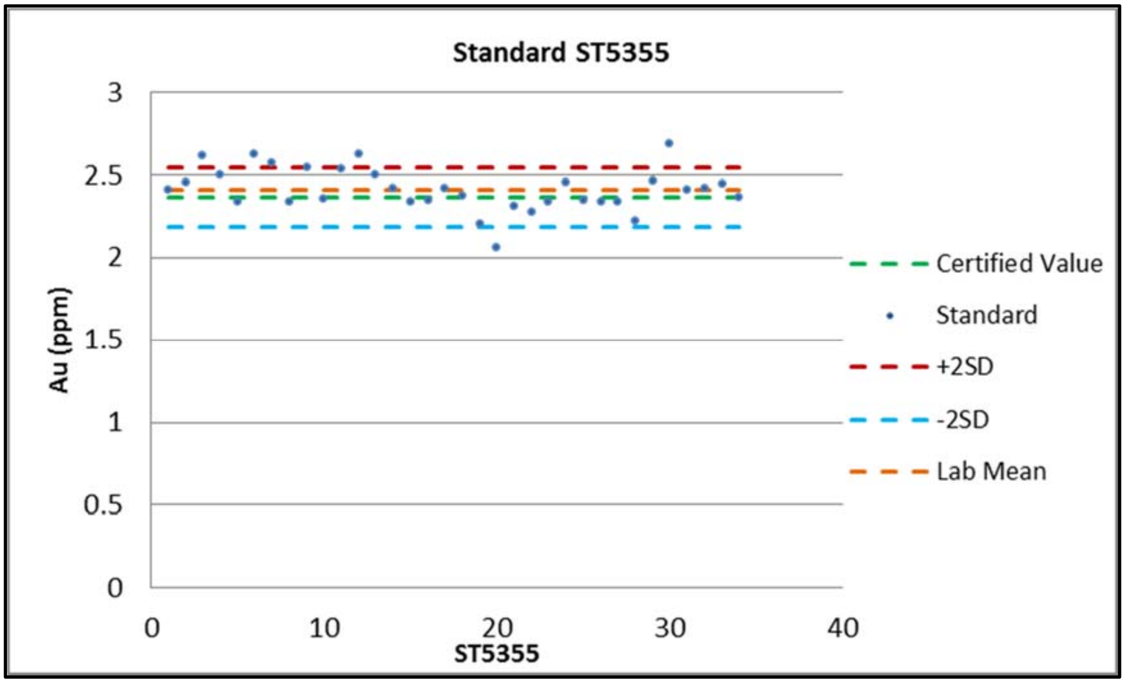


Figure 32 Standard Analysis Results for ST5355

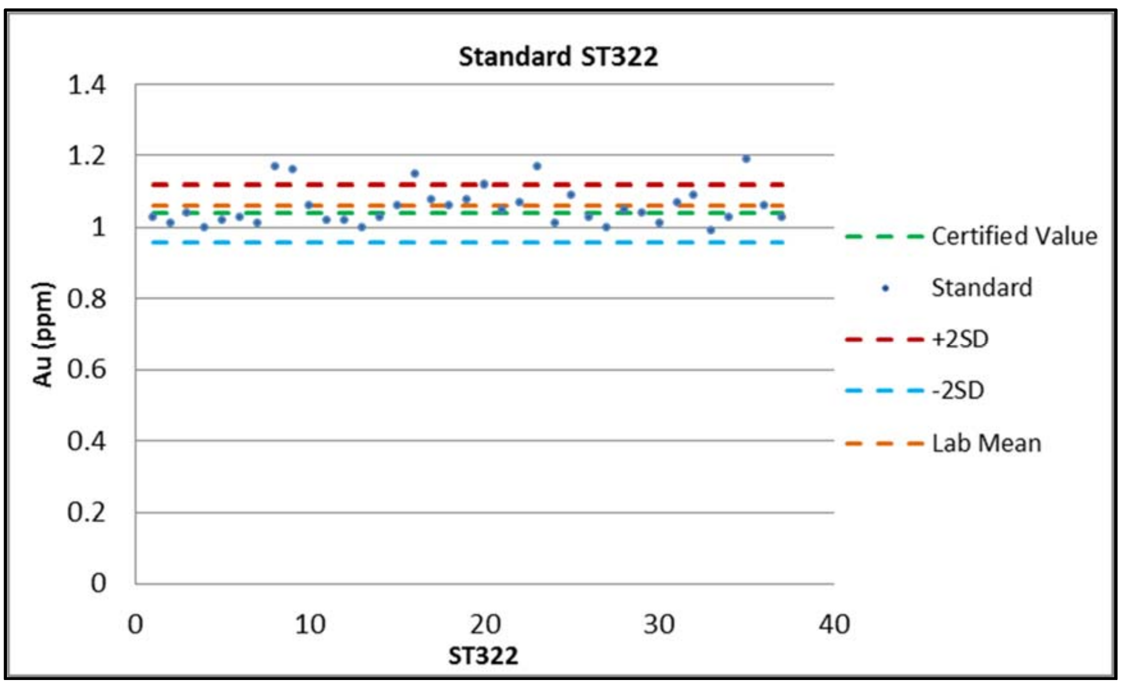


Figure 33 Standard Analysis Results for ST322

12.6 Screen Fire Assay Checks

Additional drilling was carried out on 24 L at Prestea on crosscuts 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 differences for those original pulps with values of 20 g/t gold or less are similar to those produced by the SFA of the original pulps. However, above 20 g/t gold the differences 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 included low grade samples where a small difference can exaggerate the percentage difference. For those original assays with a grade above 10 g/t gold, the average increase in grade of the SFA is 160% and for samples above 20 g/t gold the average increase is 155% (Figure 34).

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 and this could lead to an underestimation of the grade of the Prestea deposits based on the current sampling regime.

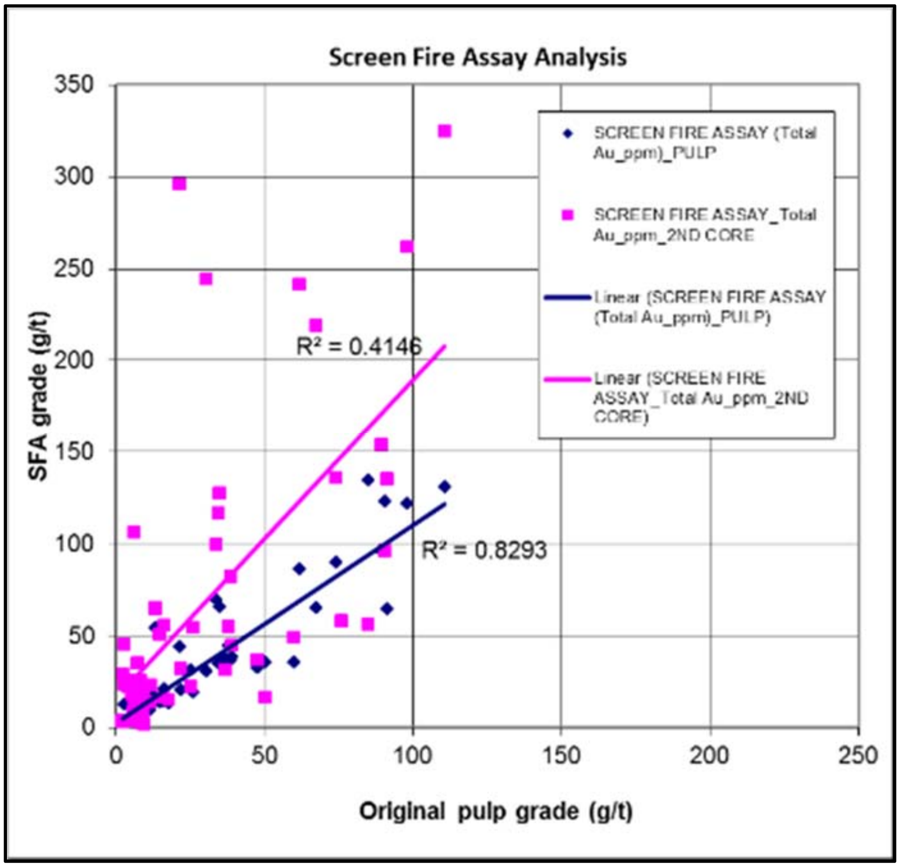


Figure 34 Results from Screen Fire Assay Analysis

12.7 Conclusions

The QA/QC 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 PUG West Reef project.

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 author also considers that the samples used for the standard analysis to date have average grades either at or lower than the cut-off grade 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.

13 Mineral Processing and Metallurgical Testing

13.1 Introduction

Three metallurgical testwork programs have been conducted on samples of mineralization from the PUG West Reef deposit.

A metallurgical testwork program was undertaken in 2008 in support of a study at that time investigating the potential for mining ore from both the West Reef and the Footwall Reef at Prestea.

The testwork supporting the current design commenced in 2013, when a different mining method (mechanized cut and fill) was being considered. That testwork was augmented with additional testwork conducted in 2015 in support of the current mining strategy.

13.2 2008 Testwork

This section will only describe the testwork conducted on samples of ore from the West Reef. This testwork was conducted by AMMTEC (now ALS) in Perth, Australia.

Interval samples from 12 diamond core boreholes were composited into Upper, Mid, and Lower composites, with an additional Master composite made up from these three individual composites. The make-up of the three individual composites is shown in Table 7, and Figure 35 shows the location of the boreholes (the boreholes used are represented by the blue and the brown diamonds).

Table 7 2008 Testwork Composite Sample Details

Composite	Borehole	From (m)	To (m)	Expected Grade (g/t Au)
Upper	UC17-274S6	238.11	240.85	29.8
	UC17-274S7	239.34	242.20	
	UC17-280S4	239.00	242.40	
	UC17-280S13	300.00	305.00	
	UC17-280S14	195.80	198.20	
Mid	UC17-280S1	349.60	352.60	23.6
	UC24-274S15	162.00	164.50	
	UC24-274S25	142.00	145.00	
	UC24-284S4	177.00	181.00	
Lower	UC24-274S2	179.40	183.00	16.4
	UC24-274S14	256.00	263.20	
	UC24-284S6	197.00	199.50	

The following testwork was conducted on the three composite samples:

- Thirty-five-element head assay;
- Semi-quantitative optical mineralogy;

- Gravity separation at a target grind size of 80% -75 µm; a two stage process using a Knelson concentrator followed by mercury amalgamation of the concentrate;
- Bottle roll cyanidation tests on the combined amalgamation and gravity tailings. Leaching conditions: 44.4% solids, initial pH 11, 0.8 kilograms per tonne (kg/t) initial NaCN addition, initial oxygen sparging, 48 hours with intermediate sampling at 1, 2.5, 5, 7.5 and 24 hours. The pH and free NaCN level were adjusted at each sampling time (pH to >10.5 and free NaCN to >160 ppm). Parallel tests were conducted, both under carbon-in-leach (CIL) conditions, with a carbon addition of 30 g/l. The first test used fresh carbon, where the carbon was replaced at the 1, 2.5, 5, and 24 hour sample intervals, in order to provide kinetic leaching data. The second test used loaded carbon taken from the Bogoso plant; in this test a single carbon addition was made and retained for the test duration; and
- The leached tailings were subjected to diagnostic leach tests in order to determine the proportions of residual gold associated with carbonaceous, sulphide and silicate phases.

The following testwork was conducted on the Master composite:

- The sample weight used was 8,250 g;
- Gravity separation at a grind size of 80% -125 µm; Knelson concentrator followed by amalgamation;
- Bottle roll cyanidation tests on the amalgamation and gravity tailings separately, following regrinding to a target grind size of 80% -75 µm. The leaching conditions were the same as for the individual composites described above; and
- Diagnostic leach tests of the leached tailings.

The head assays of the three individual composite samples are shown in Table 8.

The mineralogical investigation revealed the dominant sulphide mineral to be pyrite (51-57% v/v of the total sulphides present), followed by sphalerite (5-6%), then arsenopyrite (2-13%). Carbonaceous matter was present as ultrafine flakes of graphitic material. Only three gold occurrences were reported, varying in size from 0.5 µm to 14 µm.

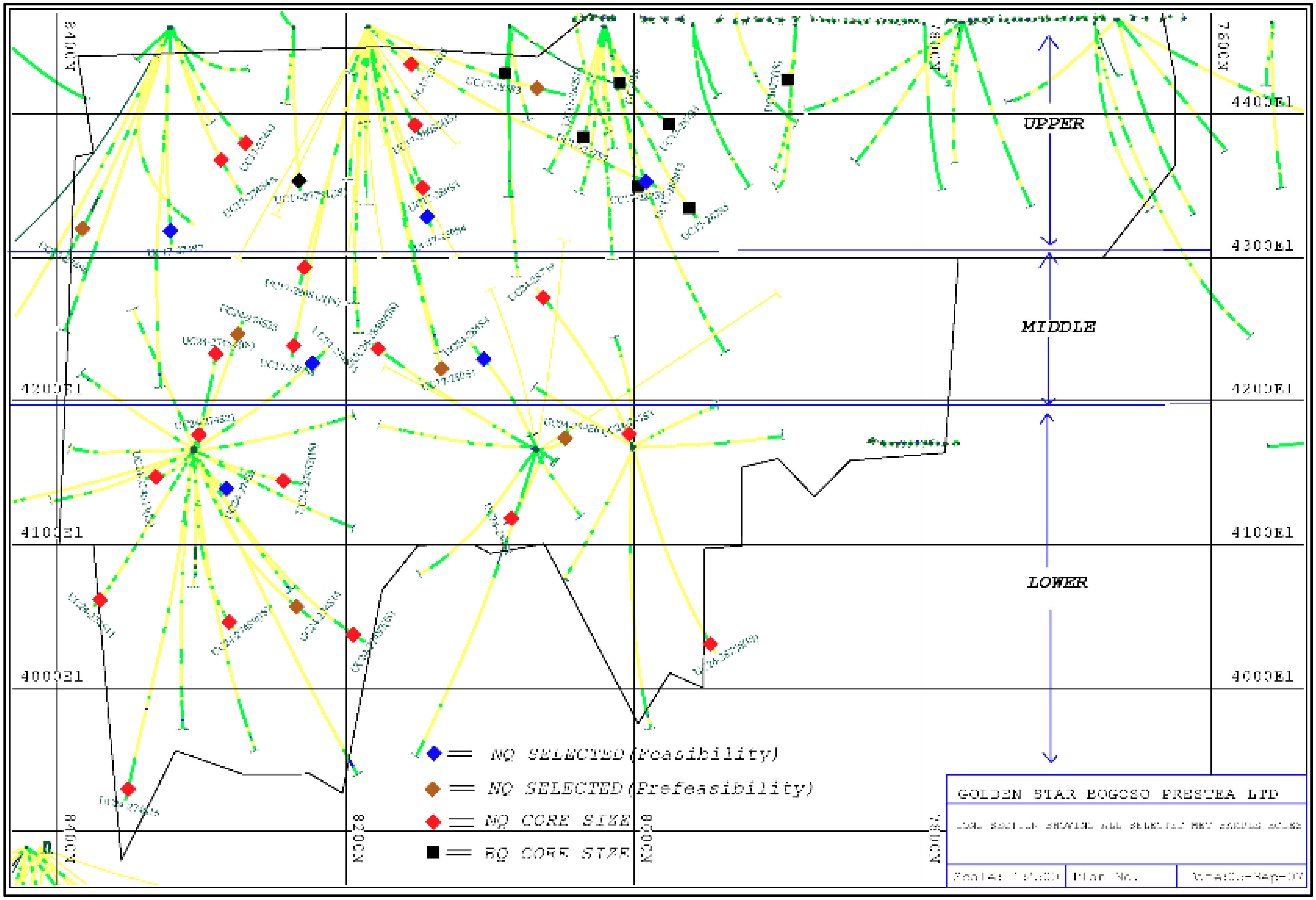


Figure 35 Borehole Locations of 2008 Testwork

Table 8 2008 Testwork Composite Sample Head Assays

Element	Unit	Upper	Mid	Lower
Au	g/t	30.9	23.1	14.8
Ag	g/t	<2	<2	<2
Cu	ppm	86	83	65
Fe	%	3.19	4.38	3.94
Zn	ppm	106	109	109
Pb	ppm	46	27	25
As	ppm	1014	342	1943
Sb	ppm	<5	<5	<5
Bi	ppm	12	<10	<10
Cd	ppm	<5	<5	<5
Co	ppm	20	25	20
Cr	ppm	756	701	662
Te	ppm	0.5	0.3	0.5
Se	ppm	1	<1	1
Hg	ppb	210	60	60
Mo	ppm	135	179	94
Ni	ppm	461	397	369
S (total)	%	0.80	0.83	1.08
S (sulphide)	%	0.62	0.65	0.64
C (total)	%	1.04	1.43	1.48
C (organic)	%	0.35	0.51	0.51
Al	%	4.04	5.43	5.18
Ba	ppm	307	441	418
Ca	%	1.37	0.90	1.47
K	%	0.90	1.56	1.37
Li	ppm	33	23	19
Mg	%	0.72	0.97	1.16
Mn	ppm	380	470	477
Na	%	0.95	1.36	1.62
P	ppm	302	442	416
Sr	ppm	177	199	235
Ti	ppm	331	495	481
V	ppm	62	77	82
Y	ppm	3	3	3
Zr	ppm	45	60	64

The gravity and cyanidation testwork results are summarized in Table 9, where the results for the Master composite show the combined leach response for the amalgamation and Knelson tailings. In the original analysis of this work conducted by GSR’s consultant, for the tests where loaded carbon from the Bogoso plant was used, the carbon assay at the end of each test was adjusted to make the calculated head for each of these tests the same as for the kinetic tests, i.e., the tests where fresh carbon was used. Using the actual final carbon gold assays gives calculated head assays significantly different to those presented. In the light of this discrepancy, these results have not been presented here.

Table 9 2008 Testwork Gravity / Cyanidation Test Results

Item	Unit	Upper	Mid	Lower	Master
Gravity grind size (P_{80})	µm	n/a	n/a	n/a	125
Gravity concentrate weight	%	n/a	n/a	n/a	2.55
Gravity Au recovery	%	20.4	39.6	34.7	75.0
Cyanidation P_{80}	µm	72	62	75	68
Cyanidation recovery					
1 hr	%	50.5	39.6	34.7	83.3
2.5 hr	%	66.1	52.1	48.1	92.6
5 hr	%	74.0	56.5	54.4	93.8
24 hr	%	76.5	58.0	56.8	95.4
48 hr	%	78.0	58.1	57.1	96.0
Gravity + CN recovery	%	97.1	98.9	97.6	99.0
CN residue Au grade	g/t	0.39	0.17	0.34	0.22
Calculated CN head Au	g/t	19.4	9.15	8.60	5.51
Calculated gravity head Au	g/t	24.3	15.5	14.5	22.0
NaCN consumption	kg/t	2.27	1.64	1.84	2.06
Lime consumption	kg/t	0.81	0.84	0.86	1.11

This testwork indicated that the West Reef material is free milling, with high recoveries reported in all cases, even for the Lower composite, which had a relatively high arsenic content. The difference in gravity recovery between the individual composites (20 – 40%) and the Master composite (75%) was not documented in the original reports received. Despite the absence of concentrate weights for the individual composites, it seems likely that the lower gravity recoveries were a due to the use of a lower mass recovery to the gravity concentrates for these samples.

The reported reagent consumptions – lime and cyanide – were reasonable, especially for the head grades of the samples. No grind size optimization was carried out, and the kinetic data indicates that there was limited additional recovery after 24 hours of leaching.

13.3 2013 Testwork

The 2013 testwork program was focussed on the amenability of processing the West Reef ore through the oxide circuit at the Bogoso operation, and on assessing the impact of the relatively high waste dilution that was anticipated from the mining method proposed at that time. Due to the excess of capacity available in the oxide circuit, no optimization of processing conditions over the standard grind size used in the oxide circuit was undertaken, nor was any significant attention paid to optimizing the leach residence time.

In the absence of any historical core, this testwork was conducted on samples of diamond drill core from the underground geotechnical drilling program that was undertaken in late 2012 and early 2013. The testwork was conducted at the SGS Lakefield laboratory in Canada.

Samples from eight boreholes were available. The aim was to sample them in such a way as to represent the mineralized shear zone that would be mined. The shear zone consists mainly of quartz within in a softer matrix of graphite. Six of the boreholes were drilled from 17 L, and the remaining two were from 24 L. The samples submitted were of half core, and were provided as intervals of up to 1-metre core length, so as to also provide data for the geological database. The principle in sampling the boreholes was to include all of the quartz vein (QV) mineralization, all footwall (FW) and hanging wall (HW) mineralization within the shear zone (SZ), as well as 0.5 to 1.0 m of additional dilution "skin" outside of the SZ.

On the basis of the interval sample head assays, five composite samples were prepared for the laboratory testwork program. Three composites were prepared initially from the first consignment of five 17 L boreholes, and the subsequent two composites were prepared from the second consignment of the one 17 L and two 24 L boreholes. The initial three composites varied according to the proportions of hanging wall and footwall mineralization that made up the dilution materials. The latter two composites focussed on generating samples of closer to the expected diluted ore feed grade, as the initial composites had been of higher grades due to the high grade nature of the quartz vein intervals in the initial five borehole samples. One of the latter composites was composed of intervals from the two 24 L boreholes only, and the other was made up of intervals from all three boreholes in the second consignment.

The make-up of the five composites is shown in Table 10, and Figure 36 shows the location of the boreholes used for both the 2013 and 2015 testwork programs.

The following testwork was conducted on the composite samples:

- Bond Ball Mill Work Index (BWi), at a closing screen size of 150 µm;
- Twenty-five element head assay;
- Preg-Robbing Test. This is an SGS standard procedure, with 1 kg of ground ore slurried in 1.5 litres of a 10 ppm gold stock solution. Sodium hydroxide (NaOH) is added to maintain a pH of 11 throughout the 24 hour duration of the test. Solution samples are taken for assay after 1, 3, 6, and 24 hours and assayed for gold. No cyanide is added to the test;
- Gravity separation at a grind size of 80% -150 µm; a two-stage process using a Knelson concentrator followed by a Mozley table separator, targeting a gravity concentrate mass recovery of 0.01 to 0.1%;
- Bottle roll cyanidation tests on the gravity tailings, reground to 80% -75 µm. Leaching conditions: 40% solids, pH 10.5 to 11, 0.5 kg/t initial sodium cyanide (NaCN) addition, 48 hours with intermediate sampling at 8 and 24 hours. Parallel tests were conducted with and without the addition of 10 g/l of carbon, i.e., carbon-in-pulp (CIP) vs. CIL format; and
- Tailings characterization tests:
  - Particle size distribution (sieves and hydrometer).
  - Atterberg limits.
  - Consolidated drained direct shear test.

Additional gravity and leach tests were conducted to generate the samples used for the tailings characterization tests.

In addition, SAG mill comminution tests were undertaken ahead of the composite formation, as these tests require intact diamond core pieces. Selected intervals were identified for this testwork, and suitable core pieces were taken from each interval. Where several intervals were combined for the SAG mill comminution test, the core pieces from each interval were tracked separately through the test and returned to their respective intervals at the completion of the test prior to being crushed to -6 mesh ahead of the head assays.

The results of the SAG mill comminution tests are shown in Table 11.

Table 10 2013 Testwork Composite Sample Details

Composite	Designation	Samples Used	From (m)	To (m)	FW Dilution (%)	HW Dilution (%)	Expected Grade (g/t Au)
1	1:1 FW:HW	GT17-274S1FW1	131.06	131.7	24.0	24.0	19.0
		GT17-274S1HW3	134.8	135.44			
		GT17-274S2FW2	269.8	270.9			
		GT17-274S2QV1	270.9	271.8			
		GT17-274S2QV3	272.65	273.5			
		GT17-274S2QV4	273.5	274.35			
		GT17-274S2QV5	257.2	274.35			
		GT17-274S2QV6	257.2	267.05			
		GT17-274S2QV7	267.05	276.85			
		GT17-280S2HW1	251.3	252.3			
2	1:2 FW:HW	GT17-274S2QV2	271.8	272.65	14.2	28.6	30.5
		GT17-280S2FW1	244.11	245.03			
		GT17-280S2QV1	246.87	247.86			
		GT17-280S2QV3	248.46	249.45			
		GT17-280S2QV4	249.45	250.44			
		GT17-280S2HW2	252.3	253.16			
		GT17-287S1AHW2	224.9	225.5			
3	2:1 FW:HW	GT17-274S1FW2	131.7	132.6	28.8	17.0	39.7
		GT17-274S1HW2	133.83	134.8			
		GT17-280S2QV5	250.44	251.3			
		GT17-287S1AFW3	220.8	221.4			
		GT17-287S1AQV1	221.4	222.3			
		GT17-287S1AQV3	222.5	223.4			
		GT17-287S1AQV5	223.9	224.3			
4	24 L	GT24-274S1QV1	169.0	169.83	23.9	23.9	13.3
		GT24-274S1QV3	170.66	171.5			
		GT24-274S2FW2	155.2	156.13			
		GT24-274S2QV1	156.13	156.7			
		GT24-274S2HW2	157.1	157.93			
5	17+24 L	GT17-287S2QV2	293.33	294.16	21.6	25.2	13.1
		GT17-287S2QV3	294.16	295.0			
		GT24-274S1FW1	167.85	168.25			
		GT24-274S1FW2	168.25	168.6			
		GT24-274S1QV2	169.83	170.66			
		GT24-274S1HW2	172.12	172.52			
		GT24-274S2HW3	157.93	158.4			

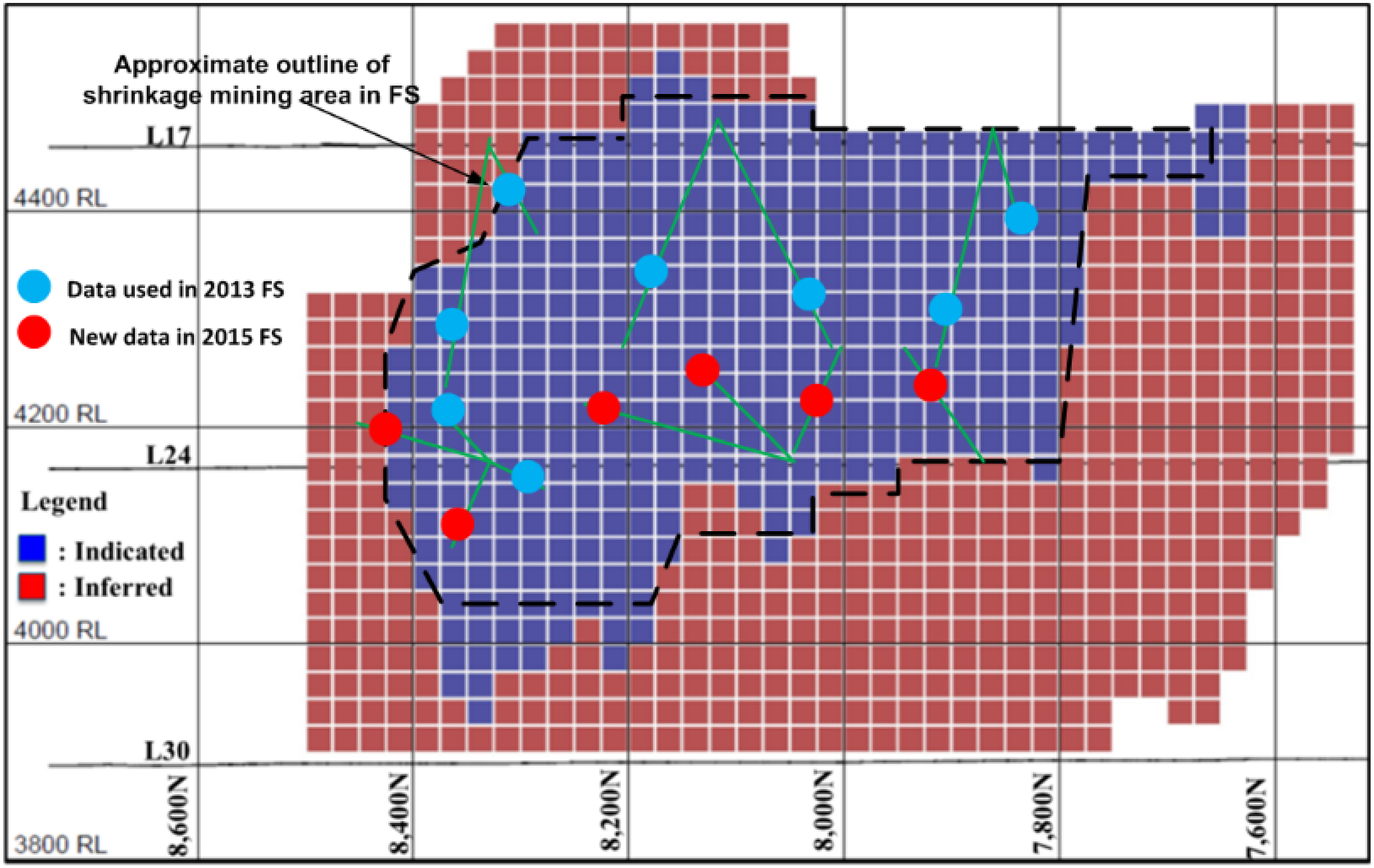


Figure 36 2013 & 2015 Testwork Borehole Approximate Locations

(Looking East)

Table 11 2013 SAG Mill Comminution Test Results

Sample	Drop Weight Index (kWh/m^3)	JK A*b parameter	JK t_a parameter	Hardness Percentile (%)
GT17-274S2QV1 GT17-274S2QV2 GT17-274S2QV3 GT17-274S2QV4 GT17-274S2QV5 GT17-274S2QV6 GT17-274S2QV7	3.5	75.1	0.75	22
GT17-280S2QV1	3.0	82.1	0.85	19
GT17-280S2QV3	3.7	70.0	0.69	25
GT17-280S2QV4	3.8	69.3	0.68	25
GT17-280S2QV5	2.4	78.0	0.76	21
GT24-274S1QV1 GT24-274S1QV2 GT24-274S1QV3	4.9	54.0	0.53	40
GT24-274S2FW1 GT24-274S2FW2 GT24-274S2HW2 GT24-274S2HW3	7.8	37.0	0.33	70

The drop weight index is a measure of an ore's resistance to impact breakage, where a smaller number indicates less resistance, i.e., a softer ore. The JK A*b parameter is an analogous figure, where a larger number indicates a softer ore. The JK t_a parameter is a measure of an ore's resistance to abrasion breakage, and again a larger number indicates a softer ore. The hardness percentile reflects the particular samples' position on the database of all SAG mill comminution tests results. The results, therefore, indicate that the 17 L samples are relatively soft, however, the results for the two 24 L samples indicate that the quartz vein sample is significantly harder than the corresponding 17 L samples, in terms of its resistance to both impact and abrasion breakage, and that the footwall and hanging wall combined sample is harder still.

The results of the BWi Tests are show in Table 12.

Table 12 2013 BWi Test Results

Composite	P_{80} (μm)	BWi (kWh/t)
1	117	15.4
2	117	15.8
3	120	15.8
4	121	15.8
5	117	15.8

The samples had BWi values averaging 15.7 kWh/t, indicating an ore with a moderately high grinding energy requirement.

The head assays for the five composite samples are shown in Table 13.

Table 13 2013 Testwork Composite Sample Head Assays

Element	Unit	Composite 1	Composite 2	Composite 3	Composite 4	Composite 5
Au 1	g/t	22.0	24.1	41.8	24.4	14.8
Au 2	g/t	15.2	21.2	37.0	16.9	15.0
Au 3	g/t	-	-	-	9.6	-
Au 4	g/t	-	-	-	5.9	-
Au (average)	g/t	18.6	22.7	39.4	14.2	14.9
Ag	g/t	8.2	4.6	1.5	1.1	<0.9
Cu	ppm	120	40	40	240	60
Fe	%	3.22	2.39	2.47	3.37	3.44
Zn	ppm	120	120	<100	160	110
As	ppm	610	490	300	300	1600
Sb	ppm	<20	<20	<20	<20	<20
Te	ppm	<4	<4	<4	<50	<50
Se	ppm	<10	<10	<10	<20	<20
Hg	ppm	<0.3	<0.3	<0.3	<0.3	<0.3
C	%	1.73	1.36	1.26	2.05	1.92
TOC	%	0.99	0.71	0.83	1.40	0.97
S	%	0.54	0.42	0.50	0.71	0.88
SiO2	%	70.90	78.40	78.80	71.80	69.10
Al2O3	%	10.70	7.55	8.04	9.60	11.40
MgO	%	1.69	1.28	1.27	1.73	1.88
CaO	%	1.50	1.43	1.00	1.42	1.63
Na2O	%	2.36	1.57	1.76	1.92	3.23
K2O	%	1.29	1.05	1.05	1.33	1.27
TiO2	%	0.41	0.29	0.29	0.36	0.40
P2O5	%	0.07	0.06	0.06	0.06	0.08
MnO	%	0.05	0.05	0.04	0.06	0.05
Cr2O3	%	0.04	0.04	0.04	0.03	0.02
V2O5	%	0.02	0.01	0.01	0.02	<0.01
LOI	%	5.80	4.41	4.29	5.51	5.28

The difference between each set of repeat gold assays indicates the presence of coarse, free gold. The gold assays were close to the expected values for Composites 1, 3, 4, and 5, but were lower than the expected value for Composite 2 (see Table 10 for the expected values). The remaining assays do not indicate any potentially problematic behaviour, with the possible exception of arsenic, indicating the presence of arsenopyrite, which is the principal refractory gold host in the Bogoso region. However, the arsenic assays were relatively low, with the exception of Composite 5, whose arsenic assay was higher, but still lower than that for the Lower composite from the 2008 program (see Table 8).

The results of the gravity separation segment of the gold recovery tests are summarized in Table 14. Where two results are shown for a composite, the first is from the initial test, which was used to generate the sample for the initial carbon-in-pulp and carbon-in-leach tests, and the second is from the subsequent test that was used to generate the sample that was leached ahead of the tailings characterization testwork.

The gold recoveries achieved to the target mass recovery of 0.05 to 0.10% ranged from 52.0 to 82.3%, and the 17 L composites reported higher gravity gold recoveries than the composites containing 24 L ore. The repeat gravity tests produced higher mass recoveries to the gravity concentrate, which tended to result in a higher gold recovery. These tests were also conducted at a finer grind size (80% -75 µm compared to 80% -150 µm for the initial tests). There was a reasonably good level of agreement between the calculated and assayed head grades for these tests.

The results of the cyanidation tests are summarized in Table 15. Again, where two carbon-in-leach results are shown for a composite, the first is from the initial test and the second is from the subsequent carbon-in-leach test used to generate sample for tailings characterization.

Table 14 2013 Testwork Gravity Separation Test Results

Item	Unit	Composite 1		Composite 2		Composite 3		Composite 4	Composite 5
Gravity Concentrate Weight	%	0.025	0.102	0.032	0.123	0.070	0.097	0.074	0.062
Gravity Concentrate Au Assay	g/t	45,400	12,100	61,100	14,600	38,200	29,000	12,200	9,880
Gravity Tailings Au Assay1	g/t	3.22	3.47	5.03	2.47	5.73	3.17	2.94	5.63
Gravity Au Recovery	%	78.1	78.0	79.6	87.9	82.3	89.8	75.6	52.0
Calculated Head Au Assay	g/t	14.7	15.8	24.6	20.5	32.4	31.2	12.0	11.7
Measured Head Au Assay	g/t		18.6		22.7		39.4	14.2	14.8

1 Calculated from gravity tailings carbon-in-pulp test

Table 15 2013 Testwork Cyanidation Test Results

Item	Unit	Composite 1			Composite 2			Composite 3			Composite 4		Composite 5	
		CIP	CIL	CIL	CIP	CIL	CIL	CIP	CIL	CIL	CIP	CIL	CIP	CIL
Grind Size P80	µm	88	81	84	85	82	85	77	81	82	77	79	76	74
CN Recovery														
8 hr	%	77	-	-	86	-	-	87	-	-	52	-	68	-
24 hr	%	78	-	-	87	-	-	88	-	-	53	-	69	-
48 hr	%	78.3	67.0	90.2	88.1	94.9	88.9	88.7	95.5	90.4	54.3	86.1	70.5	90.5
Gravity + CN Recovery	%	95.2	92.5	97.8	97.6	98.9	98.7	98.0	99.2	99.0	88.8	96.6	85.8	95.4
CN Residue Au Grade	g/t	1.15	0.37	0.33	0.56	0.28	0.26	0.60	0.28	0.29	1.34	0.40	1.60	0.54
Calculated CN Head Au	g/t	5.30	1.12	3.37	4.71	5.19	2.34	5.31	6.22	3.02	2.93	2.87	5.42	5.68
NaCN Consumption	kg/t	0.62	0.50	1.50	0.90	0.75	1.68	0.82	0.79	1.45	0.12	0.25	0.14	0.34
Lime Consumption	kg/t	0.37	0.40	0.41	0.35	0.28	0.33	0.30	0.30	0.37	0.60	0.53	0.42	0.48

The cyanidation recoveries ranged from 54.3 to 88.7% under carbon-in-pulp conditions, and 67.0 to 95.5% under carbon-in-leach conditions. The overall recoveries (i.e., cyanidation plus gravity) ranged between 85.8 and 98.0% under carbon-in-pulp conditions, and 92.5 to 99.2% under carbon-in-leach conditions. The 8- and 24-hour intermediate results for the carbon-in-pulp tests indicate rapid kinetics, with little additional recovery achieved after 8 hours.

The carbon-in-leach tests indicate an improvement in recovery under carbon-in-leach conditions. The magnitude of the improvement was relatively small for Composites 2 and 3, however, it was greater for Composites 4 and 5, the composites containing 24 L ore. This suggests that preg-robbing may become more significant with depth.

There was something of an inconsistency in the Composite 1 results. The initial carbon-in-leach test indicated a lower recovery than the carbon-in-pulp test. However, the repeat test indicated an improvement in recovery over the carbon-in-pulp case. The solids residue grades for the two carbon-in-leach tests were very similar, and were lower than the solid residue grade for the carbon-in-pulp test, indicating an improvement in recovery. However, the calculated head grade for the first carbon-in-leach test was low, 1.12 g/t gold compared to 5.30 g/t gold for the carbon-in-pulp test and 3.37 g/t gold for the second carbon-in-leach test. The calculated head grade is based on the solid residue, final solution and carbon assays, and so it would appear that the low calculated head grade for the first carbon-in-leach sample, and hence the low recovery, was probably due to an error in the carbon assay for this test. As standard laboratory practice is to assay such carbon samples in their entirety, it is not possible to further investigate the low recovery reported for this test. The second carbon-in-leach test does, however, indicate that there was an improvement in gold recovery under carbon-in-leach conditions for this composite.

The Bogoso site laboratory calculates a preg-robbing index based on the difference between the carbon-in-pulp and carbon-in-leach format recoveries, as follows:

$$\text{Preg-robbing index} = (\text{carbon-in-leach recovery} - \text{carbon-in-pulp recovery}) / \text{carbon-in-leach recovery} * 100$$

On the basis of the cyanidation stage recoveries only, and using average figures for those samples with two carbon-in-leach results, the Bogoso preg-robbing index values for the composites are:

- Composite 1: -0.4 %
- Composite 2: 4.1 %
- Composite 3: 4.6 %
- Composite 4: 36.9 %
- Composite 5: 22.1 %

These results indicate that there was more preg-robbing exhibited by the samples containing 24 L ore.

The results of the SGS preg-robbing tests are summarized in Table 16.

Table 16 2013 Testwork Preg-Robbing Test Results

Item	Unit	Composite 1	Composite 2	Composite 3	Composite 4	Composite 5
Au absorbed after						
1 hr	%	9.0	8.5	15.2	13.5	8.6
3 hr	%	7.0	5.1	9.0	9.6	13.3
6 hr	%	4.2	4.1	3.0	0.0	13.4
24 hr	%	4.4	1.3	-6.6	-20.2	17.9

These results indicate that all samples exhibited some preg-robbing behaviour initially, but that for all samples except Composite 5, this behaviour had reduced or disappeared after 24 hours. For Composite 5, the preg-robbing behaviour increased slightly over the course of the test.

The cyanide consumptions averaged 0.76 kg/t. This is a relatively low figure, particularly considering the high head grades of this material. The average lime consumption was 0.40 kg/t, which is a low figure.

The results of the tailings characterization tests are shown in Table 17. The particle size distribution results indicate that the samples consisted primarily of fines, with the majority of this fine material being silt sized grains, with only a minor fraction falling into the clay size range. The Atterberg limit tests characterize Composites 2, 3, 4 and 5 as “cohesionless inorganic silt or rock flour” type material, whereas Composite 1, with its elevated liquid limit and limited plasticity was characterized as an “inorganic silt or rock flour of low plasticity”. The direct shear test results report similar results for all samples, with peak shear stresses ranging between 36 kPa and 292 kPa, with horizontal displacements at peak ranging from 4.2 to 8.2 mm.

Table 17 2013 Testwork Tailings Characterization Test Results

Composite	Particle Size Characterization			Atterberg Limits			Direct Shear Tests			
	Sand (%)	Silt (%)	Clay (%)	Liquid Limit (%)	Plastic Limit (%)	Plasticity Index (%)	Consolidation Pressure (kPa)	Time to Failure (hr)	Peak Shear Stress (kPa)	Horizontal Displacement at Peak (mm)
							75	35.6	56.8	5.13
1	20.7	72.3	7	27	25	2	150	36.7	106	5.28
							300	44.0	284	6.33
							75	34.0	36.0	4.90
2	19.4	74.6	6	22	NP	NP	150	42.6	128	6.14
							300	43.8	290	6.31
							75	33.2	61.5	4.78
3	22.1	71.9	6	22	21	1	150	29.3	115	4.21
							300	43.9	261	6.30
							75	36.9	59.3	5.32
4	21.1	74.9	4	23	NP	NP	150	42.8	105	6.17
							300	48.5	280	6.99
							75	57.1	58.2	8.22
5	24.9	71.1	4	22	NP	NP	150	33.9	104	4.88
							300	46.6	292	6.71

13.4 2015 Testwork Program

The context of the 2015 testwork program was the change of the mining method to shrinkage stoping, which would result in less dilution and, therefore, higher plant feed grades, and the decision made that the ore would be processed in a separate process line, which is to be constructed within the existing Bogoso plant complex.

Given that a dedicated process line is to be used, the 2015 testwork program focussed on optimizing the key operating parameters of grind size and residence time in support of the new plant design.

Samples available for the testwork consisted of the remaining core samples from the 2012/13 geotechnical drilling program (see Figure 36 for the borehole locations). Three additional composite samples were made up for this testwork program, the details of which are shown in Table 18. Due to the limited amount of sample available, it was not possible to make composites that matched the target mineral reserve grade of 14.0 g/t gold while also having sufficient weight of the sample to conduct the specified testwork program.

Table 18 2015 Testwork Composite Sample Details

Composite	Samples Used	From (m)	To (m)	Expected Grade (g/t Au)
7	GT17-274S1FW3 GT17-274S1QV1 GT17-274S1HW1 GT17-274S2FW1 GT17-274S2HW1 GT17-280S1HW1 GT17-280S2FW2 GT17-280S2FW3 GT17-287S1AHW1 GT17-287S1AHW3 GT17-287S2QV1	132.6 133.0 133.2 269.3 276.9 284.3 245.0 246.0 224.3 225.5 292.5	133.0 133.2 133.8 269.8 278.0 285.3 246.0 246.9 224.9 226.1 293.3	19.20
8	GT24-274S3FW1 GT24-274S3FW2 GT24-274S3QV1 GT24-274S3HW1 GT24-274S3HW2 GT24-274S3AFW1 GT24-274S3AFW2 GT24-274S3AFW3 GT24-274S3AQV1	204.8 205.8 207.0 207.5 208.1 209.0 210.0 211.0 211.6	205.8 207.0 207.5 208.1 208.8 210.0 211.0 211.6 212.6	4.52
9	GT24-284S1QV1 GT24-284S1HW1 GT24-284S2FW1 GT24-284S2QV1 GT24-284S2QV2 GT24-284S2HW1 GT24-284S3FW1 GT24-284S3QV1 GT24-287S1FW2 GT24-287S1QV1 GT24-287S1HW1	175.9 176.7 182.1 182.8 183.5 184.2 211.6 213.1 165.6 166.6 167.2	176.7 177.5 182.8 183.5 184.2 185.4 212.4 213.7 166.6 167.2 167.8	7.35

Composite 7 consisted of virtually all of the remaining available 17 L sample intervals. Composite 8 consisted of samples from the 274 heading of 24 L, and Composite 9 consisted of samples from the 284 and 287 headings of 24 L; these two areas of the orebody had not previously been included in testwork samples (i.e., in 2013).

The testwork conducted on these samples was as follows:

- Bond Ball mill Work index (BWi) at a closing screen size of 106 µm
- Twenty-five element head assay
- Preg-robbing test, as per the 2013 testwork program
- Gravity separation at a grind size of 80% -110 µm; the same two stage process as per the 2013 testwork program

- Bottle roll carbon-in-leach tests on the gravity tailings – leaching conditions: 40% solids, pH 10.5 to 11, 0.5 kilograms per tonne initial NaCN addition, with the following combinations of grind size and residence time:
  - 80% -110 µm, 24 hours
  - 80% -110 µm, 16 hours
  - 80% -75 µm, 16 hours

Settling tests were also conducted on samples of ground Composite 7 ore, i.e., before leaching. Two measuring cylinder scale tests were conducted, one at each grind size of 80% -110 µm and 80% -75 µm, at an initial slurry density of 20% solids, and with a flocculant addition of 40 g/t of the anionic flocculant currently used at the Bogoso operation.

Gravity and carbon-in-leach tests were also undertaken on selected samples remaining from the 2013 testwork program in order to generate additional grind size and residence time data.

These tests were as follows:

- Composite 1:
  - Gravity and carbon-in-leach test at 80% -110 µm, 16-hour residence time
- Composite 2:
  - Gravity and carbon-in-leach test at 80% -110 µm, 16-hour residence time
- Composite 4:
  - Gravity and carbon-in-leach test at 80% -150 µm, 16-hour residence time
- Composite 5:
  - Gravity and carbon-in-leach test at 80% -150 µm, 16-hour residence time
  - Gravity and carbon-in-leach test at 80% -110 µm, 16-hour residence time
  - Carbon-in-leach test on a sample without prior gravity separation, at 80% -75 µm, 24-hour residence time

The results of the BWi tests are show in Table 19.

Table 19 2015 BWi Test Results

Composite	P_{80} (µm)	BWi (kWh/t)
7	80	14.3
8	80	14.2
9	82	14.8

Despite the finer closing screen size (106 µm rather than 150 µm), these samples returned BWi values slightly lower than those reported in the 2013 testwork program (see Table 12).

The head assays for the three composite samples are shown in

Table 20.

Table 20 2015 Testwork Composite Sample Head Assays

Element	Unit	Composite 7	Composite 8	Composite 9
Au 1	g/t	15.7	2.94	6.99
Au 2	g/t	7.94	4.65	6.32
Au (average)	g/t	11.8	3.80	6.66
Ag	g/t	<0.5	4.9	<0.5
Cu	ppm	50	40	30
Fe	%	4.50	4.06	3.27
Zn	ppm	<100	120	<100
As	ppm	390	1300	1200
Sb	ppm	<20	<20	<20
Te	ppm	<4	<4	<4
Se	ppm	<10	<10	<10
Hg	ppm	<0.3	<0.3	<0.3
C	%	2.71	2.31	1.82
TOC	%	1.06	1.03	0.78
S	%	0.99	0.72	0.57
SiO_2	%	59.1	63.2	69.4
Al_2O_3	%	14.3	13.8	11.1
MgO	%	2.31	2.16	1.70
CaO	%	2.46	1.74	1.58
Na_2O	%	2.90	2.15	2.45
K_2O	%	1.96	2.06	1.50
TiO_2	%	0.55	0.53	0.39
P_2O_5	%	0.13	0.10	0.09
MnO	%	0.08	0.07	0.05
Cr_2O_3	%	0.03	0.02	0.02
V_2O_5	%	0.02	0.02	0.02
LOI	%	7.93	7.37	6.01

The difference between each set of repeat gold assays, particularly for Composite 7, indicates the presence of coarse, free gold. The gold assays were close to the expected values for Composites 8 and 9, but were somewhat lower than the expected value for Composite 7 (see Table 18 for the expected values). The arsenic assays for Composites 8 and 9 were similar to those for Composite 5 (see Table 13); given that these three samples were either completely or largely made up of material from deeper in the orebody, this suggests that there may be an increase in arsenic-containing minerals with depth.

The results of the gravity separation tests, for both the new composites, and the additional tests conducted on samples from the 2013 program, are shown in Table 21.

Table 21 2015 Testwork Gravity Separation Test Results

Item	Unit	Composite 7	Composite 8	Composite 9	Composite 1	Composite 2	Composite 5
Gravity Concentrate Weight	%	0.21	0.10	0.15	0.09	0.12	0.13
Gravity Concentrate Au Assay	g/t	2,083	5,427	2,017	14,464	18,836	7,341
Gravity Tailings Au Assay	g/t	1.08	1.03	2.55	3.81	3.00	3.71
Gravity Au Recovery	%	80.5	83.7	54.6	77.0	88.4	71.9
Calculated Head Au Assay	g/t	5.51	6.32	5.61	16.5	25.9	13.2
Measured Head Au Assay	g/t	11.8	3.80	6.66	18.6	22.7	14.9

The gravity recovery tests on the new composites gave similarly high levels of gravity recoverable gold to the 2013 samples, although Composite 9 reported a lower value, similar to that reported for Composite 5 in 2013 (see Table 16). These two composites consisted solely (Composite 9) or largely (Composite 5) of material from 24 L, suggesting that the amount of gravity recoverable gold may decrease with depth. However, this was not a consistent trend, as Composite 8 and Composite 4 were also both solely of 24 L material, and both reported high gravity gold recoveries.

The level of agreement between the calculated and assayed head grades was good for Composites 9, 1, 2, and 3, but was relatively poor for Composites 7 and 8.

The results of the cyanidation tests conducted on the new composites are shown in Table 22.

Table 22 2015 Testwork Cyanidation Test Results

Item	Unit	Composite 7			Composite 8			Composite 9		
Grind Size P_{80}	µm	107	107	75	102	102	81	108	108	78
Residence Time	hr	16	24	16	16	24	16	16	24	16
CN Recovery	%	75.3	76.0	69.9	72.8	73.9	72.0	80.1	84.1	83.8
Gravity + CN Recovery	%	95.2	95.3	94.1	95.6	95.7	95.4	91.0	92.8	92.6
CN Residue Au Grade	g/t	0.27	0.25	0.34	0.29	0.27	0.28	0.51	0.40	0.42
Calculated CN Head Au	g/t	1.07	1.04	1.13	1.05	1.04	1.00	2.56	2.51	2.59
NaCN Consumption	kg/t	0.14	0.21	0.34	0.14	0.14	0.26	0.09	0.16	0.30
Lime Consumption	kg/t	0.65	0.71	0.70	0.71	0.51	0.70	0.78	0.70	0.73

These results show high levels of gold recovery achieved both at the coarser grind size and at the reduced residence times. The results for Composites 7 and 8 show negligible differences over the range of grind sizes and leach times tested, as evidenced by the solid tails grades. However, for

Composite 9, the coarser grind size (80% -108 µm) and lower residence time (16 hours) resulted in a solid tails slightly higher (0.51 g/t gold) than those for the longer residence time at the same grind size (0.40 g/t gold) or for the finer grind and same residence time (0.42 g/t gold).

The results of the tests conducted on the samples from the 2013 testwork program are shown in Table 23, together with the relevant results (i.e., finer grind size, 48-hour residence time) from the 2013 program.

Table 23 2015 Testwork Cyanidation of 2013 Composites Test Results

Item	Unit	Composite 1		Composite 2		Composite 4		Composite 5			
Grind Size P_{80}	µm	81	110	82	112	79	154	74	158	103	73
Residence Time	hr	48	16	48	16	48	16	48	16	16	24
Gravity Recovery	%	78.4	77.0	78.9	88.4	76.6	76.6	51.8	51.8	71.9	0
CN Recovery	%	88.3	92.9	94.9	83.3	86.1	74.6	90.5	79.0	84.4	95.0
Gravity + CN Recovery	%	97.5	98.4	98.9	98.1	96.7	94.1	95.4	89.9	95.6	95.0
CN Residue Au Grade	g/t	0.37	0.27	0.28	0.50	0.40	0.65	0.54	1.30	0.58	0.49
Calculated CN Head Au	g/t	3.15	3.80	5.49	2.99	2.88	2.56	5.68	6.19	3.72	9.80
NaCN Consumption	kg/t	0.50	0.18	0.75	0.21	0.25	0.14	0.34	0.12	0.25	0.29
Lime Consumption	kg/t	0.40	0.62	0.28	0.60	0.53	1.11	0.48	1.04	0.66	0.85

These results indicate the following:

- The results for Composites 1, 2, and 5 indicate that the combination of a coarser grind (to the target of 80% -110 µm) together with the reduced residence time (from 48 to 16 hours) had no significant impact on overall recovery.
- However, the tests on Composite 4 and 5 using a coarser grind size (80% -150 µm) did result in a decrease in recovery, although these tests were conducted at a 16-hour residence time.
- The test using Composite 5 without a gravity separation stage first produced a similar overall recovery to the equivalent test with gravity separation. This indicates that even if the gravity separation stage in the plant is not as efficient as in the laboratory, high overall recoveries can still be expected.

The results of the preg-robbing tests are summarized in Table 24.

Table 24 2015 Testwork Preg-Robbing Test Results

Item	Unit	Composite 7	Composite 8	Composite 9
Au absorbed after				
1 hr	%	10.4	10.4	2.2
3 hr	%	3.1	13.0	-8.7
6 hr	%	-5.6	8.6	-17.5
24 hr	%	-30.9	-31.9	-31.6

As in the 2013 testwork, these results indicate that all samples exhibited some preg-robbing behaviour initially, but that for all samples, this behaviour had reduced or disappeared after 24 hours.

The results of the settling testwork are shown in Table 25.

Table 25 2015 Testwork Composite 7 Settling Test Results

Item	Unit	P_{80} 75 μm	P_{80} 110 μm
Final underflow solid content	% w/w	63	65
Thickener underflow unit area	$m^2/t/d$	0.06	0.05
Thickener hydraulic unit area	$m^2/t/d$	0.01	0.01
Initial settling rate	$m^3/m^2/d$	445	498

13.5 Predictive Metallurgy

The metallurgical testwork programs conducted on ore from the West Reef have all indicated that the ore is free milling, with a relatively high gravity-recoverable component, and demonstrating a degree, although relatively low, of preg-robbing behaviour. Processing using a carbon-in-leach configured plant, rather than carbon-in-pulp, is indicated.

The latest testwork, seeking to optimize some key operational parameters, has indicated that:

- A grind size of up to 80% -110 μm is acceptable
- A leach residence time of 16 hours is sufficient at a grind size of 80% -75 μm, however for a coarser grind size of 80% -110 μm, a leach residence time of 24 hours is probably required

The leach testwork data from the three programs has been combined to yield a recovery-head grade relationship for use in mine optimization and for the prediction of plant behaviour. Of all of the testwork results reported above, only those at the coarse grind size (i.e., 80% -150 μm) have been excluded.

Figure 37 shows this data, together with a polynomial line of best fit. At the reserve head grade of 14.0 g/t gold, this regression line gives a recovery of 97.3%. Given the likelihood that plant performance may not be as efficient as under laboratory conditions, this figure has been discounted to 96.0%, in which case the regression line with head grade is scaled to the following:

$$\text{Recovery} = 0.0003 * Au^3 - 0.028 * Au^2 + 0.87 * Au + 87.9$$

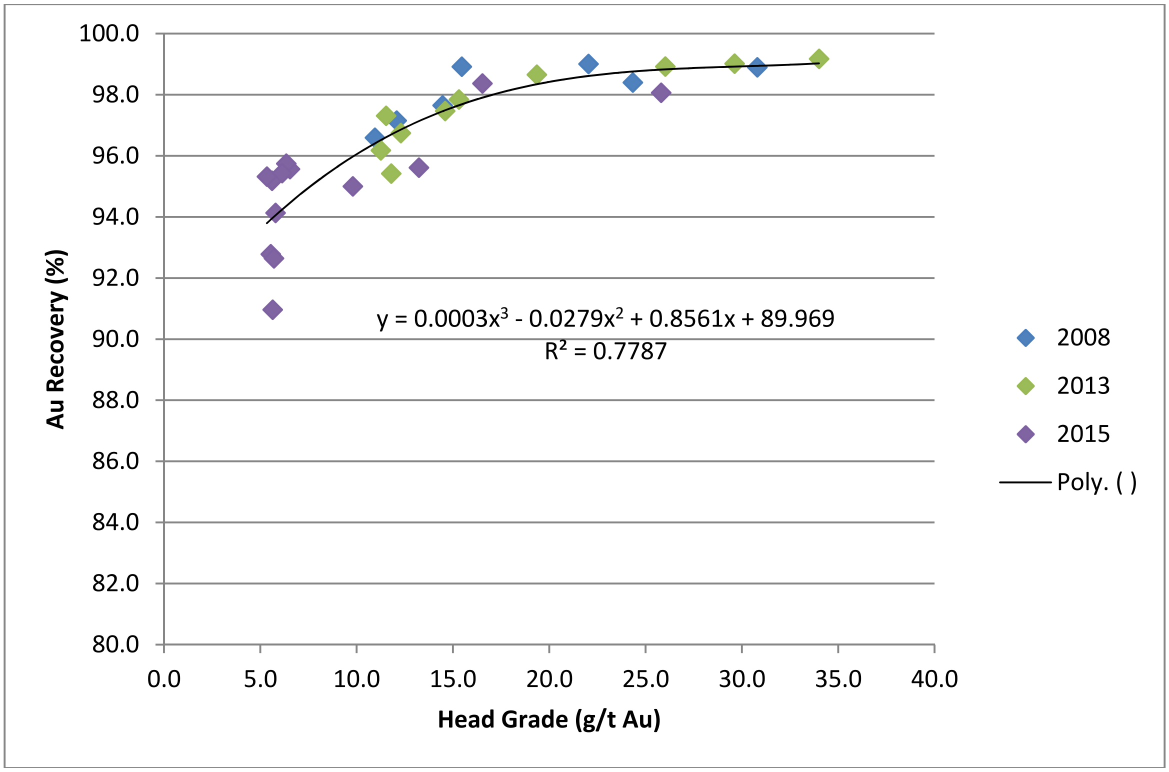


Figure 37 Recovery – Head Grade Relationship

14 Mineral Resources

14.1 Open Pit

14.1.1 Introduction

The Mineral Resource Statements presented herein represent the latest and most up to date Mineral Resource evaluation prepared for the various GSBPL Projects in accordance with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserves Best Practices" guidelines. They are reported in accordance with the Canadian Securities Administrators' National Instrument 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.

The Mineral Resource models for Chujah, Dumase, Ablifa, Prestea South and Mampon were largely prepared by SRK, with the exception of the Buesichem model, which was prepared by GSR's consultant Mr. William Tanaka. The Prestea underground resources for Main Reef, Main Reef Footwall and the shaft pillar were prepared by Dr, John Arthur (CEng MIMMM, CGeil FGS) who is an appropriate "independent qualified person" as this term is defined in National Instrument 43-101. The West Reef Resource estimation was completed by Golden Star Resources Geologists under the guidance and direction of Mr. S. Mitchel Wasel, Vice President Exploration and QP for Mineral Resources for the company. The complete resource estimation work was prepared under the direction of Mr. S. Mitchel Wasel, Vice President Exploration for Golden Star Resources. The effective date of the resource statements is 31 December 2017. In many cases, the original block grade modelling was carried out over a period between 2004 and 2017 and the updated Mineral Resource estimates are based on updated cut-off grade calculations and December 2017 topography and depletion updates.

This section describes the resource estimation methodology and summarizes the key assumptions considered by GSR and it's consultants

The drillhole databases used to estimate the GSBPL Mineral Resources is sufficiently reliable to interpret with confidence the boundaries for gold mineralization at GSBPL and that the assay data are sufficiently reliable to support Mineral Resource estimation.

Gemcom or Surpac Software ("GEMS®") was used to construct the geological solids, prepare assay data for geostatistical analysis, construct the block model and tabulate Mineral Resources. The Geostatistical Software Library ("GSLib") family of software and Isatis were used for geostatistical analysis, variography and grade interpolation.

14.1.2 Resource Estimation Procedures

GSR's Mineral Resource estimates are derived using a combination of DD and RC data, trench and grab sample data are not used for grade interpolation although it is sometimes used to assist in the original orebody modelling. The geological models used as the basis for the volumetrics of the Mineral Resource estimate were produced by GSR under the supervision of Mr. S. Mitchel Wasel. High grade data capping is performed where deemed appropriate for individual domains and the grade interpolation is typically carried out using ordinary kriging coupled with a neighbourhood analysis as a check on the quality of the kriging interpolation.

Topographic data comprises total station survey information along drill and exploration lines in conjunction with drill collars. The data is then combined with the regional topographic survey produced by the Ghanaian survey.

Geological wireframe models are typically produced by GSR using the GEMS or Surpac software packages. In the field the ore/waste contacts are defined by a combination of geological and structural factors and it was found to be more appropriate to model the contact initially using grade and to then use the structural model to subsequently define individual domains within the wireframe model.

For each of the GSBPL deposits, individual interpolation domains have been identified based on the main structural features identified from face and bench mapping during the mining process where applicable. At Chujah, for example, three main domains are delineated from west to east, the MCZ, the Fault Zone and the Footwall Splay domain respectively.

Deposit modelling has been typically carried out using a 1 g/t Au cut-off grade to define the contacts, but this grade has not been strictly adhered to and there are intervals within the deposit models with grades lower than 1 g/t Au. This approach is appropriate, given the complex nature of the deposits and the close spacing between zones of high grade mineralisation which would preclude small scale selective mining. The inclusion of narrow bands of waste material in the model is providing a degree of planned internal dilution.

The resource evaluation methodology involved the following procedures:

- Database compilation and verification;
- Construction of wireframe models for the boundaries of the gold mineralization;
- Definition of resource domains;
- Data conditioning (compositing and capping) for geostatistical analysis and variography;
- Block modelling and grade interpolation;
- Resource classification and validation;
- Assessment of "reasonable prospects for economic extraction" and selection of Appropriate cut-off grades; and
- Preparation of the Mineral Resource Statement.

14.1.3 Specific Gravity Data

Table 26 summarises the density data collected during the 2005 drilling campaign at Mampon. The number of samples available for all three oxidation domains is considered appropriate to describe the density of the material.

Table 26 Mampon and Abronye density data, 2005 drilling programme

Oxidation Domain	No. Samples	Specific Gravity (t/m^3)
Oxide	255	1.85
Transition	269	2.07
Fresh	169	2.6

GSR has continued to carry out density determination work on the core samples taken from Chujah exploration drilling. The results from drilling have not led to any significant changes in the density values used for the current Mineral Resource statement. The oxide density used is 1.9 t/m^3, transition density is 2.20 t/m^3 and the density is 2.69 t/m^3 for fresh material.

At Ablifa, test work has been completed on the determination of the density of rocks within these different material types and global average values are accepted for each material type, regardless of the lithology involved. A density of 1.90 t/m^3 has been applied to oxide rock; a density of 2.20 t/m^3 is applied to transition material, whereas a density of 2.63 t/m^3 has been used to describe fresh rock.

At Beta Boundary (Table 27), work carried out by GSR during the 2003 and 2004 drilling campaigns involved analysis on a total of 540 samples. All of the sampled holes formed part of the metallurgical testwork data and were drilled on the Plant North and Beta Boundary deposits.

Table 27 Summary SG data for Beta Boundary deposits (2004 sampling)

Oxidation State	Specific Gravity (t/m^3)
Oxide	1.8
Transition	2.2
Fresh	2.7

At Bondaye and Tuapim, values used in producing the current Mineral Resource estimate are 1.70 t/m^3 for oxide, 2.13 t/m^3 for transition and 2.74 t/m^3 for fresh.

The values in Table 26 to Table 27 are those used by GSR and are based on its experience in mining and production at the Bogoso and Prestea concessions. There is some evidence for higher SG values in certain lithologies and mineralization styles. In particular, the transition zone samples in some areas show SG values averaging 2.4 to 2.5 t/m^3. The current SG values used in the tonnage estimates are appropriate, given the mineralization style and, in particular, the deeper weathering experienced within the mineralization haloes leads to a lowering of the density around these areas relative to the surrounding country rock.

During 2012, GSR carried out additional density testwork on samples from Buesichem and Chujah/Dumasi. GSR has also taken specific gravity samples from Beta Boundary.

Table 28 Summary SG data for Chujah/Dumasi, Buesichem and Beta Boundary

Oxidation State	Chujah/Dumasi		Buesichem		Beta Boundary	
	SG (t/m^3)	*Number samples*	*SG (t/m^3)*	*Number samples*	*SG (t/m^3)*	*Number samples*
Oxide	2.01	3	1.89	2	1.76	133
Transition	2.49	72	2.49	195	2.27	136
Fresh	2.70	477	2.78	644	2.74	334

The 2004 values were used in the block models, for the Mineral Resource estimate, based on 3D oxidation and transition surfaces which were produced by GSR.

The specific gravity values derived from the 2004 programme are used by GSR. Confidence in the specific gravity values per oxidation state is further refined through experience in mining and production at Bogoso, Buesichem and Plant North. GSR note that there is some evidence for higher specific gravity values in certain lithologies and mineralization styles. In particular, the transition zone samples in some areas show specific gravity values averaging between 2.4 and 2.5t/m^3. The current specific gravity values used in the tonnage estimates are appropriate given the mineralization style and in particular the deeper weathering experienced within the mineralization halos leads to a lowering of the density around these areas relative to the surrounding country rock.

14.1.4 Data Cutting and Filtering

Data cutting of outlier high grades is not carried out unless deemed necessary for individual domains in the various deposits present on the GSBPL licence.

At Mampon/Abronye it was not considered necessary to apply any high grade cutting.

At Chujah, Dumasi and Chujah South, an initial review of the data indicated the presence of anomalous high grade values. However, it was decided that the domaining and kriging process was sufficient to reduce the impact of these high grades on the final block grade estimate.

At Ablifa, the highest grade recorded in the 3 m composite data is 50.8 g/t Au, however the Coefficient of Variance ("CV") of the data is relatively low at 1.3 and data cutting is not considered necessary.

At Buesichem, parallel grade estimates were undertaken capping the composites above 19 and 14 g/t Au, respectively. The resulting estimates from the capped data confirm that there is very little risk attached to the uncapped estimate and no grade capping is required.

At Beta Boundary, investigation of the data indicated that, although there are some significant high grades in both the Main Reef ("MR") and East Reef ("ER") domains, it is not appropriate to apply a top cut to this data. High grades tend to occur in discrete areas.

At Tuapim, the three high grade outliers in the fresh domain with values above 20 g/t Au were cut from the data which brings the fresh data in line with the oxide and transition domains. At Bondaye, the fresh domain high grades are cut at 20 g/t Au and the mean grade drops to 1.99 g/t Au and the CV drops to 1.2 which is again, in line with the results from the oxide and transition domains.

Statistical Analysis in Table 29 to Table 35 summarize the statistics for the various domains, drill methods and oxidation states identified throughout the GSBPL deposits. The DD samples are predominantly from the fresh horizon and the RAB mostly from the oxide horizon while the RC drilling has effectively sampled all three horizons. The DD and RC datasets generally show close correlation.

In all cases, RAB data was excluded from the final Mineral Resource estimate as it was felt the quality of sampling from this method is not consistent with that achieved from the DD and RC methods.

Compositing of sample grades was generally carried out using a 2 m composite with composite start and end coordinates restricted by the wireframe boundaries. The exception to this was at Ablifa where a 3 m composite was used.

The Bogoso North database contains information from a large number of historic sampling campaigns, including underground adit channel samples and open pit rip line samples. The quality of the sample collection and assaying for the adit samples cannot be adequately verified by GSR and as such all estimates made using the adit samples in the south of the Bogoso North deposit (Marlu) are classified as inferred and will remain so until adequate drill samples can be collected. The north south divide within the deposit is between y=30,500 and y=30,600 (BGL Grid). South of this the sampling is dominated by Adit channel samples and to the north the RC and DD sampling is predominant.

At Buesichem, historically the deposit had been subdivided into a northern zone and a main zone. The area further to the south, but contiguous with the main zone, was referred to as Beposo, representing a third possible domain. The identification of three zones is retained in the 2 m composite file as is the oxidation state. Statistical analysis and comparison by domain demonstrates no material difference between the three historical domains. Similarly, there is no indication of material difference in grade by oxidation state. The population characteristics of the composites are sufficiently similar to justify inclusion of all composites into a single domain

The Beta Boundary deposit can be sub-divided into three mineralised domains.

- The WR does not actually form a continuous reef, but is rather a domain in the hangingwall of the main deposit where patchy occurrences of high grade material are recorded in drill sections. These intersections generally cannot be traced between adjacent sections and cannot be modelled down dip.
- The MR is a 4 km long continuous body of quartz and sulphide mineralization. This unit is generally narrow (<10 m), but is continuous over the entire strike length when modelled at a 1 g/t Au cut off. The current modelling has made use of the underground data available for modelling purposes only and this information is not used for the grade interpolation.
- The ER is a zone in the immediate footwall of the MR. The grade of the ER is generally lower than that of the MR. In structure, the ER is made up of a large number of individual deposits generally not greater than 200 to 300 m in length, and frequently shorter. These deposits form a linear “swarm” and occur parallel to each other to produce a broad zone of mineralization generally >20 m across. The frequency and thickness of these deposits increases in areas where the MR undergoes right hand flexures in its structure. This indicates some form of structural control on the ER mineralization, possibly due to dextral movement along the structural plane which hosts the MR.

At Beta Boundary, the JCI underground drilling data was excluded from the grade interpolation, although it was used for deposit modelling.

At Bondaye and Tuapim, the DD data forms a very small component of the total available data. It was decided to produce semi-variograms from the RC data only in order to reduce the variance and allow a better definition of the models to be produced.

Table 29 Summary statistics for the Mampon and Abronye Domains*

Deposit	Method	Oxidation	No. Comp	Max (g/t)	Mean (g/t)	Variance	CV
Mampon	DD	OX	120	39.0	3.83	38	1.6
		TR	89	35.9	4.36	51.3	1.6
		FR	327	80.1	4.10	75.2	2.1
	RC	OX	307	51.0	3.66	39.3	1.7
		TR	139	67.1	5.89	136.2	2.0
		FR	224	45.9	3.89	35.7	1.5
	RAB	OX	181	16.2	1.46	3.1	1.2
		TR	3	1.3	0.95	0.1	0.3
		FR	-	-	-	-	-
		Total	**1390**	**80.1**	**3.81**	**53.4**	**1.9**
Abronye	DD	OX	-	-	-	-	-
		TR	8	0.1	0.03	0.0	-
		FR	143	2.6	0.27	0.2	1.6
	RAB	OX	265	12.5	1.39	2.9	1.2
		TR	66	23.4	1.73	8.4	1.7
		FR	57	3.5	1.42	0.1	0.6
		Total	**539**	**23.4**	**1.13**	**2.8**	**1.5**

** Highlighted statistics are considered invalid due to low number of samples or inappropriate drilling technique*

Table 30 Summary statistics for the Bogoso North/Marlu resource domains

Domain	Method	Composites	Max (g/t)	Mean (g/t)	Variance	CV
MCZ-South	ADIT	424	26.3	3.36	14.7	1.1
	DDH	68	24.2	3.89	18.8	1.1
	RC	5	1.9	1.0	0.5	0.7
MCZ North	ADIT	-	-	-	-	-
	DDH	265	26.5	4.04	16.2	1.0
	RC	314	26.5	4.23	14.2	0.9
LGZ South	ADIT	353	10.1	0.74	1.14	1.4
	DDH	57	5.28	0.74	0.73	1.2
	RC	28	6.15	1.19	2.6	1.4
LGZ North	ADIT	-	-	-	-	-
	DDH	489	12.5	0.92	1.46	0.9
	RC	460	6.8	0.74	0.56	1.0

Table 31 Summary statistics for the Chujah and Dumasi resource domains

Deposit	Domain	Oxidation	No. Comp	Max (g/t)	Mean (g/t)	Variance	CV
Chujah	CW-HGZ	OX	6	6.1	4.00	2.3	0.4
		TR	130	46.4	5.59	28.8	1.0
		FR	986	57.1	5.28	22.8	0.9
	CW-MCZ	OX	1508	16	1.71	2.2	0.9
		TR	443	53.3	2.19	10.3	1.5
		FR	2643	32	1.91	4.3	1.1
	CE	OX	3867	93.3	2.01	8.7	1.5
		TR	1055	25.2	2.31	7.8	1.2
		FR	1856	68.1	2.24	9.1	1.4
Dumasi	DW-HGZ	OX	45	18.1	3.14	10.3	1.0
		TR	261	24.0	3.63	8.2	0.8
		FR	955	33.0	3.82	8.1	0.8
	DW-MCZ	OX	572	20.4	1.46	3.3	1.2
		TR	750	14.2	1.91	2.5	0.8
		FR	2632	83.9	1.90	5.3	1.2
	DE	OX	1163	45.2	1.86	6.0	1.3
		TR	194	12.5	1.84	2.8	0.9
		FR	546	21.1	2.26	4.2	0.9
	DN	OX+TR+FR	506	13.1	1.74	2.26	0.9

Table 32 Summary statistics from 3 m composites at Ablifa

Source	Count	Min	Max	Mean	Std Dev	Variance	CV
RAB	14,061	0	18.51	0.91	1.12	1.27	1.23
RC/DD	7,169	0	50.84	0.97	1.40	1.95	1.43
Total	**21,230**	**0**	**50.84**	**0.93**	**1.22**	**1.50**	**1.31**

Table 33 Summary statistics for the Buesichem resource domains

Domain	Campaign	No. Comp	Max (g/t)	Mean (g/t)	Variance	CV
Main	Exploration	3549	28.16	2.83	4.8	0.86
North	Exploration	272	8.42	1.92	1.8	0.75
Beposo	Exploration	1105	11.55	1.61	2.5	0.82
Main	Grade Control	5615	34.17	2.72	5.7	0.86

Table 34 Summary Statistics for Beta Boundary Domains

Domain	Sample Data	No Comp	Max (Au g/t)	Mean (Au g/t)	Variance	CV
MR Oxide	DD	46	32.4	2.73	23.5	1.8
	RC	151	39.1	2.96	30.1	1.9
	RAB	477	46.4	2.36	15.9	1.7
MR Trans	DD	32	10.02	2.4	3.7	0.8
	RC	270	49.6	3.1	2.6	1.7
	RAB	51	9.5	2.14	4.9	1.0
MR Fresh	DD	97	22.8	3.3	16.3	1.2
	RC	525	133	3.01	51	2.4
ER Oxide	DD	141	12.3	1.76	2.4	0.9
	RC	850	18.8	1.51	2.9	1.1
	RAB	1416	24.9	1.61	4.0	1.2
ER Trans	DD	85	7.2	1.56	1.8	0.9
	RC	551	81.8	1.62	14.3	2.3
	RAB	39	3.2	1.3	0.7	0.7
ER Fresh	DD	200	18	1.35	2.6	1.2
	RC	1685	89.9	1.45	9.1	2.1

Table 35 Summary 2 m composite statistics for Bondaye and Tuapim

Deposit	Drill Method	No. Holes	No. 2m Comps	Max (g/t)	Mean (g/t)	Variance	CV
Bondaye	RC	182	1288	63.3	2.17	14	1.7
	DD	13	93	56.6	2.68	38	2.4
	RAB	737	629	40.0	1.89	13	1.9
Tuapim	RC	198	1052	75.8	2.41	15	1.6
	DD	10	76	13.0	1.60	3	1.1
	RAB	591	394	18.1	1.94	6	1.3

14.1.5 Block Model Grade Interpolation

Introduction

Generally, the grade interpolation was carried out after domaining the composite data into various possible weathering domain combinations. In some cases, the oxide and transition was treated as a single domain (Bogoso North). However, it was more common where data was sparse or the statistics allowed, for the data from all three domains to be combined into a single dataset (Mampon/Abronye, Chujah/Dumasi, Ablifa, Buesichem, Beta Boundary, Bondaye, Tuapim). Grade interpolation was carried out by ordinary kriging in all cases. The composites were sourced from DD and RC drilling assays only, except in the case of Ablifa where the RAB data was combined with the RC results. In most cases, the semi-variogram analysis was carried out after data cutting and conversion of the resulting composite data to a normalized gaussian model through the process of gaussian anamorphosis. Results from the Gaussian semi-variograms were back-transformed to a normal data space and grade estimates were carried out using the untransformed composite files.

Kriging was carried out in two phases. The first search was based on the results of the semi-variogram modelling and, in those cases where an anisotropic variogram could be modelled, the search ellipse was structured to honour the anisotropy. A second search was then carried out using a much larger radius than implied by the semi-variogram model range. The second search was carried out to fill remaining blocks not already interpolated in the first search. For the purpose of applying grade classification categories, the blocks in the wider second search were generally classified as Inferred.

Mampon/Abronye and Opon

Directional semi-variograms produced in the plane of the mineralized zone indicated a directional anisotropy with a possible plunge to the north-west Table 36. However, the semi-variograms modeling was inconclusive in this regard and it was decided that the along strike direction should be considered the primary axis with the secondary axis in the down dip direction to the west.

Table 36 Semi-variogram modelling results for the Mampon deposit

Structure	Variance	Range 1	Range 2	Range 3
nugget (C_0)	28.6			
spherical (C_1)	15.14	17		
spherical (C_2)	9.63	115	55	
Orientation	Dip dir°/dip	000°/00	270°/65W	

At Aboronye (Table 37 to Table 40), the original anisotropic variograms produced in the plane of the deposit exhibited poor structure with a high nugget variance and poorly defined sill. The current model has a range of 125 m, but this is poorly defined and, for this reason, it was decided to restrict the search for the kriging.

Table 37 Semi-variogram modelling results for the Abronye deposit

Structure	Variance	Range 1	Range 2	Range 3
nugget (C_0)	1.75			
spherical (C_1)	1.08	125m	95m	
Orientation	Dip dir°/dip	330°/00	240°/90	

Table 38 Mampon/Abronye block model parameters

Direction	Origin	No of cells	Size of cells
X	14750	100	12.5
Y	61350	70	25
Z	5200	80	8

Table 39 Mampon combined oxide/transition/fresh kriging profile

Direction	Initial Search (m)	Infill Search (m)	Search Direction	Search Dip
X	120	300	000°	00°
Y	60	300	270°	65°W
Z	20	30	90°	25°E
Min Samples	5	2		
Max Samples	36	36		
Descritization	6x12x3	6x12x3		

Table 40 Abronye combined oxide/transition/fresh kriging profile

Direction	Initial Search (m)	Infill Search (m)	Search Direction	Search Dip
X	60	300	330°	00°
Y	30	300	240°	90°
Z	10	30	060°	00°
Min Samples	5	2		
Max Samples	36	36		
Descritization	6x12x3	6x12x3		

Bogoso North

The only domain which produced a meaningful downhole semi-variogram was the MCZ sulphide in the Marlu domain. The range was calculated at 18 m and, although hole effect is evident beyond this distance, the early part of the semi-variogram exhibits a reasonable structure.

In general, the nugget variance is high for all the semi-variograms and confidence in these is low. In all domains, most of the directional variograms exhibit pure nugget effect, reflecting the sparseness of data other than in the along strike direction (Table 41).

Table 41 Bogoso North Semivariogram Parameters

Data	SU South	SU North	OX/TR
Ore Strike	010	340	010/340
Deposit Dip	55W	35W	55W/35W
Pitch	8N	11N	8N/11N
½ Angle	25	25	25
Lag (m)	20	23	40
Co	10.7	6.5	3
C1	4.7	8.6	1.56
a1	35.7	32	66

Table 42 details the parameters used for the grade interpolation. Rotation is defined in the Gemcom™ rotation system and specifies the orientation of the primary axis. The anisotropy details the search radius in the three principle directions. A total of three passes were made with increasing search radii in order to fill all blocks with grade, however, only the first pass was based on the semi-variogram models discussed in the previous section. The wider searches were effectively multiples of the primary search distances and all blocks filled using these were classified as inferred.

Table 42 Bogoso North – Gemcom™ Search ellipse Parameters

Domain	Zone	Rotation			Anisotropy 1 (m)			Anisotropy 2 (m)			Anisotropy 3 (m)		
		X	Y	Z	X	Y	Z	X	Y	Z	X	Y	Z
MCZSU	South	86	-55	8	35	19	18	60	30	20	200	100	50
	North	127	33	11	32	21	18	60	30	20	200	100	50
MCZOX/TR	South	86	-55	8	60	40	20	90	60	30	200	100	50
	North	127	33	11	60	40	20	90	60	30	200	100	50

Grade interpolation was carried out using Ordinary Kriging ("OK") with a minimum of five and maximum of 36 assays for the first two passes and a minimum of two assays for the final long distance run. A discretisation matrix of 3 x 3 x 3 was utilized. The interpolation was carried out on untransformed composite values and no additional high-grade cutting was carried out other than that already described previously.

Chujah

Grade interpolation was carried out separately for five domains with the production of semi-variograms and ordinary kriging estimates for gold grade, as shown in Figure 38. Semi-variograms were oriented using the general dip and strike of the deposit in the individual domains as modeled by GSR. Where sufficient information was available, directional semi-variograms and variogram maps were produced in the plane of the deposit as defined by the wireframe models. These plots helped to define the potential for modeling plunging structures within the main deposit thus improving the quality of the overall grade estimates.

Initially, semi-variograms were produced for the individual oxidation domains. However, the lack of data points in the oxide and transition domains precluded the production of reliable semi-variogram models and, as the statistics indicated little variation in grade between the oxide states it was decided to combine the data.

Variogram structures are generally well developed in the downhole orientation which directly compares pairs of composite values at increasing distances along the hole trace with ranges modeled at between 6 to 20 m reflecting the average thickness of individual zones within the domains and relative continuity of the grade distribution at local scales. The reliability of the downhole variograms gave confidence to using the nugget variance (Co) value from these semi-variograms when modeling the directional semi-variograms.

The variogram structures in the principle direction in the plane of the deposit are poorly developed and the interpretations show only minimal evidence for the range of grade continuity in each direction. Where possible, the directional semi-variograms were produced at closely spaced intervals which allowed the delineation of plunging structures within the plane of the deposit. Results of modeling the experimental semi-variograms are contained in Table 43 to Table 47.

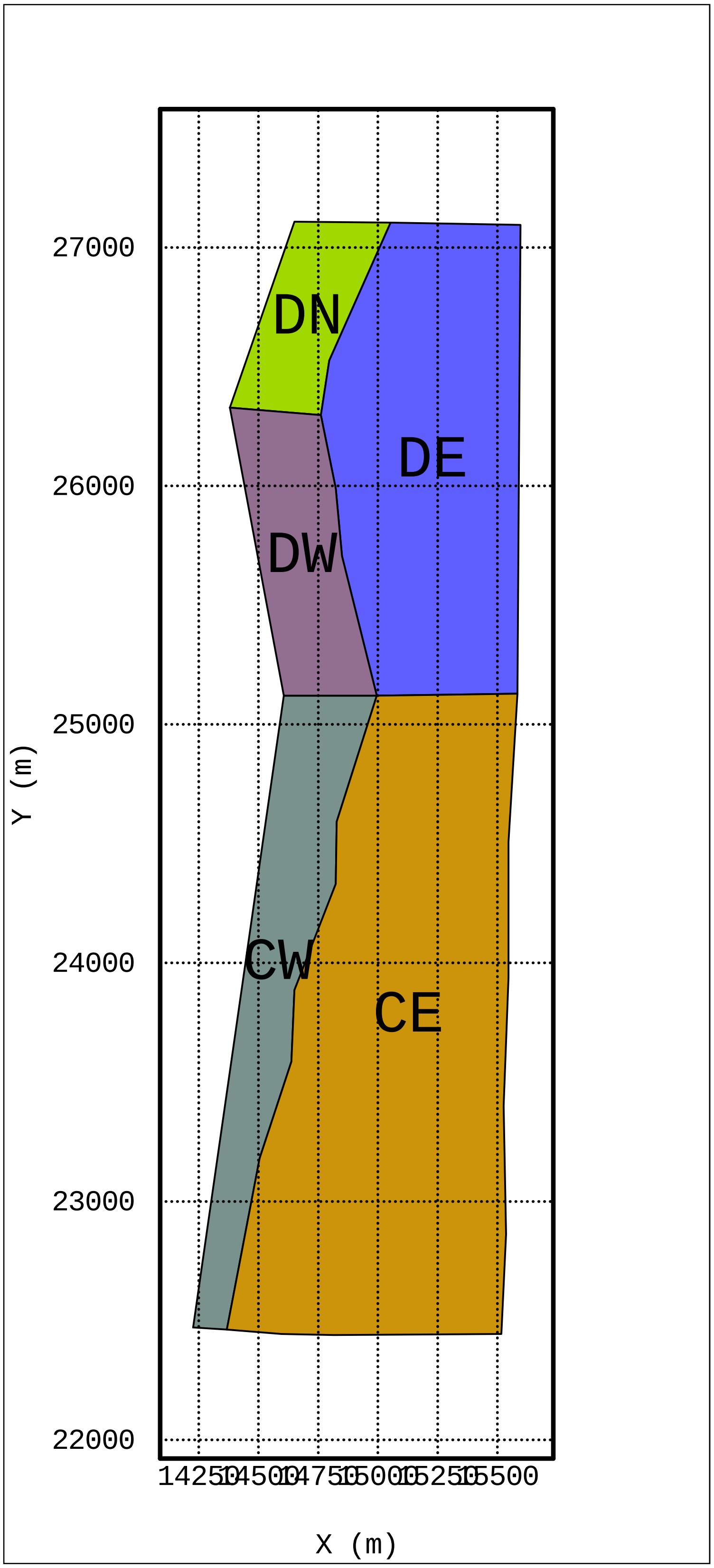


Figure 38 Geostatistical domains covering the Chujah and Dumasi deposit

Table 43 Semi-variogram modelling results for the Chujah Dumasi CW Domain

Structure	Variance	Range 1	Range 2	Range 3
nugget (C_0)	1.42			
spherical (C_1)	2.34	45.7		
spherical (C_2)	0.42	150	120	
spherical (C_3)				
Orientation		0° to 010	0° to 280	

Table 44 Semi-variogram modelling results for the Chujah Dumasi DW Domain

Structure	Variance	Range 1	Range 2	Range 3
nugget (C_0)	3.05			
spherical (C_1)	1.04	48 m		
spherical (C_2)	0.31	150 m	250 m	
spherical (C_3)				
Orientation		0° to 350	5° to 260	

Table 45 Semi-variogram modelling results for the Chujah Dumasi CE Domain

Structure	Variance	Range 1	Range 2	Range 3
nugget (C_0)	0.00			
spherical (C_1)	4.43	3 m		
spherical (C_2)	4.1	30 m	40 m	
spherical (C_3)				
Orientation		0° to 000	0° to 270	

Table 46 Semi-variogram modelling results for the Chujah Dumasi DE Domain

Structure	Variance	Range 1	Range 2	Range 3
nugget (C_0)	0.00			
spherical (C_1)	4.06	14 m		
spherical (C_2)	1.11	70 m	65 m	
spherical (C_3)				
Orientation		0° to 340	0° to 250	

Table 47 Semi-variogram modelling results for the Chujah Dumasi DN Domain

Structure	Variance	Range 1	Range 2	Range 3
nugget (C_0)	0.18			
spherical (C_1)	1.05	40 m	30 m	
spherical (C_2)	1.0	85 m	60 m	
spherical (C_3)				
Orientation		0° to 020	5° to 290	

Grade was calculated for individual blocks using ordinary kriging and a search radius generally greater than the modeled semi-variogram range. The block model was constructed using the parameters in Table 48. The block height has been increased from previous years from 6 to 8 m and the depth of the model has also increased to reflect the continuing deep drilling.

Table 48 Chujah Dumasi block model parameters

Direction	Origin	No of cells	Size of cells	Maximum
X	14,200	136	12.5 m	15,700
Y	22,400	192	25 m	27,200
Z	4,480	90	8 m	5,200

For all domains, the first pass interpolation used a minimum of 10 and maximum of 36 composites. The second search (wider search used to fill outlying blocks) utilized a minimum of two composites and a maximum of 36. Discretization was set at 6x12x3 m (xyz). Search radii used for the various domains were as follows:

- DW – 150x150x15 m, 500x300x50 m;
- CW – 175x150x15 m, 500x300x50 m;
- CE – 50x50x15 m, 300x300x50 m;
- DE – 75x75x15 m, 300x300x50 m; and
- DN – 100x75x10 m, 300x300x50 m.

Grade control data from the mining operation at Chujah South was provided by GSR and was modelled as a check against the variography produced from the exploration drilling. The majority of the data is from blast hole sampling with some RC, rip line and RAB drilling data also included. The majority of the sample points are either 1 or 4 m composites and the database was re-composited to a 2 m standard composite. The resulting semi-variograms show a similar trend and range to those produced from the exploration data with a principle direction roughly along strike at 010º and a range of 80 m, compared with a range of 95 m in the MCZ, 85 m in the FW Splay and 185 m in the Fault domain from the exploration data. The long range modelled in the fault data is not considered to be of a high confidence and the search parameters used for the kriging reflect this.

The conclusion from the grade control study is that the parameters modelled from the exploration data can be confirmed from the grade control data and therefore the exploration data semi-variograms can be considered a valid model for use in grade interpolation.

Ablifa

The experimental variograms suggested that the greatest range occurs in the strike-parallel direction (Table 49 and Table 50). Within the Ashanti trend, steeply plunging mineralised shoots have been observed, particularly well defined within the quartz vein hosted mineralization. The data available in the Ablifa area are predominantly present within the oxide domain (RAB drilling data), with the RCDI drilling providing the deeper data below the transitional zone. The observed anisotropy may be an artefact of the spatial distribution of the sample data.

Table 49 Ablifa semi-variogram modelling results

Structure	Variance	Range 1 (m)	Range 2 (m)	Range 3 (m)
nugget (C_0)	0.717			
spherical (C_1)	0.789	19	21	11
spherical (C_2)	0.206	125	60	45
Orientation		00° to 015°	50° to 285	

Table 50 Ablifa combined oxide/transition/fresh kriging profile

Direction	Initial Search (m)	Infill Search (m)	Search Direction	Search Dip
X	125	300	015°	00°
Y	60	300	285°	50°
Z	10	30	105°	00°
Min Samples	5	2		
Max Samples	36	36		
Descritization	6x12x3	6x12x3		

Buesichem

Global (360º) variograms were generated for both untransformed Au and LN Au to establish the nugget effect as well as permit a general sense of range for non-nugget sills (Table 51 and Table 52). Directional variograms for untransformed Au were then generated for the principal axes determined from interpretation of the variance contour maps. The relative sills were drawn directly from the model fitted to the global variogram.

Table 51 Semi-variogram modelling results for the Buesichem deposit

Structure	Variance	Range 1 (m)	Range 2 (m)	Range 3 (m)
nugget (C_0)	0.357			
spherical (C_1)	0.393	25	20	10
spherical (C_2)	0.250	55	55	20
Orientation		00° to 180°	30° to 270°	

Table 52 Buesichem combined oxide/transition/fresh kriging profile

Direction	Initial Search (m)	Infill Search (m)	Search Direction	Search Dip
X	60	120	000°	00°
Y	60	120	270°	30°
Z	20	40	090°	60°
Min Samples	4	4		
Max Samples	36	36		
Descritization	4x643	4x6x3		

Beta Boundary

The statistical analysis indicated that the data from the oxide, transition and sulphide domains could be combined for the purpose of grade interpolation. However, the MR and ER were treated as two separate domains for the purpose of semi-variogram modelling and grade interpolation.

Semi-variograms were produced after cutting the datasets to 12 g/t Au, this topcut was only applied for semi-variogram modelling and not for the final grade interpolation. Downhole semi-variograms exhibited good structure and allowed detailed modelling of the nugget variance, generally with a range of 12 m for both the MR and ER domains. Directional variograms were generally poor quality, although at MR there is evidence for a steeply plunging component. The along strike and down dip directions were those which produced the best quality variograms and are the ones modelled in Table 53. Table 54 summarises the block model parameters for the Beta Boundary model. This block model is contiguous with that for the Plant North deposit.

Table 53 Beta Boundary Semi-variogram model parameters

Domain	Variable	Variance	Range dir1 (m)	Range dir2 (m)
Main Reef 3m lag downhole (cut <12)	Co	0		
	C1	2.08	3	
	C2	2.7	12	
Main Reef 25m lag anisotropic (cut <12) 005 deg, 70W	Co	0		
	C1	4.17	35	
	C2	0.88	86	70
East Reef 3m lag downhole (cut <12)	Co	0.81		
	C1	0.37	6.2	
	C2	0.28	13.5	
East Reef 25m lag anisotropic (cut <12) 010 deg, 70W	Co	0.81		
	C1	0.4	30	10
	C2	0.63	130	100

Table 54 Beta Boundary block model parameters

Direction	Origin	No of cells	Size of cells	Maximum
X	11,200	168	12.5	13,300
Y	5,000	192	25	9,800
Z	4,600	108	6	5,248

Block grade interpolation was carried out using OK. The first pass was carried out using a search radius of 200 x 200 x 15 m for MR, ER and WR and a minimum number of 10 samples required for grade interpolation. Blocks not filled by kriging were assigned values based on the average of the kriged blocks after applying a high grade top cut to the block grades, the values used are as follows:

- MR – cut at 3.5 g/t Au, value applied = 1.84 g/t Au.
- ER – cut at 2.0 g/t Au, value applied = 1.23 g/t Au.

Bondaye/Tuapim

Semi-variograms were produced separately for Bondaye and Tuapim. Initially, only the RC data was used, data was cut to 20 g/t Au, which had the effect of bringing the statistics from the three weathering domains into line. As a result, the semi-variograms produced for the Mineral Resource estimate have used a combination of data from the oxide, transition and fresh domains.

At Tuapim, it was possible to model directional semi-variograms and the best direction was modeled in a direction of 239º, which represents a steeply plunging structure. However, the along strike direction could not be as reliably modelled, but it appears that the range in the perpendicular direction is very close to that seen in the 239º direction indicating an isotropic search radius is appropriate.

Table 55 summarises the semi-variogram models used for block grade interpolation. The models for both domains show very similar results, although the data variance for the Tuapim domain is higher. Ranges modelled are still low with a maximum range of only 40 m. Significant infill drilling would be required on at least 25 m spacing to allow the definition of any Measured Mineral Resource material at either of these deposits.

Table 55 Semi-variogram modelling results for Bondaye and Tuapim, 2007 models

Domain	Variable	Variance	Range dir1 (m)	Range dir2 (m)
Bondaye 2m lag downhole (<20)	Co	0.64		
	C1	0.98	4.7	
	C2	1.01	11.6	
Bondaye 12m lag isotropic (<20), 342° 80°W	Co	0.64		
	C1	0.697	3.3	
	C2	1.305	40	
Tuapim 2m lag downhole (<20)	Co	0.64		
	C1	0.70	3.2	
	C2	2.89	5.9	
Tuapim 12m lag anisotropic (<20), 003° 70°W	Co	0.64		
	C1	1.244	3.6	
	C2	1.24	40	

Table 56 and Table 57 summarise the parameters for the block models based on the block centroid of the lower left corner of the model. In both cases, block size was set to 12.5 m across strike (hole spacing approximately 25 m along lines); 25 m along strike (line spacing approximately 50 m) and 8 m vertically to match the planned mining bench height.

Table 56 Bondaye 2007 Block model parameters

Direction	Origin	No of cells	Size of cells
X	12,006.25	140	12.5
Y	3,512.5	120	25
Z	4,452	100	8

Table 57 Tuapim 2007 Block model parameters

Direction	Origin	No of cells	Size of cells
X	10,756.25	100	12.5
Y	12.5	200	25
Z	4,452	100	8

In addition to estimating the grade, the slope of regression and sum of positive weights were also estimated for each block in order to assess the quality of the final estimate and to aid in classification. The following parameters were used for the kriging operation:

- Search radius 1 (xyz) = 50x50x10 m
- Search radius 2 (xyz) = 200x200x30 m
- Minimum number of composites = 2
- Number of sectors = 4
- Maximum number of composites per sector = 5
- Discretization = 6x12x4 m
- Minimum distance between composites = 2 m

14.1.6 Mineral 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.

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 RAB drilling on sections spaced at 25 to 50 m.

Generally, for mineralization exhibiting good geological continuity investigated at an adequate spacing with reliable sampling information accurately located, blocks estimated during the first estimation run considering full variogram ranges can be classified in the Indicated Mineral Resource category within the meaning of the CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014). For those blocks, that the level of confidence is sufficient to allow appropriate application of technical and economic parameters to support mine planning and to allow evaluation of the economic viability of the deposit. Those blocks can be appropriately classified as Indicated Mineral Resource. Where Mineral Resources are categorised as Measured Resources, the level of drill information and the quality of the kriging estimate to be appropriate and generally is reserved for areas of ground where the average slope of regression from the kriging analysis exceeds 90%.

Conversely, blocks estimated during a subsequent search pass, considering search neighbourhoods set at beyond the variogram ranges, should be appropriately classified in the Inferred category because the confidence in the estimate is insufficient to allow for the meaningful application of technical and economic parameters or to enable an evaluation of economic viability.

14.2 Underground

The Mineral Resource Statement reported herein is presented in accordance with the guidelines of NI 43-101.

The GSR exploration team was responsible for the modelling portion of the resource estimate exercise, which included all topographic surfaces, weathering surfaces, and geological and grade wireframes. SRK Consulting (UK) Inc. (SRK) was commissioned to construct a mineral resource model with estimated gold grades for the Prestea underground deposit. A portion of that model, the West-Reef deposit, was subsequently updated internally in 2015 by GSR using the methodology established by SRK. A portion of the West Reef Resource was updated in 2017 using additional diamond drilling and raise/drift channel sampling data that was gathered in the northern, down dip portion of the WR deposit between 21 and 27 levels,. The methodology differs slightly but is based largely on the methodology outlined by SRK.

The methodology section of the estimate has been altered to show the variation where appropriate from the methodology outlined by SRK. The generation of the Mineral Resource Statement was conducted by GSR under the supervision of S. Mitchel Wasel, a Qualified Person pursuant to NI 43-101.

This section describes the mineral resource estimation methodology and summarizes 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 the generally accepted CIM *Estimation of Mineral Resource and Mineral Reserves Best Practices Guidelines* and are reported in accordance with 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.

14.2.1 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. Figure 39 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 of December 2005 to December 2017 led to the development of updated geological models for the MR, FR, 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 has 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 the grade and the distribution of the drilling.

The West Reef deposit which lies to the south of the Central shaft area has been worked historically (Figure 39). Production records and historic sampling identified this deposit as having good prospects for future exploitation as an underground deposit.

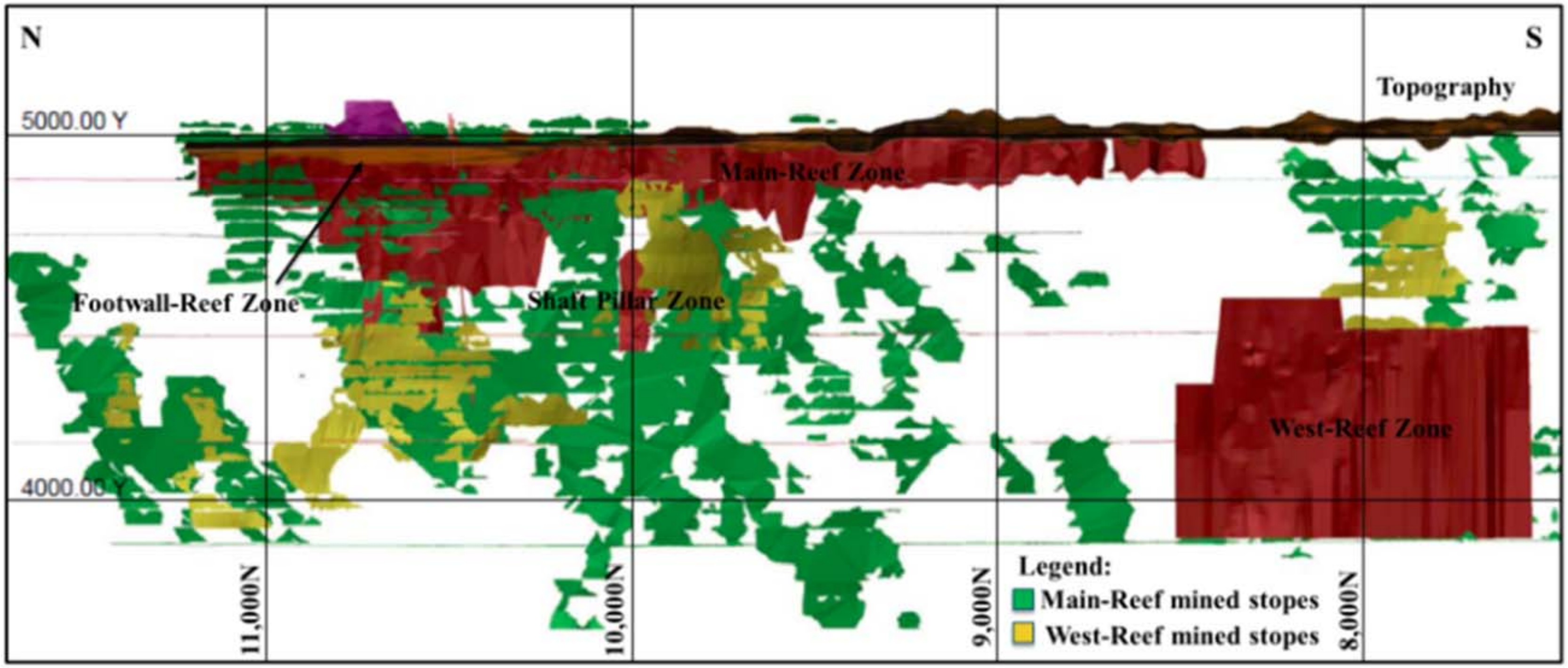


Figure 39 Prestea Deposits Long-Section

14.2.2 Data Sources

In the past estimates SRK was provided with GEMS project directories containing the Prestea data. These directories contain the relevant borehole databases, geological wireframes, oxidation and topographic surfaces, and block model parameters. Additional information was provided as Excel spreadsheets documenting QA/QC data and results of density determinations.

All drill hole data for the 2017 update was exported from the GSR centralised Acquire database. Wireframes were constructed using ore contacts based on drilling, channel samples and on survey pick up of the quartz reef in exposures underground.

Gold grade 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 (Table 58). The MR, FR, and HW statistics are mostly from surface diamond drilling. The WR statistics are derived from underground boreholes and channel sampling collared at locations along the 17 and 24 Levels and targeting reef intersections as deep as the 30 L.

The 2017 Update was done using 0.7m down hole composites rather than reef width composites to give equal weighting to samples. As seen in the summary statistics, the wider full reef composites do not have a lower standard deviation or CV, so there is no reason to composite further.

Table 58 Summary Statistics for the Prestea Underground Resource Domains

Domain	Count	Min (g/t)	Max (g/t)	Mean (g/t)	Std. Dev	Variance	CV
Main Reef	700	0.0	108.72	7.72	9.9	97.6	1.3
West Reef	200	0.1	234.6	16.40	26.8	680.2	1.6
Foot Wall	79	0.1	108.7	9.68	17.5	305.3	1.8
Shaft Pillar	222	0.2	72.3	8.71	9.3	86.9	1.1

Summary statistics for West Reef 2017 Update

Domain	Count	Min (g/t)	Max (g/t)	Mean (g/t)	Std. Dev	Variance	CV
West Reef	775	0	80	15.33	18.8	353.5	1.23

The above statistics highlight the relatively high grade nature of the shoots present in the West Reef when compared with the other domains at Prestea. The statistics also show that the addition of the new data has dropped the grade of the composites within the recent 2017 drilling and sampling areas.

The West Reef has a significantly higher grade distribution than the Main Reef and the drilling has confirmed the continuity of these high grades both along strike and down dip between levels.

14.2.3 Data Capping/Filtering

The composites database was analysed zone by zone to determine if capping was required. The WR capping study was conducted by GSR by analysing probability plots and histograms of composites which are illustrated in Figure 40. The WR zone was capped at 80 g/t gold based on the capping analysis. Even with the narrower composites, the 80ppm Au capping was kept, as it still appeared to be justified in the populations of the new composites.

The MR and FR deposits were capped to 30 g/t gold by SRK following studies that 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.

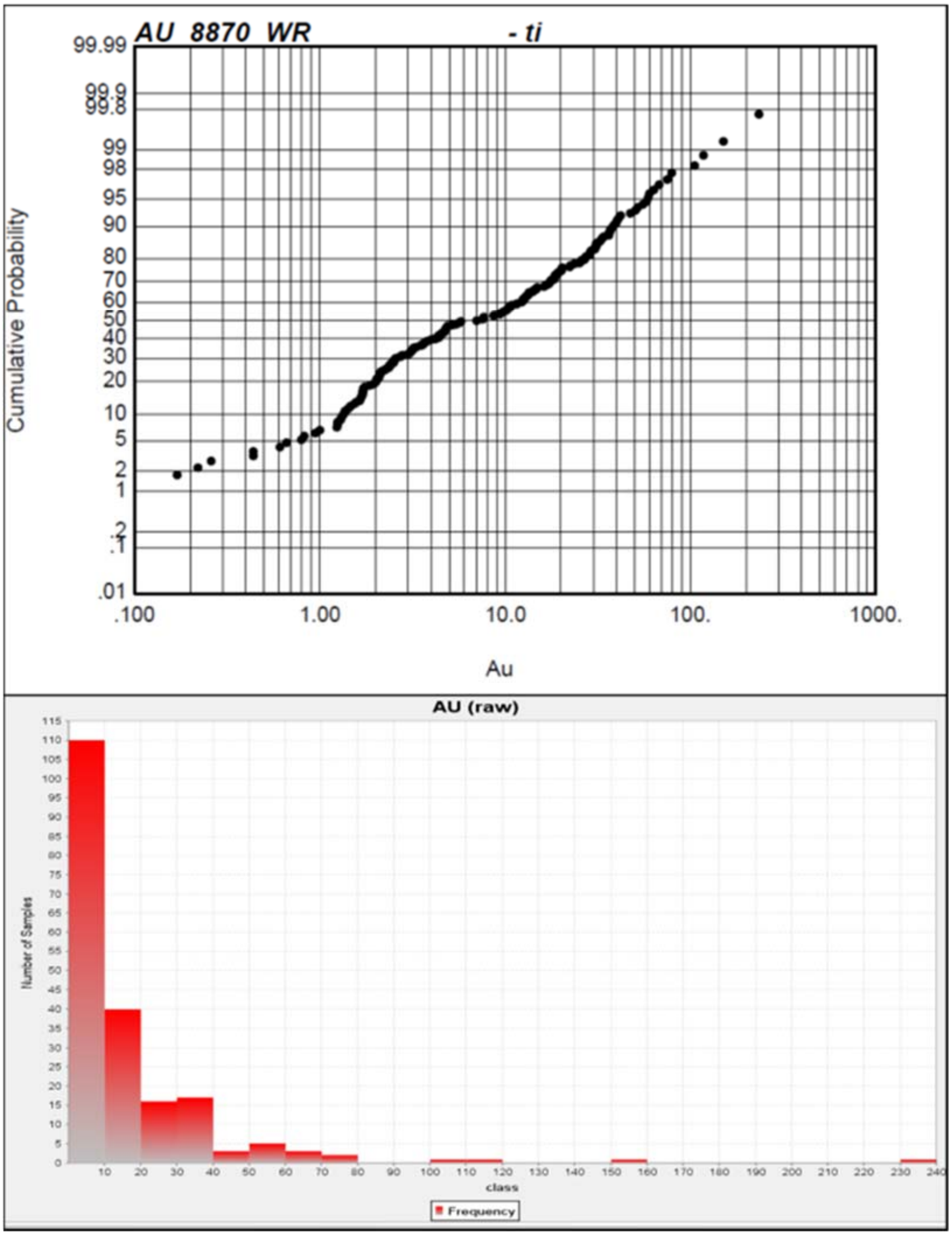


Figure 40 Probability Plot and Reef Composite Data Histogram for the West Reef Deposit

14.2.4 Variogram Analysis

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 that appears to be highlighting a plunge to the mineralization of approximately 42 degrees (°) to the northwest 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 southwest.

The following plot (Figure 41) 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 northwest in the plane of the structure. The secondary direction to this is plunging 34° to the southwest (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 40 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.

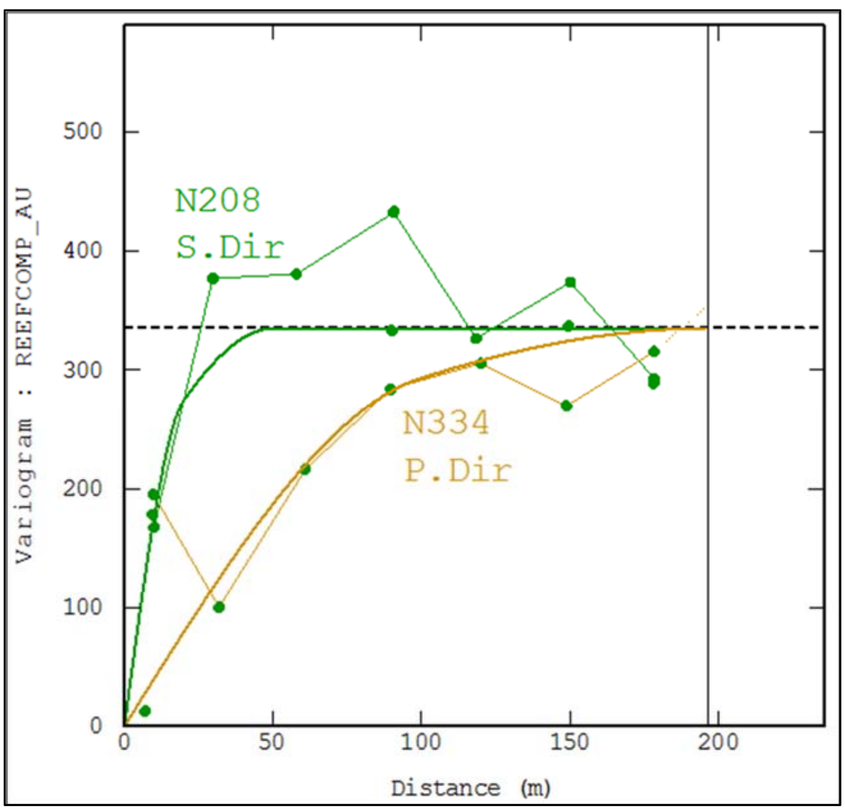


Figure 41 Directional Semi-Variograms

Produced in the Principal Directions in the Plane of the West Reef Deposit with Associated Model Semi-Variograms

Table 59 Semi-Variogram Modelling Results for the West Reef Deposit

Domain	Structure	Variance	Principle Direction plunge 42° to 334°	Secondary Direction plunge 34° to 208°
WR	nugget (C0)	0		
WR	spherical (C1)	195	100 m	20 m
WR	spherical (C2)	140	195 m	50 m

The 2017 Partial West Reef update based the anisotropy directions on those of the previous model, and reasonable variograms were produced. There were significantly more composite pairs available, which enabled slightly higher resolution for the variogram modelling. Results showed that there was a 20% nugget variance, 90% of the variance had been reached by 40m (showing significantly higher shorter scale variability than previously, likely due to lack of short scale pairs previously).

The remaining 10% of the variance is reached by 100m range, with samples outside this showing no correlation. This is a contrast to the previous work, which showed 40% of the variance between 100 and 195m. The new drilling and channels show that the west reef is less continuous on a long range than previously modeled.

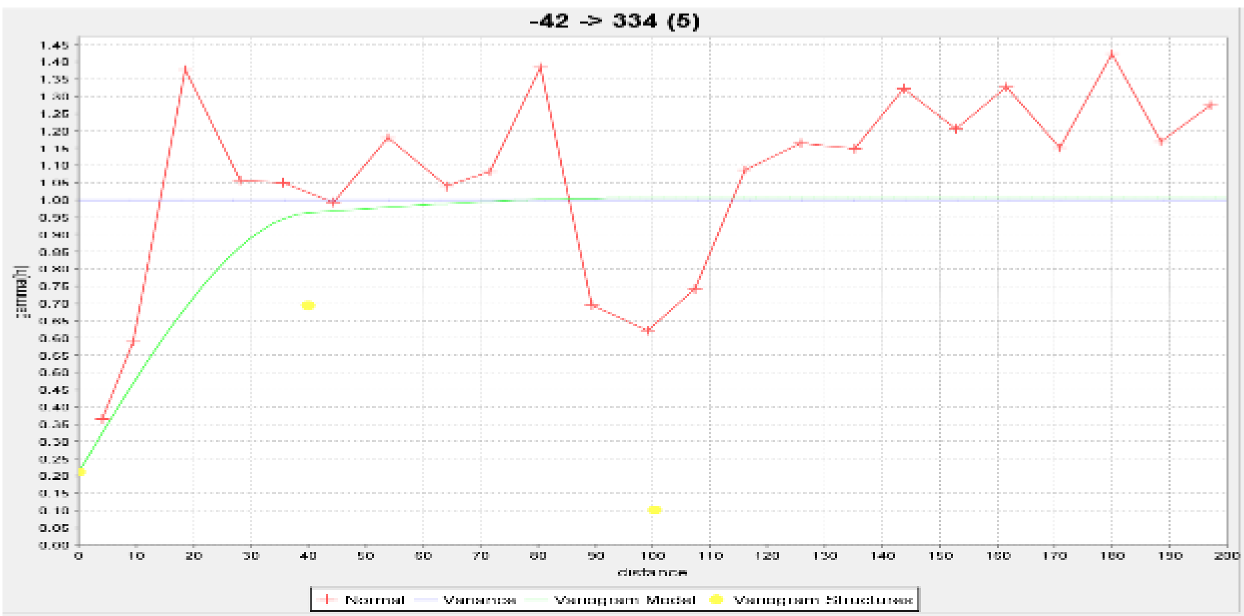


Table 60 Semi-Variogram Modelling Results for the West Reef

Domain	Structure	Variance	Principle Direction plunge 42° to 334°	Secondary Direction plunge 34° to 208°
WR	nugget (C0)	0.2		
WR	spherical (C1)	0.7	40m	20 m
WR	spherical (C2)	0.1	100m	50 m

Main Reef and Footwall

The MR semi-variograms produced good directional information and the maximum modelled 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 modelled 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 modelled with a range of approximately 100 m.

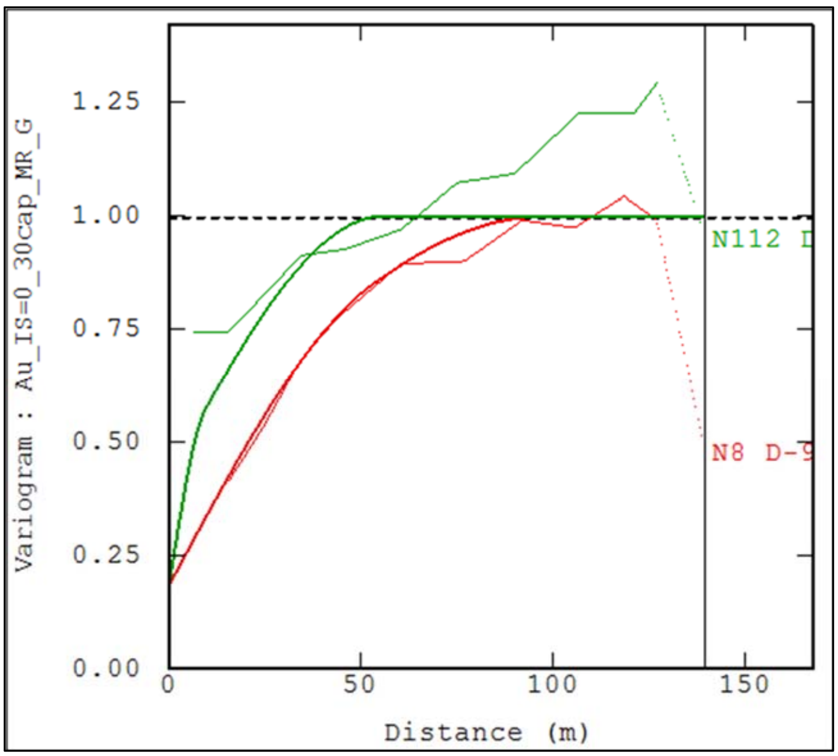


Figure 42 MR Semi-Variogram

Along Strike (N8) and Down-Dip (N112) Directions (Gaussian Anamorphosis Transformation)

Table 61 Semi-Variogram Modelling Results for the Main Reef Deposit

Domain	Structure	Variance	Principle Direction plunge 00° to 005°	Secondary Direction plunge 60° to 275°
MR	nugget (C0)	13.5		
MR	spherical (C1)	10.8	90 m	44 m
MR	spherical (C2)	21.3	105 m	57 m

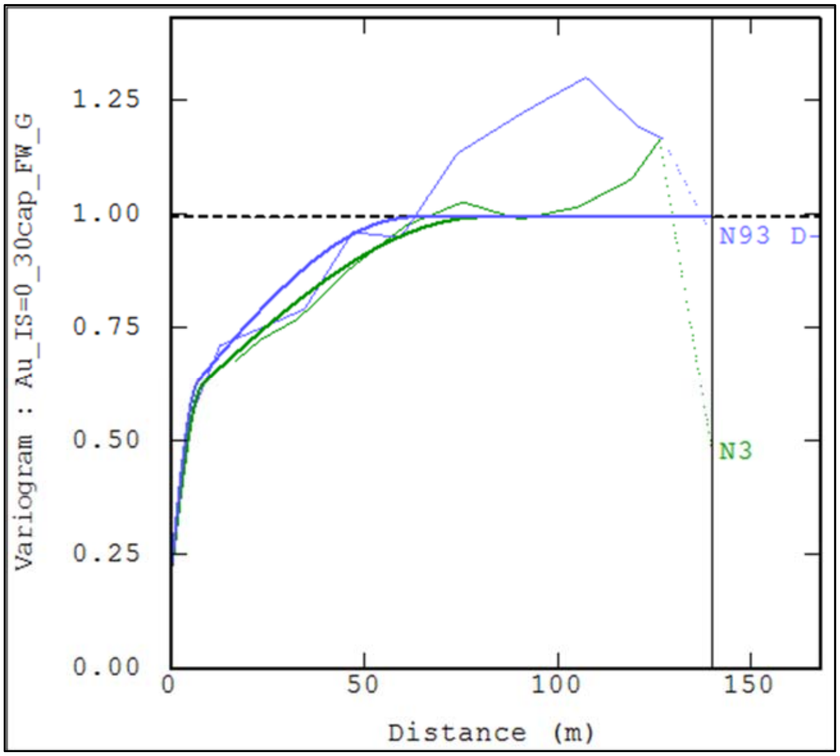


Figure 43 FR Semi-Variogram

Along Strike (N3) and Down-Dip (N93) Directions (Gaussian Anamorphosis Transformation)

Table 62 Semi-Variogram Modelling Results for the Footwall Deposit

Domain	Structure	Variance	Principle Direction plunge 00° to 005°	Secondary Direction plunge 60° to 275°
FR	nugget (C_0)	10.8		
FR	spherical (C_1)	13.0	9.2 m	9.2 m
FR	spherical (C_2)	10.7	103 m	52 m

14.2.5 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 hanging wall 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 ordinary kriging with block size set at 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. The detail parameters are shown in Table 63 to Table 66.

Table 63 WR Kriging Search Parameters

Search	Radius (m)			Samples	
	Strike	Dip	Perpendicular	Min	Max
a	100	50	10	10	10
b	150	75	15	5	15
b	500	250	25	2	20

Table 64 MR Kriging Search Parameters

Search	Radius (m)			Samples	
	Strike	Dip	Perpendicular	Min	Max
a	100	60	20	3	16
b	500	500	50	3	16

Table 65 FR Kriging Search Parameters

Search	Radius (m)			Samples	
	Strike	Dip	Perpendicular	Min	Max
a	100	60	20	3	16
b	500	500	50	3	16

Table 66 Block Model Dimension Parameters (lower left corner)

Coordinate	Value	No. of blocks	Spacing (m)
x	11600	92	12.5
y	7400	152	25
z	3800	56	25
Rotation	000°		

Figure 44 shows a representation of the grade distribution in the Prestea deposits viewed from the Footwall. Figure 46 is a more detailed plot of the WR deposit showing the relative distribution of composites used for the interpolation against the interpolated blocks. The plunging structure can be clearly seen as zones of high grade dipping to the north both in the composites and the block grades.

14.2.6 Mineral Resource Classification

Mineral resource classification is typically a subjective concept, and 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 these 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 mineral resource evaluation. The sampling information utilized for the resource classification was acquired primarily by diamond drill core, reverse circulation, and reverse air blast 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) were assigned an Inferred classification category. Wireframes were constructed around blocks with a slope of regression 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 slope of regression 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 borehole 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 mineralization intersections both along strike and down dip contribute to the increased confidence in the block grades, a longitudinal section of the classified blocks is illustrated in Figure 48. At MR, the number of blocks with a slope of regression value of greater than 0.5 was low and the classification here has been based largely on the visual confirmation of grade continuity from analysis of boreholes sections.

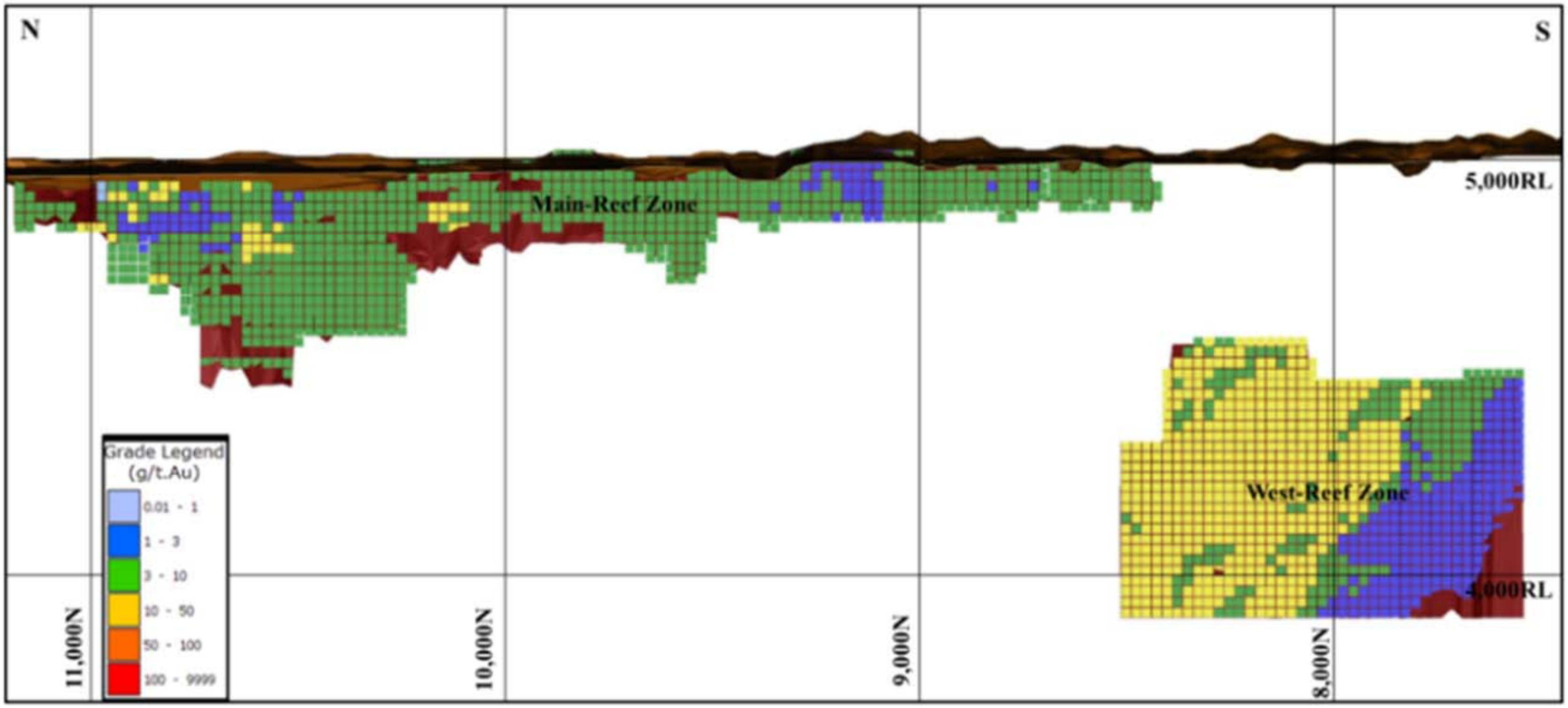


Figure 44 Long-Section Showing the Block Grade Distribution of the Main and West Reefs

Highlighting the High Grade Nature of the West Reef

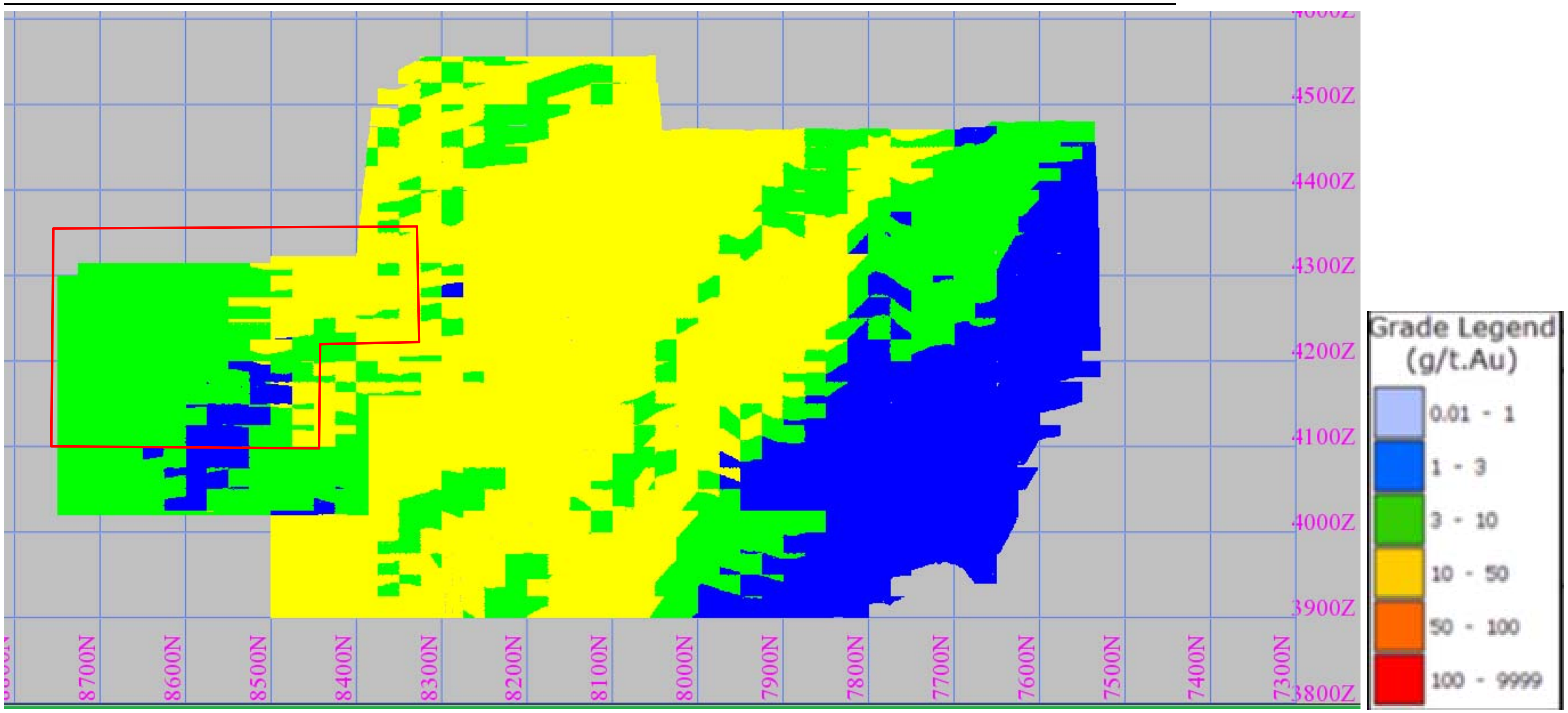


Figure 45 Long-Section Showing the Block Grade Distribution of the West Reef

Highlighting the High Grade Nature of the West Reef

Area of 2017 partial update shown inside red outline.

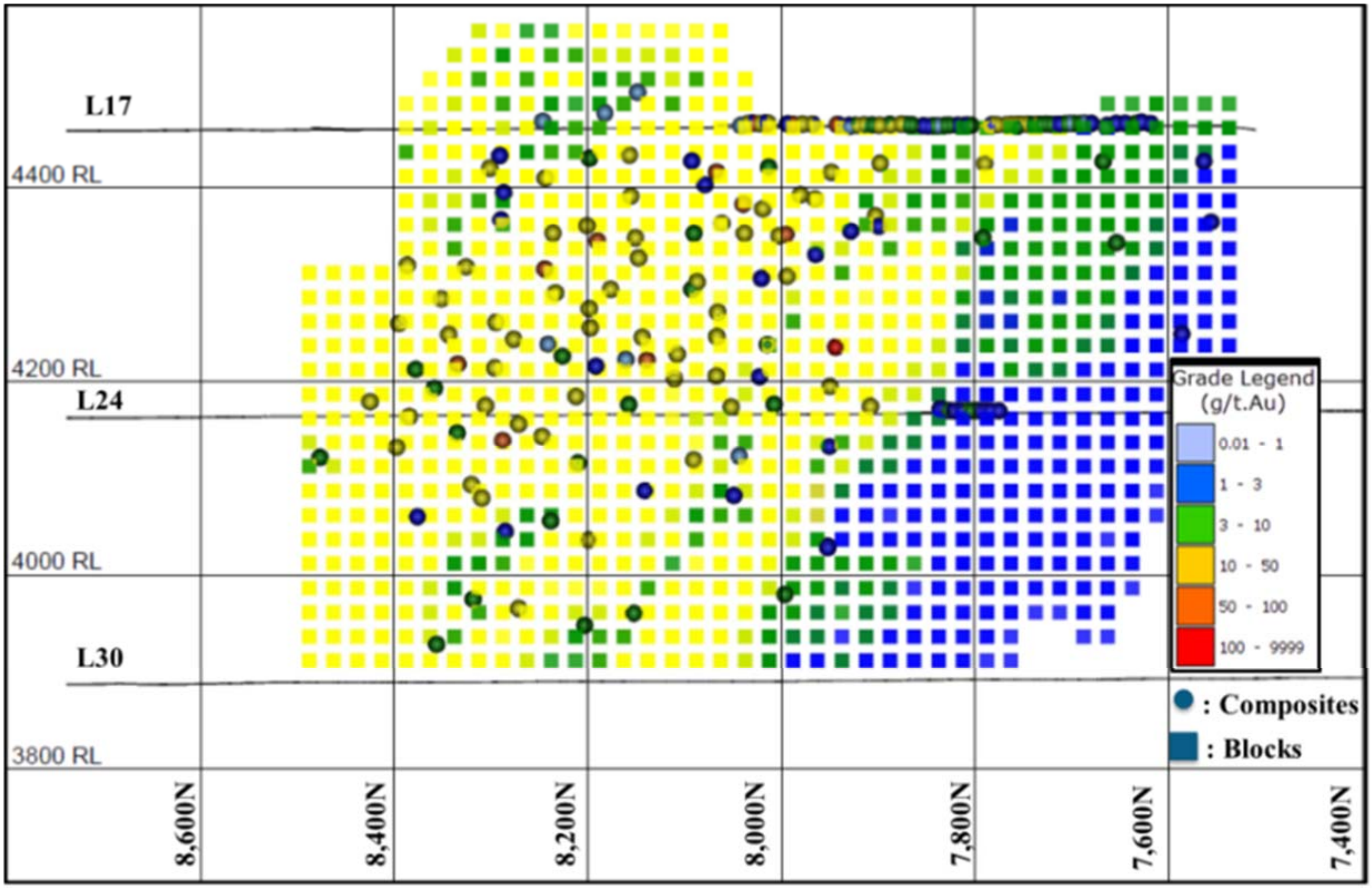


Figure 46 Vertical Longitudinal Projection View of the West Reef 2015 Block Model

Showing the Distribution of Gold Grade Values after Kriging and the Uncapped Composites

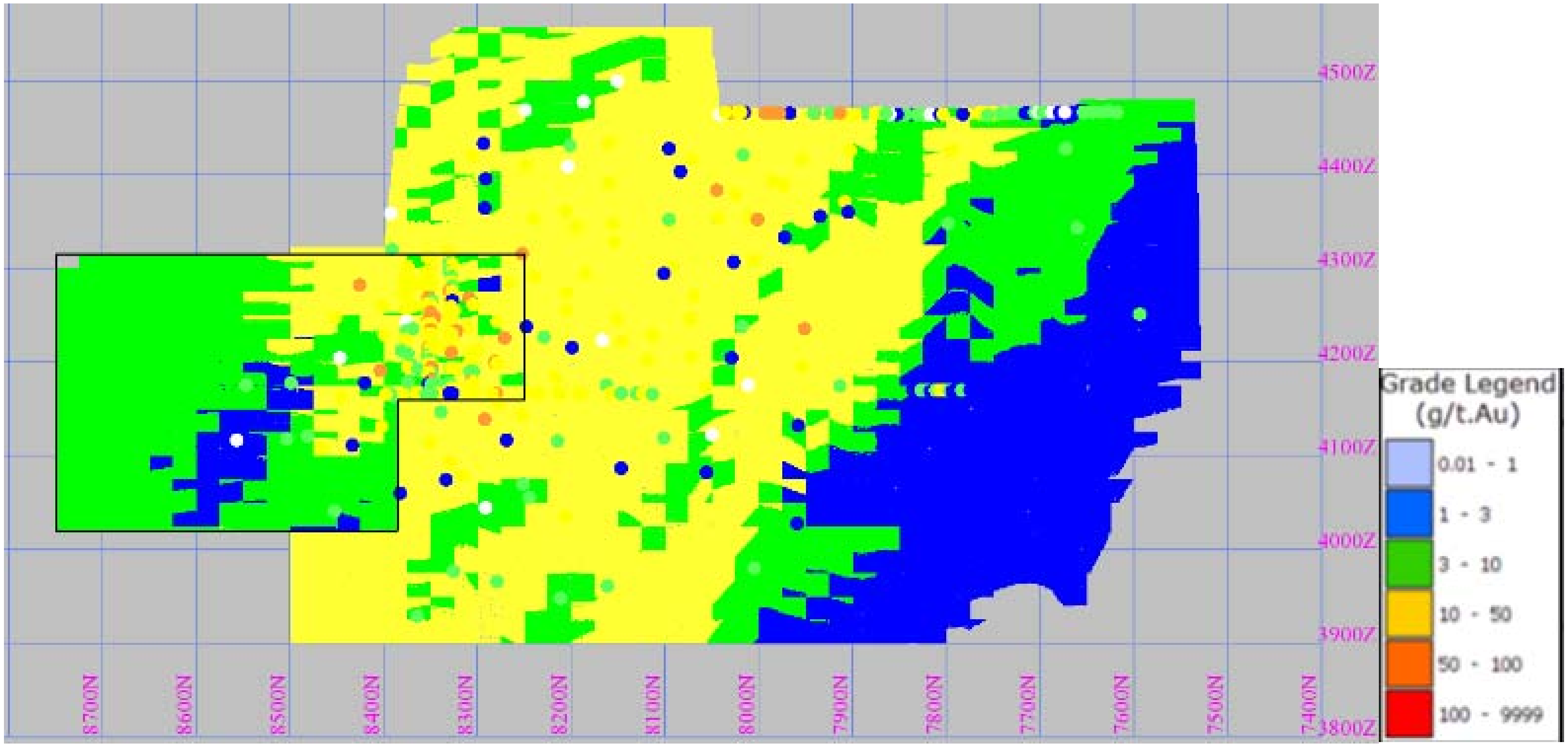


Figure 47 Vertical Longitudinal Projection View of the West Reef Block Model

Showing the Distribution of Gold Grade Values after Kriging and the Uncapped Composites

Area of 2017 partial update shown inside black outline.

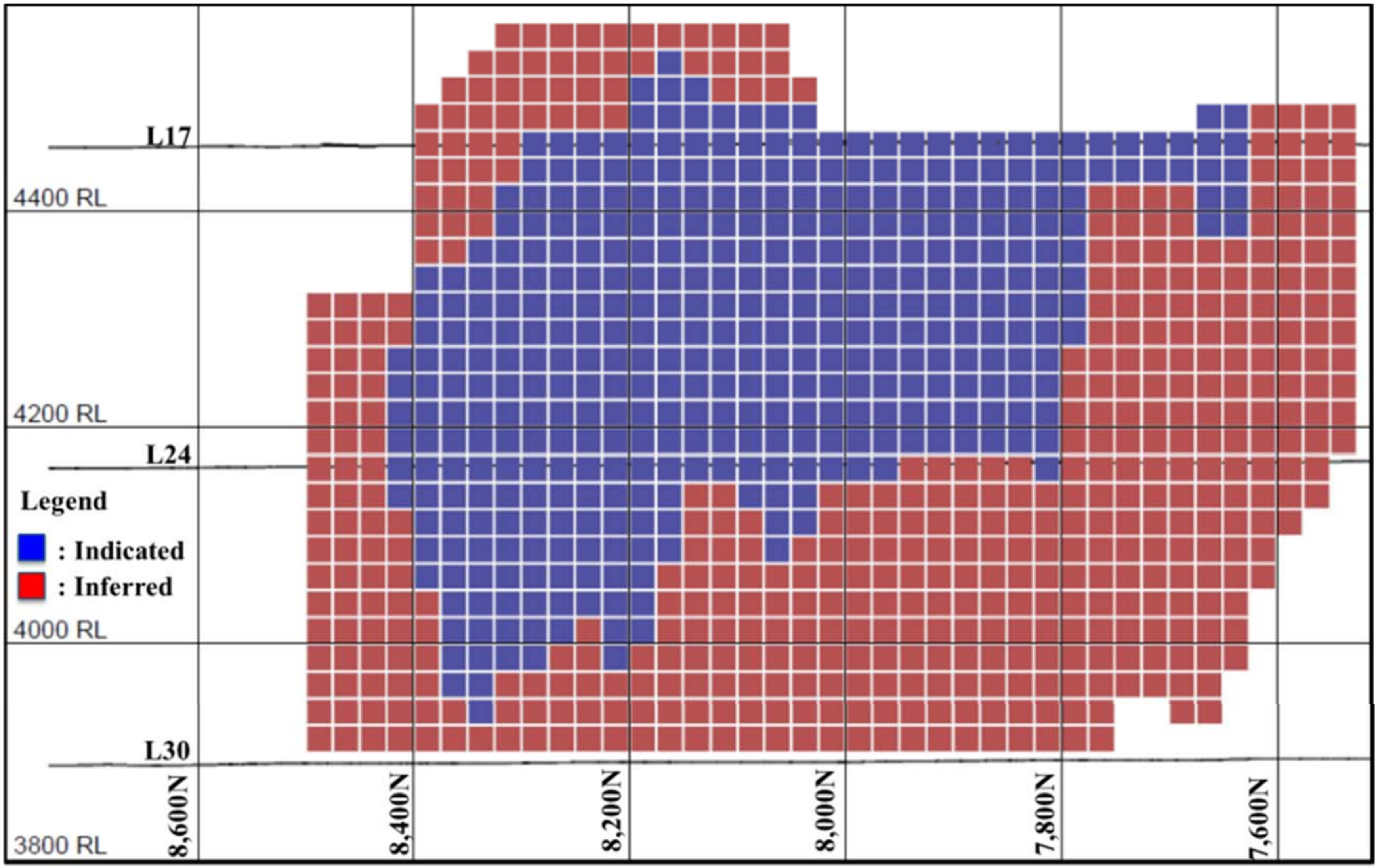


Figure 48 Vertical Longitudinal Projection View of the West Reef 2015 Block Model

Showing the Distribution of Block Classification Categories

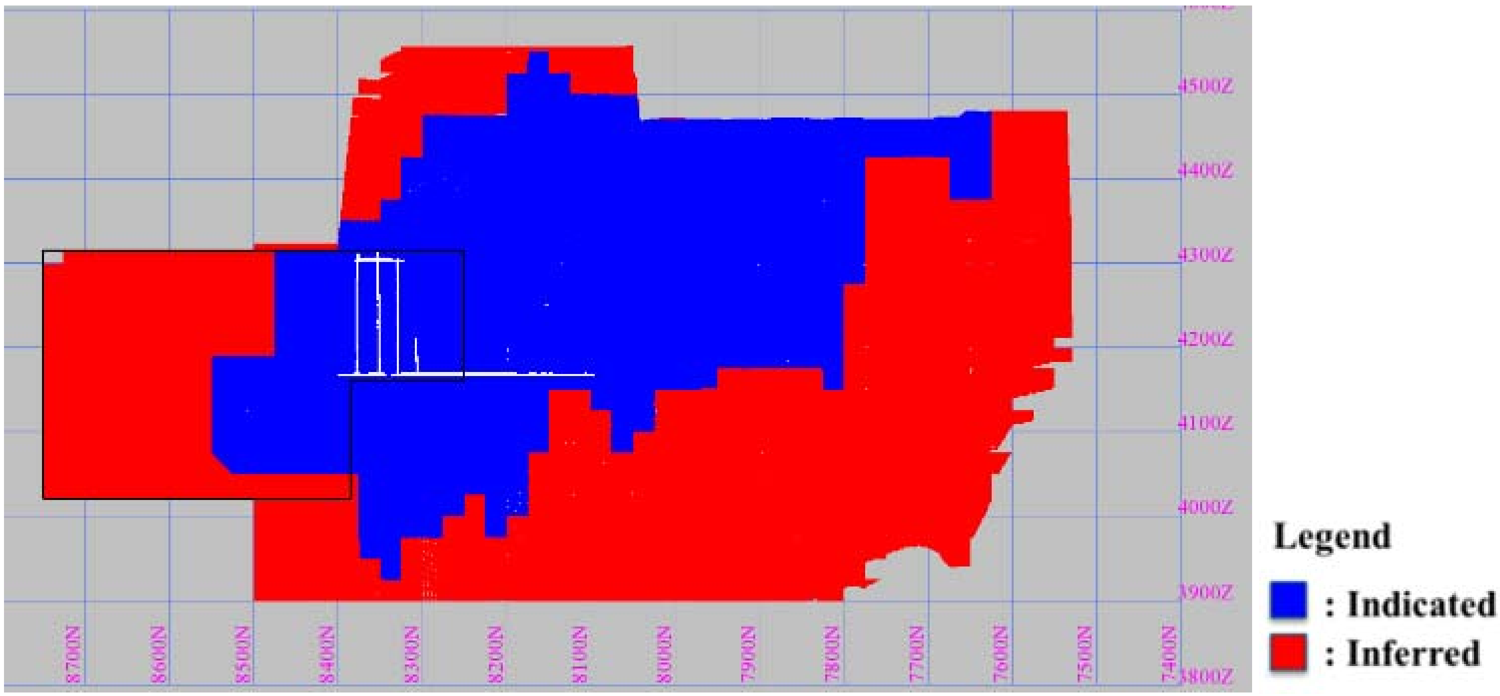


Figure 49 A Vertical Longitudinal Projection View of the West Reef 2017 Block Model

Showing the Distribution of Block Classification Categories
Area of partial update shown inside black outline.
Area shown in white has been depleted (mined out 2017).

14.3 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.

The Mineral Resource Statement has been prepared using a block cut-off grade of 5.54 g/t gold based on a US$1,450 per ounce gold price and appropriate costing data to produce a mineral resource that matches the requirements that the deposit should have "reasonable prospects for economic extraction" as defined by the CIM.

The statement was prepared by Mr. S. Mitchel Wasel who is a Qualified Person pursuant to National Instrument 43-101. Mr. Wasel is employed by GSR as Vice President of Exploration and is not independent of the company. The effective date of the Mineral Resource Statement is December 31 2017. The Mineral Resource Statement for Bogoso/Prestea is given below in Table 67.

Table 67 Mineral Resource Statement

Mineral Resources	Dec 31, 2017 Measured Mineral Resource			Dec 31, 2017 Indicated Mineral Resource			Dec 31, 2017 Inferred Mineral Resource		
	tonnes	grade	ounces	tonnes	grade	ounces	tonnes	grade	ounces
	(000)	g/t Au	(000)	(000)	g/t Au	(000)	(000)	g/t Au	(000)
Bogoso/Prestea Refractory	-	-	-	17,809	2.84	1,625	922	2.60	77
Mampon	-	-	-	103	1.59	5	14	1.68	1
Prestea South	-	-	-	1,627	2.12	111	68	1.89	4
Prestea Underground	-	-	-	1,649	15.16	804	3,193	8.46	868
Bogoso/Prestea Other	-	-	-	2,414	1.65	128	470	1.50	23
TOTAL	-	-	-	**23,602**	**3.52**	**2,673**	**4,667**	**6.48**	**973**

Notes to Mineral Resource Estimate:

- The Mineral Resources for "Bogoso/Prestea Other" include Chujah, Dumasi, Bogoso North, Buesichem, Opon, and Ablifa.
- The open pit resources for Bogoso/Prestea Other has been estimated using a gold cut-off grade ranging from 0.72 to 1.04 g/t for oxide material; from 1.33 to 1.75 g/t for transition material and from 1.20 to 1.52 g/t for fresh material.
- Prestea Underground Mineral Resource has been estimated using a gold cut-off grade at 5.54 g/t Au.
- Open pit Mineral Resources were estimated using optimized pit shells at a gold price of $1,450 per ounce. Other than gold price, the same optimized pit shell and underground parameters and modifying factors used to determine the Mineral Reserves were used to determine the Mineral Resources.
- Mineral Resources are inclusive of Mineral Reserves.
- Numbers may not add correctly due to rounding.

15 Mineral Reserves

15.1 Break-even Cut-off Grade Estimation

The following methodology and inputs were used to estimate the cut-off grade:

- Mining operating costs based on zero-based estimates combined with actual costs observed since mining commenced in 2017 of $111/t
- Processing costs based on historical costs and zero-based estimates of the modified process stream of $75/t
- G&A costs for the mine including both the Prestea mine site and the Bogoso plant site and services of $45/t
- Ore haulage costs are based on actual costs in 2017 of $3.70/t
- Estimated process recovery of 94% based on testing and current plant performance
- Dilution of 33% based on stope designs containing planned dilution
- Government royalty of 5% on gross revenue

Table 68 shows the estimate result of 6.5g/t.

Table 68 Break-even Cut-off Grade Estimate

Parameter		Unit	PUG
Mining		$/t	111.00
Ore haulage		$/t	3.70
Process and tailings cost		$/t	75.00
G&A		$/t	45.00
Total operating cost		**$/t-ore**	**234.70**
Gold price		$/oz	1,250
Government royalty	5%	$/oz	63
Process recovery		%	94%
Break-even cut-off grade		**g/t**	**6.5**

15.2 Mine Design

The orebody is steeply dipping and tabular. The mining is targeting a plunging high grade shoot within the West Reef. The cut-off grade is used primarily to define the lateral limits of the orebody as the grade reduces away from the core of the pay shoot.

A mechanized shrinkage mining method is being used. This method utilizes Alimak raise climbers to develop raises; drill horizontal blastholes from the raise; and load and blast the ore in a shrinkage manner retreating out of the top of the stope. With each blast the swell ore is mucked out of the bottom of the stope to make room for the next blast but the stope remains full of broken muck until the blasting is complete and the stope is in final drawdown.

The mine design comprises panels of stopes separated vertically by sill pillars. Each panel consists of:

- footwall drive, crosscuts to ore, ore drives along the vein and hangingwall Alimak nests at the base of panel extraction level;
- Alimak raises along the centre of the planned stopes;
- A top of Alimak panel sub-drift driven along the vein to connect the top of the Alimak stopes; and
- Shrinkage stopes.

A continuous ore drive is developed in the vein immediately above the sill pillar. These are developed large enough to allow access by the LHD's and create the initial void for the stope blasting. The cross section size is 2.7 m high by 2.7 m wide.

The Alimak raises are 3.0 m wide along strike of the vein and 2.7 m wide perpendicular to strike.

The sub-drifts connecting the top of the Alimak stopes are 2.5 m wide by 2.5 m high as they need to be large enough to allow passage of the parts of the dismantled Alimak between raises. The sub-drifts are developed and mucked hand held.

The Alimak stopes are drilled off from the Alimak platform using top access. Stopes are blasted from the bottom up, two or three rings at a time. A shrinkage method is used whereby only the blast swell is drawn out of the stope leaving enough room for the next blast. When stope blasting is complete to the full height of the raise, the stope is drawn empty.

The stopes are designed to include: hangingwall dilution from a weak graphitic layer in the immediate hangingwall; and a minimum mining width of 1.5m. The average stope dilution in the design is 33% (ore/waste).

The ore reserve presented in this report contains a small amount of open pit material which will be completed mining at the publication of this report. As such there will be no discussion of open pit mining in this technical report.

15.3 Mineral Reserve Statement

The Mineral Reserve Statement is presented in

Table 69. The Reserve contains small quantities of open pit mining which is expected to be complete in Q1 2018. The details around the open pit mining will not be discussed due to the immateriality of the quantities.

Table 69 Mineral Reserve Statement

Mineral Reserves	Dec 31, 2017 Proven Mineral Reserve			Dec 31, 2017 Probable Mineral Reserve			Dec 31, 2017 Proven and Probable Mineral Reserve		
	tonnes	grade	ounces	tonnes	grade	ounces	tonnes	grade	ounces
	(000)	g/t Au	(000)	(000)	g/t Au	(000)	(000)	g/t Au	(000)
Mampon	-	-	-	105	2.23	8	105	2.23	8
Prestea South	-	-	-	103	1.80	6	103	1.80	6
Prestea Underground	-	-	-	1,165	12.35	463	1,165	12.35	463
Stockpiles	547	1.21	21	-	-	-	547	1.21	21
TOTAL	***547***	***1.21***	***21***	***1,373***	***10.79***	***476***	***1,920***	***8.06***	***497***

Notes to Mineral Reserve Estimate:

- The stated mineral reserves have been prepared in accordance with the requirements of NI 43-101 and are classified in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum's "CIM Definition Standards – For Mineral Resources and Mineral Reserves". Mineral reserve estimates reflect the Company's reasonable expectation that all necessary permits and approvals will be obtained and maintained. Mining dilution and mining recovery vary by deposit and have been applied in estimating the mineral reserves.
- Mineral reserves are the economic portion of the measured and indicated mineral resources. Mineral reserve estimates include mining dilution at grades assumed to be zero.
- The 2017 mineral reserves were prepared under the supervision of Dr. Martin Raffield, Senior Vice President Project Development and Technical Services for the Company. Dr. Raffield is a QP as defined by Canada's NI 43-101.
- The mineral reserves at December 31, 2017 were estimated using a gold price assumption of $1,250 per ounce.
- The slope angles of all pit designs are based on geotechnical criteria as established by external consultants. The size and shape of the pit designs are guided by consideration of the results from a pit optimization program.
- Cut-off grades have been estimated based on operating cost projections, mining dilution and recovery, government royalty payment requirements and applicable metallurgical recovery. Marginal cut-off grade estimates for the open pits are as follows: Mampon 1.3 g/t; and Prestea South 1.1 g/t. Break-even cut-off grade estimates for Prestea Underground is 6.5 g/t;
- Numbers may not add due to rounding;
- Only non-refractory ore is included in mineral reserves.

16 Mining Methods

16.1 Introduction

This report summarizes the mine design and planning work completed to support the determination of the mining inventory available for processing utilizing the Alimak raise stoping method on the West Reef.

The ore reserve presented in this report contains a small amount of open pit material which will be completed mining at the publication of this report. As such there will be no discussion of open pit mining in this technical report.

The Prestea Underground Mine (PUG) consists of an underground mine with a life of approximately 5 years which will feed the existing processing plant at Bogoso. The nominal mill feed rate is 245,000 tonnes per annual (tpa), or 650 tonnes per day (tpd).

The Prestea deposits comprise a series of interbedded greywacke and phyllites dipping at approximately 65° west. The host rock mass contains varying amounts of graphite which in many places is sheared or heavily fractured. Through this sequence two mineralised quartz veins which contain gold mineralisation have been intruded - the Main Reef and West Reef. The veins are associated with a decrease in rock mass properties and increase in fracturing and sheared material. The West Reef is located to the west of the Main Reef in the hangingwall. Drive development has been focused within the footwall of the Main Reef and a series of ore drives have been extended west to locate the West Reef. The underground development at Prestea is shown in Figure 50.

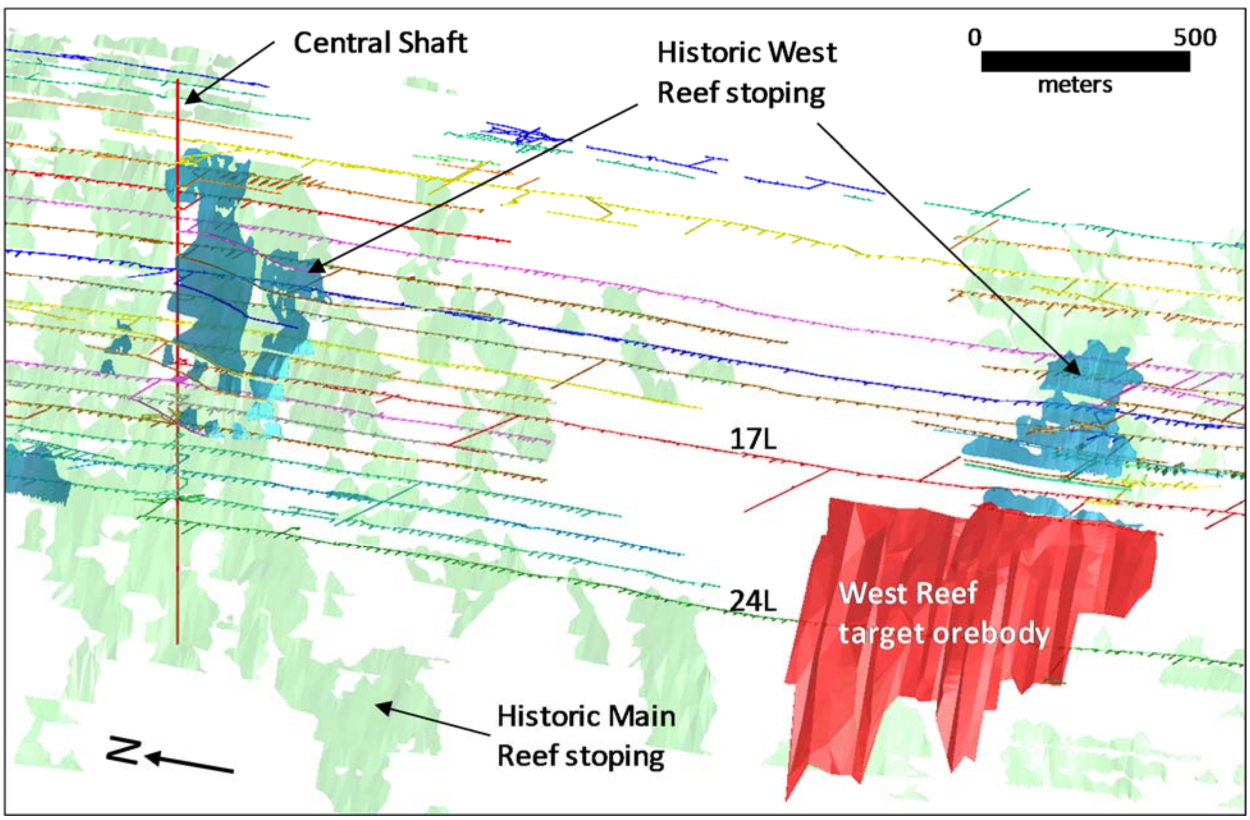


Figure 50 Location of West Reef and Main Reef

16.2 Hydrogeology

16.2.1 Groundwater Flow

This section was prepared utilising information sourced from: "Technical Report on Mechanized Shrinkage Mining Method at Prestea Underground Mine, March 2016" an unpublished internal GSR document prepared by SRK Consulting (UK) Ltd; "Prestea Environmental Impact Assessment Groundwater and Geochemistry Baseline Report", April 2016; prepared for GSBPL by Golder Associates (South Africa); and "Prestea West Reef Underground Mine: Environmental Impact Assessment", March 2017 prepared for GSBPL by Golder Associates (Ghana).

Weathered substrate extends over most of the Prestea area and is considered to be the main groundwater resource. The depth of weathering is highly variable (up to 120 m below ground surface) and is dependent on topography. Clay, silts, and sandy clays, which form as a result of weathering, produce a shallow aquifer with a high porosity and high storage that has a low permeability and low effective porosity/storage due to high silt/clay content.

The underlying, deeper, and less weathered rock is considered to have a low permeability and storage due to the lack of primary porosity. Secondary porosity is likely to be discrete in nature comprising fissures, fractures, and shear zones. These secondary porosity features have relatively high storage and transmissivity and provide preferential groundwater flow paths. It is considered that the bulk of groundwater flow occurs in these secondary porosity features. Packer testing (SRK 2001) showed highly heterogeneous hydraulic conductivity values in the competent fractured rock aquifer, ranging from 1E-06 m/s in the MCZ to 1E-09 m/s in un-weathered phyllites. The typical conductivity of the aquifer outside the MCZ are consistently low at 1E-08 m/s.

Average annual groundwater recharge is considered to be between 115 and 150 mm. Recharge models (Houston 1988) indicate an average rainfall recharge of 200 mm or around 11% of the mean annual precipitation (Bogoso data from 1937 to 1990). Due to the mostly unconfined nature of the aquifer, lateral groundwater flow is driven by topographic gradients towards the Ankobra.

Dewatering of the underground mine is via two principal vertical access shafts, the Central and Bondaye shafts. Dewatering was operated on a care and maintenance basis since mining activity ceased in early 2002 until the Prestea Underground recommencement. Mine water levels at the Central shaft are maintained at or below 25 L.

The groundwater flow regime is considered to be predominantly influenced by geological structures, with higher permeability coinciding with the dominant structural trend of north-northeast–south-southwest. Studies by Gorman (1990) and Golder (2017) found that the infiltration of water from the shallow to the deeper fractured rock aquifer system (vertical leakage) is strongly heterogeneous and dependent on clayey layers (especially at hilltops where the regolith is sparse), interconnected fracture systems or mine workings (many caused by unauthorized mining activity) act as conduits. However, vertical flow is generally limited and as a result, the impact on the regolith aquifer is negligible away from the immediate vicinity of the former mine voids.

Due to the anisotropic nature of the fractured rock aquifer and mine dewatering, the groundwater flows in this aquifer are generally directed towards the historic mine voids within the MCZ. This may result locally in spatially different groundwater flow directions for the shallow and deeper aquifer systems. The anisotropy of the hydraulic conductivity in relation to the structural setting is of prime importance as it determines the mine inflows and the spatial extent and shape of the subsequent cone of dewatering. Resource drilling, remote sensing data and core logging has identified two principal orientations of regional structures, a NE-SW striking orientation following the strike of the MCZ; and a more dominant N-S striking orientation. In the vicinity of the ore body, water bearing strata are encountered at greater depth than in the surrounding rocks or are not encountered at all.

Studies have highlighted the key geochemical and flow processes that will affect the quality of mine waters. These are illustrated as a conceptual model of the mine water system for the historic mines (Golder 2013) and their potential interactions with the proposed West Reef mine (Figure 51).

Water accumulating in the historic mine voids is primarily sourced through the following key flow processes:

- Groundwater inflow into the mine workings from dewatering aquifer;
- Cross flow of mine influenced water from interconnected underground mines;
- Inflow of mine influenced water from surface mine workings either through direct connections or indirectly through mining induced preferential pathways (cracks);
- Direct rainfall recharges on surfaces directly overlying the voids; and
- Water pumped into the mine for use in mining activities (drilling and excavation, dust control).

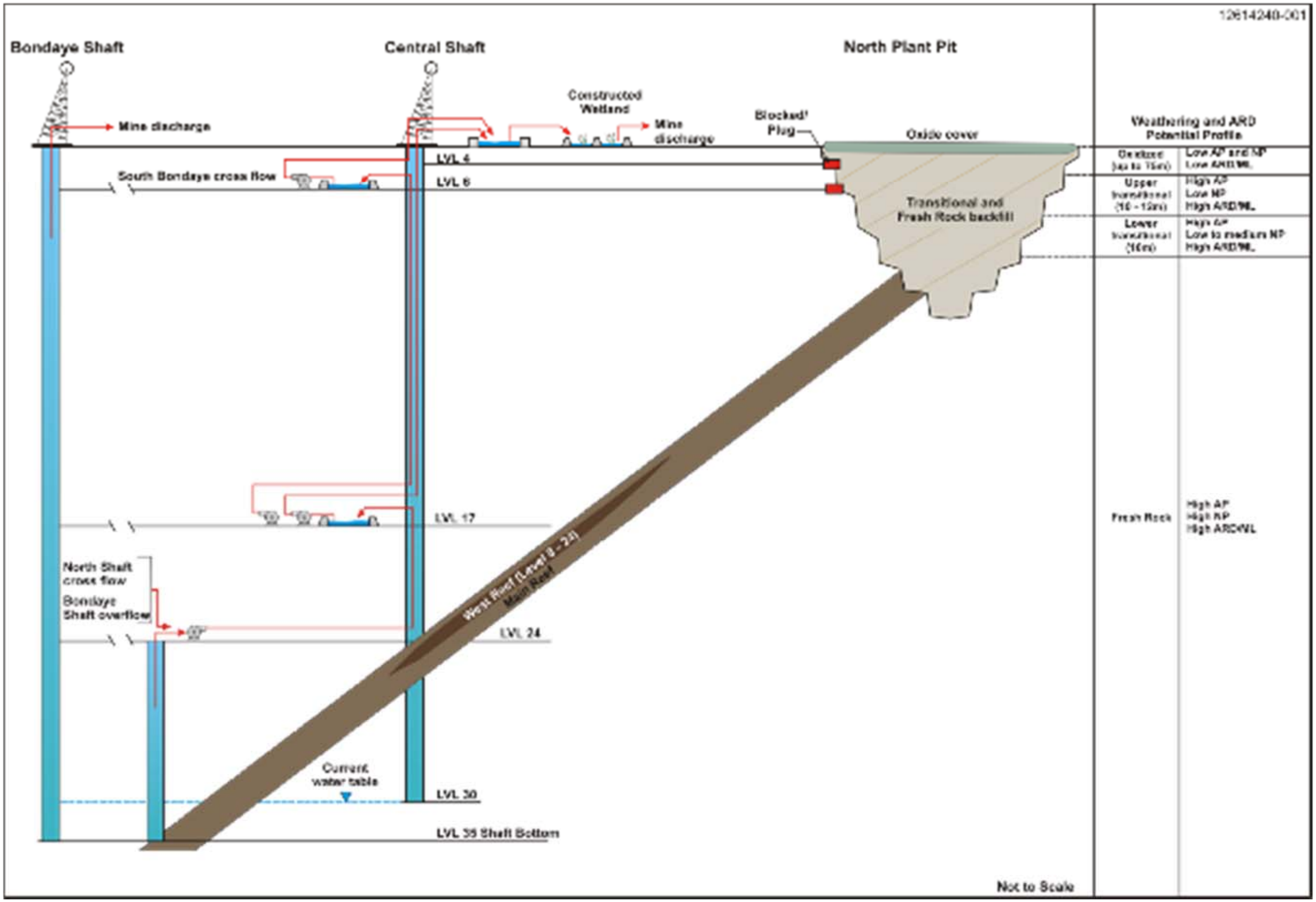


Figure 51 Schematic cross-section of conceptual mine water system (Golder 2013)

16.2.2 Predicted Inflows

Estimation of groundwater inflow into underground mine workings in a fractured hard rock setting is inherently problematic. This is due to a lack of realistic analytical solutions or models for hard-rock settings and also due to the permeability of the fractured hard rock which may vary by several orders of magnitude over short distances.

However, using abstraction data for the period 2013 – 2014 and the volume of mine workings it is possible to calculate the inflow of water per cubic metre of mine workings. This ratio can then be applied to the volume of the West Reef development to coarsely predict the inflow volume. It should be noted that this method is very basic and does not take into account any change in water level or any variation in material properties. It also conservatively assumes:

- that in the later stages of the West Reef mine the workings intersect parts of the MCZ that have not been dewatered;
- that the entire void volume below ground surface would fill with water;
- that there are no water losses in the system, e.g. mine ventilation, moisture in rock mass, consumption, etc; and
- that there is direct hydraulic connection between the future West Reef mine workings and the existing Main and West Reef mine workings along the entire length of the historic field.

For this reason the estimations provided are to be treated with caution.

Table 70 shows the predicted inflows for the West Reef workings at the end of mine life. The inflow per cubic metre calculation has been based upon inflows originating from the same elevation as the proposed development, i.e., between 17 L and 24 L, and the corresponding volume of existing workings. This ratio has then been applied to the volume of the proposed West Reef development in the life of mine plan.

It should be noted that predicted inflow to the West Reef increases from zero at start of mine development and will very steadily increase with the cumulated mine void to an approximately 150 (best case) to 300 (worst case) m^3/d, equivalent to around 1.7 to 3.5 litres/second at the end of mine life. This final inflow rate though somewhat conservative is itself relatively small, but the predicted inflows are much smaller than those predicted in the previous studies.

Table 70 Predicted Inflows for the PUG West Reef Project at the End of Mine Life

Volume of Existing Mine Workings between 17L and 24L (m^3)	Estimated Average Daily Inflows from 17L to 24L (m^3/d)		Inflow per Cubic Meter	Proposed West Reef Volume at End of Mine life (m^3)	Predicted Inflows from West Reef workings at End of Mine Life	
					(m^3/d)	(l/s)
1,429,000	Minimum (Average daily inflow for 2013)	555	0.0004	396,000	153	1.78
1,429,000	Average (Mid-point between 2013 and 2014 average daily inflow values)	826	0.0006	396,000	228	2.64
1,429,000	Maximum (Average daily inflow value for 2014)	1,096	0.0008	396,000	303	3.51

Source: SRK (2016).

This volumetric approach prevents true time-variant inflows from being calculated as there is no consideration for change in water levels or water released from storage. However, inflows can be varied according to the increased mine void volume as the West Reef is progressively mined. As such, predictive inflow volumes over the life of the mine are shown in Figure 52.

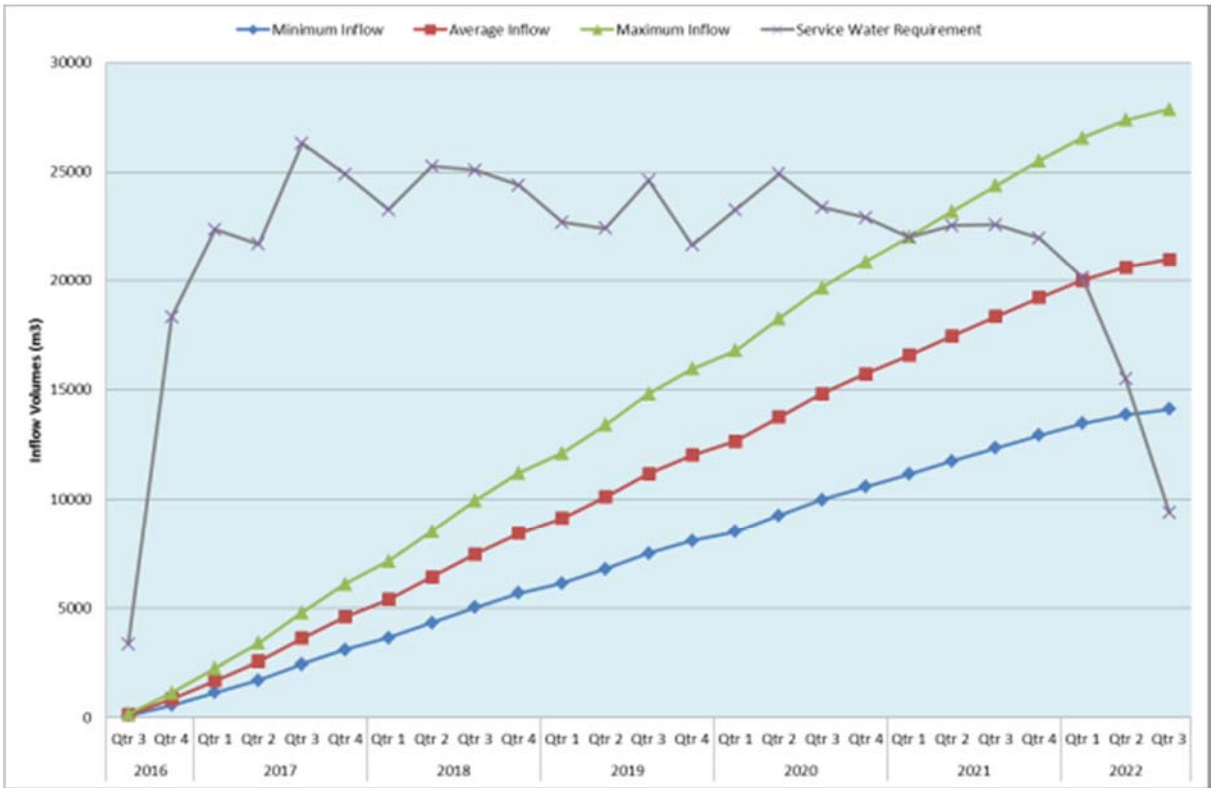


Figure 52 Predictive Quarterly Inflow Volumes for the PUG West Reef project

Source: SRK (2016)

The volume of service water required over the life of mine has been calculated and is shown as a grey line in Figure 52.

The graph shows that for the first 4.3 years of mining the inflows from the West Reef will be significantly less than the service water requirement (recirculation), meaning that the area is a net consumer. It is only in the last six quarters (Q2 2021 to Q3 2022) that the inflows increase beyond the service water demand, indicating a potential surplus of water which may require dewatering.

Since the commencement of West Reef mine development in 2016 and later commercial production commencement in 2018, the mine remains a dry mine and is a net water consumer.

16.2.3 Water Quality Issues Associated with the West Reef

An indemnity is currently in place which exonerates GSR from any environmental liability relating to historic mining practices within the licence area. This includes any water generated by the former mining areas or contaminated by these sources.

Whilst the indemnity is in place, this does not diminish GSBPL's liability for impacts from its mining operations at the West Reef and as such any water that is generated from the West Reef development will need to meet water quality standards before it is discharged.

West Reef Geochemistry

Extensive programmes of concession wide geochemical analysis (Senes 1998, 1999) were complemented by a dedicated geochemical assessment programme for the West Reef (Golder, 2016). Diamond drill core samples were selected to represent the proposed spatial distribution, depth, and rock units to be intersected by the West Reef mine, for examination and logging of key geochemical and hydrogeological features.

Thirty-four (34) samples were subjected to acid-base accounting, trace metals and whole rock x-ray fluorescence (XRF) analyses. Of those, 25 samples also underwent modified synthetic precipitation leaching procedure (SPLP), net acid generation (NAG) tests, x-ray powder diffraction (XRD) analysis and Rietveld quantification, and trace metals analysis of the NAG leachate products to determine leachate (mine water drainage) quality.

Whilst sulphide minerals are recorded as alteration minerals across the field, they were not detected by XRD in the rock samples from the West Reef, suggesting that if present, they were in quantities that were below the limit of detection of the XRD method. Sulphur speciation produced similar results and showed no distinct trend in the distribution of sulphur between hanging wall, footwall and ore samples.

Carbonate minerals, dolomite and siderite occurred as minor phases in the hanging wall, footwall and ore samples. Plagioclase and mica occurred major phases in all the samples. The dolomite is a fast reacting (dissolving) carbonate expected to result in substantial acid-buffering capacity. Additional buffering is expected from the silicate minerals though at a slower rate, as they are intermediate to slow weathering minerals. This was confirmed by high bulk neutralisation potential (Bulk NP) in the hanging wall, ore and footwall samples. The CaNP (29-170 kg CaCO3 eqv/t) was equivalent to or slightly below the Bulk NP suggesting that dolomite represented a significant proportion of total carbonates in the samples. This is in agreement with XRD results, which found dolomite to be the dominant carbonate mineral. The paste pH was alkaline (8.1-9.7) indicating sufficient reactive NP to buffer acidity generated by the initial oxidation of sulphides in all the samples.

Classification of ARD potential using the methods of Morin & Hutt 2007 and MEND 2009 showed that the bulk of the rock material at West Reef is not acid generating (NAG). Classification using the guidelines of Price et al. (1997) and Soregaroli and Lawrence (1997) also confirmed that the majority (>85%) of footwall samples and half of the hanging wall and ore samples have no acid generating potential, with remaining samples having a low acid generating potential. Net acid generation pH and TNPR of selected hanging wall, ore and footwall samples classify the samples as not acid generating (NAG), with only one footwall sample classified as uncertain. These findings are consistent with the highly alkaline paste pH result of 8.1-9.7.

West Reef Drainage Chemistry Analysis

To complement the direct sampling of water qualities from the West Reef mine area, net acid generation and de-ionised water leach tests were conducted on West Reef samples to confirm potential drainage quality (Golder 2014). These tests measure readily soluble components of geological materials but do not predict long term water quality (INAP 2010).

Results from the de-ionised water leach (Table 71) indicate that short-term interaction of materials in the West Reef mine with clean water is likely to produce near-neutral to alkaline drainage, with all parameters expected to be within EPA guidelines.

Table 71 Summary of West Reef leach test results

Deionised water leach	Net acid generation (NAG) leach
Near-neutral to alkaline (6.6-8.9) with low TDS (< 100 mg/L). All parameters within Ghanaian EPA guidelines.	Low TDS (< 500 mg/L). All parameters within Ghanaian EPA guidelines except for SO_4, As, Mn, ammonia, Ni and Sb.

Source: Golder (2016)

Leachate generated by net acid generation represents complete and instantaneous oxidation of all reactive minerals. These tests were completed to assess the worst case quality of drainage from the mine, waste rock dumps and ore stockpiles (Golder 2016). Under field conditions, sulphide oxidation and release of elements occurs gradually and concentrations in mine drainage are expected to be lower than NAG leachate chemistry at any given time.

The drainage is classified (INAP 2010) as neutral mine drainage (Figure 53) and the risk of acid rock drainage from the West Reef, including both ore and waste rock, is low to none (Golder 2016). Using the INAP method, it is suggested that there is some minor risk of slightly acidic drainage should West Reef drainage or rocks become exposed to ARD from the historic mine voids (as represented by the net acid generation leach).

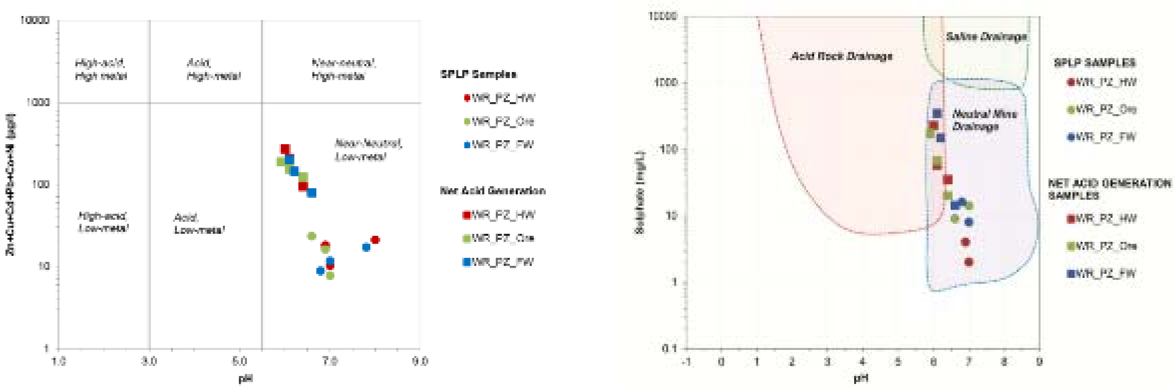


Figure 53 Classification of SPLP and NAG leachate (Golder 2016)

Conclusions of Geochemical Studies

Host rocks of the West Reef are non-acid generating, and mine drainage from the area is predicted to be near neutral to alkaline drainage, with all parameters expected to be within EPA guidelines.

As the major geochemical impacts at Prestea relate to historic mining activities and discharge from the historic mine voids (Golder 2016), to limit the potential for impact on the West Reef mine, a dedicated mine water management system should be established.

The geochemical analysis, assessment of historic studies and analyses, combined with underground field visits to the West Reef deposit in 2012 and 2015, has led to the development of a geo-environmental model for the proposed West Reef mine (Figure 54).

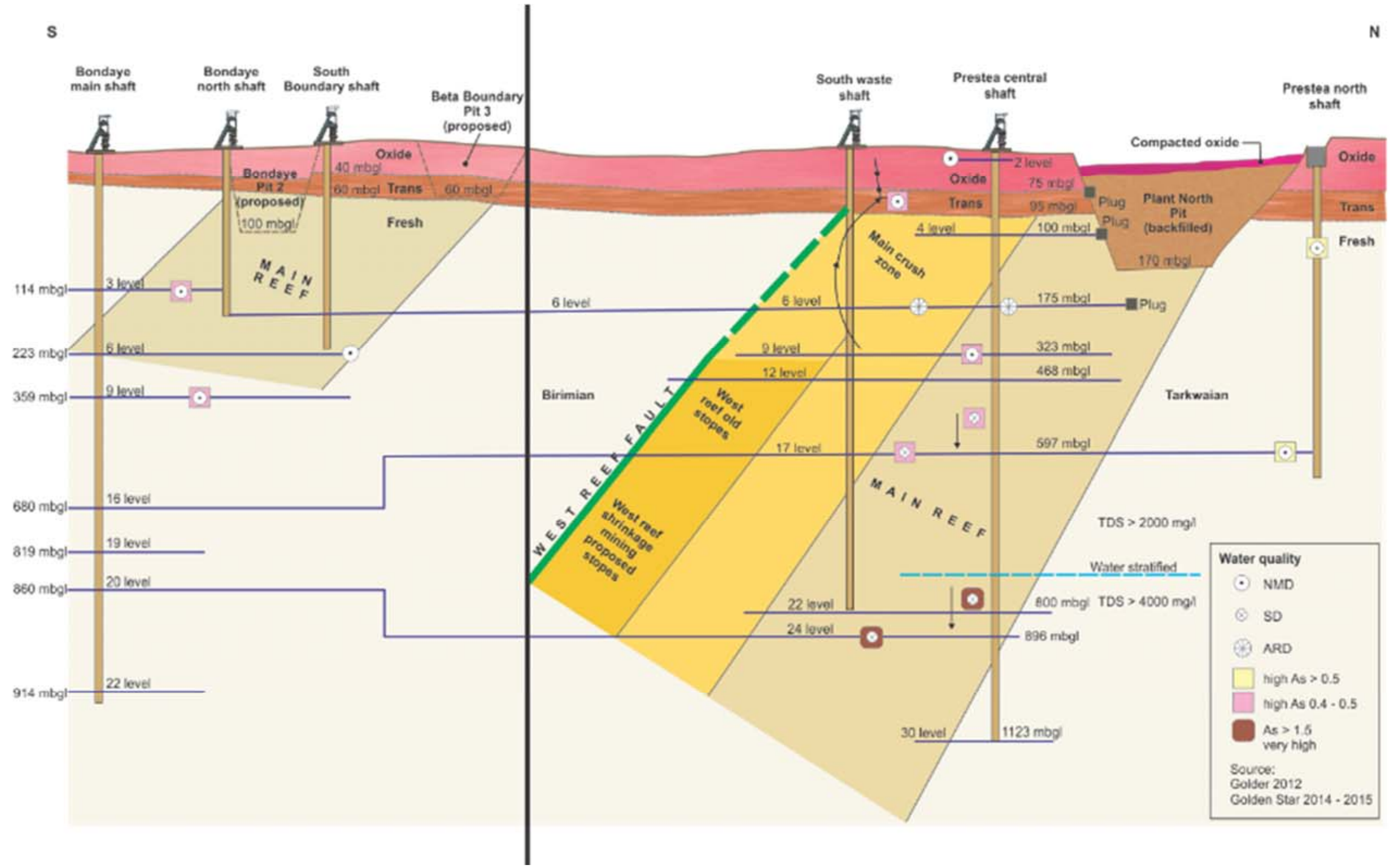


Figure 54 Conceptual West Reef geo-environmental model (Golder 2016)

16.3 Geotechnical

This section was prepared by SRK Consulting (UK) Ltd. as part of an unpublished internal GSR document titled, "Technical Report on Mechanized Shrinkage Mining Method at Prestea Underground Mine, March 2016".

16.3.1 Introduction and Summary

Geotechnical stope design parameters requiring confirmation are rib pillar and sill pillar dimensions for a 150 m vertical height by 20 m strike length Alimak raise stope. This work has been carried out utilising geotechnical input data and models generated for previous West Reef geotechnical studies. Details of the work undertaken are presented in the following sections with a summary below. Note that the terms Factor of Safety, Strength Factor and Strength:Stress ratio are variously used in this report to describe the stability condition of the pillars analysed. These values are equivalent. A numerical value of any of these parameters less than unity indicates a condition of potential instability. For short term design a Factor of Safety of 1.1 or greater is considered to be appropriate.

16.3.2 Stope Design Parameters

- Based on its assessment of geotechnical conditions it is considered that a 20 m wide stope up to 150 m vertical height should be stable. The Alimak raise mining method require the raises to be bolted which should improve stope hangingwall stability and reduce dilution to some extent.
- The minimum rib pillar width should be 2 m although to allow for some rib loss resulting from production blasting damage the design rib pillar width should be set at 3 m. This will provide a rib pillar having a strength factor of about 1.2 for the average orebody thickness of 1.75 m, reducing to 1.1 for the maximum orebody thickness of 3.5 m.
- The minimum sill pillar vertical height should be 10 m. A pillar of this dimension yields an average strength factor, based on 3D modelling, of 1.8. A minimum strength factor of slightly above unity is recorded along the upper and lower margins of the sill pillar. Any access or production development located in these areas will require to be systematically supported with rock bolts and mesh.

16.3.3 Geotechnical Assessment

Empirical Assessment of Stope Stability

This section describes the empirical assessment of stope stability based on an Alimak raise mining method utilising nominally 20 m wide stopes extending between the 17L and 21L, the 21L and 24L and the 24L and 27L. Vertical stope height is about 135 m.

Analysis Input Parameters

Data collected from a programme of core logging and underground mapping from previous studies were used for this assessment as described below. Parametric data was collected for the calculation of the Q Index classification system:

$$Q = \frac{RQD}{J_n} x \frac{J_r}{J_a} x \frac{J_w}{SRF}$$

where RQD is the Rock Quality Designation
J_n is the Joint set number
J_r is the joint roughness number
J_a is the joint alteration number
J_w is the joint water reduction factor
and SRF is the stress reduction factor

The rock mass quality term Q' has been used to assess the competency of the hangingwall and footwall of the West Reef for stope stability purposes, where Q' is defined with the effects of water and stress are not taken into account in:

$$Q' = \frac{RQD}{J_n} x \frac{J_r}{J_a}$$

Mapping of the drives and West Reef, along with analysis of the additional geotechnical boreholes has produced Q parameter ranges as presented in Figure 55.

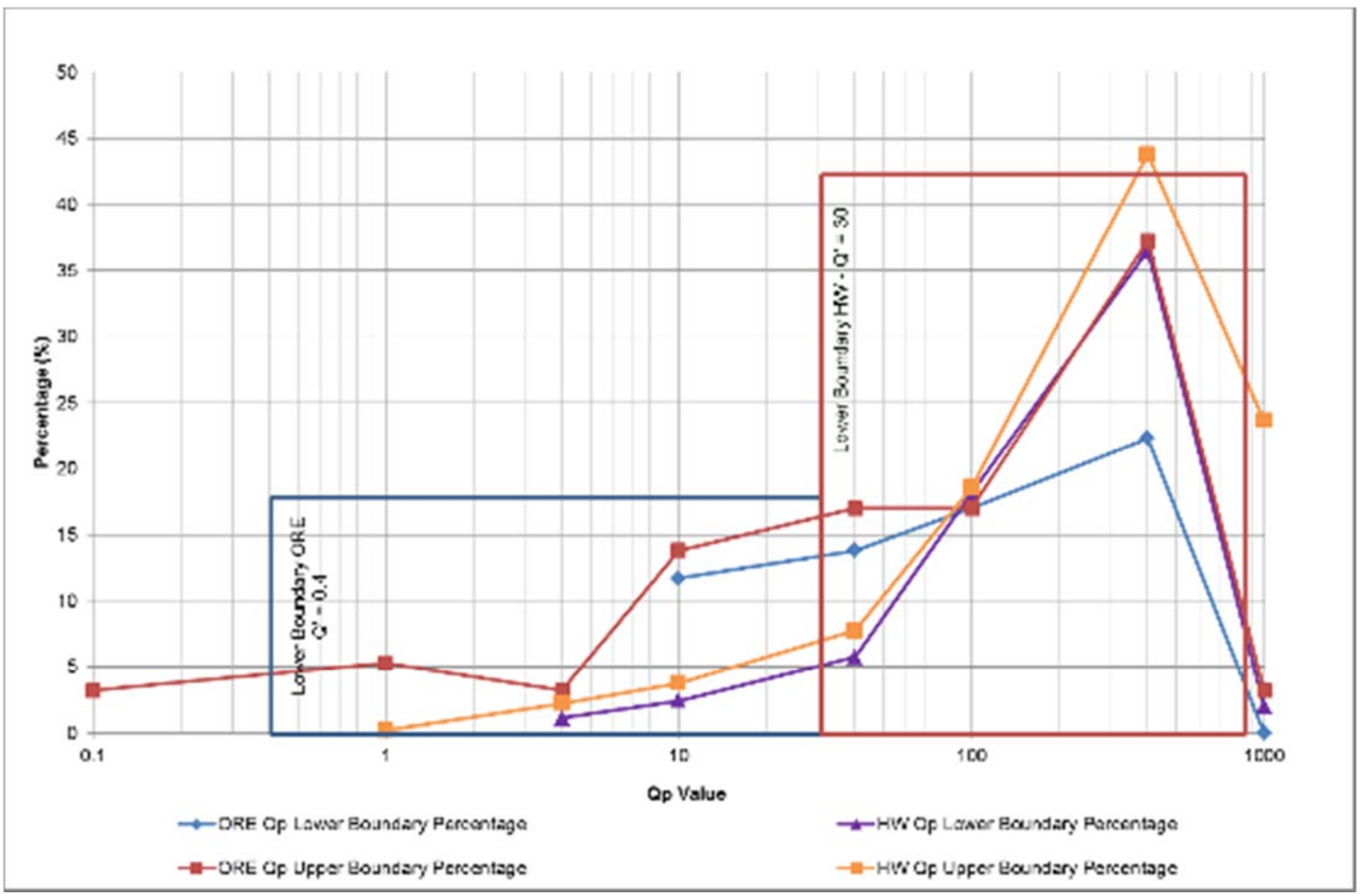


Figure 55 Q' Parameter Range for Orebody

For the hangingwall a range encompassing 70% of the Q' data set was used which yielded a lower bound Q' value 30, with a mean of about 120. For the orebody, the higher Q' values were omitted as these values were considered to not reflect the potential influence of the orebody shears on stope stability. The resulting orebody data set yielded a lower bound Q' of 0.4 with an average of about 10.

The lower bound Q' parameters for both orebody and hangingwall were utilised in the Mathews *et al*., (1981), stability graphs to determine the modified stability number N', which is defined as follows:

$$N' = Q'.A.B.C$$

Where Q' is as defined previously, and

A = rock stress factor,

B = joint orientation factor, and

C = gravity adjustment factor

The derivation of N' is shown in Table 72 which also shows the values for factors A, B and C. Characteristic values of Q' parameters were selected from the data in Table 72 with values close to lower bound being used to characterise the ore zone and upper bound values used to characterise the more competent footwall and hangingwall units. Note that the mining induced stresses were determined from numerical modelling described in the next Section of the report.

Table 72 Summary of Derivation of N'

	Stability Number Calculations	
Parameter	**Stope Back**	**Hangingwall**
Rock Quality Designation - RQD %	20	80
Number of joint sets - Jn	12	4
Joint surface roughness - Jr	1	3
Joint wall alteration - Ja	4	2
Q'	0.4	30.0
UCS (MPa) Sigma C	30	50
Sigma 1 (MPa)	60.00	5.00
Stress:strength ratio	0.5	10.0
Factor A	0.0	1.0
Angle between stope face and daylighting joint	30	32.5
Factor B	0.2	0.25
Potential Failure Mode	Sliding	Slabbing
Dip of Stope Face	0.00	70.00
Dip of Critical Joint	73	-
Factor C	3.8	5.9
N = Q' x A x B x C	0.01	44.6

Analysis Results

The approximate dimensions of the Alimak stopes are indicated in Table 73.

Table 73 Alimak Stope dimensions

	Span	Strike length	Vertical height
Dimensions (m)	1.7 - 3.5	20	130 - 160

Table 74 indicates the stope back stability condition for a range of orebody horizontal thickness (table rows) and open stope strike length (table columns). The red boxes outline the range of stope spans and strike lengths identified in Table 73. The results of the analyses indicate that the stope back may require some form of support in order to maintain stability. However considering the practicalities of Alimak mining, wherein the stope back is only a temporary feature, support of the back is unlikely to be required. However the sill drive and Alimak nests will need to be supported.

Table 75 indicates the stope wall stability condition for a range of stope vertical heights (table rows) and open stope strike length (table columns). The red boxes outline the range of stope heights and strike lengths identified in Table 14. The results of the analyses indicate that a 20m strike length stope should remain stable without support. However because of the long wall height the stresses in the stope walls will be relatively low. Whilst this will not lead to stress related stope instability it may result in rock loosening which would result in waste sloughing into the stope the result of which will be increased dilution. However the normal approach to Alimak raise mining is to support the walls of the raise and install cable bolts into the hangingwall. This approach is planned for Prestea and will go some way to limiting any hangingwall dilution.

Table 74 Stability Graph Results – Stope Back

Stope Back														
Hydraulic Radii for Various Stope Geometries														
OB Thickness	Stope Length (m)											N	0.01	
Stope Span (m)	2	4	6	8	10	18	20	30	40	50	100	Key		Hydraulic Radii (m)
1	0.33	0.40	0.43	0.44	0.45	0.47	0.48	0.48	0.49	0.49	0.50		Stable	0.40
1.2	0.38	0.46	0.50	0.52	0.54	0.56	0.57	0.58	0.58	0.59	0.59		Unsupported Transitional	1.1
1.5	0.43	0.55	0.60	0.63	0.65	0.69	0.70	0.71	0.72	0.73	0.74		Stable w ith Support	3.1
1.75	0.47	0.61	0.68	0.72	0.74	0.80	0.80	0.83	0.84	0.85	0.86		Supported Transitional	4.5
2	0.50	0.67	0.75	0.80	0.83	0.90	0.91	0.94	0.95	0.96	0.98		Unstable	4.5
2.5	0.56	0.77	0.88	0.95	1.00	1.10	1.11	1.15	1.18	1.19	1.22			
3	0.60	0.86	1.00	1.09	1.15	1.29	1.30	1.36	1.40	1.42	1.46		Stope Dip	0.00
3.5	0.64	0.93	1.11	1.22	1.30	1.47	1.49	1.57	1.61	1.64	1.69			

Table 75 Stability Graph Results – Stope Walls

Stope Walls														
Hydraulic Radii for Various Stope Geometries														
Vertical	Wall/Stope Length (m)											N	44.6	
Wall Height	2	4	6	8	10	18	20	30	40	50	100	Key		Hydraulic Radii (m)
20	0.91	1.68	2.34	2.91	3.40	4.88	5.16	6.23	6.95	7.46	8.77		Stable	10.17
40	0.96	1.83	2.63	3.37	4.05	6.33	6.80	8.80	10.31	11.50	14.93		Unsupported Transitional	13.2
60	0.97	1.88	2.74	3.55	4.32	7.02	7.61	10.21	12.30	14.02	19.48		Stable w ith Support	14.3
80	0.98	1.91	2.80	3.66	4.47	7.43	8.10	11.09	13.61	15.75	22.99		Supported Transitional	16.3
100	0.98	1.93	2.84	3.72	4.57	7.70	8.42	11.70	14.54	17.01	25.78		Unstable	16.3
150	0.99	1.95	2.89	3.81	4.71	8.09	8.89	12.63	15.99	19.04	30.74			
175	0.99	1.96	2.91	3.84	4.75	8.21	9.03	12.92	16.46	19.71	32.53		Stope Dip	70.00
200	0.99	1.96	2.92	3.86	4.78	8.30	9.14	13.15	16.84	20.24	34.02			

2D Numerical Modelling

Whilst 3D modelling generally provides more a more accurate assessment of mine element stability than 2D modelling, 2D models are easier to create and mine element dimensions are quicker to change. Therefore a number of 2D models were created to assess the stability of different sizes of stope, rib and sill pillar. The outcome of these analyses is presented below.

Input Parameters

Material parameters used in the 2015 FS shrinkage stoping study were used. The Hoek Brown constitutive model has been used to represent the lithologies at Prestea because it allows the scaling from intact laboratory and logging derived rock parameters to a rock mass strength. UCS, Young's Modulus, and Poisson's Ratio values are collated from laboratory testing for the BGW and BPH. The UCS of the ore units was calculated from laboratory tests in 2013 and Young's Modulus and Poisson's Ratio were obtained from the additional testing in 2015. The GSI values are collected from core logging (directly converted from RMR=GSI). The mi values are empirically derived using knowledge and experience of similar rock masses. A disturbance Factor (D) of zero was applied as production blast holes will be drilled parallel to the orebody strike and it is assumed that good practise blasting methods will be carried out to minimise the propagation of blast damage into the rock mass. The parameters used are presented in Table 76.

Table 76 Rock mass properties for modelling

Unit	UCS (MPa)	GSI	mi	D	Rock Mass Young's Modulus (MPa)	Poisson's Ratio
BGW (Greywacke)	116	62	18	0	26 356	0.32
BPH (Phyllite)	65	59	10	0	33 355	0.33
ORE	50	37	20	0	2 600	0.30
GBX (Orebody Shear)	15	19	19	0	4 350	0.30

The in situ stresses used in the 2015 FS shrinkage stoping study were used in the modelling. The vertical field stress material loading was calculated using the weight of the overlying rock. Calculations of the horizontal stresses from values documented in previous studies were used to formulate the orebody parallel and perpendicular horizontal stresses. The field stress gradients as applied to the 2D models are presented in Table 77.

Table 77 Field stress for modelling.

	Gradient (MPa/m)	Stress at 17L (MPa)	Stress at 30L (MPa)
Vertical Stress	0.044	12.0	31.8
Horizontal Stress Perpendicular to Orebody	0.042	13.5	32.4
Horizontal Stress Parallel to Orebody	0.040	14.9	32.9

Results of 2D Rib Pillar Modelling

A horizontal section was cut through the orebody model at an elevation of 4220 m which corresponds to a mid-point elevation between the 21L and 24L. A mid-strike central rib pillar of 25 m strike length was modelled centred around Northing 8080N.

The orebody north and south of the central rib pillar was split up into 20 m long stopes with rib pillars of varying thickness left between the stopes. Figure 56 shows a part of the Phase^2 rib pillar model showing 3 stopes to the north of the central rib pillar. Also shown are the outlines of the Alimak raises (2.7 m square) which were excavated before stope mining. Although not visible in Figure 56 a 0.3 m thick shear was modelled at the hangingwall and footwall contacts of the orebody. The model assumed that all stope when mined out remained as open voids.

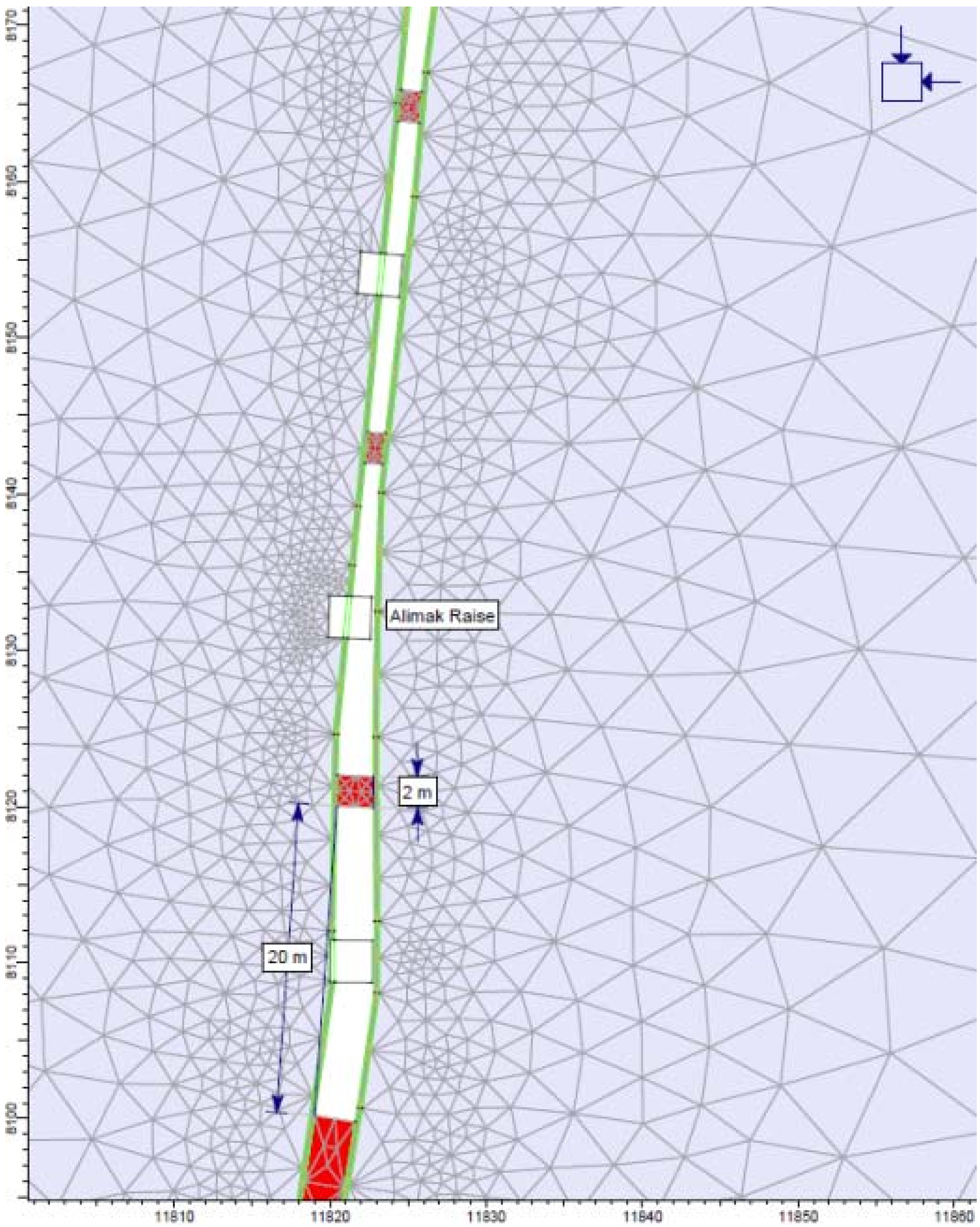


Figure 56 Phase2 Rib Pillar Model at 4220m El, 20m stope, 2m rib pillars

Constant boundary stresses were applied to the model using the field stress gradients presented in Figure 56. At the 4220 m level, which is approximately 825 m below surface, the following stresses were applied to the model:

- Vertical (out of plane) stress – 36 MPa
- Horizontal stress perpendicular to orebody – 35 MPa
- Horizontal stress parallel to orebody – 33 MPa

The results of the analysis are presented graphically in Figure 57 which shows for the 2 m and 3 m rib pillar sizes modelled the average pillar factor of safety plotted against the horizontal orebody thickness for pillars located north of the central permanent rib pillar.

For both pillar sizes there is a trend of decrease in pillar strength factor with increase in orebody width. The pillar strength factor for the average orebody width of 1.75 m is slightly less than 1.2 for both the 2 m and 3 m rib pillars. For the maximum orebody width of 3.5 m the pillar strength factor reduces to just above 1.1

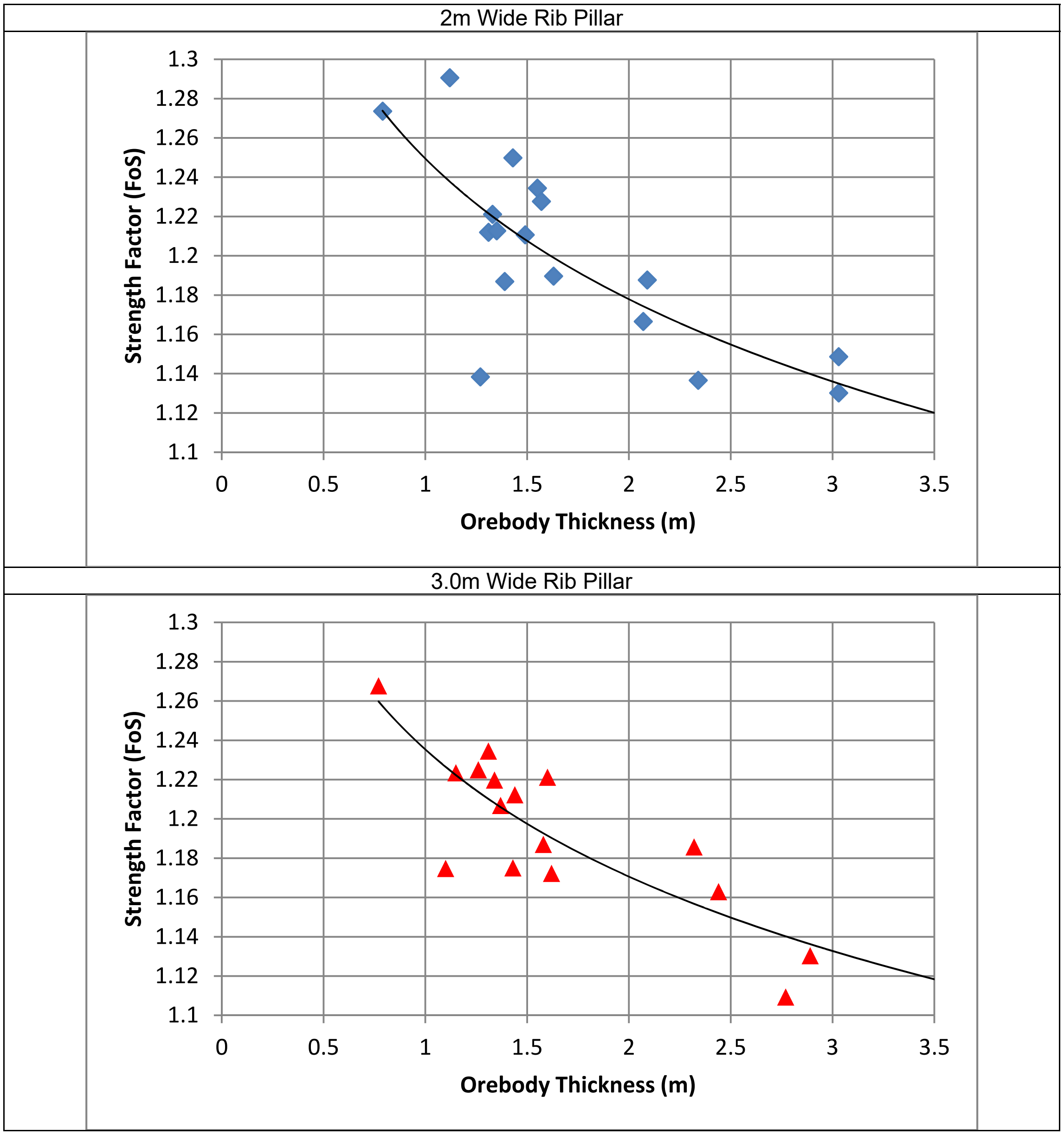


Figure 57 Results of 2D Rib Pillar Analysis

Results of 2D Sill Pillar Modelling

A cross section was cut through the orebody wireframe at 8260N. This is an area where the vertical extent of west reef is at a maximum and allowed sill pillars to be placed on 17L, 21L, 24L and 27L. The Phase^2 model constructed for this analysis is presented in Figure 58. The top of the model is at ground surface.

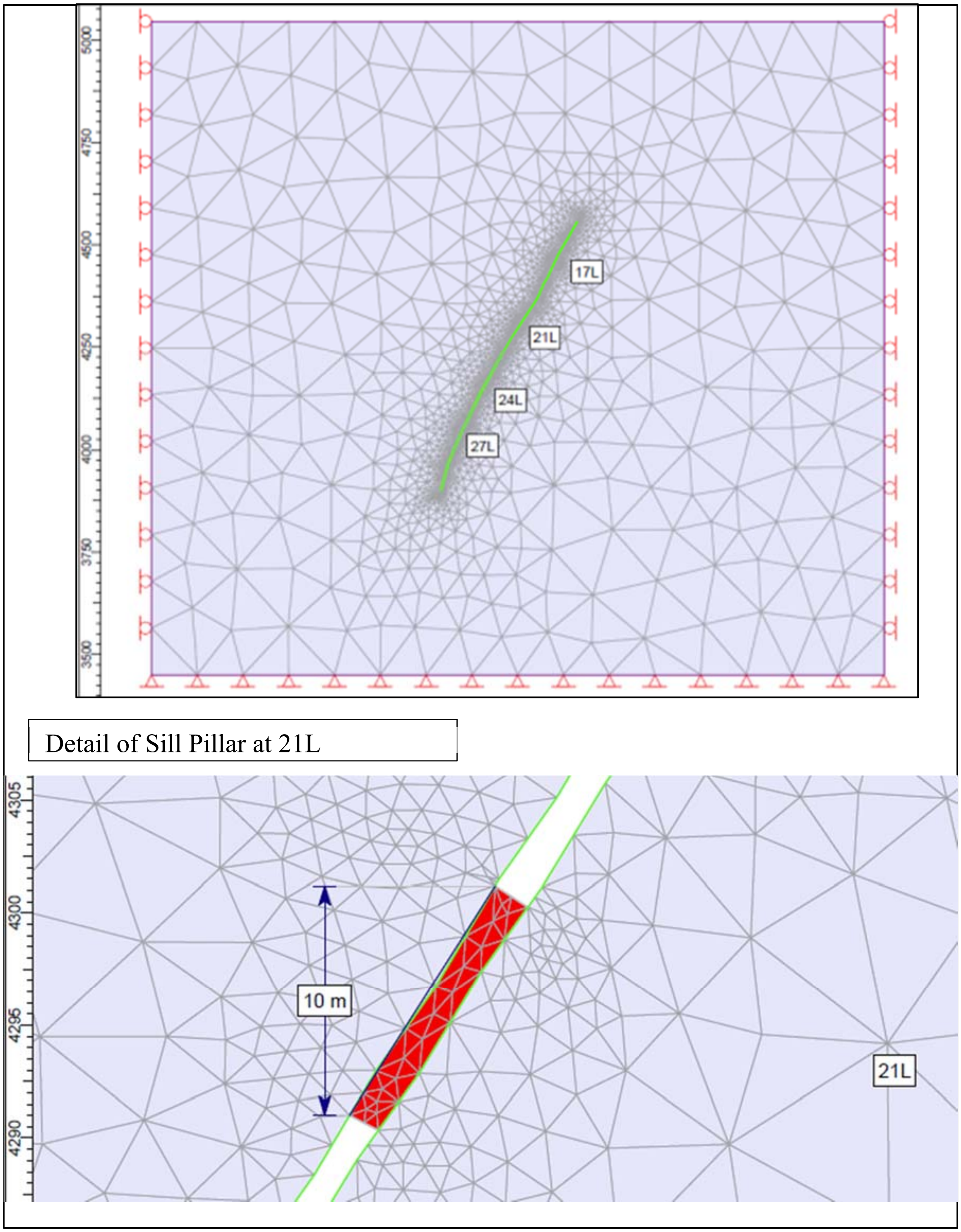


Figure 58 Phase2 Sill Pillar Model at 8260N

A number of models were created with sill pillars of varying vertical height. The results of the analysis are presented graphically in Figure 59. The graph shows the relationship between vertical pillar height and average factor of safety measured down dip through the middle of the sill pillar.

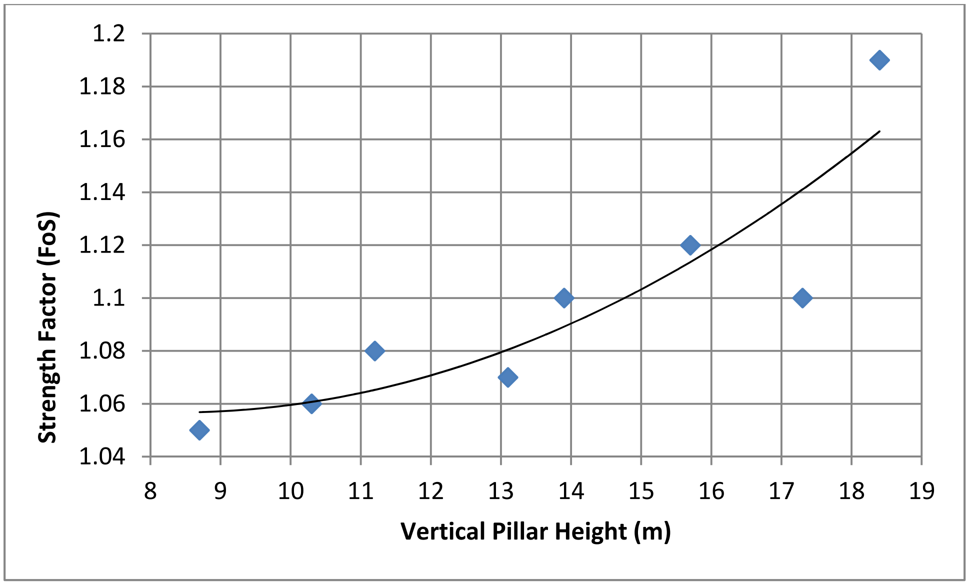


Figure 59 Results of 2D Sill Pillar Analysis

The results of the 2D modelling indicate that the factor of safety of a 10 m high sill pillar is about 1.06, whilst that of a 15 m high sill pillar increases to 1.1. The choice of sill pillar size should also take into account the results of the 3D modelling described in the next section.

16.3.4 3D Numerical Modelling

As part of the technical study, SRK has undertaken 3D numerical modelling analyses to confirm the stability of Alimak raise stope dimensions developed from the results of the 2D modelling.

Model Geometry, Geology and Limitations

Two models were analysed in FLAC3D, each model presented a stope width of 20 m including a 2 m rib pillar and an approximate vertical height of 135 m (corresponding to 160 m along the ore body). The sensitivity of the sill pillar thickness was analysed through the two distinctive geometries as follow:

- Case 1: 15 m sill pillar
- Case 2: 10 m sill pillar

For the purpose of the stope stability analyses, only a part of the stopes was modelled in 3D as presented on Figure 60. The modelled stopes correspond to the Northern part of the West Reef orebody, comprised between 17L and 24L. This compromise enabled to define a finer mesh around the area of interest (rib pillars, sill pillars) and get more accurate results while reducing greatly the software computing time.

Alimak stope wireframes have been created from the June 2015 updated ore body. The thickness of the stopes matches the thickness of the ore.

The greywacke (BGW) wireframe used for 2015 FS shrinkage stoping study was used in the modelling. The Phyllite (BPH) was assumed as host rock as per the 2015 FS study.

No mining sequencing was applied to the 3D numerical modelling, due to the complexity of the Alimak raise sequencing and the restricted time frame to deliver the study. The FLAC3D results therefore represent the last stage of mining, when all modelled Alimak stopes are mined out and are empty.

The 15 m sill pillar model was also used to check the stability of the rib pillars.

Note that the 3D model was created using the stope dimensions suggested by Manroc, 18 m wide stopes with 2 m wide pillars. GSR's concept design utilised 20 m wide stopes and 2 m pillars. Whilst it was possible to modify the 2D models to reflect the 20 m stope width, because of the complexity of building a 3D model it was not possible to change the 3D stope size. The results have been compared the results of the 2D 18 m and 20 m stope models for which there is little difference. It is concluded therefore that the results of the 18 m stope 3D model will reasonably represent the behaviour of a 20 m stope model.

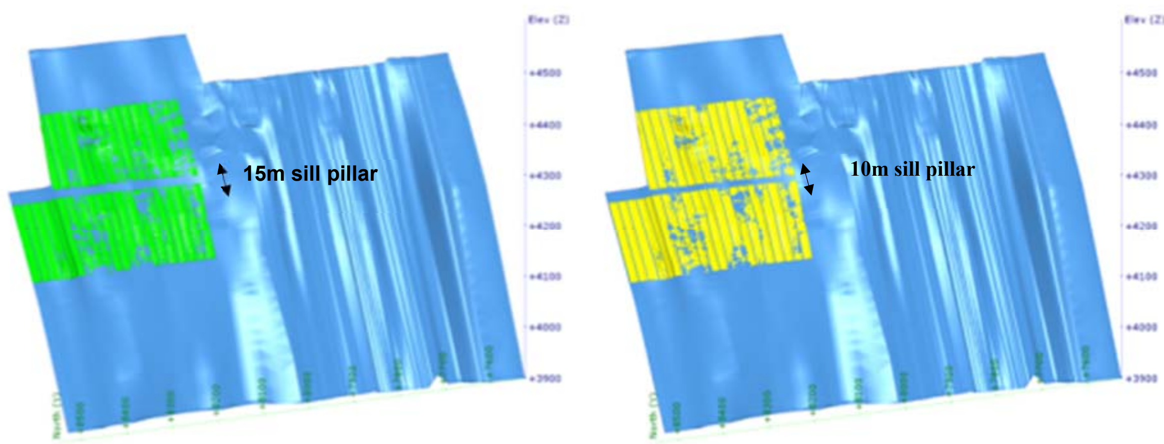


Figure 60 Updated June 2015 West Reef ore body and modelled Alimak stopes*

** Case 1: green stopes; case 2: yellow stopes*

Input Parameters

Material parameters and in situ stresses used in the 3D modelling are the same than the ones used in the 2D modelling presented in Table 76 and Table 77.

The Young's Modulus reduction technique was applied for the FLAC3D model to represent mining. By reducing the Young's Modulus value by 50% before extracting the stope completely, the breaking then drawing of the ore is simulated. As stipulated above, the 50% Young's Modulus reduction step was applied to the whole of the modelled Alimak raises prior to the extracting of the whole of the material, without taking into account any mining sequencing.

Rib Pillar Analysis

Figure 61 to Figure 63 compare the results of Phase^2 and FLAC3D analyses, in terms of total displacements, strength/stress ratio (strength factor or factor of safety) and maximum principal stress distributions across a plan section at mid stope level 4220 m. There are 18 stopes on the north side of the West Reef but the plots focus on the 5 stopes immediately north of the central rib pillar. To enable the comparison, the stope configuration for both analyses is identical with 18 m open stope and 2 m rib pillar.

A summary of the range of rib pillar parameters generated from both the Phase^2 and FLAC3D analyses is presented in Table 78. Numbers in brackets provide Phase^2 results for 2 m pillars separated by 20 m strike length stopes.

Table 78 Comparison of 2D and 3D Modelling Results, 2m Rib Pillar.

Parameter	Phase^2 2D Results	FLAC3D Results
Total Displacement (cm)	35 – 45 (30 – 60)	2 - 5
Strength Factor	1.1 – 1.4 (1.1 – 1.3)	1.0 - 1.5
Maximum Stress (MPa)	15 - 45 (25 – 45)	10 - 30

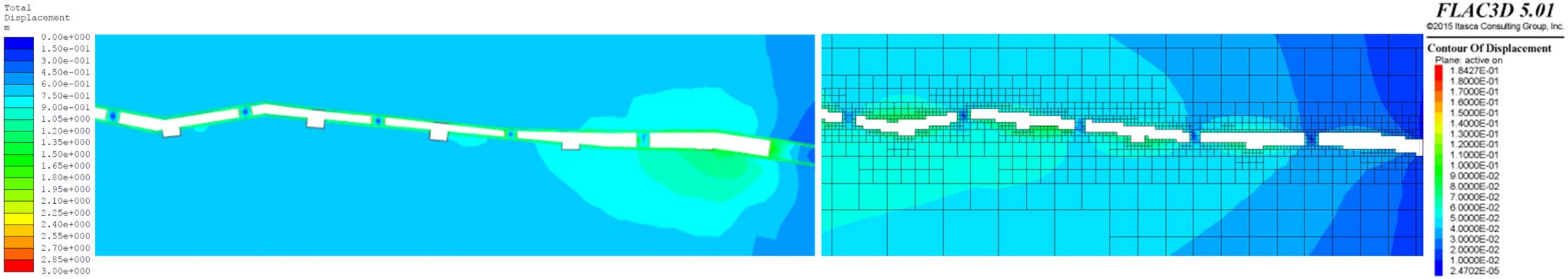


Figure 61 Total displacements 4220m plan (Phase² left, FLAC3D right)

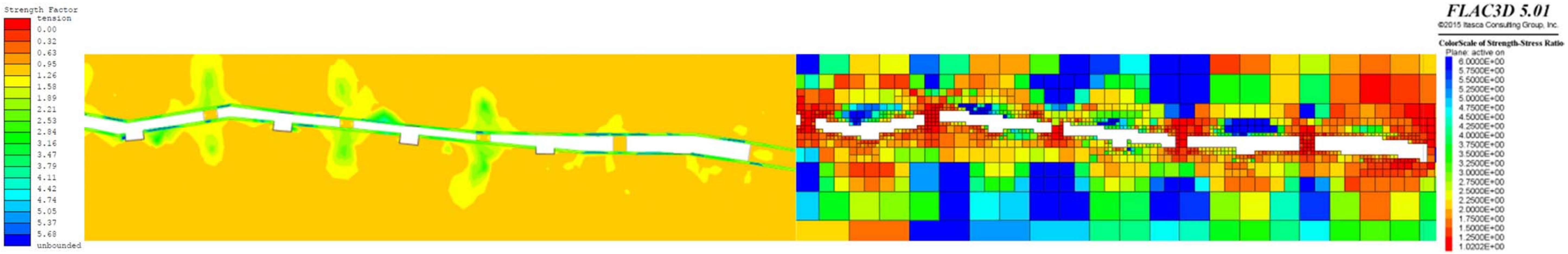


Figure 62 Strength/Stress ratio 4220m plan (Phase² left, FLAC3D right)

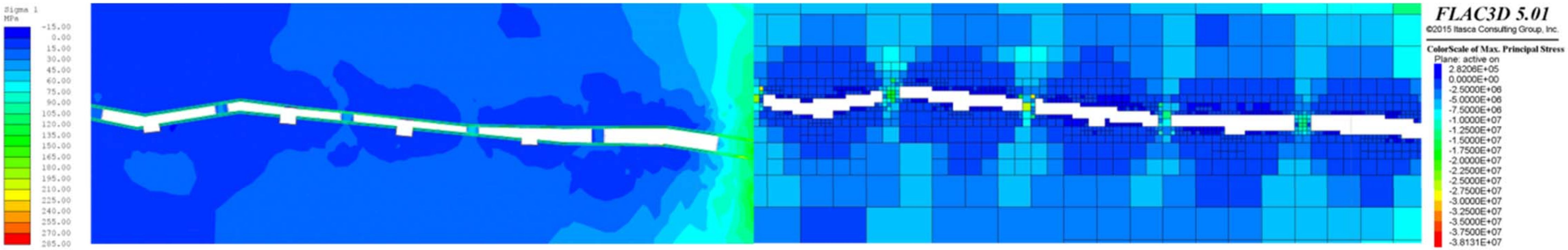


Figure 63 Max principal stress 4220m plan section (Phase² left, FLAC3D right)

The results indicate that the 2D modelling results yield higher displacements (an order of magnitude higher), slightly higher stresses and slightly lower strength factors than the 3D modelling results. This corresponds to the effect of the sill pillar, which cannot be modelled in 2D, deflecting the vertical stress field in the 3D model away from the underlying stopes as illustrated on the vertical cross section Figure 64. This results in lower deformation of the rib pillars. Also note that the 2D results for both 10 m and 20 m strike length stopes are very similar in magnitude. This implies that the results of the 3D 18 m strike length stope model are also applicable for a 20 m wide strike length stope.

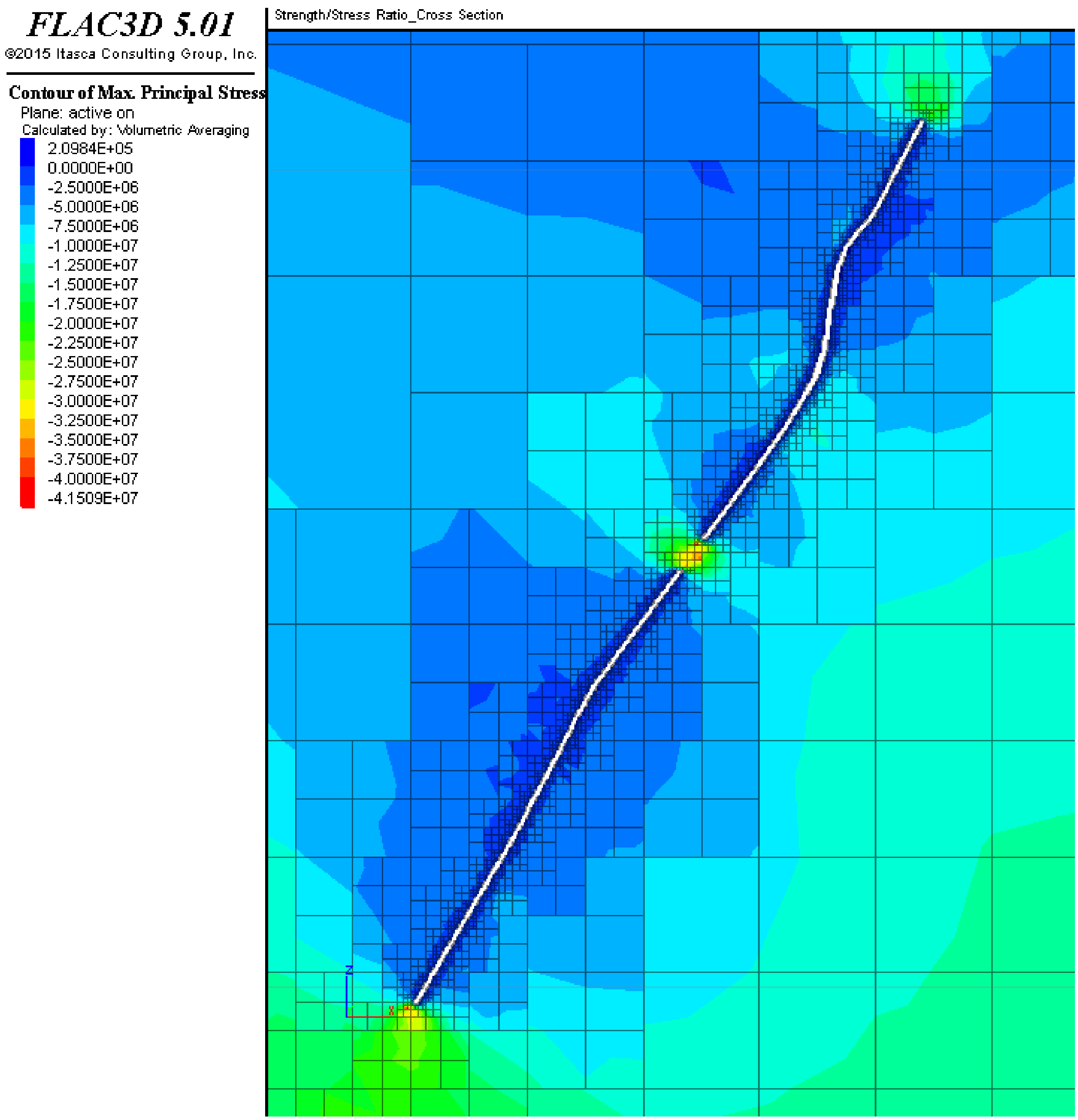


Figure 64 Maximum principal stress on vertical section

Sill pillar stability assessment

For the two cases, the strength/stress ratio (SSR) is reported for a set of points located on a plan section at mid-height of the sill pillar, as shown of Figure 65. This enables the average SSR across the section to be calculated and its sensitivity to the sill pillar thickness to be assessed. Strength/stress ratio statistics for the 2 cases are summarised in Table 79.

The minimum SSR calculated on a section across the middle of the sill pillar is 1.02 for the 10 m sill pillar and 1.08 for the 15 m sill pillar. These results support the 2D modelling results. Higher displacements are observed on the section with thinner sill pillar and higher maximum principal stresses are observed with a thinner sill pillar. Strength/stress ratios are higher in case 1 with 15 m sill pillar.

Table 79 Strength/stress ratio comparison on mid sill pillar section for the 2 cases

	Case 1 (15m sill pillar)	Case 2 (10m sill pillar)
Minimum strength/stress ratio (Factor of Safety)	1.08	1.02
Average strength/stress ratio (Factor of Safety)	3.7	1.8

It is also noted that the 2D analysis of a 10m sill pillar returns an average FoS of 1.06 according to the graph in Figure 59. The 2D analysis does not include any support that will be provided by rib pillars up dip and down dip of the sill. This 2D assessment of the stability of the sill pillar therefore represents a conservative (lower bound) estimate of the 10m sill pillar stability.

For the 3D analysis the software calculates a FoS for every element in the model. The lowest FoS values (which may only have occurred in one or a very small number of elements forming the sill pillar) is 1.02 for the 10m sill pillar and 1.08 for the 15m sill pillar. The average factor of safety is 1.8 for 10m and 3.7 for a 15m pillar respectively.

The 3D modelling returns higher factors of safety for the 10m sill pillar and as such are recommended.

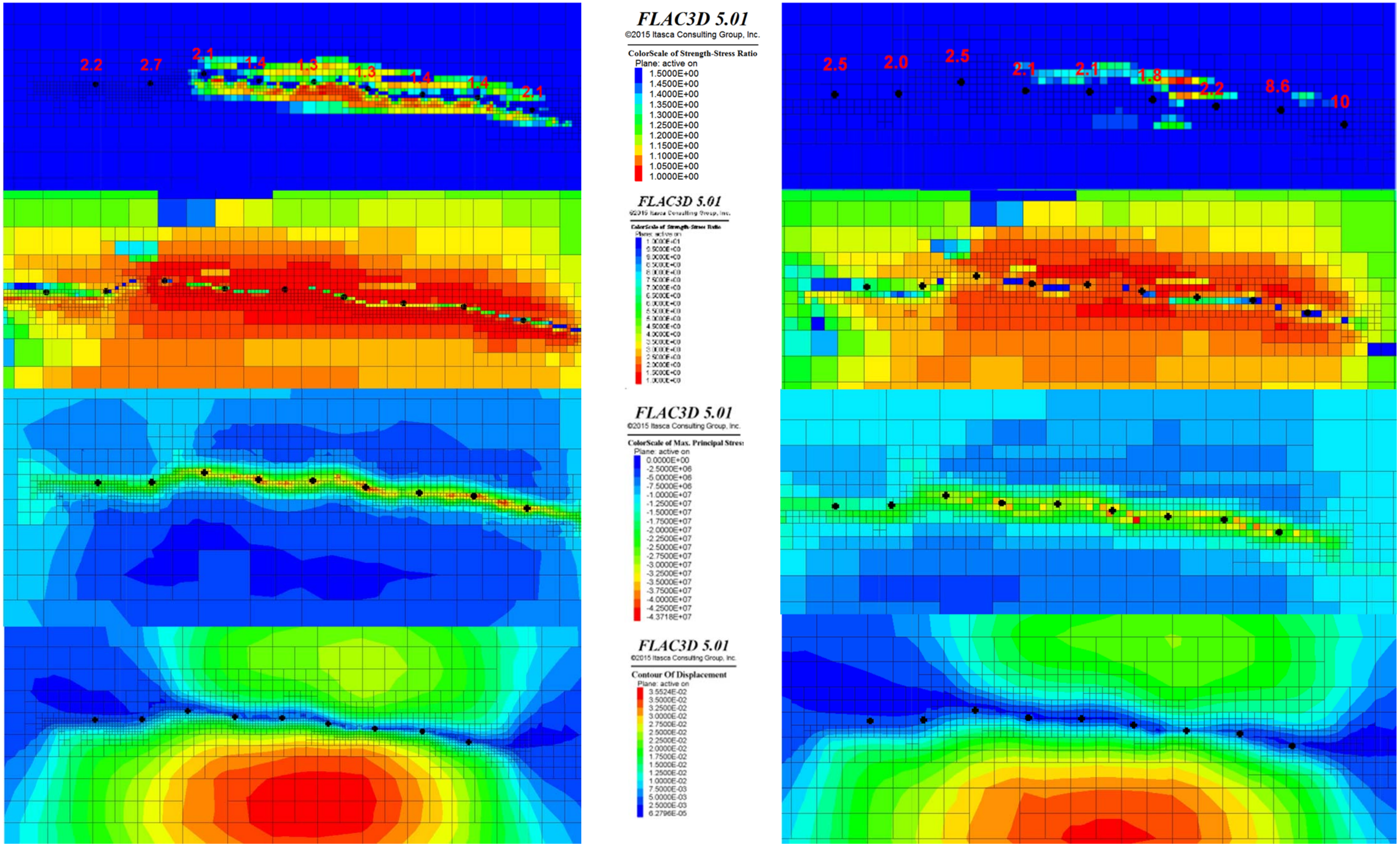
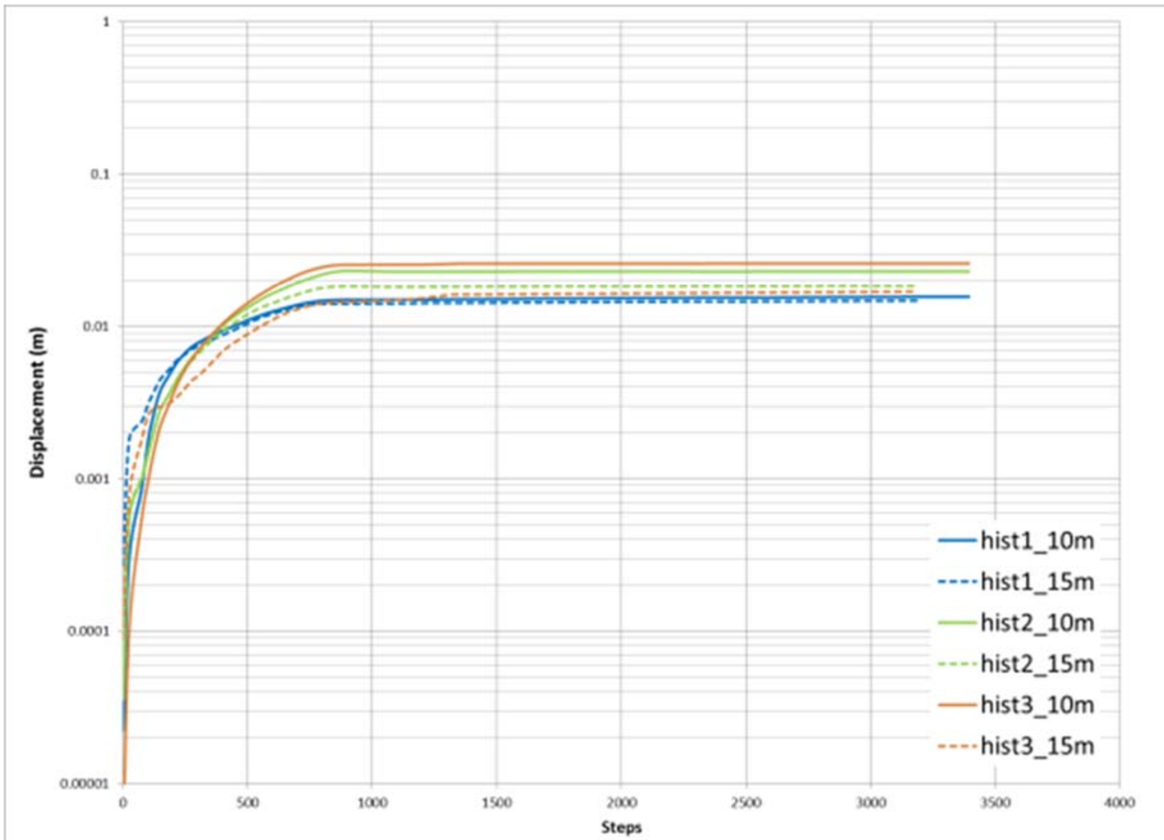


Figure 65 Strength/Stress ratio, Maximum Principal Stress and Displacement

Overall results

The movements of 3 theoretical points located at level 17L (1), on top of sill pillar (2) and at mid Alimak raise (3) were recorded during the 3D modelling computation. Results are presented in Figure 66 for both the 10 m and 15 m sill pillar. Results show that each point displacement is greater with the 10 m sill pillar. Displacements of these points range between 15 and 26 mm. The greater displacement is observed for the point located at mid Alimak raise.

The overall minimum SSR ratio calculated on the whole models is similar between the two cases, 1.0034 for the 15 m sill pillar case and 1.0028 for the 10 m pillar case as presented in Figure 66. These minimum values occur at the underside of the pillar. Both cases therefore show overall stable stopes when mined.



Hist1 : point at level 17L
Hist2 : point on top of sill pillar
Hist3 : point on rib pillar at mid raise

Figure 66 Displacements of theoretical points

The overall minimum SSR ratio calculated on the whole models is similar between the two cases, 1.0034 for the 15m sill pillar case and 1.0028 for the 10m pillar case as presented in the grey scale contour plot of strength: stress ratio (Figure 67). These minimum values occur at the upper and lower margins of the sill pillar. The plot also shows the distribution of strength: stress ratio through the modelled rib pillars. These values are all greater than one throughout but with the lowest values occurring close to the sill pillar.

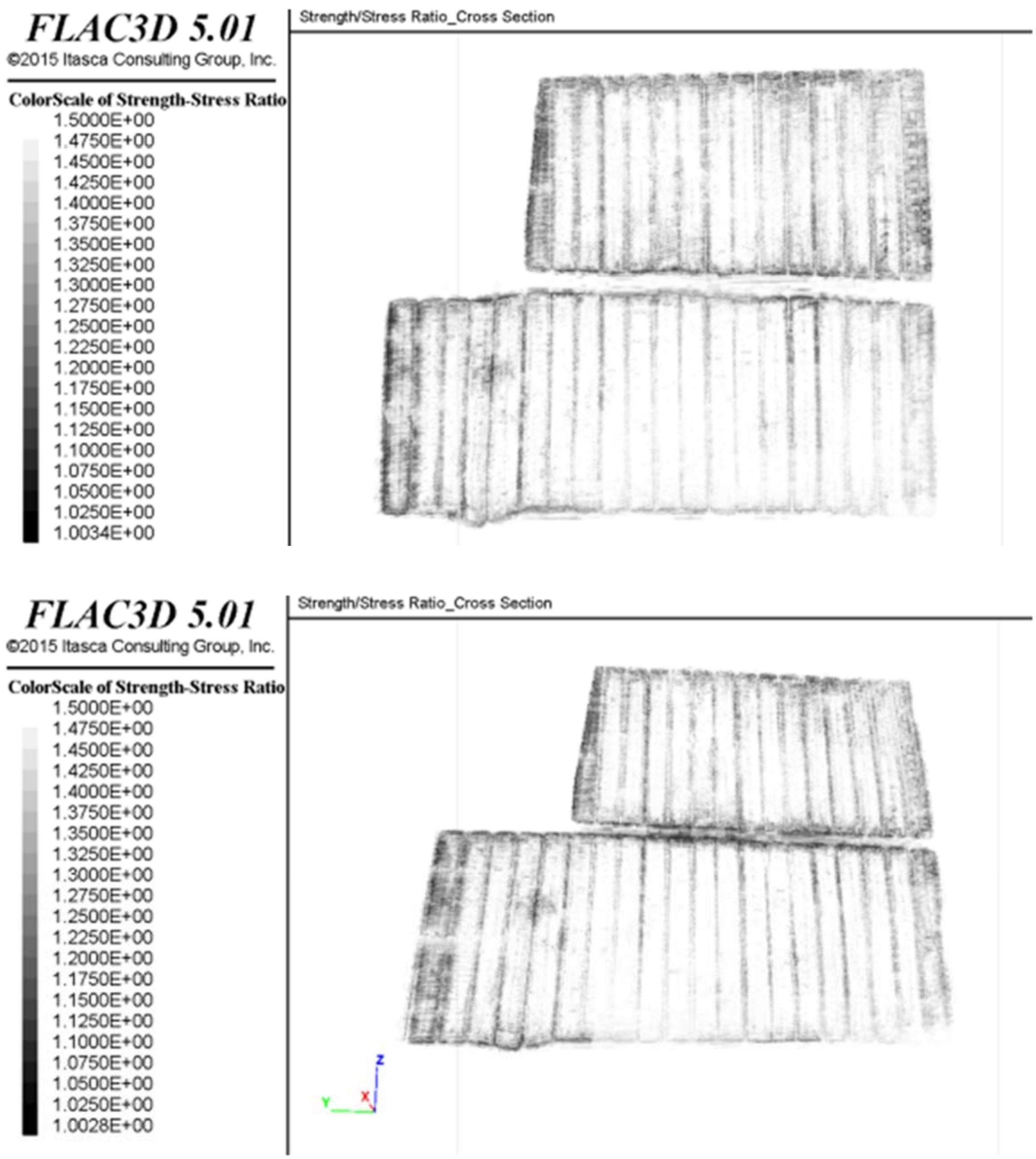


Figure 67 Strength/stress ratio – modelled Alimak stopes

16.3.5 Stope Sequencing

No specific stope sequencing studies were carried out for the Alimak raise mining option. These studies were undertaken for a previous shrinkage stoping study with stope dimensions similar to those of the Alimak stopes. That study considered both advance and retreat mining directions. The analysis indicated that both mining directions produced similar stress concentrations it was concluded that there was no preferential mining sequence for mining induced stress control or limitation. The stope sequence should therefore be governed by operational efficiency.

16.3.6 Recommended Pillar Design Dimensions

Based on the results of both the 2D and 3D analysis SRK recommends the design pillar dimensions to be used for Alimak raise stoping at Prestea are as follows:

- Rib pillar width – 3 m
- Minimum sill pillar vertical height – 10 m

16.4 Mining method

The planned mining method is mechanized shrinkage mining. This method uses mechanical raise climber technology (Alimak) together with longholes drilled from the raise climber. The method is an advance in terms of safety and productivity from conventional hand-held shrinkage mining. Safety is greatly enhanced both in raise development and stoping operations by having miners protected from rockfalls and removed from working on the broken muck pile. The productivity (per miner) is enhanced by introducing short longholes for stope production.

A typical cycle for Alimak stoping is:

- Raise development in the centre of a stoping block;
- Raise hangingwall rock support and screening;
- Drilling of longhole blast rings;
- Relocated of the raise climber infrastructure to the top of the stoping block, blasting of longhole rings and swell mucking; then,
- Final draw-down mucking and possible waste backfilling.

The cycle is shown in Figure 68.

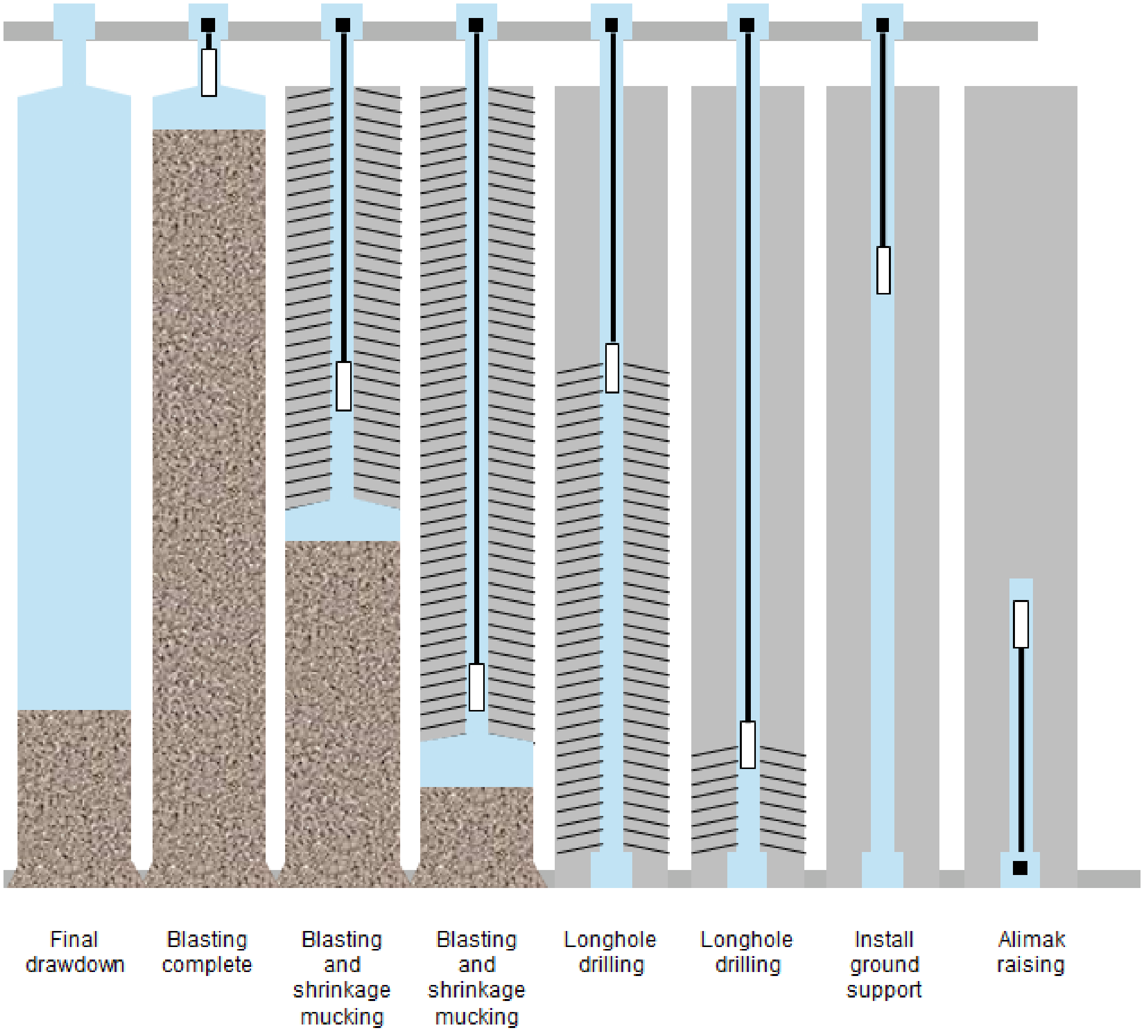


Figure 68 Typical Alimak stoping cycle

Small diesel and electric 2 m^3 LHD's will be used for rock haulage from the base of the stoping panel. Mine development will be by hand held drills with LHD mucking. Tramming of ore and waste along the two main levels (17 and 24) will be via an upgraded rail haulage system to Central Shaft.

16.5 Mineral Inventory for Mine Planning

16.5.1 Economic cut-off grade

The cut-off grade calculations are discussed in Section 15.1 of this document.

16.5.2 Alimak panel layout

The mine design comprises panels of stopes separated vertically by sill pillars. Each panel consists of:

- footwall drive, crosscuts to ore, ore drives along the vein and hangingwall Alimak nests at the base of panel extraction level;
- Alimak raises along the centre of the planned stopes;
- A top of Alimak panel sub-drift driven along the vein to connect the top of the Alimak stopes; and
- Shrinkage stopes.

A continuous ore drive is developed in the vein immediately above the sill pillar. These are developed large enough to allow access by the LHD's and create the initial void for the stope blasting. The cross section size is 2.7 m high by 2.7 m wide.

The Alimak raises are 3.0 m wide along strike of the vein and 2.7 m wide perpendicular to strike.

The sub-drifts connecting the top of the Alimak stopes are 2.5 m wide by 2.5 m high as they need to be large enough to allow passage of the parts of the dismantled Alimak between raises. The sub-drifts are to be developed and mucked hand held.

The Alimak stopes are drilled off from the Alimak platform using top access. The minimum stoping width applied is 1.5 m.

The general panel layout is shown in Figure 69.

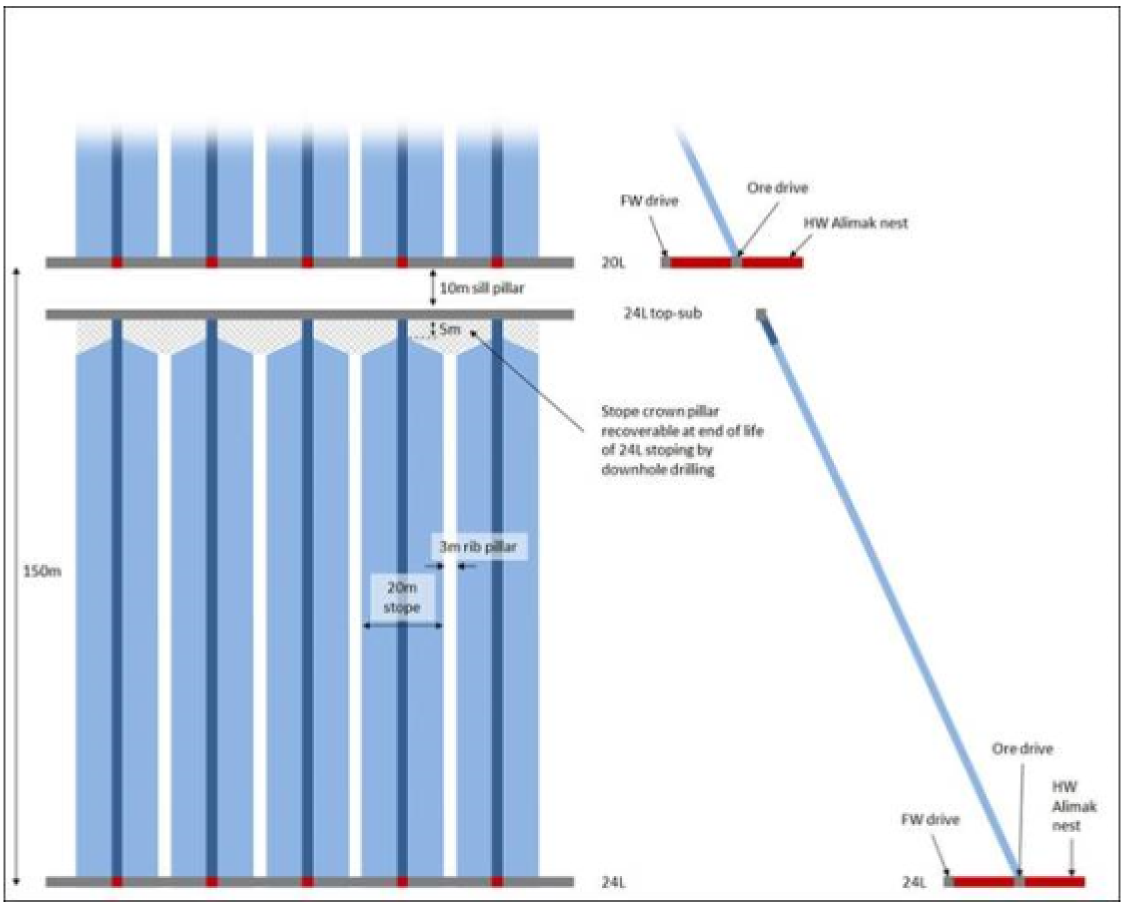


Figure 69 General Alimak stoping block arrangement

As part of the planning process, assessments have been made of the modifying factors that are required to be applied to the in-situ resources in order to classify them as Reserves in the context of a life of mine plan. The following subsections outline the factors for recovery of the orebody and the dilution that is expected to be incurred during mining. These factors are based upon the following:

- A detailed geotechnical assessment (Section 16.3); and
- Experience from other similar orebodies, for example Bulyanhulu Gold Mine in Tanzania; and
- Observation and measurements made from the existing West Reef workings.

16.5.3 Recovery

The rib pillars are planned to be 3 m wide along the strike of the orebody. They will run the entire length of each of the Alimak stopes. They are not planned to be recovered. 10m sill pillars will be left between a top sublevel above the stopes and the next main production elevation. 5m sill pillars will be left between the top of the stopes and the top sublevel (see Figure 69).

The application of these design parameters results in an overall recovery of around 80%.

16.5.4 Dilution

Dilution of the in-situ orebody is expected to come from the following sources:

- The development of 2.7 mH x 2.7 mW ore drives along the strike of the ore body. These drives will be often wider than the orebody and waste dilution will occur;
- The driving 2.7 mH x 3.0 mW Alimak raises up the dip of the orebody. These drives will be often wider than the orebody and waste dilution will occur;
- The application of a minimum mining width of 1.5 m across the entire orebody. This was applied assuming a dilution grade of zero;
- A waste rock skin of 0.2 m from the hangingwall containing zero grade. This is the average expected thickness of the graphitic material that will be mined as a result of blasting to the minimum width of 1.5 m; and
- A waste rock skin of 0.1 m from the footwall containing zero grade. Due to the dip of the orebody there is expected to be less dilution on the footwall and less overbreak is expected.

The application of these parameters results in an overall stope dilution of 33% and development dilution of 75%.

With respect to the graphitic schist, it should be noted that this comprises part of the orebody in places because it carries mineralisation. Therefore it has no impact on dilution in these areas. In other parts of the orebody where the schist comprises the hangingwall, it has been deliberately modelled with a low Q' value to simulate the effect of being sheared. During the subsequent modelling the hangingwall was found to be stable at the proposed mining widths. This conclusion is supported by observation of the historic existing mined out areas of the West Reef.

16.5.5 Material properties

A total of 392 samples were taken at Prestea for density analysis with an average density of fresh material of 2.71t/m3. Samples were taken from underground drilling in 2005 and 2013.

16.6 Mine Design

16.6.1 Principal parameters for stope design

The geotechnical parameters for design are summarised in Table 80.

Table 80 Principal Geotechnical design parameters

Individual stope strike length	20 metres
Stope height	up to 160 metres
Rib pillars (sacrificial) between stopes	3 metres
Main sill pillar thickness	10 metres

16.6.2 Mine design

Figure 70 shows the target West Reef mineralization in relation to the existing Prestea underground infrastructure and historical mining voids. The mine plan will fully take advantage of the existing infrastructure.

The Central shaft will be the main access for hoisting ore and waste materials, personnel and supply materials, major fresh air entry, and other mine services. The existing 24 L will serve as the main haulage level. The existing 17 L will be another main level for personnel and supply materials, and temporary haulage level during the West Reef development stage.

The Bondaye shaft will serve as the main second egress, with functions of supplemental fresh air supply and the route of West Reef dewatering if excessive water needs to be dewatered during the later stage of mine life.

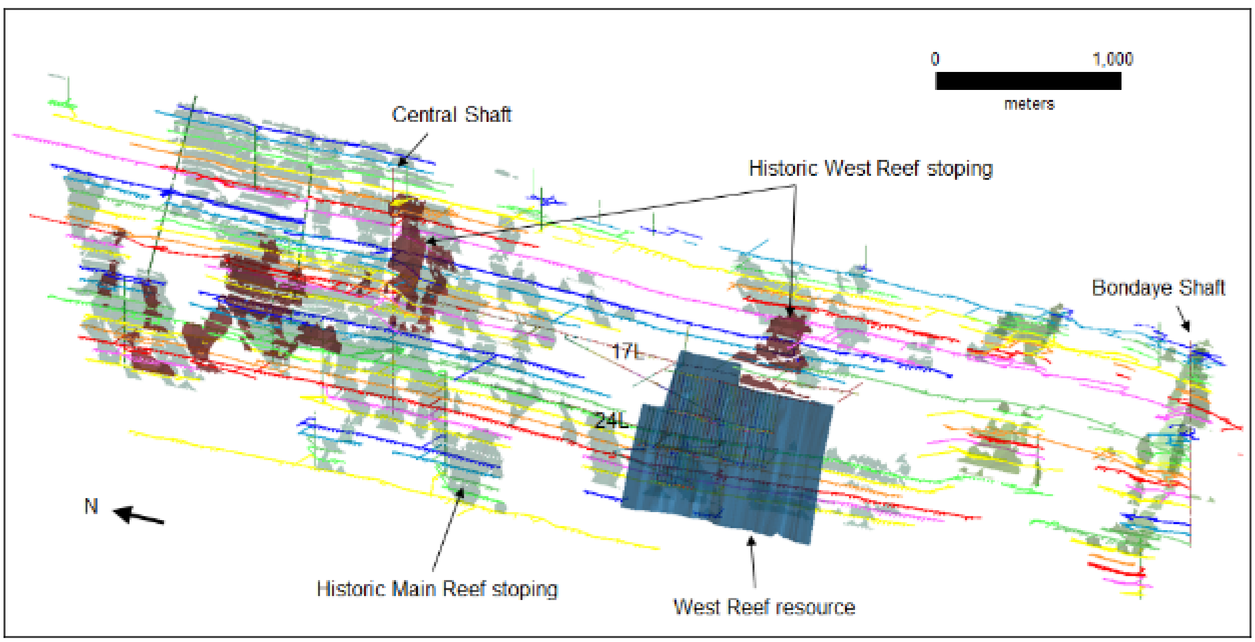


Figure 70 Prestea Mine general long-section

The West Reef has two primary access points from the existing mine network, 17 and 24 levels approximately 590 m and 890 m below shaft collar level from the Central Shaft. The West Reef is located 1,700 m south of the Central Shaft through which normal operations shall be accessed and 2,000 metres north of Bondaye Shaft which provides the emergency egress.

The existing 17 and 24 level rail systems are located on or about main reef, 150 m east of the West Reef location.

Three stoping panels require access to their base elevations; these are vertically spaced approximately 150 m apart on 21L, 24L and 27L. The 24L panel can be accessed directly from the existing 24L rail drive. The 21L panel is accessed via a decline from the existing 17L rail drive. The 27L panel is access via a decline to facilitate trucking with the start of the decline near the 24L rail drive.

Apart from the existing rail drives and extending two spur lines through 24L north and south crosscuts to the West Reef, the mine design is for small trackless equipment. The set-up for each panel is from the level. Each level has a full length of vein footwall waste development drive with a nominal 11m pillar to the vein. Crosscuts from the footwall drive to the vein are spaced at 23 m centres to line up with the spacing of 20 m strike length stops plus a 3 m wide separating rib pillar.

Alimak nests are excavated into the immediate hangingwall of the vein in each of these crosscuts. A typical nest arrangement is shown in Figure 71. The Alimak nests and ore drive intersections require rebar and screen support. Cable-bolting equipment will be on hand if required for additional support.

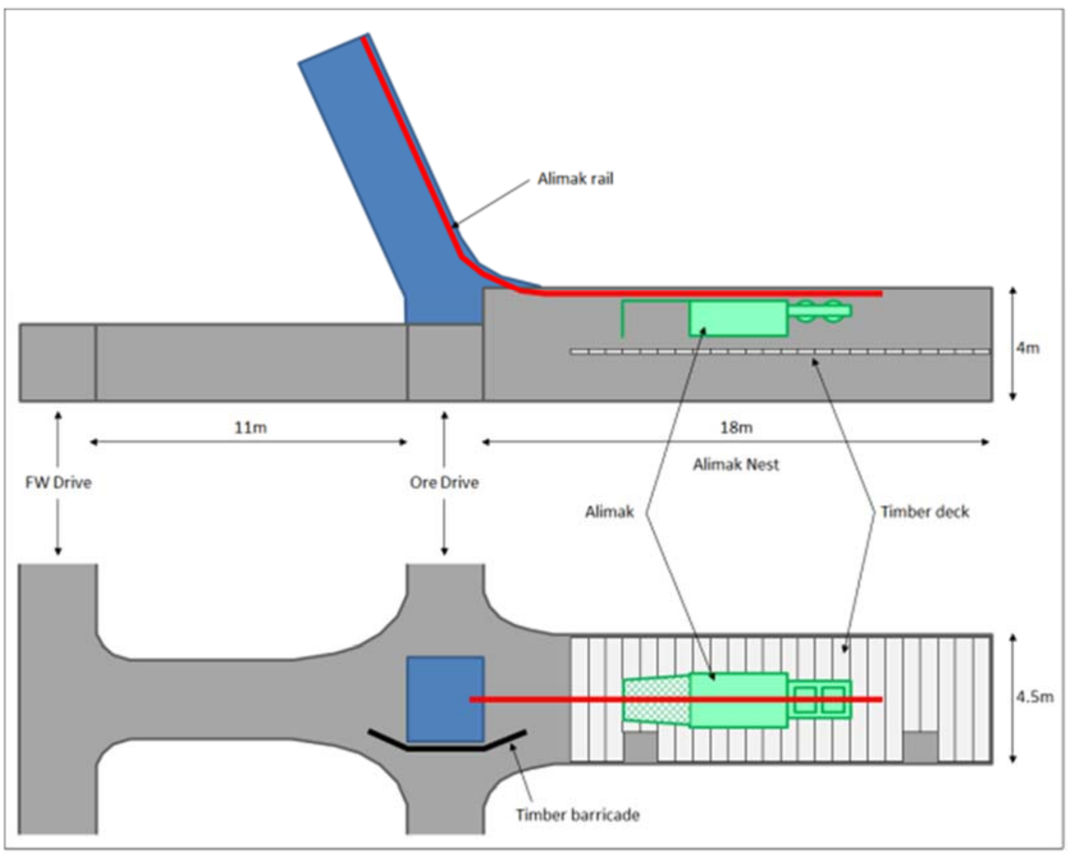


Figure 71 Typical Alimak nest layout

The mine design allows for LHD mucking of both raise development ore and stoping ore both directly from the crosscut and along strike from adjacent crosscuts.

A small decline for LHD access is designed from the 17L rail drive at a 1:7 gradient to the 21L. This results in a 1,000 m long decline. There is a connection midway to the primary return raise from 24L through to 17L. All material will need to be hauled out of the decline with LHD's until the 21L is reached. Once reached access to the ore-pass will be gained and waste dumping can occur into the voids of the Alimak stopes from the 24L panel immediately below. Access to the 27L is via a larger decline to allow the passage of 16 tonne class haul trucks. Trucks shall be used to haul up all development and stoping material from below the 24L.

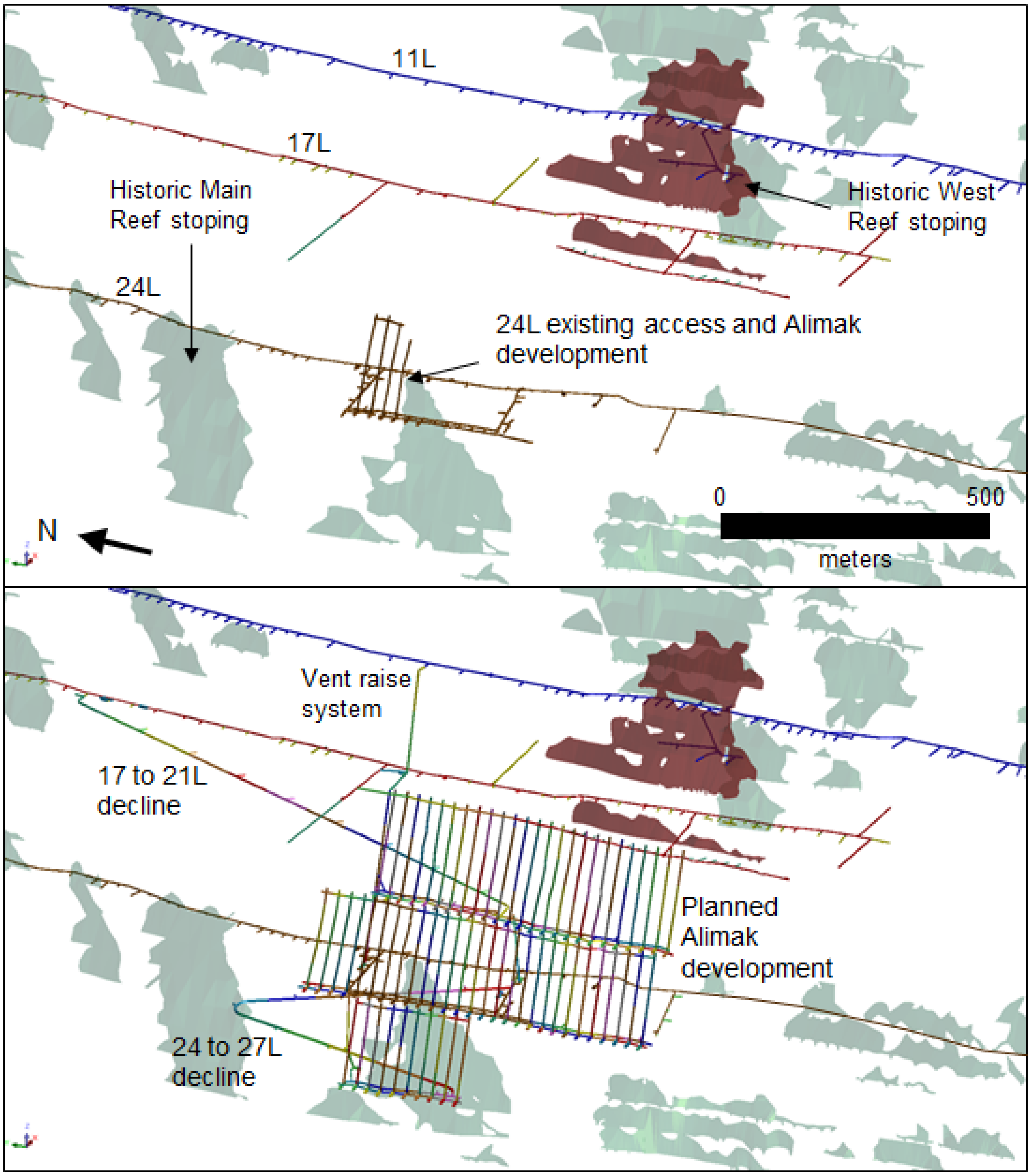


Figure 72 Current and final development design

Table 81 shows the development heading design specifications.

Table 81 Mine development specifications

Design String number	Heading type	Height	Width	Length	Comments
1	ramp	2.7	2.7	varies	
2	footwall drive	2.7	2.7	varies	nominal 11m pillar
3	crosscuts (footwall to vein)	2.7	2.7	11	
4	other waste	2.7	2.7	varies	
5	remucking bays	2.7	2.7	8	
6	orepass		1.8	varies	circular
7	ventilation raise	2.7	2.9	varies	
8	Alimak nests	4	4.5	18	
9	sump	2.7	2.7	8	
10	ladder-raise	1.2	1.2	varies	Ore-pass finger raise
11	main access to west reef	2.7	2.7	varies	
12	workshop	3	4	45	total length of excavation
16	ore drives on vein	2.7	2.7	varies	
17	Alimak stope raise	2.7	3	varies	3m perpendicular to vein
18	Alimak top connection	2.5	2.5	varies	
20	trucking decline below 24 level	3.5	3.5	varies	
21	footwall drive of trucking size	3.5	3.5	varies	

16.7 Development and Production Schedule

16.7.1 Material handling capacity

The Central Shaft hoisting capacity from the 24 Level loadout facility is calculated to be more than 1,300 tonnes per day. The capacity will increase when hoisting from 17L and skipping could be further increased. Hoisting duty cycle calculations are in Table 82.

The average hoist production requirement throughout the commercial production phase is 19,000 tonnes per month.

Waste development will be disposed of in suitable underground openings or batched to surface for disposal in the Prestea waste dump.

The waste associated with 17 L development will remain separate as the 20 L loading pocket will be used. This waste will be hoisted separately, although when connection of empty 24 L stopes and the 21 L footwall drive is available waste will be stored underground.

Table 82 Central shaft hoisting capacity from 24L load-out

Production inputs		
Payload	5.433	tonnes
Operation	16	hours per day
	25	days per month
Conveyance speed	7.12	meters per second
Allowance for creep in/out, acceleration, etc		
Production outputs		
Cycle time	234	seconds
Trips per hour	15.4	trips per hour
Hoisting capacity	84	tonnes per hour
	1,338	tonnes per day
	33,453	tonnes per month

Rail tramming on both 24 and 17 levels is via 8 tonne battery locomotive with ten 2.3m^3 rail cars. The system has a capacity range of 682 tonnes to 929 tonnes per day depending on the crosscut trammed from.

Table 83 Train cycle time

Activity	units	Comments	
Average loaded speed	km/h	7.5	
Average empty speed	km/h	8	
Number Of Cars In a Train	ea	10	
Capacity per car	m³	2	
Average Payload per car	tonne	3.4	
fixed time calculations			
Manoeuvre at West Reef	minutes	3	
Loading Time	min		
Passes to fill a Mine Car	ea	1.20	3.24 tonnes per LHD bucket
Time to load one rail car	minutes	1.64	
Tipping Time per car	minutes	0.8	
Total fixed time per train per cycle	minutes	27.45	

Mucking time analysis was conducted such that machine hour requirements could be calculated throughout the project, however the maximum machine hours was used to generate the LHD requirement to satisfy the production profile. Stope mucking was de-rated to 75% bucket capacity and limiting the LHD engine hours to 400 per month per machine to account for mechanical availability. This results in an average production capacity of 37.3 tonnes per hour for each LHD, and to achieve the average stope production rate of 19,000 tonnes per month (650 tpd) approximately 1.3 LHDs are required.

Direct train loading during production on 24 L will occur in many cases, however there are remucks included in the centre of the block to limit stope tramming distance. The remuck locations limit tramming for the rehandling to 130m resulting in train loading production capacity of 35.2 tonnes per hour for each LHD and 1.4 LHDs to achieve the average stope production rate of 19,000 tonnes per month (650 tpd).

Development of 17 level decline and later the production from 21 level requires an additional 2 LHDs. Similarly the sub-24 level decline and production will utilize 2 LHDs in concert with truck haulage, however as the 24 level production mining will be complete prior to the sub-24 level decline development starting the LHDs assigned to 24 level will be utilised.

A total of 5 LHDs have been included to account for materials handling, stope mucking and remuck rehandle throughout life of mine. This comprises:

- Three (3) LHDs for 24 level, of which two (2) will be utilized later for the sub-24 level decline; and
- Two (2) LHDs for 17 level decline and 21 level production

16.7.2 West Reef Scheduling Inputs

Shift roster is planned to be two shifts per day of ten hours duration each. The key scheduling parameters are listed in Table 84. All equipment is limited to 400 hours per month or approximately 6.7 hours per shift. Locomotive haulage similarly expected to be 12-14 hours per day 6-7 hours per shift (once crew travel to/from west reef deducted). The maximum allowable mechanised development advance is 14.4 m per day from all available headings. This is based on three 2.4 m rounds per shift in three independent headings. The first two months development schedule has been de-rated by 50% and allows only a maximum of 7.2 m per day during the ramp up period. This is based on blasting a single 2.4m round each day in each of the three headings.

Table 84 Maximum individual heading advance for scheduling

Maximum individual heading advance	per 24 hours	units	comments
All ramps and lateral mechanised waste	3.6	metres	average round advance is 2.4 metres
Alimak raising	3.7	metres	
Development on ore	2.4	metres	allows geological mapping time
Alimak nests	1.8	metres	oversize development
Sub-drifts (top of Alimak panels)	1.2	metres	reduced for slusher mucking

The daily ore production target is 650 tonnes. A stope in the blasting/shrink mucking cycle will produce 150-200tpd. A stope in final drawdown will produce 350-450tpd. The difference between the target of 650tpd and the stope ore production will be made up with development ore.

16.7.3 Schedule outputs

The mine development and stoping solids are scheduled in detail using Geovia MineSched software.

Mine development commenced in November 2016 on 24 level. Two crosscuts from the 24L rail drive (north and central) were driven initially and connect up to form a loop via the footwall drive.

This established the primary ventilation circuit into the workings; provides a secondary egress; and allows stoping to commence on 24L by driving the Alimak raises.

Stope raises are driven and connected by the raise crews horizontally at the top of the stoping block. Figure 59 shows the current development on 24L.

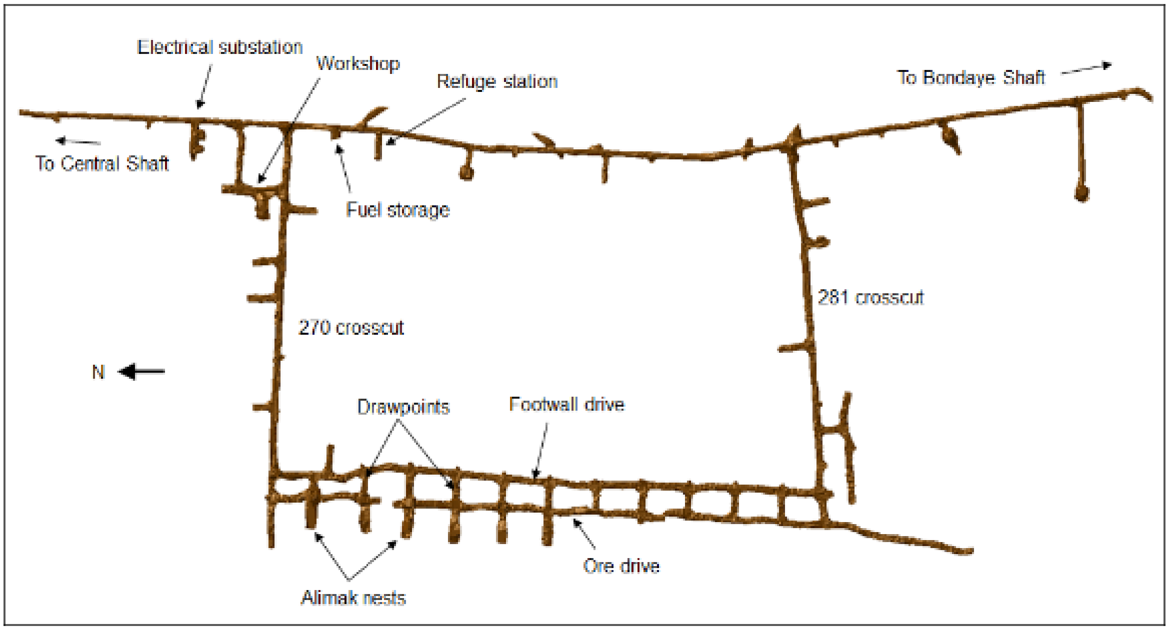


Figure 73 Current 24L development as-built

Figure 74 shows the current Alimak raise and stope development.

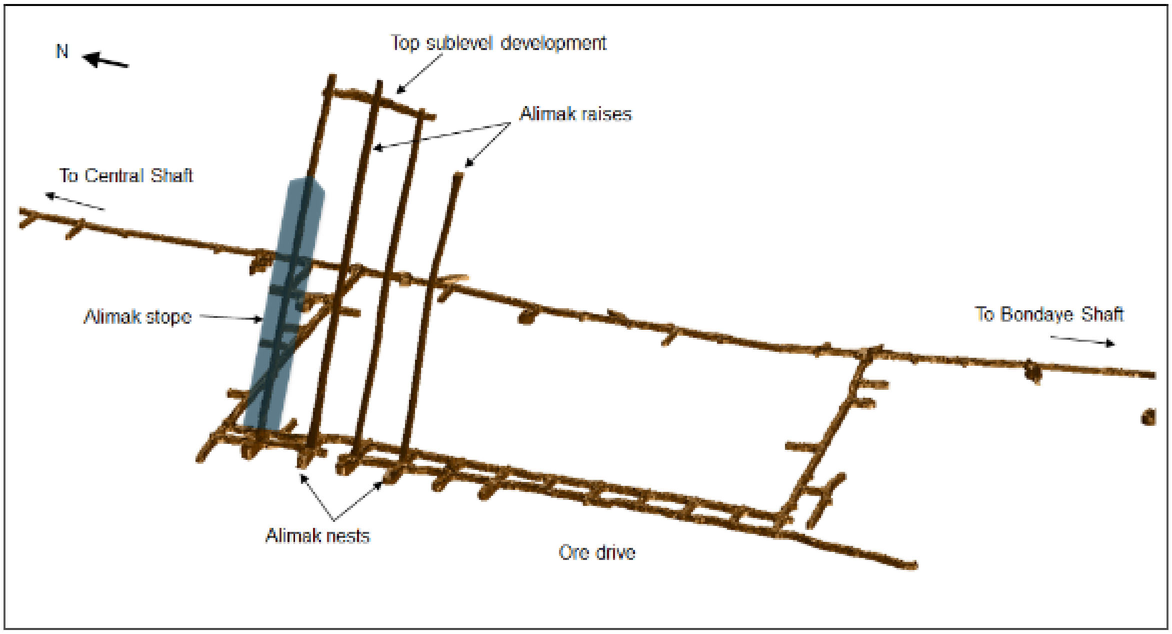


Figure 74 Alimak stopes and raises - as-built

Figure 75 shows the annual stoping and development plan.

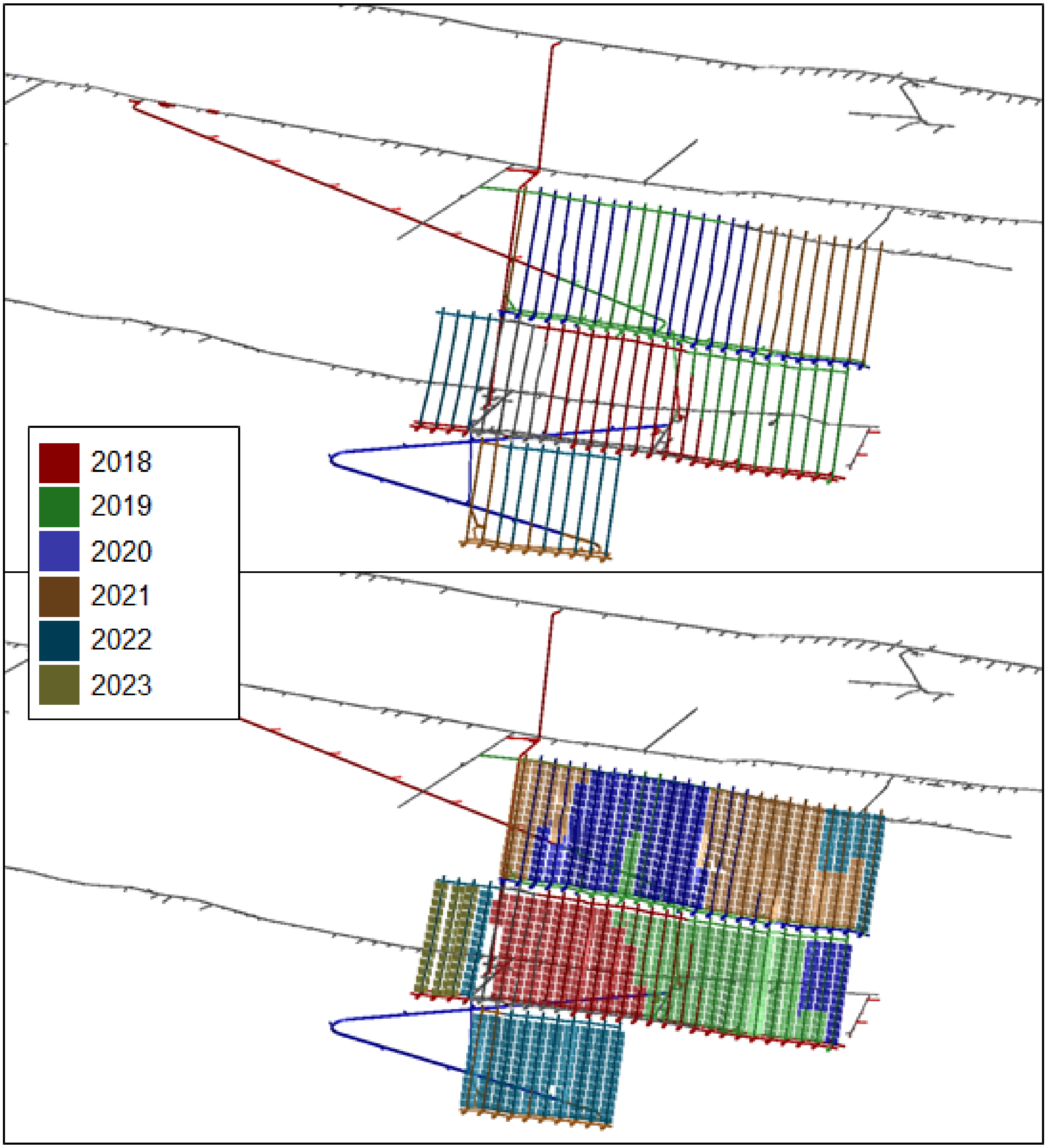


Figure 75 Annual stoping and development plan

Table 85 shows a summary of the production statistics by year.

Table 85 Summary statistics by year

Item			2018	2019	2020	2021	2022	2023	TOTAL
Development	Decline	m	846	410	908	227	-	-	**2,391**
	Ventilation raise	m	485	-	58	102	-	-	**646**
	Footwall drive	m	372	46	-	-	-	-	**418**
	Alimak nest	m	492	504	390	322	-	-	**1,708**
	Total waste	m	2,611	1,791	1,356	924	-	-	**6,682**
		t	67,312	42,133	45,977	26,208	-	-	**181,630**
	Ore drive	m	280	660	232	194	-	-	**1,366**
	Alimak raise	m	1,135	2,029	2,289	2,131	1,342	-	**8,925**
	Total ore	m	1,979	2,921	2,532	2,336	2,273	57	**12,099**
		t	45,325	62,169	55,054	50,837	49,307	1,199	**263,891**
		g/t	10.62	8.97	10.91	9.36	9.40	5.68	**9.80**
Stoping	Stope production	t	116,244	167,400	179,987	188,848	187,655	61,171	**901,306**
		g/t	14.34	13.25	14.93	13.01	11.54	9.94	**13.10**
TOTAL	Total waste mined	t	67,312	42,133	45,977	26,208	-	-	**181,630**
	Total ore mined	t	161,569	229,570	235,042	239,684	236,962	62,369	**1,165,196**
		g/t	13.30	12.09	13.99	12.24	11.10	9.86	**12.35**

16.7.4 Material flow

All material hoisted out of the mine will pass through either the 20.5L or 25L shaft load-out bins.

Currently there is only one rock pass at either location which means that scheduling of rock types (i.e. ore and waste) is of high importance.

Similarly on surface the bin is currently configured to handle only one material type. The surface bins discharge by way of a chute into trucks for surface haulage.

The 17L decline development waste will be trammed along 17L and pass through the 20.5L shaft load-out bins. To avoid dilution of the 24L ore this waste will be campaigned skipped separately to the surface bin. Once the ramp from 17L reaches the 21L footwall drive the material handling changes. The ore-pass from 24L is reached and ore is dumped into this pass to a chute loadout directly into 2.3m^3 rail cars on 24L. Waste can at the same time by dumped into empty Alimak stopes directly below the 21L sill pillar.

As such from this time on all hoisted material is handled along 24 level haulage.

Material handling requires a change once waste development from the 24L ramp commences in month 31. A total of 66kt of waste will require handling. A new shaft loading bin is excavated from 24 to 25L with independent load-out infrastructure directly below the existing loadout.

16.8 Mining Equipment

The production mining will be done utilising five Alimak units.

Mucking of all development and production material is by small LHD's (Sandvik LH203). Rail wagon loading is also by LHD. Five LHD units in total are eventually required, three of which are dedicated to the main production panel on 24L. Of the three LHDs one will be dedicated to stope mucking to the nearest remuck, another to rail car loading and the third as back-up to the other two LHD's.

For the 27L panel, trucks are introduced for haulage back to the 24L. Three trucks of the 16t capacity are included in the schedule.

Material transport vehicles are included for hauling mining materials from the rail network into the footwall drives. These units will be most utilised when serving the 27L and 21L panels via the ramps. There are also personnel and supervisor transport tractors.

Regarding rail haulage along both the existing 24 and 17 levels, the rail network has been upgraded to 24" gauge rail and utilizing 8 tonne battery locomotives pulling ten 2.3m^3 rail cars. Two locomotives are utilised on 24L which is the main haulage for all production and 24L and 27L development. One locomotive is to be utilised on 17L which will haul waste during ramp development to 21L and waste from 17-12 level raise.

16.9 Mining Personnel

The mining department personnel are summarised in Table 86.

Shaft operations include winder operators, pump station attendants, banksman, cage attendants, shaft loading and shaft maintenance miners.

Mining (development and tramming) includes hand-held miners, LHD operators, truck operators and locomotive operators.

Alimak miners include all raise miners, longhole drill and blast miners and direct Alimak maintenance personnel.

The personnel numbers for the mining department have been developed in line with the mine schedule and effective work time available.

Table 86 Mining department personnel by year (at December 2017)

Department	2018	2019	2020	2021	2022	2023
Management and Technical Services	17	17	17	17	17	11
Shaft Operations	72	72	72	72	72	37
Mining - development and tramming	78	78	49	49	49	44
Alimak Mining - Expats	9	9	9	9	9	9
Alimak Mining - National	31	31	31	31	31	0
Safety and training	5	5	5	5	5	5
Maintenance	46	46	45	45	45	35
Total	**259**	**259**	**228**	**228**	**228**	**141**

16.10 Mine Services and West Reef Infrastructure

16.10.1 West Reef Infrastructure

The West Reef area has two independent accesses by rail from the Central Shaft, one each on the 17 (590 m below shaft collar level) and 24 levels (890 m below shaft collar level).

On 17 level the key infrastructure includes:

- A workshop for assembling and maintaining the LHD's, drills and material transport vehicle;
- A workshop for Alimak maintenance
- A self-contained refuelling facility (Sat-stat system);
- Explosives magazines;
- The primary ventilation circuit booster fan;
- Refuge chamber;

On 24 level the key infrastructure includes:

- A workshop for assembling and maintaining the LHD's, drills and material transport vehicle;
- A workshop for Alimak maintenance
- A self-contained refuelling facility (Sat-stat system);
- Explosives magazines;
- Refuge chamber;
- A waste tip for rail cars into the no.6 shaft;
- The base of the orepass from the 21-17L stoping panel with chute loading of rail cars.

To support the stoping panel on27L, the decline from 24-27L will provide fresh air and an extension to the return ventilation system and egress system installed that finishes on 24L. Slashing will also be done from the top of decline to the existing workshop for maintenance purposes.

16.10.2 Power

Electrical power is be reticulated to the West Reef on both main levels at the mine high voltage of 3.3 kV or 6.6kV. It is reduced down to 415 V for:

- Primary ventilation booster (firstly on 23L and finally on 17L);
- Secondary ventilation fans;
- Fixed and mobile dewatering and recirculation pumps;
- Electric LHD on 24L; and
- After further reduction used at 240 V for lighting, sat-stat refuelling and uses around the workshops.

16.10.3 Dewatering

West Reef dewatering uses both submersible face pumps and positive displacement pumps. Hydrological investigations estimate that conditions will be dry with a potential negative water balance (i.e. the West Reef consumes water).

The settling sumps will be installed to collect groundwater flowing under gravity along installed drives and from service water from mining operations (drilling and wash down after blasting) pumped from the face. The settling sumps are constructed with sufficient size for solids settling and a floor gradient of 1:7 to allow LHD mucking for removal of settled solids.

16.10.4 Service water

Service water is piped from the Central Shaft system reticulated to the West Reef.

The Alimak units maintain their own water pressure system for providing service water to the decks.

16.10.5 Communications

A leaky feeder radio system is installed to give coverage from surface, throughout the length of the shaft and to the loading pockets, along both main haulage levels and throughout the west reef.

16.11 Ventilation

16.11.1 Introduction

Five phased ventilation models were completed for this study that aligns to the development and production schedule developed that commences November 1, 2016: one development and four production models representing the mine until Q20. In the development stage, the current surface mine fans and a booster fan on the 23 Level operate to provide ventilation to the 24 Level. Four phased ventilation models were completed during key production stages in the West Reef mining area representing the time after the completion of the West Reef return raise system. Two new fan installations are proposed; a replacement of the main fans at the top of the South Waste Shaft, and a booster fan located between 17 Level and the top of the central West Reef raise. In this configuration the 23 Level booster fan is removed. The required fan duties for production mining are shown in Table 87.

Table 87 Main fan summary for West Reef production phase

	Main SWS Exhaust Fan			**17 Level Booster Fan**			
Date	Pressure (kPa)	Quantity (m^3/s)	Motor Power (kW)*	Pressure (kPa)	Quantity (m^3/s)	Motor Power (kW)*	**Total Motor Power (kW)***
Q8	1.38	93	184	0.93	45	186	185
Q10	1.40	93	186	0.65	45	42	230
Q16	1.11	93	147	2.89	66	273	420
Q20	1.18	93	157	2.19	60	188	345

*assume 70% fan efficiency

16.11.2 West Reef Layout

The planned mining method for the West Reef is Alimak mining. There are three main production levels. The 21 sill level is accessed from a decline from the 17 Level. The 24 Level sills are located just off of the level development off of the 24 level. The sills below the 24 Level are accessed by a ramp connecting to the 24 Level. Intake air will come from the Central and Bondaye shafts. A main ventilation raise which connects the main levels will also necessary for the return air of the West Reef. The layout of the West Reef is displayed in Figure 76.

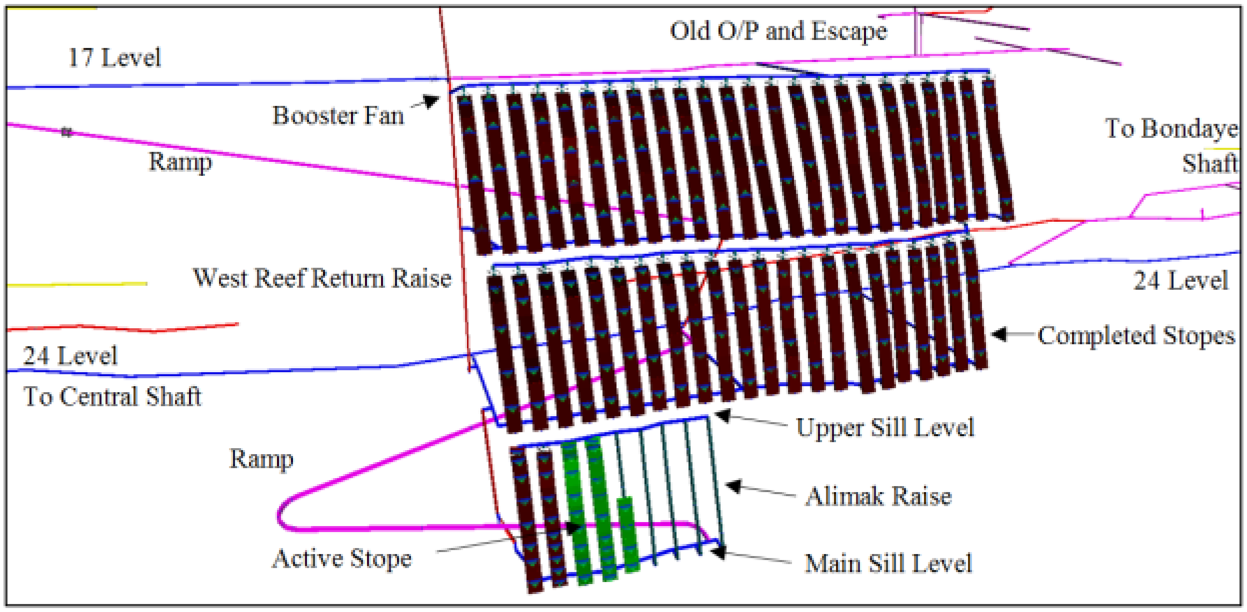


Figure 76 West Reef development layout

16.11.3 Airflow Requirements

While most of the haulage is done with electric rail cars and drilling completed with air powered jackleg drills, some diesel equipment is proposed to be used at the mine. The airflow requirements for each diesel vehicle were based on the vehicle motor power (kW). MVS engineers recommend a factor of 0.08 m^3/s per kW of engine power to ensure safe working conditions in areas with diesel equipment. Although this number may in some cases be higher than the manufacturers suggested airflow requirement, MVS engineers recommends the use of this factor in order to ensure sufficient dilution of airborne contaminants and reduction of heat. A breakdown of the equipment, power and airflow requirements for headings are displayed in Table 88. The airflow requirements were based on production equipment using a 100% utilization factor. This is to ensure there is enough airflow when the equipment is being used to its full ability. The average utilization over a shift will be less than 100% allowing room for smaller diesel equipment to be used. Service equipment have much lower utilization factors over a shift.

Table 88 Diesel equipment airflow requirements

Type	Power (kW)	Airflow (m^3/s)
LHD	80	6.4
Haul Truck	165	13.2

Each main sill level is predicted to have 2 LHD's in operation requiring 12.8 m^3/s of air on the main sill level. Below the 24 Level, 2 Haul trucks are expected to be in operation along with 2 LHDs requiring 39.2 m^3/s of air on the main sill level. Development headings using auxiliary ventilation assumed an additional 25% airflow to prevent recirculation.

Each stope requires a raise that is developed with the use of an Alimak. During the initial raise development the Alimak will be in dead end ventilation. The proposed method to ventilate the dead end heading is from compressed air. Ghana regulations require at least 0.05 m^3/s of airflow per worker. Golden star have designed a system to meet this airflow with the compressed airline. Additional breaks may be required during this type of development to prevent heat stress. Up to three men are assumed to work in a development raise requiring a minimum of 0.15 m^3/s to be provided by compressed air. GSR will be installing a compressed air system to deliver this minimum flow of air per raise that does not yet have flow though ventilation. Additional information is provided in Section 18.5.

After flow through ventilation is established in the raise MVS recommends maintaining an air velocity of at least 0.3 m/s (2.5 m^3/s of airflow in the raise). This will aid in reducing heat and dust around the workers on the Alimak.

16.11.4 Model Development

VnetPC Pro+ software was used to establish a model of the Prestea Underground Mine and West Reef development. The VnetPC program is a self-contained package designed to assist mine engineers in the planning of mine ventilation layouts. Using data obtained from ventilation surveys or determined from known airway dimensions and characteristics, existing or projected ventilation networks can be simulated in such a manner that airflow rates, frictional pressure drops, and fan operating points approximate those of the actual system. The program has been developed based on the assumption of incompressible flow and consistency with Kirchhoff's Laws. It uses an accelerated form of the Hardy Cross iterative technique to converge to a network solution.

The initial ventilation model of the Prestea Underground Mine had been constructed based on measurements completed by MVS engineers. Using drawings provided by Golden Star, the proposed West Reef airways were added to the ventilation model. The data that populates the model is provided in the following sections.

Friction Factors

Most of the resistance values for the branches in each model were calculated using k-factors drawn from data gathered under similar conditions for the entries and based on the drift dimensions and airway type. Shock losses at bends, inlets and exhausts were accounted for by adding appropriate shock loss factors to the airway resistances where necessary. A list of the various k-factors used in the creation of the models in this study is provided in Table 89.

Table 89 List of k-Factors used

Airway Type	Friction (k-) Factor (kg/m^3)*
Sill Level	0.012
Decline	0.012
Alimak Raise	0.022
Raise with escapeway	0.020

*at mine density 1.25 kg/m^3

Development Phase Model

Additional Airflow must be delivered to the 24 Level for the development stage of the West Reef. From the climatic study outline in the MVS report "*February 2015 Site Visit Summary for Golden Star Resources' New Century Mine,*" approximately 20 m^3/s of airflow is needed to maintain safe working conditions. To obtain this airflow, it is proposed by providing fresh air from the Bondaye Shaft and exhausting through the SWS. To do this a booster fan must be installed on the 23 Level at the top of the No. 4 Winze. Several other modifications to the ventilation system will also be required. These modifications were made to the ventilation model representing the system as of April 2015, to estimate the fan requirements. Additional information regarding the modifications is outlined in the *February 2015 Site Visit Summary*. The following is a recommended list of modifications before the 23 Level Booster fan can be turned on.

20 Level

- Removal of Old Infrastructure on 20 Level Bondaye

23 Level

- Clearing top of No. 4 Winze
- Doors – Reverse Mounting
- SWS Regulator Removal
- Removal of Old Infrastructure on Central Shaft (Between SWS and No. 4 Winze)

24 Level

- Removal of Existing Doors Between SWS and No. 4 Winze
- Removal of Old Infrastructure/muck pile on 24 Level Central Shaft
- Install New Airlock Doors Between Central Shaft and No. 4 Winze (1.8 kPa)
- Pump Station Ventilation Modifications
- Clearing Top/Bottom of Bondaye Ore Pass
- Removal of Booster Fan (Last Step Before Turning on New Booster Fan)

25 Level

- Install Bulkhead in the Central Shaft Access Drive (incorporates a sump dam into the bulkhead) or a concrete and steel cap on the No. 4 Winze at the 24 Level

In order to achieve approximately 20 m^3/s of airflow across the 24 Level the new booster fan will be required to operate at approximately 35 m^3/s with a delivered pressure of nearly 2.0 kPa. This will be a temporary installation used for the initial development of the West Reef. The 23L fan installation will only be needed until the West Reef return raise system is completed from 24L to 17L. The airflow distribution for the 24 Level is displayed in Figure 77.

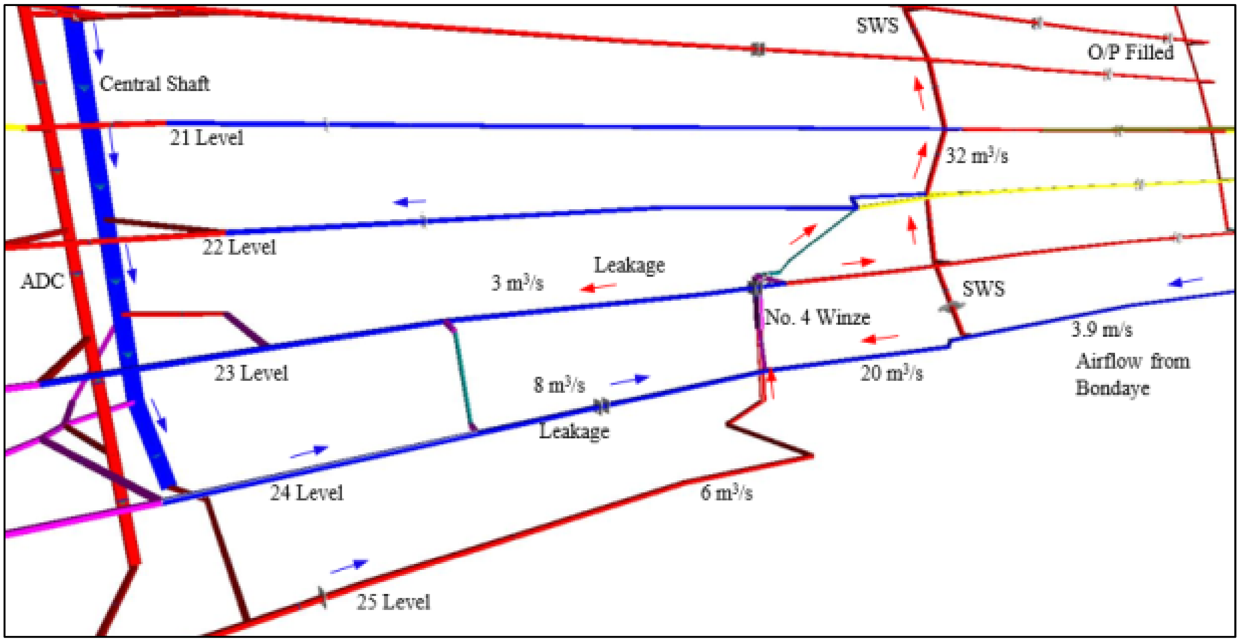


Figure 77 24 Level airflow distribution

The 24 level ventilation configuration will remain in place until the completion of the West Reef return raise system. The West Reef return raise system is a series of raises that connects from below the 11 level. A ventilation model was completed representing the ventilation system in Q4 showing beginning stages development on the West Reef where the West Reef return raise has started construction. From the timing schedule two production stopes have started to be mined on the 24 level as well as additional Alimak Raises. The model assumes that the new surface fan installations are installed. This provides enough ventilation to the 17 Level for 2 LHDs to develop the 21 Level ramp. If the new surface fans are not installed then there will only be enough ventilation for one LHD on the 17 Level. The main fan results for Q4 are shown in Table 90. The ventilation model representing the mine in June Q4 is displayed in Figure 78.

Table 90 Fan operating points for Q4 Development Phase

Fan Location	Pressure (kPa)	Quantity (m^3/s)	Air Power (kW)	Motor Power (kW)*
Main SWS Exhaust Fan	1.19	93	110	158
23 Level Booster Fan	1.95	35	68	97

*assume 70% fan efficiency

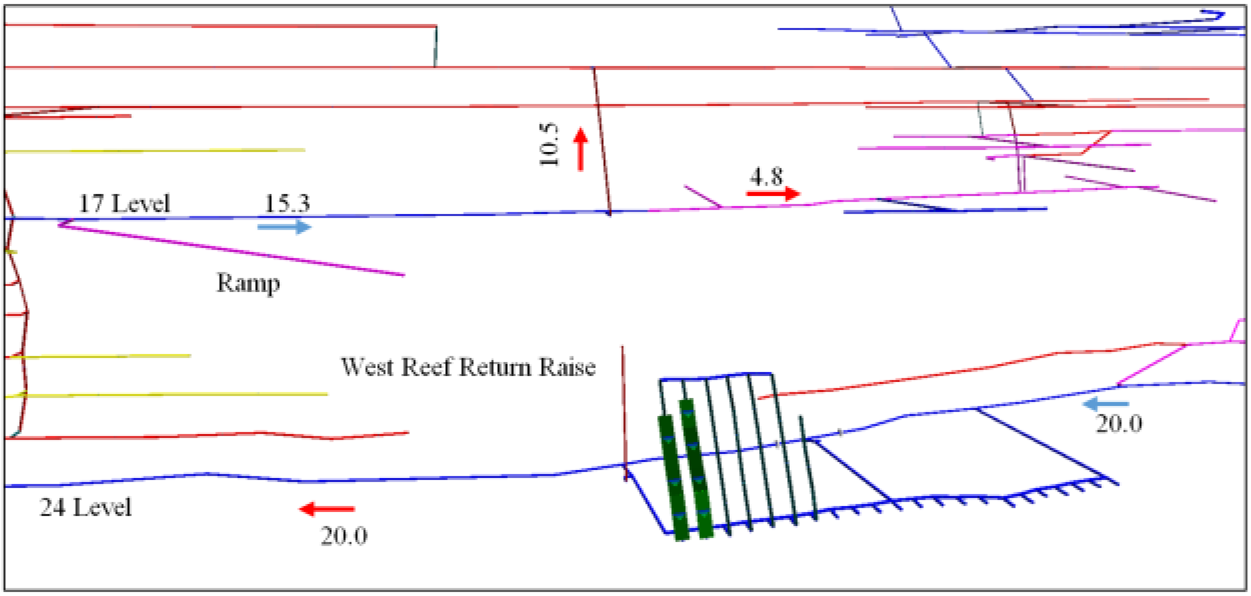


Figure 78 Q4 ventilation model (m^3/s)

The longest section of auxiliary duct needed is for the 17L decline to the connection to the West Reef RAR, which is approximately 700m. Only 1 LHD will be operating in the decline, therefore 6.4 m3/s of airflow is required at the end of the duct. The diameter of the duct is 750mm. Using only 1 fan at the beginning of the duct, a pressure of at least 5.16 kPa is required. At this pressure the duct will be harder to maintain as this pressure is directly applied to the ducting, therefore more than one fan is required. With 2 fans in the duct (one at the intake and one approximately 300m along the duct) the pressure on the duct is reduced to approximately 3.0 kPa. Due to the high pressure and total distance, the duct will need to be well maintained and free of holes. The fan duties for the duct are displayed in Table 91.

Table 91 Auxiliary fan duties for the 17 Level decline

700m Duct with 2 Fans	Pressure (kPa)	Quantity (m^3/s)	Air Power (kW)	Motor Power (kW)*
Fan at Intake	3.0	8.8	26	40
Fan at 300m	2.4	7.9	19	29

*assume 65% efficiency

West Reef Production Airflow Distribution

Once the West Reef return raise system is completed the ventilation system will need to be configured for production. Fresh air is supplied to the West Reef primarily from the Central Intake Shaft on 17 and 24 Level, and some air from the Bondaye shaft on the 24 level. Intake air is then delivered to the West Reef development. The return air exits the area through the West Reef return raise, where air is pulled up the raise to a connection on the 17 Level where the West Reef booster fan is located. The air is then pushed through another raise connecting to the 12 level. The return air then splits between the 12, 11 and 10 levels and travels to the South Waste Shaft. Air is then pulled up the shaft to the main surface fan installation, and exhausted to atmosphere. Figure 80 is an example of the main airflow distribution for the entire mine.

Once intake air enters the sill level it is distributed between open Alimak raises and to ventilate loading equipment. The air entering the Alimak raises can be regulated on the top sill level so that there is just enough air velocity in the stope to remove heat and dust away from the personnel working the Alimak raise. In Alimak raises where flow though ventilation is not possible, compressed air is proposed to ventilate the working area. The air travels along the top sill level and flows down the last open raise where a final brattice regulator is placed and joins the air on the main sill level. Mined out stopes should be closed to airflow in order to maintain enough airflow for working equipment on the main sill level. The air on the main sill level is maintained high enough for the amount of equipment working on the level. The air exits the end of the sill level through a regulator and into the West Reef return raise.

Figure 80 shows an example of the airflow distribution through the stope panels.

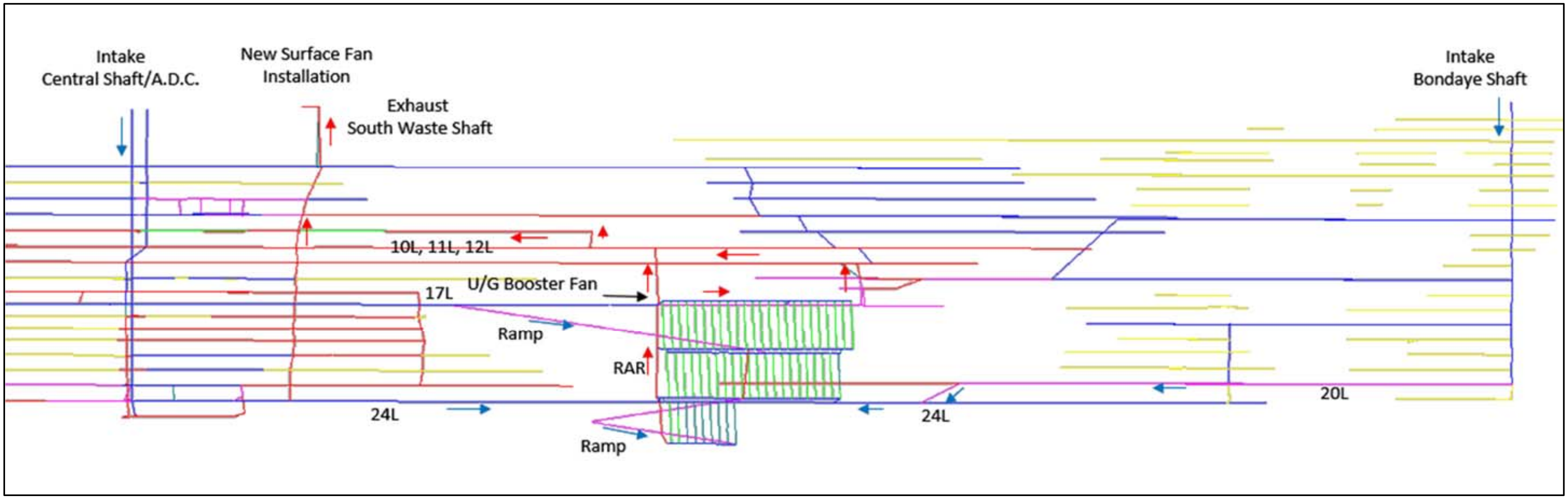


Figure 79 Example mine airflow distribution configured for production

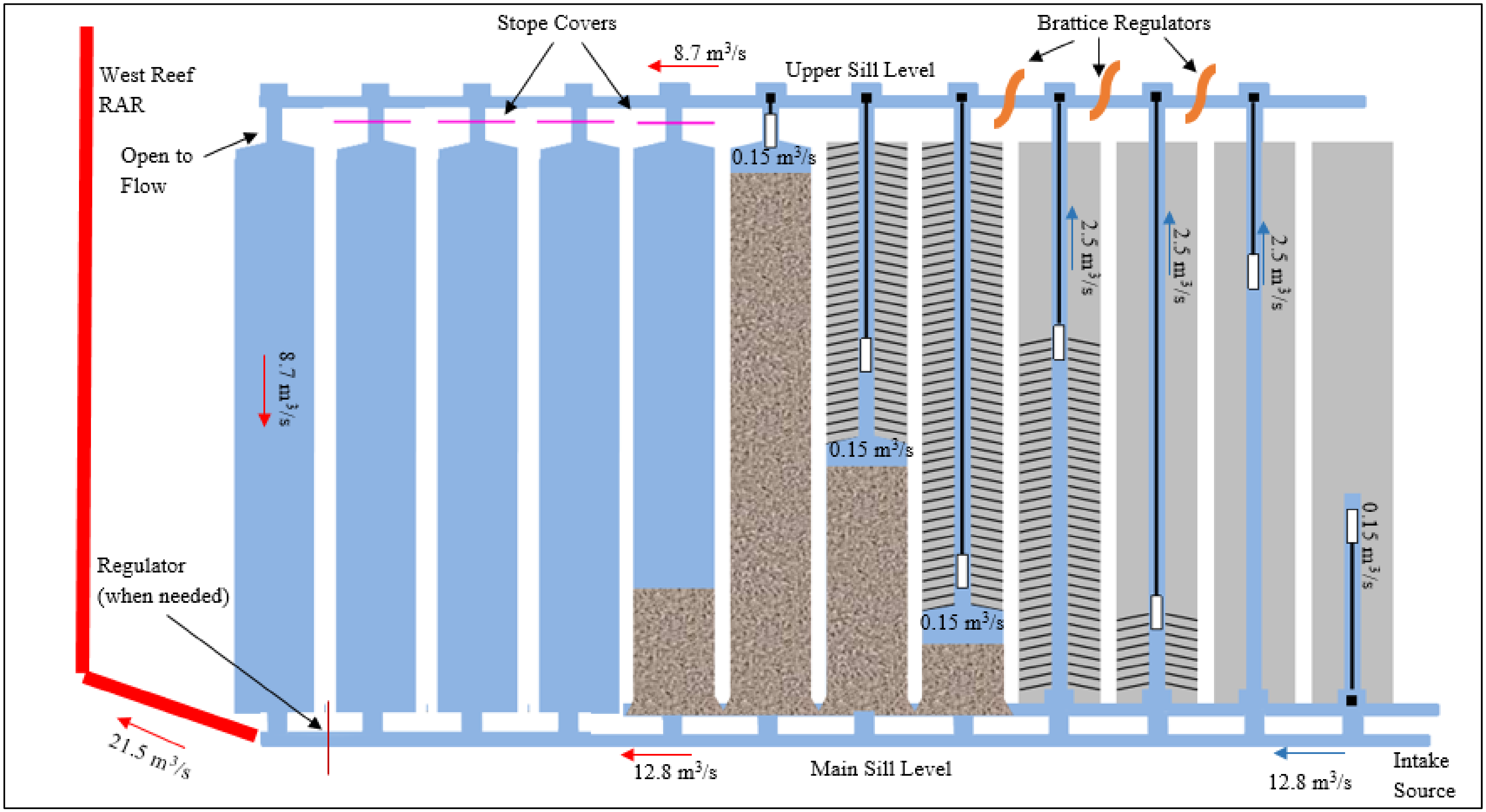


Figure 80 Example West Reef airflow distribution

West Reef Raises

For return air in the West Reef a raise system will be needed. This system extends to the bottom of the West Reef and connects as high as the 10 Level. The sizing of each section of raise was examined for pressure loss while maintaining reasonable velocities, and constructions methods to determine recommended dimensions. The main factor in the recommended size for the main Raise system is the need for an escapeway to be installed. The escapeway can add considerable pressure loss to the system. The escapeway components also reduce the available area to flow in the raise. An escapeway is proposed to be installed from the 21 level to the bottom of the raise system. Table 92 displays the recommended raise and drift dimensions for the West Reef return raise system.

Table 92 West Reef return raise system dimensions

Infrastructure	Modeled Dimensions w x h (m)
Upper Raise (11L to 17L)	2.4 × 2.0
Main Raise System (Sub 17L)	2.7 × 2.9

Changes between Development Models to Production

From the initial site visit and ventilation modeling several changes and upgrades were noted and need to be completed to provide adequate ventilation to the West Reef expansion. In order to ventilate the West Reef mining area the ventilation system will need to be reconfigured for production. These modifications are outlined in Figure 81. This figure also displays the critical path for installing the new fans. Additional details on the recommended changes are outlined in the MVS report *"February 2015 Site Visit Summary for Golden Star Resources' New Century Mine."*

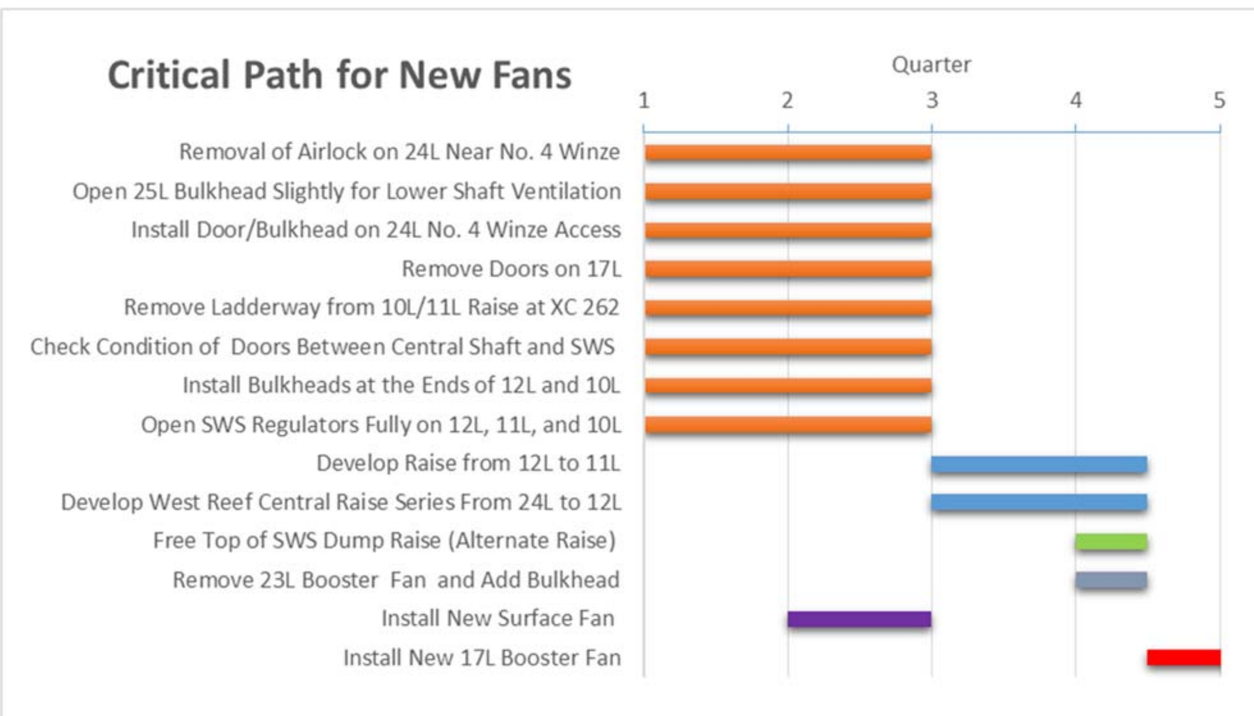


Figure 81 Production ventilation critical path

16.11.5 Productions Phase Ventilation Models

After determining the primary ventilation method, several staged models were completed. The staged models were developed to determine fan operating points and possible operating challenges. Key development stages were selected to be represented with the ventilation model. Using the timing map provided by Golden Star timeframes modeled are approximately: Q8, Q10, Q16 and Q20. Each of the modeling results are discussed in the following sections.

Q8 Ventilation Model

The Q8 ventilation model represents the completion of the West Reef return raise system. Increased airflow is now delivered to the active stopes on the 24 Level. This stage also represent the time immediately after the installation of the 17 Level booster fan. The decline between the 17L and 21L is also complete. The model assumes enough airflow for two LHDs on the 24L and two on the 21 Level. The model also accounts for ventilation of the Alimak raises. The main fan results are displayed in Table 93. The ventilation model representing the mine in Q8 is shown in Figure 82.

Table 93 Fan Operating points for Q8

Fan Location	Pressure (kPa)	Quantity (m^3/s)	Air Power (kW)	Motor Power (kW)*
Main SWS Exhaust Fan	1.38	93	129	184
17 Level Booster Fan	0.93	45	42	60

*assume 70% fan efficiency

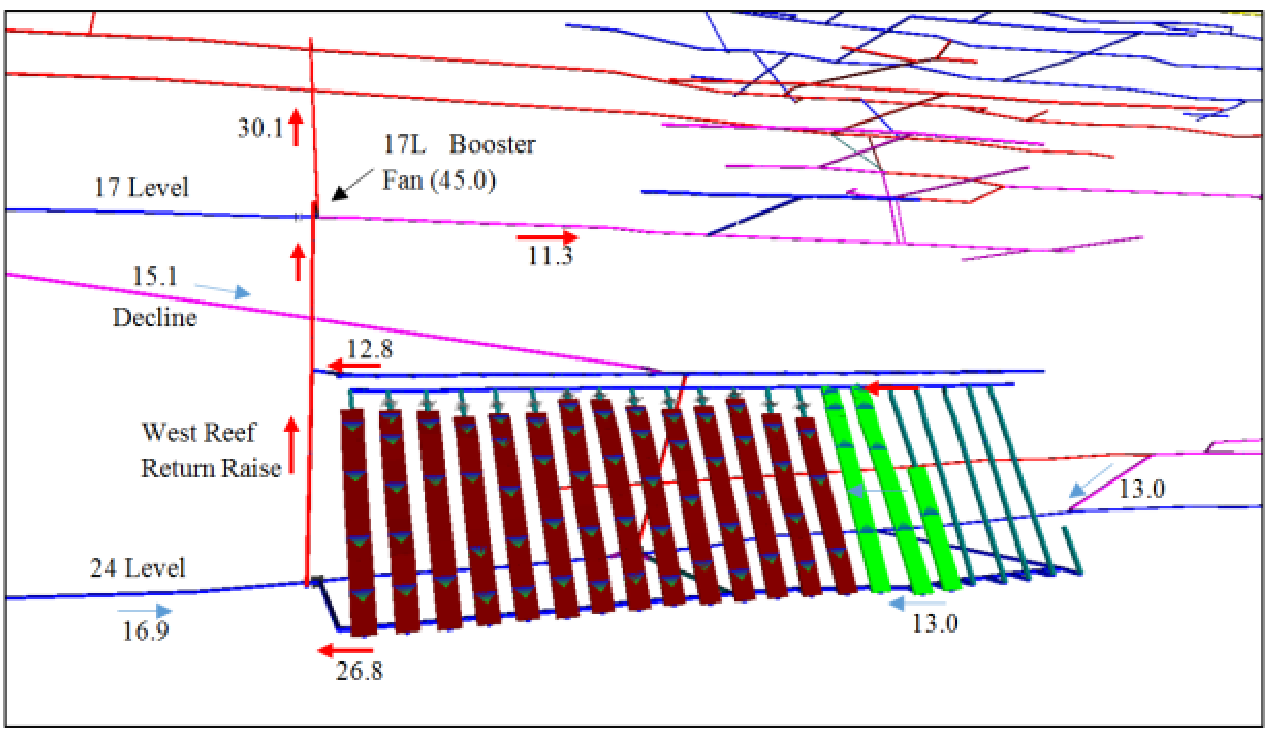


Figure 82 Q8 Ventilation model (m^3/s)

Q10 Ventilation Model

The Q10 ventilation model represents a time where production mining is taking place on both the 21L and 24L. The last Alimak raise from the 24 level is assumed to have been connected to the 21 Level to provide additional intake air to the 21L. The model assumes enough airflow for two LHDs on the 24 Level and two on the 21 Level. The model also accounts for ventilation of the Alimak raises on the 21 Level. The fan results are displayed in Table 94. The ventilation model representing the mine in August 2017 is shown in Figure 83.

Table 94 Fan operating points for Q10

Fan Location	Pressure (kPa)	Quantity (m^3/s)	Air Power (kW)	Motor Power (kW)*
Main SWS Exhaust Fan	1.40	93	130	186
17 Level Booster Fan	0.65	45	29	42

*assume 70% fan efficiency

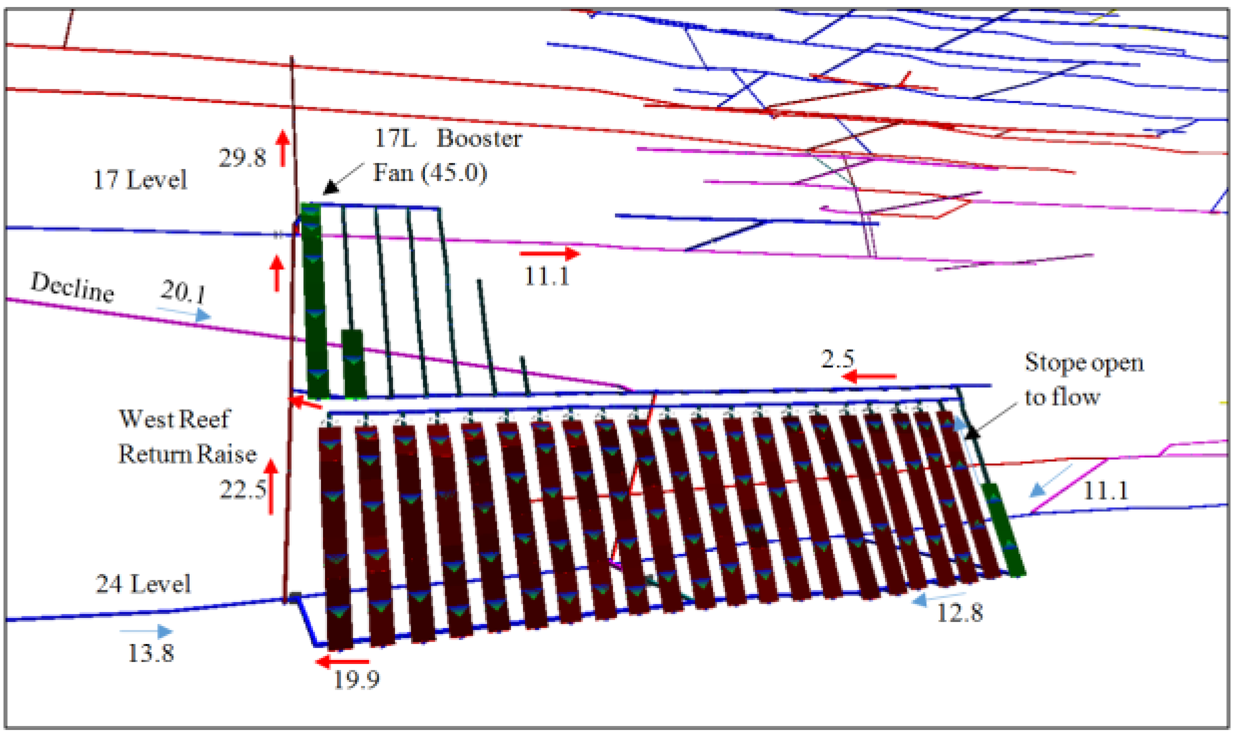


Figure 83 Q10 Ventilation model (m3/s)

Q16 Ventilation Model

The Q10 ventilation model represents production mining on the 21L and development of the ramp below the 24 Level. Mining below the 24 Level is unique in that diesel truck haulage is proposed to be used. This requires an increase in the total required airflow. Production mining has moved outside of the area where the 17 Level decline can provide ventilation to the 21 Level. The last

Alimak raise on the 24 level is proposed to break through to the 21 level to provide intake airflow. A set of equipment doors will be required in the17 Level decline to promote airflow through the Alimak raise from the 24L. The fan results are displayed in Table 95. The ventilation model representing the mine in May 2018 is shown in Figure 84. This model represents the highest fan duty of all the models because the 17 level is not used to provide intake air to any of the mining sections.

Table 95 Fan operating points for Q16

Fan Location	Pressure (kPa)	Quantity (m^3/s)	Air Power (kW)	Motor Power (kW)*
Main SWS Exhaust Fan	1.11	93	103	147
17 Level Booster Fan	2.89	66	191	273

*assume 70% fan efficiency

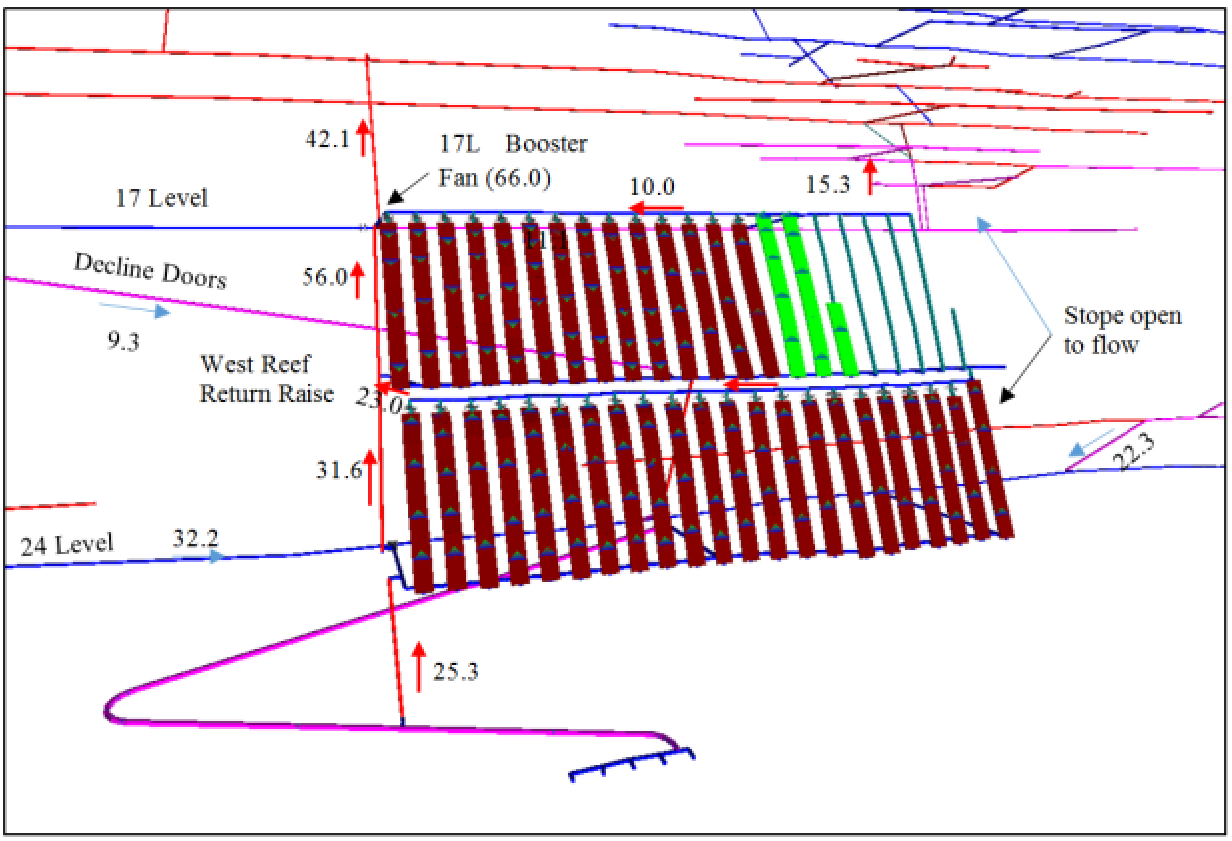


Figure 84 Q16 Ventilation model (m3/s)

Q20 Ventilation Model

The Q20 model represents production mining below the 24 Level in the West Reef. Additional airflow is now needed due to trucks being used for primary haulage below the 24 Level. To reduce fan pressures air is also allowed to flow from the 21 Level through an open stope to the 24 Level and be used as intake air. The fan results are displayed in Table 96. The ventilation model representing the mine in Q20 is shown in Figure 85.

Table 96 Fan operating points for Q20

Fan Location	Pressure (kPa)	Quantity (m^3/s)	Air Power (kW)	Motor Power (kW)*
Main SWS Exhaust Fan	1.18	93	110	157
17 Level Booster Fan	2.19	60	132	188

*assume 70% fan efficiency

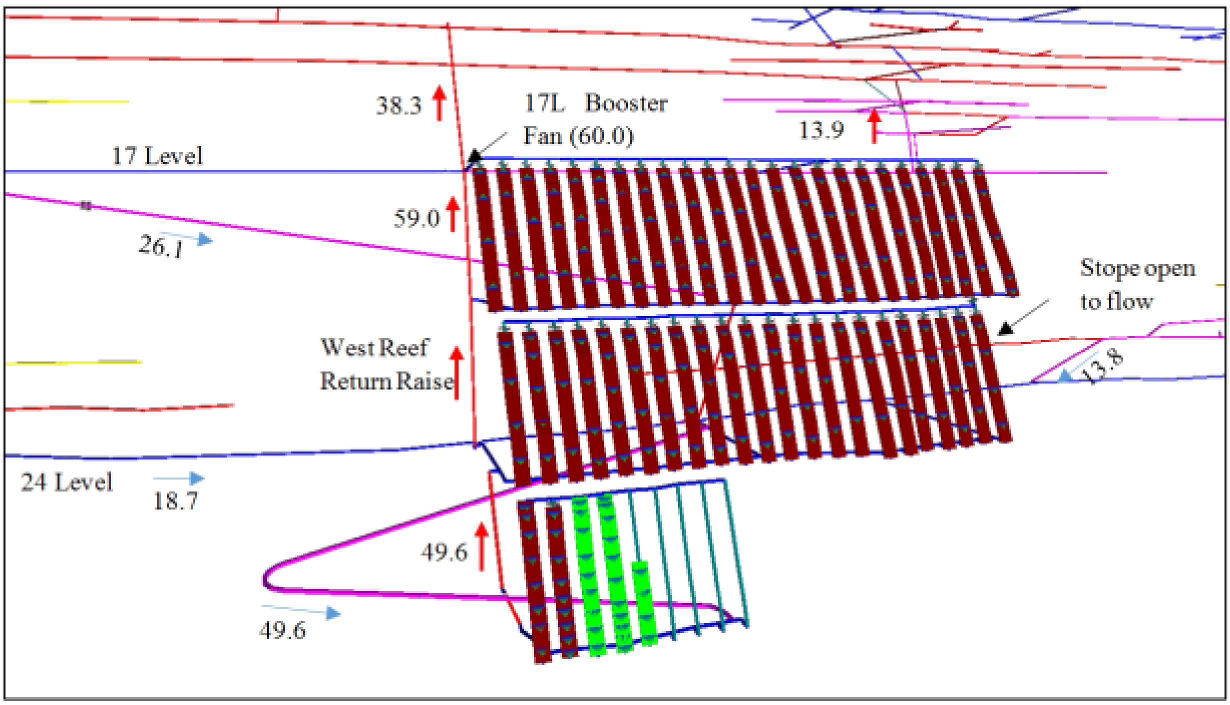


Figure 85 Q20 Ventilation model (m^3/s)

16.11.6 Fan Requirements

A list of the fans operating points (pressure and quantity) and total power requirements for development stages are provided in Table 97 and production phases in Table 98. The fan motor powers were calculated using an estimated fan/motor efficiency of 70%. The use of variable frequency drives is not justified based on capital cost against a short mine life. It is important to note that the fan operating pressures should be considered applied pressures; that is, the pressures reported do not account for losses associated with fan housings, ducts, or diffusers/evases. The fan manufacturer should provide an estimate of the pressure losses associated with fan installation and provide a suitable fan to fit these requirements. The manufacturer can recommend the type of fan and how many stages based on obtaining high power efficiencies and adhering to space limitations.

Table 97 Main fan summary for West Reef development phase

Fan Location	Pressure (kPa)	Quantity (m^3/s)	Air Power (kW)	Motor Power (kW)*
Main SWS Exhaust Fan	1.19	93	110	158
23 Level Booster Fan	1.95	35	68	97

Table 98 Main fan summary for West Reef production phase

	Main SWS Exhaust Fan			17 Level Booster Fan			
Date	Pressure (kPa)	Quantity (m^3/s)	Motor Power (kW)*	Pressure (kPa)	Quantity (m^3/s)	Motor Power (kW)*	Total Motor Power (kW)*
Q8	1.38	93	184	0.93	45	186	185
Q10	1.40	93	186	0.65	45	42	230
Q16	1.11	93	147	2.89	66	273	420
Q20	1.18	93	157	2.19	60	188	345

*assume 70% fan efficiency

17 Recovery Methods

17.1 Metallurgical Testing

Metallurgical testwork programs have been conducted on samples of mineralization from the PUG West Reef deposit. A metallurgical testwork program was undertaken in 2008 in support of a study at that time investigating the potential for mining ore from both the West Reef and the Footwall Reef at Prestea. More recent testwork supporting feasibility studies for mechanised cut and fill (2013) and the recently published shrinkage mining feasibility study (2015), the latter containing a summary of programs conducted.

Further to the metallurgical testwork program noted above a comminution study was completed by MDM / AMEC Foster Wheeler / KEMIX (Appendix A) that focussed on the operation of the comminution circuit, specifically the use of both the Ball and SAG mill in a batch operation, as opposed to the operation of the SAG mill only in a continuous operational mode.

The comminution study incorporated two operating scenarios for the treatment of PUG ore. The first is where oxide and PUG ore are mined and processed together at a RoM rate of 1.154 Mt/a d.b and 0.237 Mt/a d.b. In such a scenario, the ore would be processed through the existing Bogoso oxide plant over 7076 operating hours per annum (81% overall availability).

In the second scenario, which occurs when the oxide ore supply is exhausted, 0.237 Mt/a of PUG ore will be processed through the same oxide milling circuit, but on a batch basis. In a batch mode, the comminution and CIL circuits are decoupled from one another by incorporating a medium term slurry storage capability between the plant comminution and CIL sections.

In the analysis of the comminution operating scenarios considered the testwork data was used to develop a JKSimMet® model, which was calibrated against actual plant operating data. This model was then used to run simulations for changing parameters in order to achieve the maximum throughput at a desired system product P80 of 75 µm.

A key parameter used in the simulation was ore hardness. The previous metallurgical testwork programs obtained ore hardness results were the basis, including:

- Mechanised cut and fill program (2013) averaged BWi at 15.7 kWh/t
- Shrinkage mining program (2015) averaged BWi at 14.4 kWh/t

The simulations at P80 of 75 µm were run on two ore hardness to simulate the initial blending and later only PUG ore. The ore hardness used in the simulation includes:

- Soft oxide (blended ore) BWi at 12.0 kWh/t
- Hard ore (PUG only) BWi at 16.0 kWh/t

In summary, the blending of oxide ore with PUG ore in the existing oxide circuit and later running the comminution section in a batch mode when processing PUG ore alone is seen as the most appropriate decision for GSR from a techno-economic perspective. When processing PUG ore alone MDM/Amec Foster Wheeler agrees that, based on the testwork sighted, gold recoveries in excess of 94% are achievable on an annualised basis.

The main findings of the work are summarised below and provided in detail in the body of comminution study report (Reference).

- The comminution section as currently configured is suitable for the required duties, both when operated in a continuous mode and in a batch mode over the range of ore types reviewed. Furthermore, greater operational flexibility and cost savings will be realised when operating in a batch mode, compared to the continuous mode, at the lower plant throughputs.
- In the CIL leach, there are two interrelated rate determining mechanisms, namely leaching and adsorption as noted below.
  - The leach tests carried out by SGS, suggest that the leach is relatively fast, and likely to be complete within 15 to 20 hours. Thus, leaching could theoretically be complete in the first two PUG CIL tanks (circa, 23 hours of residence time). This compares to the 63 and 95 hours of residence time available if five or six of the PUG tanks are used, herewith referred to as the PUG CIL tanks.
  - Gold adsorption onto carbon however, is a more a function of the number of stages, solution carbon concentration, carbon activity and gold on regenerated carbon rather than residence time. For the purpose of this study, gold grade on the regenerated carbon has been set at 100 g/t. More recently Bogoso have sought to reduce this to between 50 and 100 g/t.

The simulations were also used for process plant design with the primary objectives of

- Maximising gold recovery:
- Minimising carbon inventory and carbon residence time, thereby minimising:
  - Gold inventory in the CIL circuit, and
  - Carbon fouling.
- Align the adsorption and elution circuits.
- Optimising the number of CIL tanks required on a techno-economic basis, thereby also minimising carbon lockup.

The results of the simulation and interpretation are summarised below.

- The CIL tanks are oversized for the lower throughput, possibly by a factor of 2 to 4 times. However, there is no economic justification for building a new CIL circuit specifically for the PUG ore.
- In order to minimise carbon inventory and fouling, set the carbon flow rate and carbon inventory by using a relatively low carbon upgrade ratio (900:1) and staggered carbon concentrations in each CIL stage. The optimum carbon loadings are believed to be 10 g/L in the first stage and 5 g/L in the four subsequent stages. Whilst GSR have both 5 and 16 t elution columns, it is recommended that the 5 t elution column be used on the basis that:
  - A 5 t column (5 t batches) will give one elution every 5 days, thereby ensuring a steady cash flow stream. Whilst a 16 t column would result in elution cost savings, this comes at the cost of delayed revenue and increased carbon management issues and costs.
  - Each CIL stage increase operating cost and contributes to increased carbon and gold lock up in the CIL circuit. Thus, the objective should be to minimise the number of CIL tanks required, subject to the provision that placing a CIL tank in a standby duty should

not incur gold losses in excess of the CIL stage operating costs. Whilst simulations indicate that six stages are required, a reduction to 5 stages or less might be achievable. The number of stages required will only be established under real operating conditions.

17.2 Process Description

Ore from the PUG West Reef is trucked to GSR's Bogoso operation, a distance of approximately 15 km, where the ore will be processed through the existing process facility producing 2,480 kg/a or 80,000 oz/a of gold at an overall recovery of 94%.

The processing facility at Bogoso consists two plants to separately process refractory and non-refractory (oxide) ores. As the PUG ore is non-refractory ore it will be processed through a modified Oxide circuit that utilizes conventional CIL for gold recovery. The circuit includes the following unit processes:

- Run-of-mine (RoM) receiving
- Crushing
- Milling and classification circuit (cyclones)
- Gravity concentration
- Thickening
- Feed storage tanks
- Carbon-in-leach (CIL) circuit
- Upgrade / refurbishment of high intensity cyanide leach reactor (Acacia)
- Gold room and recovery
- A new CIL tailings disposal line
- Services (compressed air, instrument air, oxygen, gland service water, raw and process water make-up)

The proposed flow sheet for the Bogoso plant is shown in the next section.

17.2.1 Flowsheet

The modified Oxide plant proposed will utilize the majority of the existing Oxide circuit and include the following unit process augments:

- Inclusion of feed storage tanks (for batch process of PUG ore)
- Inclusion of the existing high intensity cyanide leach reactor (Acacia)

Figure 86 shows the proposed block flow for processing the PUG ore (batch process) and Figure 87 the general layout of the Bogoso process plant and, more specifically, the major components utilized for PUG ore processing.

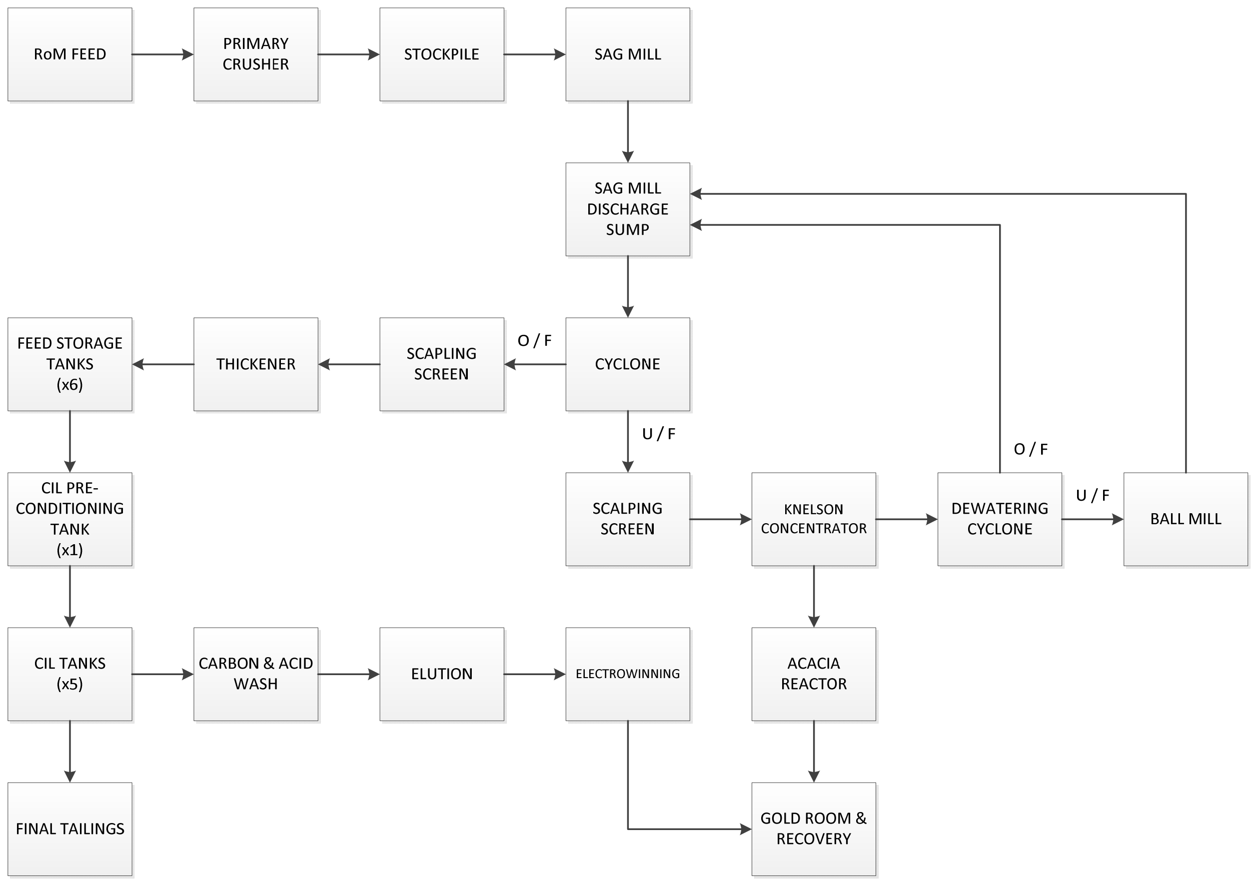


Figure 86 Block process flow schematic

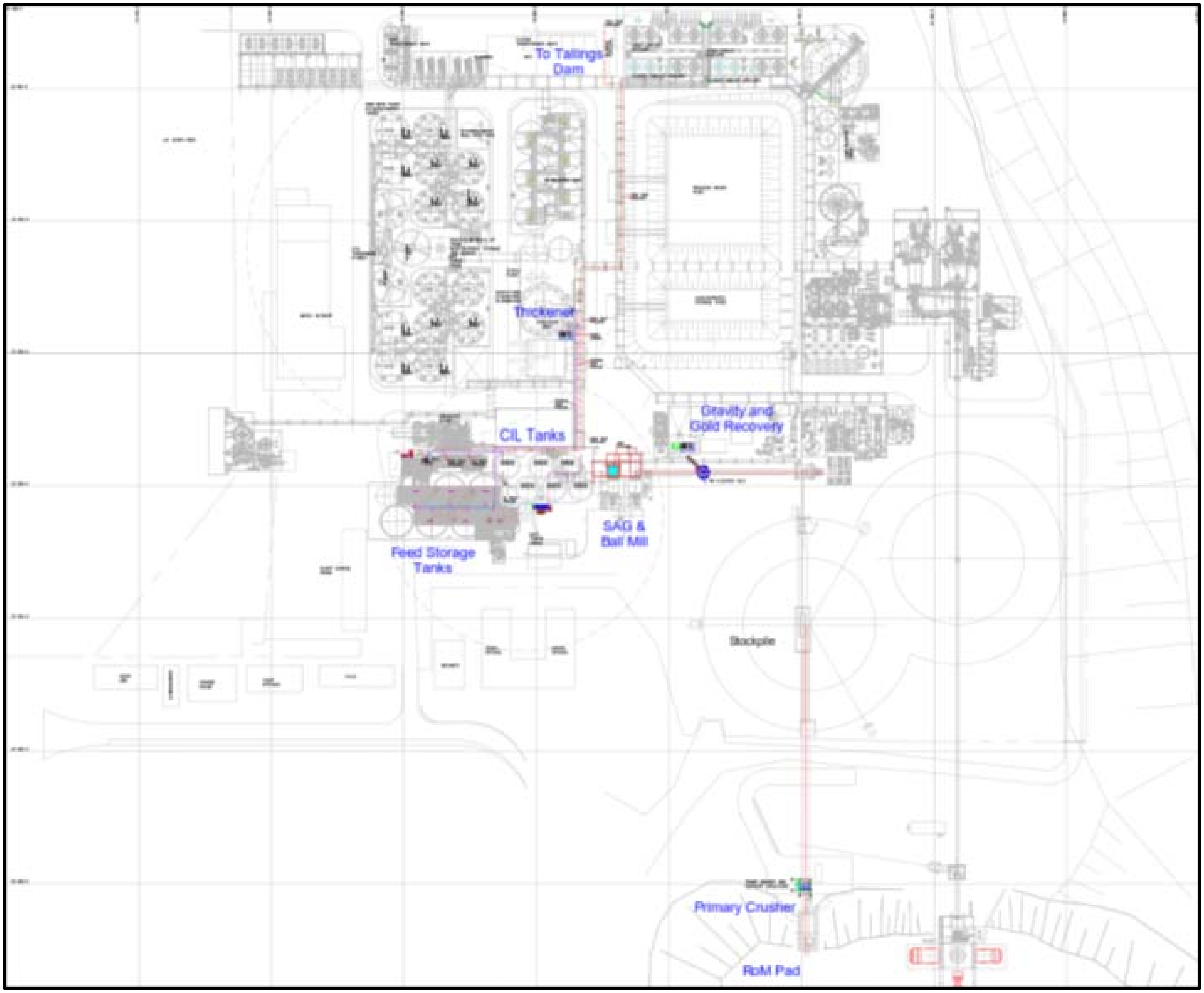


Figure 87 General layout of Bogoso process plant (blue - PUG ore process areas)

17.2.2 Run-of-Mine (ROM) Pad

Run of Mine Ore of about -350mm will be delivered by truck directly from the Prestea Underground Mine to the Bogoso ROM stockpile. The ore will be stored under the newly constructed ore shed at the stockpile and the area will be fenced and under 24hour security surveillance. Security cameras will be installed at the ROM pad shed, ROM Bin area, Crusher chamber and CV01 tail pulley areas.

17.2.3 Crushing

A CAT988 front end loader (FEL) will reclaim ore from the stockpile on the ROM pad and deliver to the ROM bin. A 150mm aperture vibrating grizzly (01FE01) fitted with variable speed drive will allow material of -150mm to bypass the crusher and report to a belt conveyor (CV01). The +150mm oversize will gravitate into the KEMCO C160 Jaw crusher producing product of P80 -150mm which also falls onto CV01 and the combine product collected at the stockpile. CV01 belt will be fitted with conveyor canopy to prevent washing of the ore during transport and rain.

Given the over capacity of the crushing unit, crushing will be done in batches. The high security risk associated with the PUG ore requires that large stocks will not be kept at the ROM pad.

The crushed material will be stockpiled on a 6000t capacity stockpile fitted with overhead canopy to keep out rainwater and also for dust suppression.

17.2.4 Milling and Classification

The crushed ore is removed from beneath the stockpile by a slot feeder (02-CH06) to a variable speed conveyor (02-CV-02) driven by hydraulic motor, which delivers the ore to the Mill feed conveyor (02-CV-03). The hydraulic drive unit of 02-CV-02 and the adjustment bars in the slot feeder allow the feed to the mill feed conveyor to be controlled.

The Mill feed conveyor is equipped with a weightometer that allows accurate control of Mill feed and associated controls. The mill feed conveyor will deliver the ore at nominal rate of 120tph into a 1.5MW 5.2x4.8 Morgardshammer SAG mill operating in open circuit. For maximum flexibility in adapting to ore competency changes, the Mill has a hydraulic coupling that allows the mill speed to be varied from 60% to 80% of critical speed. SAG mill discharge slurry of product size of P80-700 micron enters the Mill discharge distribution box, which then directs the slurry into the Combined Mill discharge pumps hopper.

The Mill discharge pumps, delivers slurry with adjusted density of 55% solids to a bank of 8 GMAX 15-3117 Krebs cyclones. Cyclones overflow of P80 -75 micron gravitates to the vibrating trash screen before feeding into the 14M pre-leach thickener. The cyclone underflow is fed to the gravity circuit consisting of two(2) Operational 30" Knelson concentrators and one on standby, via a distribution box and scalping screen. The Knelson tails and scalping screen oversize (+4mm) are pumped to a cluster of 2 GMAX26-3332 Krebs dewatering cyclones.

The dewatering cyclones underflow feeds the 1.5MW 4.2x5.4 Morgardshammer Ball Mill. The overflow however gravitates to the Mill discharge hopper and serves as additional dilution water. The Ball mill is operates at a fixed speed of 75% critical speed through an air clutch system.

17.2.5 Feed Storage and CIL

Cyclone overflow thickened to density of 45% solids is pumped to the Feed automatic sampler ahead of the feed box situated on top of the feed storage tanks. There are six (6) 1200m^3 capacity storage tanks providing storage of about 3600t – 4200t depending on achieved thickener density. All the six storage tanks are equipped with a 55kw agitator and linked by a manifold system for easy withdrawal of feed to the leach.

Feed is pumped to the CIL feed auto sampler on top of the precondition tank by means of variable speed pumps. The feed lines have flow and density measuring devices that allow the exact CIL feed tonnage to be measured and controlled. The volumetric flow rate to the CIL will be regulated by the Programmable Logic Controller (PLC) and will be calculated from the dry feed tonnage set-point (selected by the operator) and the slurry density measured by the densitometer. The flow rate will be controlled by adjusting the speed of the CIL feed pump.

The CIL feed dry tonnage is calculated by using the following formula:

$$T = \frac{V}{\frac{1}{SG} + \left({}^{W}/_{S}\right)}$$

Where,

T = dry tons

V = volume of slurry in m^3

SG = dry density of solids in t/m^3

W = 100 - % Solids of slurry

S = Slurry % solids

Feed conditioned with lime and oxygen flow via interconnecting launders to the CIL head tank.

The CIL circuit consists of five (5) 800m3 leach and adsorption tanks where sodium cyanide, oxygen/(hydrogen peroxide optional) and activated carbon are added in various propositions to enhance the leaching process. Optimum cyanide addition is achieved by the use of the Automatic Cyanide analyser/controller, TAC 1000, and rationed to the CIL feed rate (tph).

Activated carbon is advanced from the back tanks and eventually recovered at the head tank as loaded carbon. The slurry exiting the last tank of the CIL plant gravitates to a carbon safety screen, which ensures that all carbon is retained within the circuit. Screen oversize (carbon) is recovered and returned to the circuit, screen undersize reports to the final tailings hopper where it is pumped to TSFII cell ½ and 2A.

The loaded carbon goes through a standard 3% 1BV acid washing with dilute hydrochloric acid followed by a standard pressurised Zadra elution process. There is the flexibility to utilise either the 5T or 16t Elution circuits.

On a weekly basis the electrowinning cell cathodes are remove, washed and the concentrate calcined for a minimum of 12hrs at 750ºC. The calcined material is fluxed, smelted and cast into bullion bars.

Barren carbon is regenerated in a 275kg/hr Electric Kiln at temperatures of between 650 –750 degrees Celsius in an inert atmosphere in order to remove organic foulants that cannot be removed by acid washing.

Concentrate from the Knelsons will be processed through the Consep Acacia CS2000 within the gravity fence and the existing Acacia electrowinning cell in the Gold room. This unit uses heating of the leach system to maximise both the leach rate, and to allow very high gold solutions to be achieved. The higher the temperature of the solution, the more gold can be dissolved into that solution.

17.2.6 Mass Balance

The comminution circuit mass balance is shown in Figure 88. The CIL receives thickener underflow in a relative density (RD) range of 1.42-1.44 via a variable speed pump and densitometer at a rate of 25-40 tph. All other downstream mass balance is compatible with this mass balance for slurry tonnes and water.

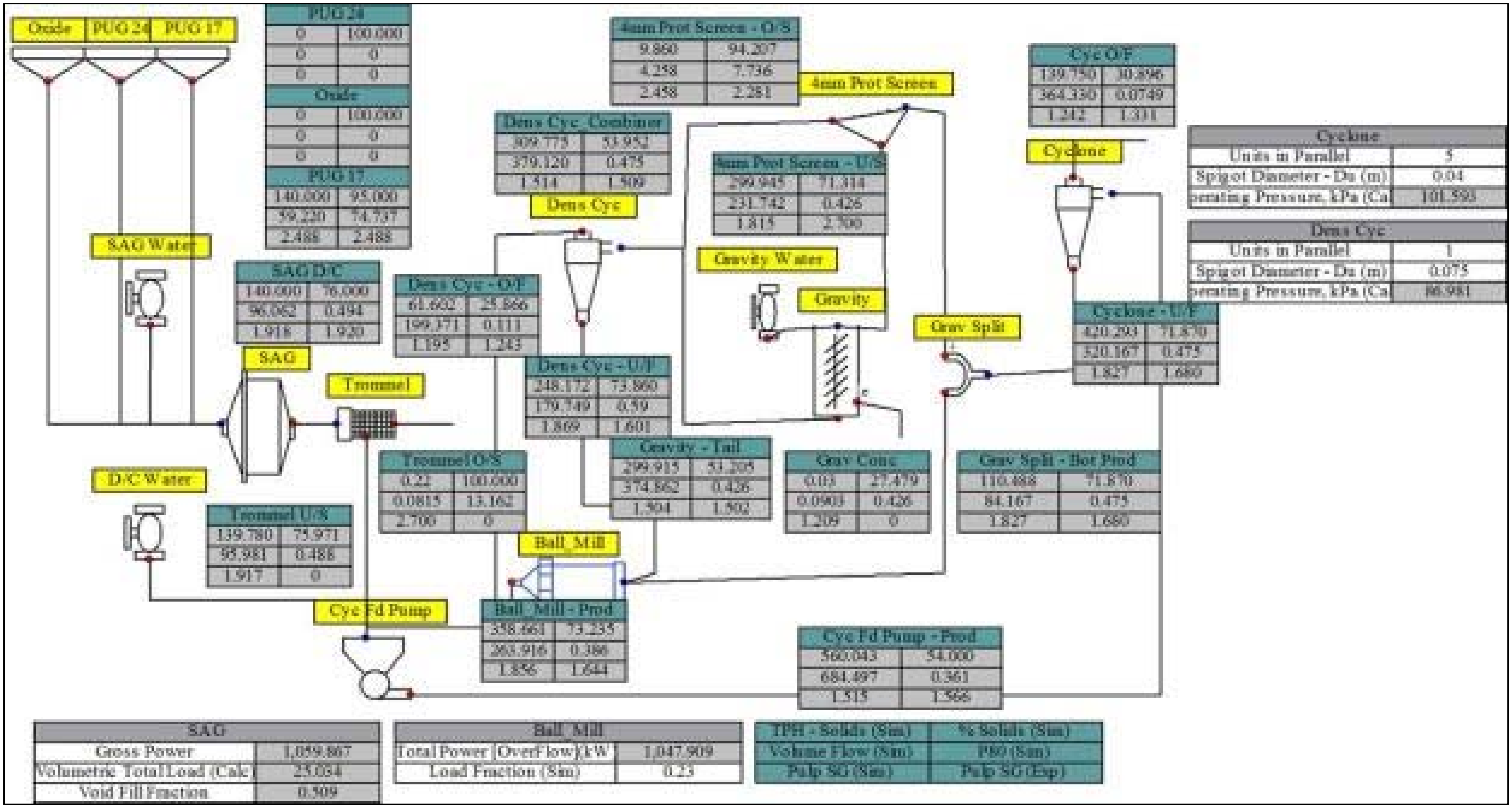


Figure 88 Comminution mass balance

17.2.7 Process Design Criteria

Key operating parameters for the process plant are shown in Table 99.

Table 99 Prestea West Reef Process Design Criteria

Process Design	Parameter	Value	Units
Milling Schedule			
	Hours per day	24	hrs
	Days per month	30	days
	Months per year	12	months
	Mill Utilisation	23%	%
	Effective days per week	1.58	days
Material Properties			
	Solids SG - TSF 1	2.7	
	Liquid SG	1.0	
Plant Capacity			
	Throughput	120	t/h
		650	t/d
		19,500	t/m
		234,000	t/y
	Feed Grade	16.50	g/t
	Gravity Recovery	50	%
	CIL Recovery	92	%
	Overall Recovery	94.0	%
	Gold Produced	15.51	g/t
		10,082	g/d
		324	oz/d
		116,686	oz/y
Crushing			
	Overall availability	80	%
	Feed rate – nominal	250	t/h
	Feed rate – design	400	t/h
	RoM F80	350	mm
	Crushed ore P80	120	mm
Milling			
	Operating days (Batch)	1.58	days/week
	Overall availability	96	%
	Feed rate – design	120	t/h
	Bond Ball Mill Work Index	16	kWh/t
SAG	Size (Outside)	5200 X 4850	mm
	Size (Inside)	5050 X 4400	mm
	Installed Power	1500	Kw
	Speed	14	rpm
	Ball Charge	10	%
	Discharge grate aperture	12 X 30	mm
	Trommel screen aperture	12	mm
Ball Mill	Size (Outside)	4200 X 5400	mm
Process Design	**Parameter**	**Value**	**Units**
	Size (Inside)	4080 X 5250	mm
	Installed Power	1500	Kw
	Speed	15.7	rpm

	Ball Charge	35	%
	Trommel screen aperture	12	mm
	Recirculation load	300	%
	Milled Product	80% -75µm	
Gravity / Intensive Cyanidation			
	Concentrator	KC XD30	
	Feed Rate	300	tph
	Cycle Time	45	Minutes
	Leach Reactor	CS2000	Acacia
	Batch size	3,000	kg
	Cyanide consumption	200	kg/batch
	Caustic soda consumption	16	kg/batch
	Leach-Aid consumption	8	kg/batch
Thickening & CIL			
	Thickener size	20	m
	Thickener underflow density	45	% solids
	Flocculant consumption	30	g/t
	Number of CIL tanks	1 pre-leach, 5 CIL	
	CIL Feed Rate	27	tph
	CIL Feed Density	40	% solids
	CIL Tank volume	763	m3
	CIL residence time	60	hr
	Inter-tank screen type		
	Inter-tank screen aperture	630	µm

17.2.8 Gold Security and Ore Transport

Ore from underground will be hoisted through the Central Shaft to the surface bin. Road trucks will be utilised to transport the ore from Prestea Central Shaft to the RoM pad at the Bogoso process plant. The Prestea Central Shaft loading area is secured by a site perimeter fence, security personnel controlling access and supported through a CCTV (camera) security system.

The trucks used for transportation will contain GPS for tracking and monitoring movements and the load/ore covered (for security and to reduce runoff and free gold loss during rainfall). The route from Prestea Central Shaft will predominantly be on GSR haul roads to the Bogoso process plant RoM Pad. As at Prestea, security personnel will be controlling access and be supported through a CCTV (camera) security system.

Existing security measures will be tightened through the process plant to reflect the modified Oxide plant. There will be CCTV security cameras on RoM Pad and throughout the process plant, including additional installations on CIL circuit. The existing high security areas containing the Knelson concentrators, the recommissioned Acacia and the gold room itself will remain. A weighbridge at RoM Pad has been installed to measure incoming loads, confirm unloading and reconciled to the plant weightometer on the feed conveyor.

Further measures and upgrades may be undertaken to mitigate runoff of free gold during rainfall events. This includes rain covers at RoM Pad, refurbishing the stockpile umbrella roofing and conveyor canopies.

17.3 Operating Strategy

17.3.1 Batch Milling & Continuous Leach

The oxide milling circuit will process blended ore from existing oxide pits and underground ore from Prestea through the existing oxide milling and gold recovery circuits. Until such time as the open pit oxide resource is depleted, the oxide milling and oxide CIL circuits will operate 24 hours per day, 365 days per year, such that the plant will process some 1 154 889 dry tonnes of ore over 7,076 operating hours per annum (81% overall availability).

When the oxide pits are exhausted, the Bogoso Plant will nominally process some 237,250 dry tonnes per annum from Prestea underground. Given that the RoM throughput is dropping by approximately 80%, GSR will operate the comminution section in a batch mode and the CIL circuit in a continuous mode. The two sections will be decoupled from one another by using the CIL pre-leach thickener and one or more of the six oxide circuit CIL leach tanks in a storage duty.

Each Oxide CIL tank is 1200 m3 in size, with a live volume of approximately 1140 m3. From these tanks, a nominal 30 t/h of solids or 48 m3/h of slurry (45% w/w solids) will be withdrawn 24 hours per day, 365 days per year, such that the CIL leach plant will operate for 8037 operating hours per annum (equivalent to an overall availability of 92%). The slurry will be pumped to the "PUG" CIL circuit (only run as CIL for these ores), comprising six CIL tanks-one serving as preconditioner unit and 5 others operating as CIL units in series. Each tank is 800 m3 in size, with a live volume of approximately 760 m3.

In all scenarios, two 30" Knelson gravity concentrators operating in parallel will treat 300 t/h of cyclone underflow solids. Prior to entering the Knelson, a scalping screen removes the +4 mm material and gravitating to join the Knelson tails. The Knelson tails report to the dewatering cyclone and the underflow is returned to the feet chute of the ball mill.A standby Knelson will be installed.

The circuit classification cyclones, the overflow of which is the CIL feed, are fed SAG and ball mill discharge and the dewatering cyclone o/f. Prior to the CIL leach the cyclone overflow reports to a CIL thickener and a linear screen for trash removal.

The recovered gravity concentrate will be treated in the existing Acacia circuit and gold recovered from the CIL circuit will be further processed in the existing acid wash and elution circuits. Final product will be handled in the existing gold room. The elution circuit to be used will be the 5-tonne batch unit that is part of the Bogoso non-refractory circuit.

17.3.2 Transition from Prestea Oxide to Prestea UG Fresh Ore Treatment

As per the current mining plan, oxide ore and Prestea underground ore runs as a mixture /blend continuously until May 2018 when PUG will commence as feed to the existing batch operating comminution plant.

The current Oxide plant forms the majority of the proposed plant for treating exclusively the PUG ore under the batch process scenario. Modifications are required to incorporate the major unit process augments including the feed storage tanks, which are currently used for Oxide CIL processing and the existing high intensity cyanide leach reactor (Acacia). Additional piping changes, various upgrades including the thickener and tailings systems are required. Most of the upgrades, refurbishment and modifications will be installed prior, however a 15-day plant shutdown is planned to tie-in and prepare the plant for commissioning of modified Oxide plant for batch operation.

17.4 Process Plant Manpower

The staff and labour requirements for the process plant has been developed based on current manning levels, including management and professional staffing levels for this size of operation, operating 2 x 12 hour shifts a day and 7 production days per week, 350 days per year. The breakdown of the manpower requirement for mill operation and maintenance is shown in Table 100.

Table 100 Manpower requirements for the proposed PUG ore processing plant

Department	Number
Metallurgy- Plant Operation	55
Metallurgy- Tailings Dam	13
Metallurgy – Assay & Met Labs	24
Plant Maintenance	42
Total	134

18 Infrastructure

18.1 Introduction

Prestea is a long-established mining town. The entire basic infrastructure required for the project is in place.

Figure 89 is a general plan of the existing site, showing Prestea underground site infrastructure with four major areas.

Figure 90 to Figure 92 show the more detailed surface infrastructure layouts in the Central Shaft, Bondaye Shaft, South Waste Ventilation Shaft, respectively.

Key infrastructure shown in the Central Shaft area includes:

- Central shaft headframe and winder house
- Compressor house
- Mine equipment workshop
- Warehouse
- Workshops
- Electrical substations – Prestea township and Job 600
- Mine control room
- Emergency response station
- Marsh area (for discharge water management)

Key infrastructure shown in the Bondaye Shaft area (Figure 91) includes:

- Bondaye shaft and winder house
- Compressor house
- Outdoor substation

Key infrastructure in the South Waste Shaft area (Figure 92) includes:

- South Waste shaft
- Main exhaust fan and fan drift
- Transformer station

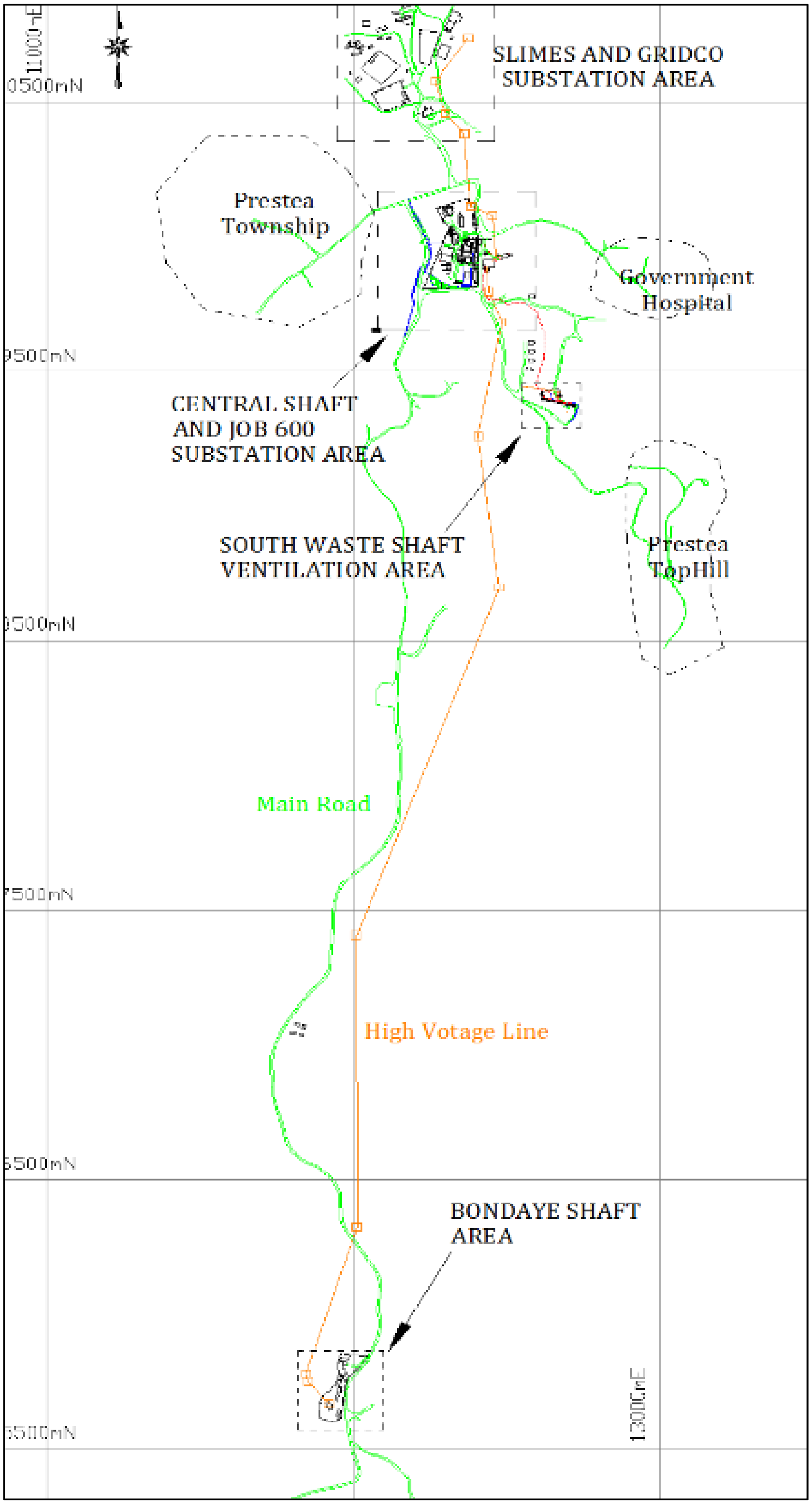


Figure 89 General site plan

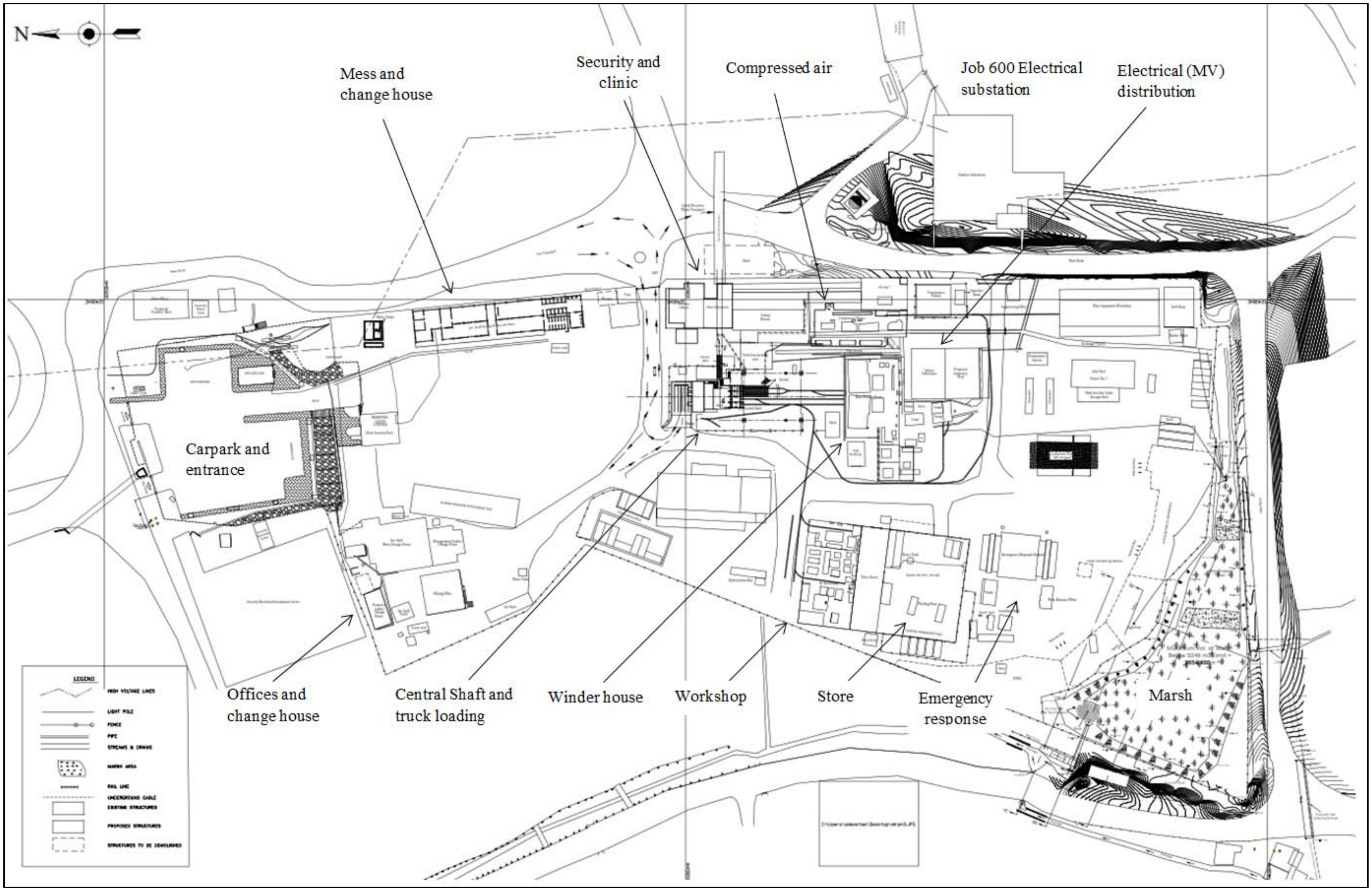


Figure 90 Surface Infrastructure Layout in the Central Shaft Area

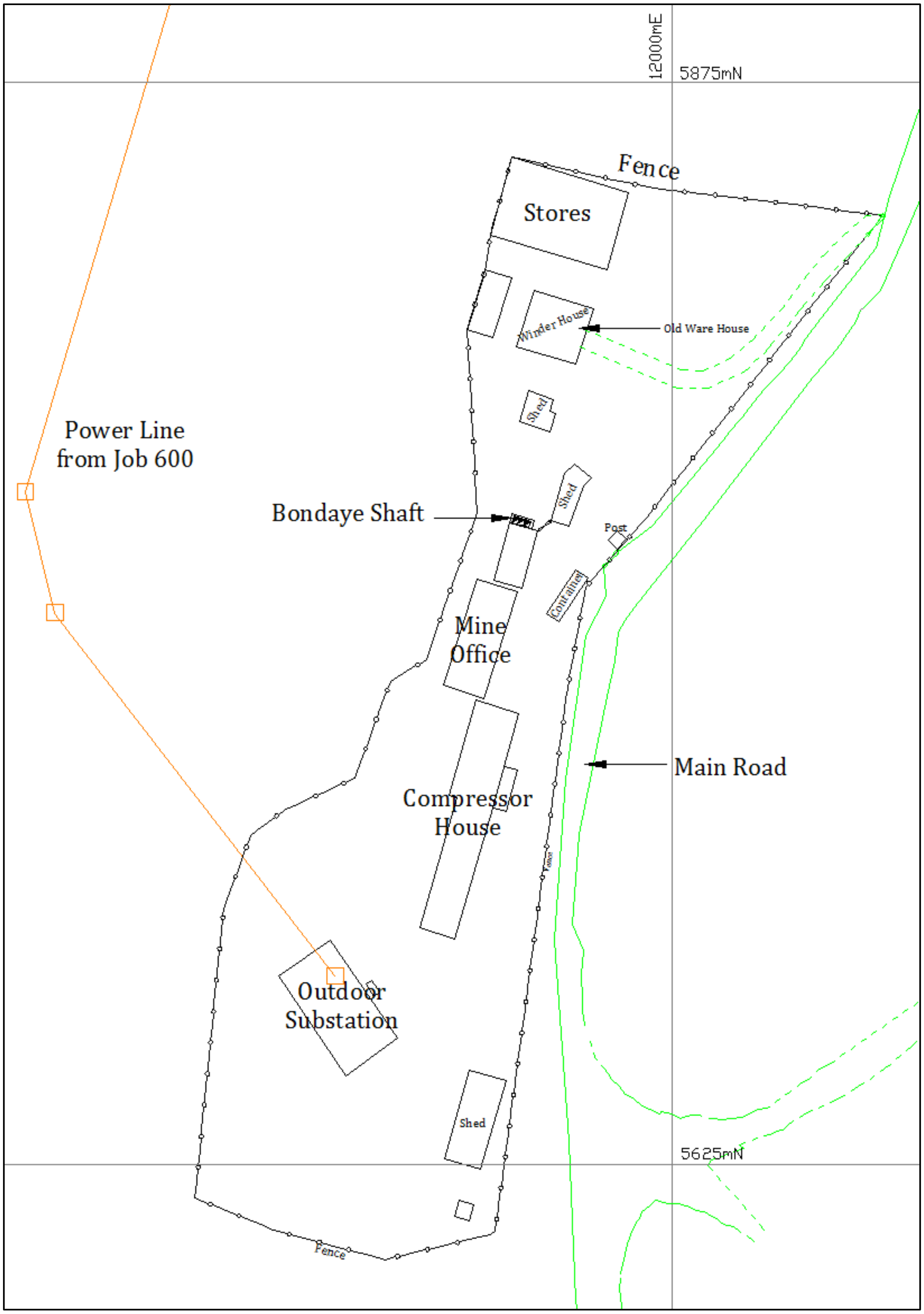


Figure 91 Surface Infrastructure Layout in the Bondaye Shaft Area

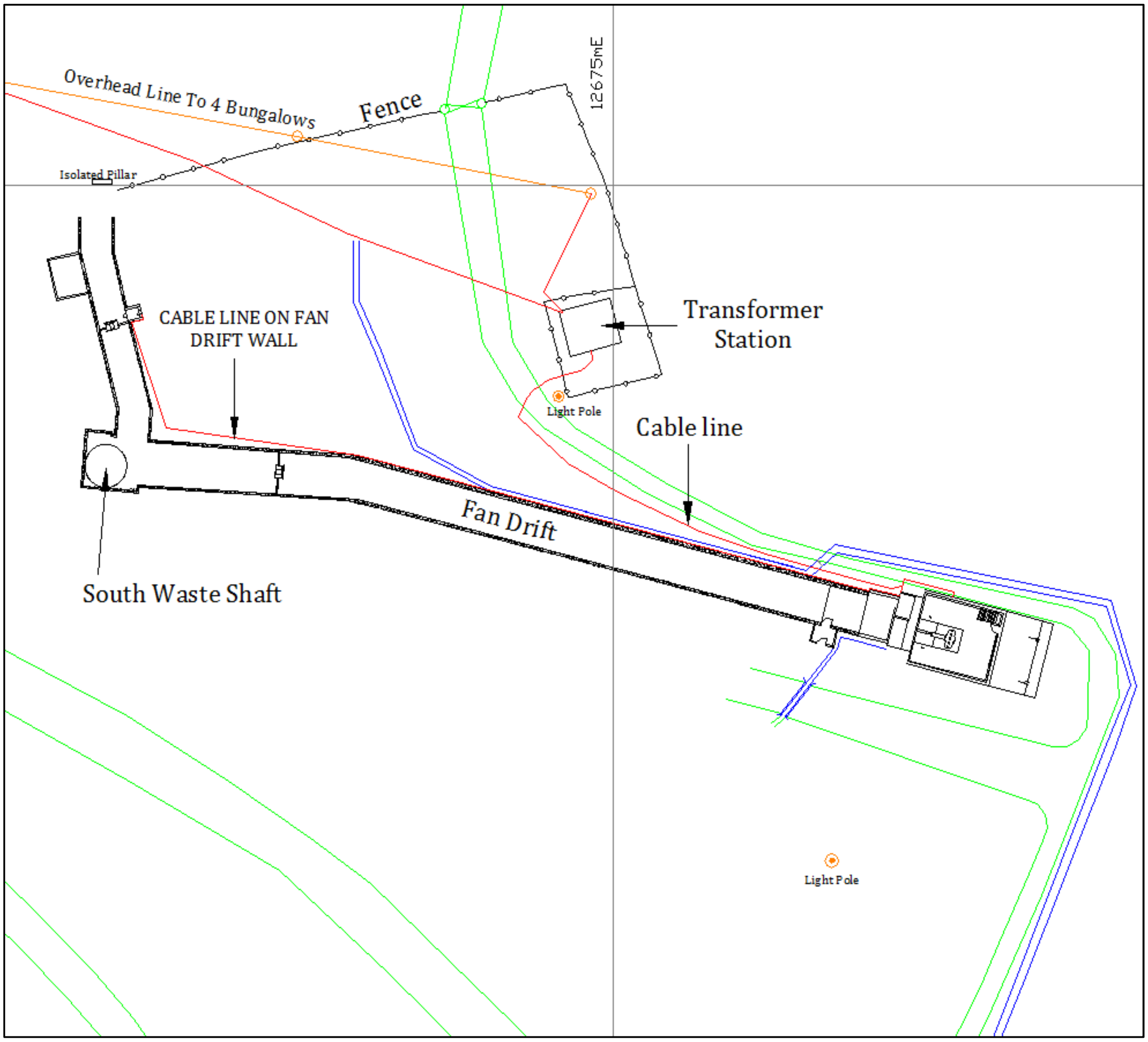


Figure 92 Surface Infrastructure Layout in the South Waste Shaft Ventilation Area

18.2 Shafts

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 1,204.5 m below collar level, with skip loading pockets below the stations at 20 L and 24 L. The sectional plan of the Central Shaft steelwork is shown in Figure 93. The shaft is generally unlined having been excavated through competent rock, although there are some sections of concrete lining where poor ground was encountered during sinking. 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 5.4t. 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 rock and material cars. The shaft services and ladder way compartments are to the south side of the skipping compartments, running the full depth of the shaft.

The shaft steelwork has been fully rehabilitated in recent years in accordance with recommendations by shaft engineering consultants to meet hoisting duties and also serve as the main egress and services shaft (e.g.: dewatering, compressed air).

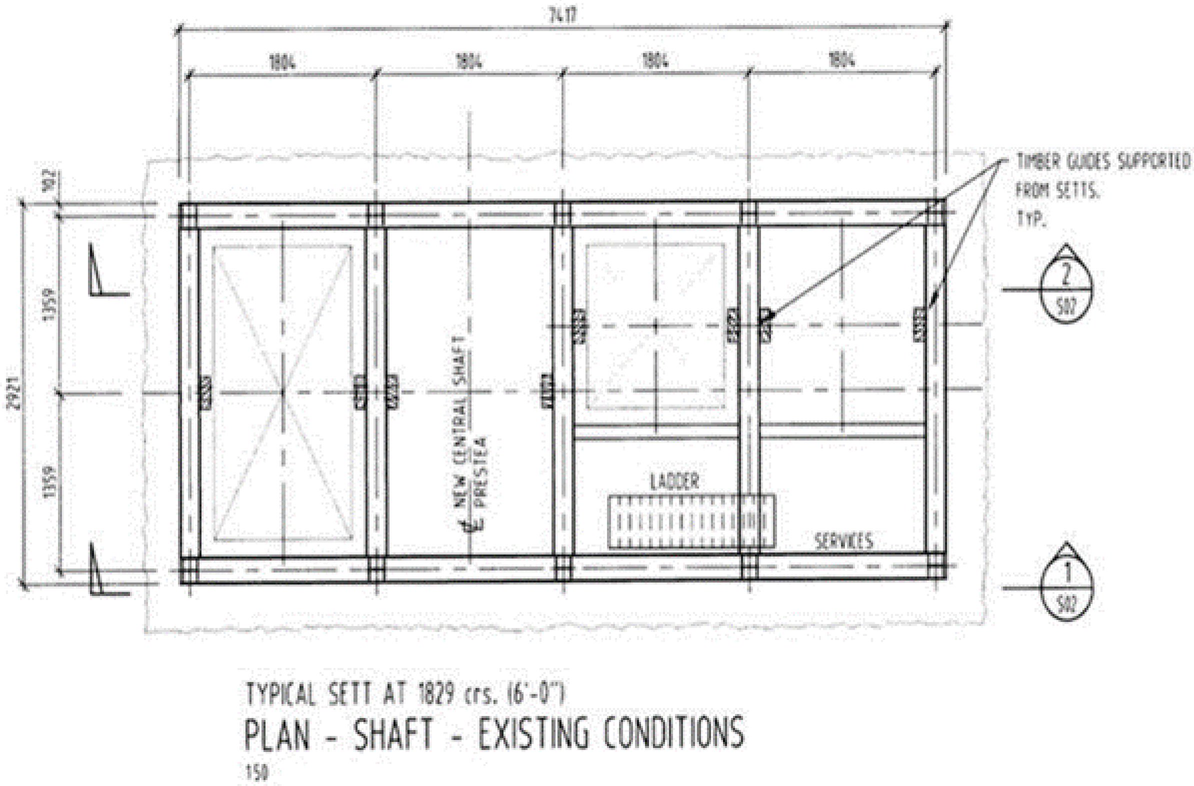


Figure 93 Central Shaft steelwork plan

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 currently 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 9 L is poor and in need of immediate attention. Beyond 9 L conditions improve; however there are localized areas that require rehabilitation.

Like the central Shaft, the Bondaye Shaft rehabilitation recommendations have been provided by shaft engineering consultants. Rehabilitation is ongoing with surface to 4 L complete.

18.3 Winders

There are three winders that service Prestea underground. Two are located at Central Shaft that comprises a production winder and a service winder, with the third a service winder located at Bondaye Shaft. All three winders were supplied by NEI-Davy Markham (UK) in 1992 and are geared, double drum configuration with a single hydraulic operated clutch.

A review of the winder condition and operation by Winder Controls (Pty) Ltd, based in Bedfordview, South Africa, found the equipment to be generally in good condition and well maintained. There were electrical and mechanical upgrades recommended to align with recent legislation changes in Ghana (L.I. 2182, 2012) as well as international safety norms, which were completed by Winder Controls (Pty) Ltd in 2017 and included:

- Doubling of brake capacity to have two independent brake systems, each capable of resisting two times the normal hoisting load;
- Brake paths linished and restored to original condition;
- Replacement of hydraulic system to integrate additional braking capacity and new clutch;
- Replacement of output shaft due to visible backlash;
- Installation of a dynamic braking system including PLC based regulator, encoders and automatic slowdown;
- Install a variable speed drive (VSD) for the 3.3kV hoist motor to replace the liquid controller and improve the control of the hoist (for the Central Shaft service winder);
- Replace liquid-controller speed control system (for Central Shaft winders);
- Installation of new motors, with the existing motors becoming spares;
- Install a hoist recording sand trending system (for Central Shaft winders); and
- Improvements to safety circuits and temperature monitoring system for motor and liquid-controller.

18.4 Electrical Infrastructure

In recent years GSR commissioned PPE Technologies, based in Nelspruit, South Africa, to complete an inspection of the primary electrical infrastructure at the GSR Prestea operations, and design an upgraded electrical distribution system to service the site – surface, underground and the West Reef mining area. As the Prestea mine electrical infrastructure dated back to the 1950s and 1960s, it was in typically poor condition,did not meet current day safety and protection standards and had several different voltages across the site. As a result extensive modifications were required to the electrical system and a progressive standardisation of voltages, which have been substantially completed.

The upgraded Prestea mine electrical distribution is supplied with electrical power from the Ghana electrical distribution company, GridCo, at a voltage level of 34.5 kV (typically referred to as 33 kV) to GSR's Job 600 outdoor substation. Job 600 substation transforms power to 6.6kV and 3.3kV for distribution at Central Shaft as well feeding 34.5kV power to the Bondaye Shaft. A new indoor medium voltage (6.6kV) switch room at Central Shaft distributes power on the surface and underground where transformers local to the equipment and power need step down to 415V.

Diesel backup generators (2 x 2MVA) step-up from 415V to 6.6kV and tie into the medium voltage switchroom at Central Shaft. These generators have sufficient capacity to power all critical equipment at Central Shaft. A separate generator at Bondaye Shaft does likewise at that location.

There is some equipment at Central Shaft, mainly pumps underground, on the existing 3.3kV system, however these will be progressively changed over to the new 6.6kV reticulation in line with planned pump system upgrades.

Bondaye Shaft medium voltage distribution will remain at 3.3kV with upgrades made to the substation to transform from 34.5kV to 3.3kV and hardware to meet modern safety and international norms.

18.5 Compressed Air

Upgrades to the compressed air system have been completed. GSR commissioned SRK Consulting (South Africa) Pty Ltd. (SRK SA) to provide a detailed compressed air distribution system for the previous shrinkage mining feasibility study, which forms the basis for mechanised shrinkage.

The upgrades are complete and comprise new compressed air generation at the Central Shaft in the form of five, oil-injected rotary screw air compressors, four on duty and one on standby at maximum demand. The compressed air is reticulated underground via the existing 300 NB shaft column and main offtakes and feed pipelines installed at key mining levels, namely 24 L and 17 L, to support the development and mining operations at the West Reef mining operations. The compressed air reticulation for the West Reef is shown schematically in Figure 94 with offtakes to various other levels omitted for clarity.

The main feed lines production level (24 L) is complete and will continue to advance with lateral development in line with the mine plan. The new feed on 17L will be installed ahead of the decline development starting and will be extended with development. Off-takes as required on the compressed air distribution network throughout the West Reef area are installed on 24 Level (planned also for 17 L) and are typically at 50 NB compete with an isolation valve, to service equipment for development and operation.

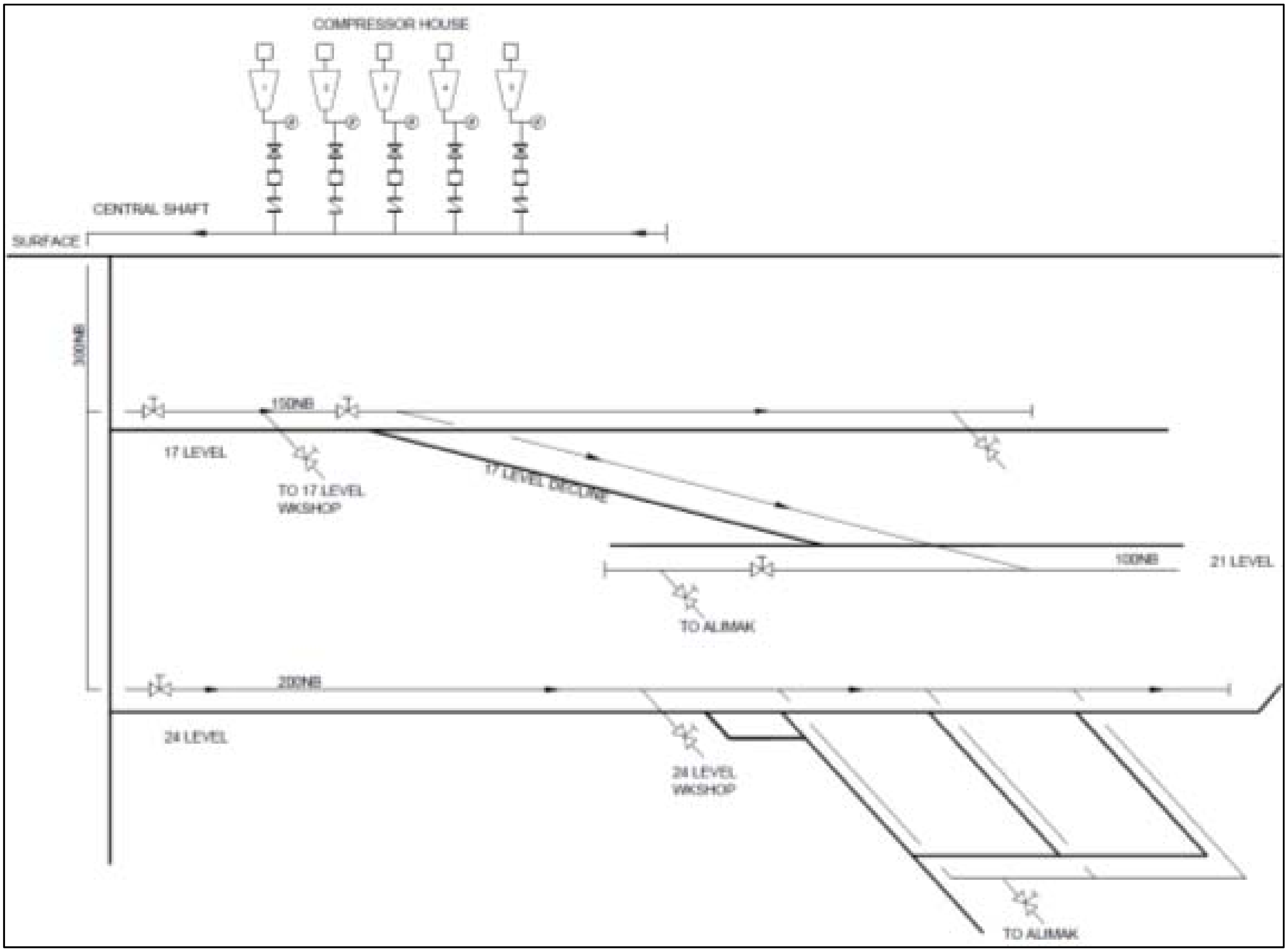


Figure 94 Schematic of the Underground Compressed Air Reticulation

18.6 Pumping

18.6.1 Existing Infrastructure

Dewatering of the Prestea underground mine continued whilst on a care and maintenance basis since mining activity ceased in early 2002 and is a requirement of the 2002 Memorandum of Understanding. Dewatering has been, and continues to be, undertaken at the two main vertical access shafts: the Central shaft and the Bondaye shaft. These shafts are connected at certain elevations by various drives or mine levels which in turn are connected to a variety of features, including natural fissures and fractured/mineralized zones of elevated permeability, mining exploration boreholes, ventilation shafts, open and partly backfilled stopes and adits.

Each of the two main shafts has a number of pump stations and associated collection sumps. The pumping stations pump water from the sumps either directly to surface or to pumping stations at higher levels and then to surface. This water is then discharged to the natural surface water drainage system. Pump operators at the mine check the sump levels on an hourly basis and manually shut down or activate the pumps depending on sump levels. The conveyance system comprises pipework with an internal diameter of 152 mm (or 6 inches).

Figure 95 and Figure 96 show the pumping station configuration for both the Central and Bondaye shafts. The configurations and number of pumps change, therefore, the pump numbers shown on the schematics are indicative only. The current mine water level is below 25 L, therefore, the pumping stations shown on Figure 95 as being below 25 L are currently flooded.

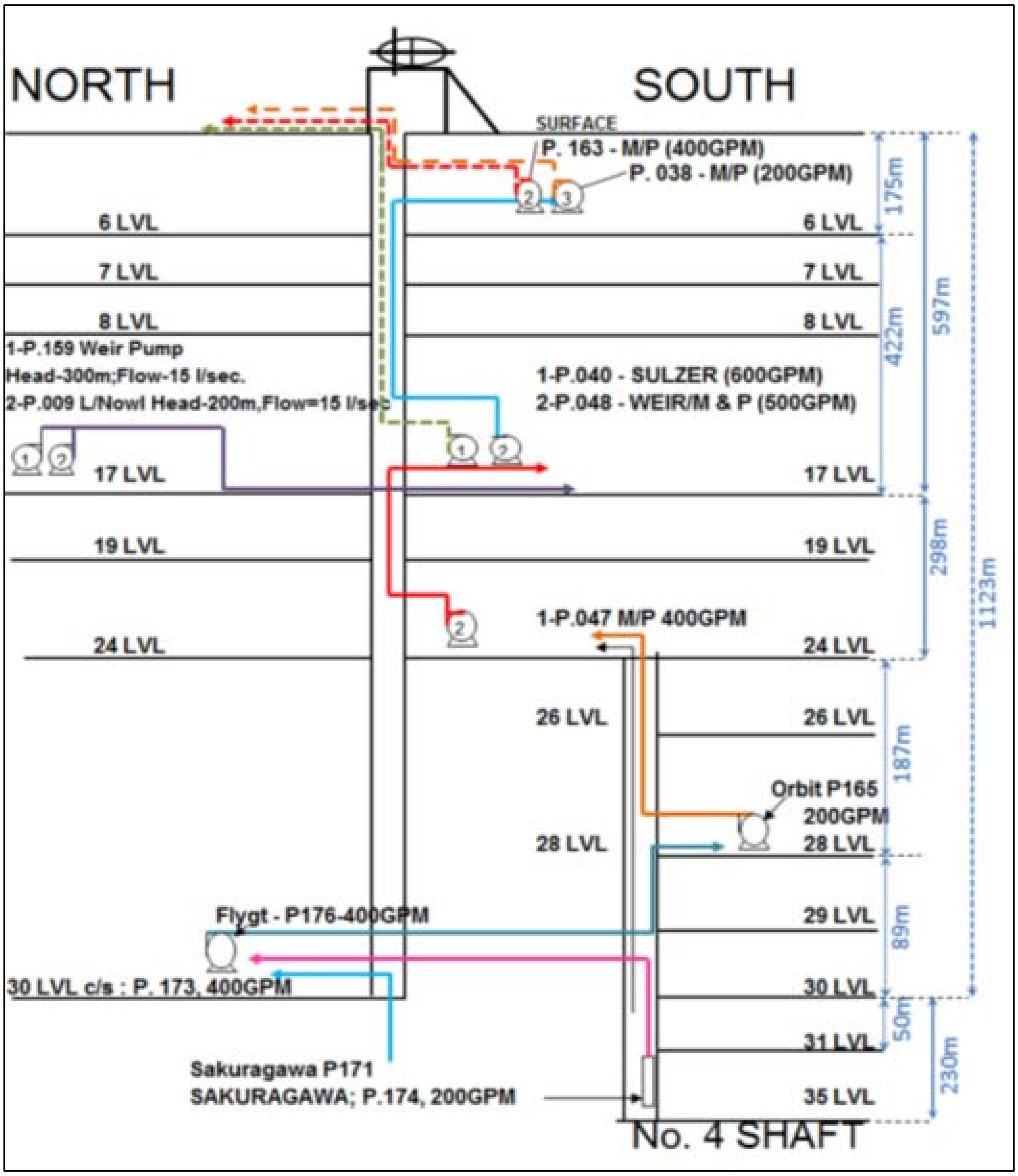


Figure 95 Dewatering System Configuration at Central Shaft (October 2012)

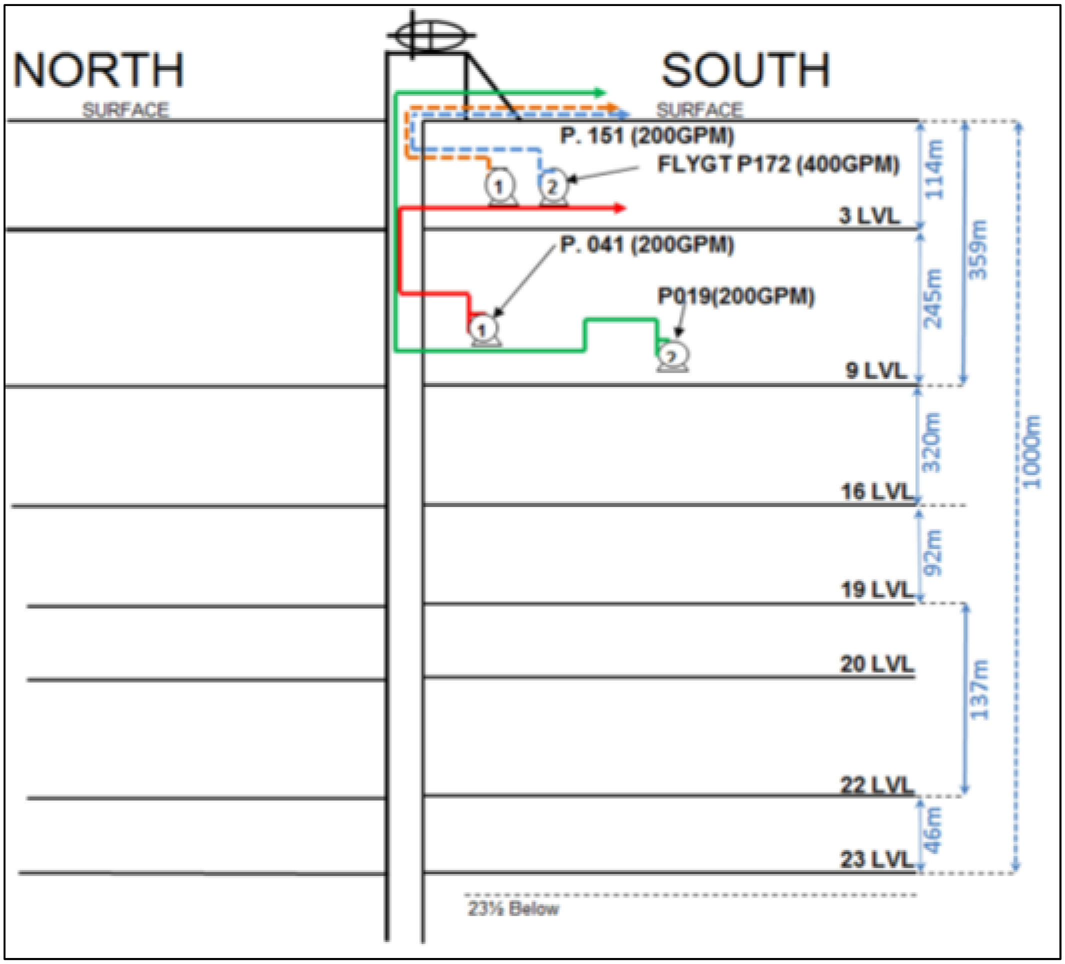


Figure 96 Dewatering System Configuration at Bondaye Shaft (October 2012)

Mine water pumped from the Central shaft is currently discharged into an open collector tank, which then gravity flows into a pipe that discharges the mine water into a reed bed system. The reed bed system then discharges the mine water via a culvert to the Nsuo Kofi stream. Mine water pumped from the Bondaye shaft is currently discharged directly to the Anobaka stream.

18.6.2 Upgrade of the Existing Dewatering Infrastructure

As part of the proposed West Reef development, the existing mine dewatering systems at the Central and Bondaye shafts will be rationalized and upgraded. Central shaft dewatering includes the following:

- At 6 L, the pump station is equipped with duty and standby pumps sized appropriately to suit the duty and the capacity required. These pumps discharge to surface.
- At 17 L, the 17 L Central shaft pump station has been equipped with duty and standby pumps sized appropriately for the duty. These pumps will pump to surface.
  - At 17 L, the 17 L Central shaft North pump station will be re-equipped with one duty pump and one standby pumps for the lateral pump to the 17 L Central Shaft pump station. It is intended that these pumps will be small submersible pumps.

- At 24 L, the 24 L Central Shaft pump station is equipped with duty and standby pumps. The pump station is located on a sub-level accessed from 24 Level and provides poor equipment and personnel access. This pump station will be decommissioned once a new pump station on 24 L is constructed with new pumps that discharge to surface. Shaft dewatering will continue through submersible pumps to the new 24 L water storage dam.

There are some pump motors that operate on 3.3kV and these will be upgraded in line with the proposed site electrical voltage (6.6kV and 415V).

Bondaye shaft dewatering includes the following:

- Decommissioning the Bondaye shaft 3 L pumping station. Water currently collected on the 3 L will be redirected down to the 9 L pump station.
- At 9 Level, the Bondaye shaft pump station, the existing 7 stage Weir MMD pumps with 3.3 kV 2 poles 275 kW motor will be supplemented with a new pump as a standby unit.
- A new pipeline will be constructed to collect water from Bondaye shaft 20 L and Central shaft 24 L haulages to the new 24 L Central shaft pump station. The pipeline will have booster sump pumps before the West Reef area and will prevent potential contamination by West Reef drain water.

The Bondaye Shaft 20 L water may be used for service water feed owing to its availability locally to the West Reef and water quality suitability, in which case the water consumed within the West Reef will be contained to the West Reef and dewatered as required via the new in Bondaye shaft pipeline noted above.

18.7 Waste Handling

Development both on 24 L and 17 L uses a combination of rail bound and trackless rocker shovels in order to install service areas (workshops, refueling bays) and tipping areas for LHD to rail mine cars. Materials handling in the stope areas is done by LHDs loading the rail mine cars.

Waste on 24 L and 17 L is hauled approximately 2km from the West Reef to the Central Shaft by locomotive for hoisting. The 24 L material is directed through a grizzly, pass system and into the 25 L loading pocket for hoisting to surface. The 17 L material passes through a grizzly, pass and chute system down to 20L before a lateral haul utilising mine cars into the 20 L loading pocket.

Waste materials are hoisted to the surface bin for truck loading and disposal in approved locations or dumped into mined voids underground as backfill materials.

18.8 Tailings Disposal

The existing tailings storage facilities (TSF) at Bogoso have been largely designed and constructed under the geotechnical oversight of Knight Piésold (KP), the engineer of record, and are operated by GSBPL. GSR will continue to utilise these facilities for the deposition of tailings generated from the processing of the Prestea underground ore.

The tailings storage facilities (Figure 97) at the Bogoso operational area can be summarized as follows:

- TSF 1 consists of a paddock style facility from which tailings was hydraulically re-mined from 2013 until August 2015 for reprocessing. In the period of reprocessing some 3 Mt tailings was removed from this facility. TSF 1 has been permitted by the EPA and engagement is underway with the Minerals Commission to enable the recommencement of tailings deposition into this TSF in the future. The void created by the tailings re-processing itself provides sufficient capacity for the LoM PUG West Reef tailings storage.

- TSF 2 is a paddock style facility, consisting of three cells: a combined cell 1/2, 2A, and 3. A total of 12 embankments separate and border the cells.
  - Cell 1/2, and Cell 2A have traditionally impounded cyanide bearing floatation tailings, and occupy a total combined area of ~60 hectares (ha).
  - Cell 3 has impounded a combination of non-cyanide BIOX tailings and more recently, cyanide bearing non-refractory tailings. Cell 3 occupies a total area of ~170 ha and is presently subject to paddock deposition and revegetation trials ahead.

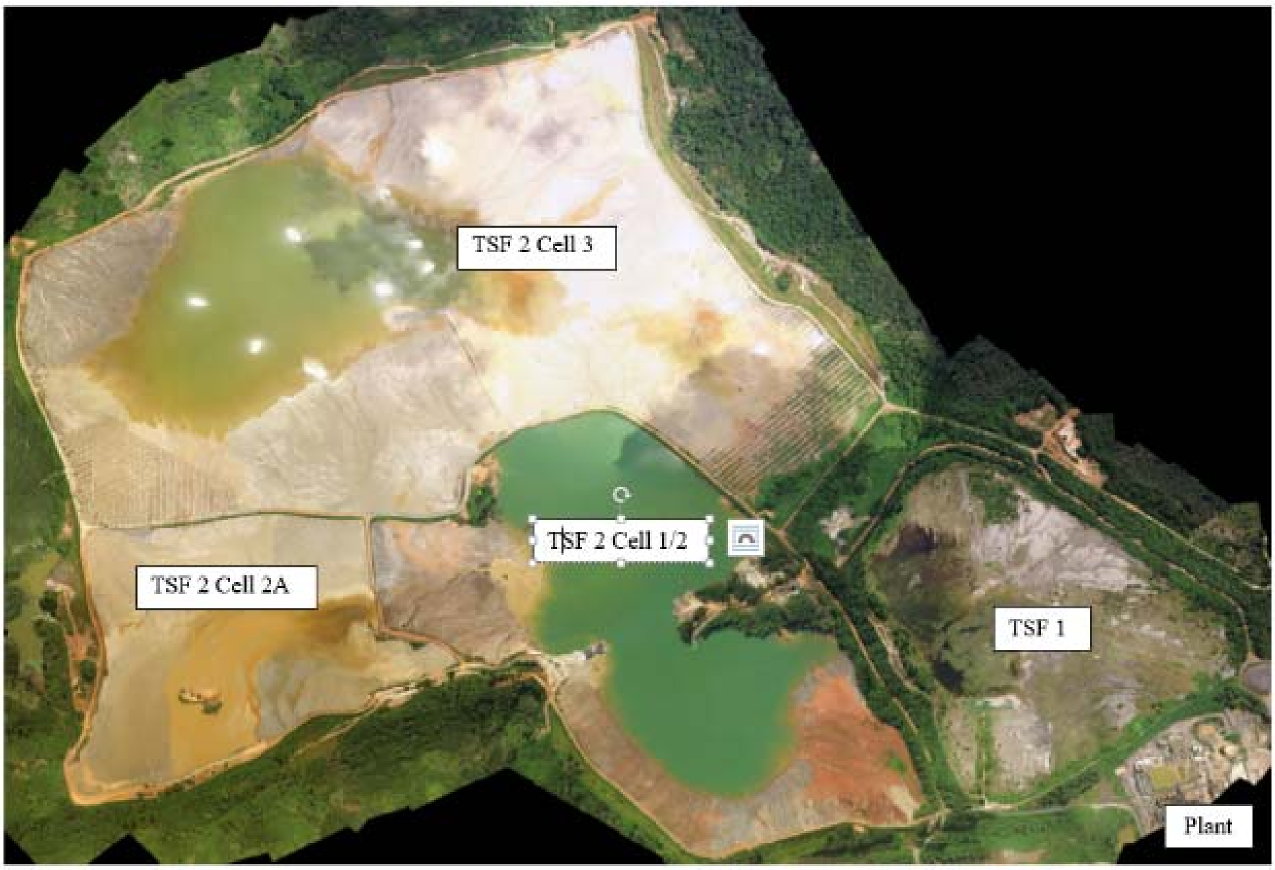


Figure 97 Bogoso tailings storage facility - aerial image (Nov 2017)

18.8.1 Future Tailings Operation Strategy

The Prestea Underground (PUG) tailings throughput has been estimated based upon the ore throughput included in the technical report mining schedule. The current schedule assumes that the plant will treat Prestea ore from commercial production in early 2018 and continuing through 2023. A total of 1.16 Mt ore will be processed over the LoM period.

GSR is utilizing the existing, approved Bogoso TSFs to store tailings produced from the Prestea Underground operations. The re-processing of tailings from TSF 1 has created a 3 Mt void, well in excess of the volume of tailings expected to be produced over the life of the West Reef project. Additional tailings storage capacity is also available in TSF 2, which is presently receiving process tailings.

18.8.2 Volumetric Assessment

In 2015 SRK completed volumetric analyses of the existing facilities and confirmed that there was sufficient storage capacity for all tailings and supernatant expected to be produced during the LoM, including the tailings expected to be produced from the Prestea Underground West Reef operations.

The analyses conducted demonstrated not only did the TSF 1 contain sufficient volume for the LoM tailings, but that there was also sufficient overall volume in the TSF 2 design to safely contain the forecast throughput of tailings until 2022. Thus, either TSF 1 or TSF 2 contain sufficient capacity for the forecasted LoM tailings from the combined surface and underground GSBPL operations.

18.8.3 Overview of Embankment Design and Construction

The Bogoso TSF 2 initially comprised a dual-cell paddock-type facility located in a broad valley-system approximately 1.5 km northeast of the Plant Site. It was designed to provide storage for flotation tailings to be produced by the Bogoso plant in the period 2004 - 2013.

The starter facility comprised two cells, a western cell (Cell 1) and an eastern cell (Cell 2), separated by a dividing wall constructed across a natural neck in the valley. The starter facility was commissioned with the commencement of tailings deposition to Cell 2 in April 2004 and by October 2004 Cell 2 was nearing capacity. Cell 1 was complete and commissioned in March 2005.

A major expansion of the TSF 2 was undertaken in April 2006 and entailed the construction of two new cells, Cell 2A and Cell 3. The design for Cell 3 encompassed 7 embankment with the cell to provide storage for non-cyanide bearing flotation tailings from the then proposed BIOX plant.

Due to depleting freeboard, in January 2010 a detailed design for Cell 3 commenced which included the proposed raises of 6 embankments to an elevation of 5080 m RL. A further raise (stage 3) in late 2010 to 5082 m RL was completed in August 2011. Further raising of Cell 3 (stage 4) was completed in December 2012 to an elevation of 5084 m RL. The total area of Cell 3 is approximately 170 hectares.

In November 2014 construction commenced to raise the Cell 1/2 and 2A embankments to elevation 5083.3 m RL to accommodate stored water rather than tailings in these cells. Work was completed on these raises during April 2015.

Permitted elevations for the Bogoso TSF 3 are given in Table 101. Each of the TSF 2 cells is either at or within permitted tailings embankment elevations.

Table 101 Bogoso TSF 3 permitted embankment elevations

TSF 2 Cells	Permitted crest elevation (mRL)
Cell 1/2 eastern embankment	91.8
Cell 1/2 south western embankments	90.8
Cell 2A	83.3
Cell 3	86.0

18.8.4 Tailings Deposition

The facility employs the sub-aerial tailings deposition method in which tailings slurry is discharged from a pipeline onto a section of previously deposited tailings by means of spigots off-takes located along the upper edge of the confining embankment. The discharge slurry flows gently over the sloping beach and forms a uniform layer. Once the section of beach has been covered to a depth of approximately 200 mm, tailings discharge is moved to another section of beach and the newly deposited layer is left to release excess slurry water. The released slurry water flows down the beach surface to the supernatant pond and is reclaimed as a clarified water through the decant system. The method ensures the supernatant pond forms remote from the external confining embankments. Barge-mounted decant pumps return the supernatant pond water to the process plant for re-use. Quarterly TSF inspections and operational monitoring conducted since the construction of the TSF continues to show no evidence of seepage from the TSF 2 into the external environment.

18.8.5 Stability Analysis

As the engineer of record, Knight Piésold conduct slope stability analyses for external embankment, critical conditions as a routine component of each design raise and/or expansion. These analyses, assessed against the Ghana Minerals and Mining Commission Regulation L.I. 2182 (1), Canadian Dam Association (CDA) and International Committee on Large Dams (ICOLD) standards/guidelines, are required for raise permitting and approvals for construction.

In 2015 SRK undertook a review of the slope stability analyses completed by KP for each of the TSF 2 cells with a focus on critical external embankments as these represent the greatest risk of significant downstream consequences associated with failure

The assessment found that the critical embankment sections analyzed in Cell 3 met the acceptability criterion under all loading conditions. At that time, it was noted that the position of the pond in relation to the external embankments should be monitored closely with regards to the phreatic surface in the embankment. This was the result of a period of excess water inventory build that had resulted from the positive water balance of the refractory plant operations. With the cessation of refractory plant operations in late 2015, the resulting net negative water balance and ongoing treatment of process water inventory for release, as at 31 January 2018, the supernatant pond volume on the three TSF 2 cells was less than 1.18 Mm^3, well within normal operating parameters. Of this volume, less than 83,000 m^3 was contained within Cell 3.

In relation to cells 1/2 and 2A, the SRK 2015 analysis disagreed with some slope stability assessments of KP for reasons of being conservative with regards to original construction conditions. A subsequently process of review by SRK, KP and Golder Associates failed to achieved an aligned opinion, although SRK subsequently acknowledged that factors of safety were within international standards requirements. As a result of the review process GSBPL has conservatively installed a toe buttress at TSF 2 embankment 19 and is installing a similar buttress at embankment 4 and later embankment 1. A buttress for the northern extent of TSF 1 is also under consideration and if necessary would be installed prior to the recommencement of deposition into TSF 1.

18.8.6 Tailings Water Balance

GSBPL maintains a detailed, daily time-step water balance model that is coded using the GoldSIM simulation package. The model represents the interdependencies and interactions among the various control structures and users of the water system.

A dynamic stochastic model allows the quantification of these variabilities thereby laying the foundation for effective integrated water management, with management of water at Bogoso driven by:

- The need to provide an adequate and reliable supply of process water;
- The need to store/reuse water where possible; and
- The need to discharge water of complying water quality to the environment.

The daily dynamic continuous probabilistic model gives a picture of the various possible outcomes of different scenarios and assesses the impact of risk, allowing for better decision making to address the risk arising from the uncertainties.

In 2015, with the suspension of refractory processing operations, the water balance at the Bogoso operations was significantly simplified. The GoldSim model outputs (Golder 2015) were used to populate a visual schematic representation of the key water balance components (Figure 98) and actual water balance for 2017 are illustrated in Figure 99.

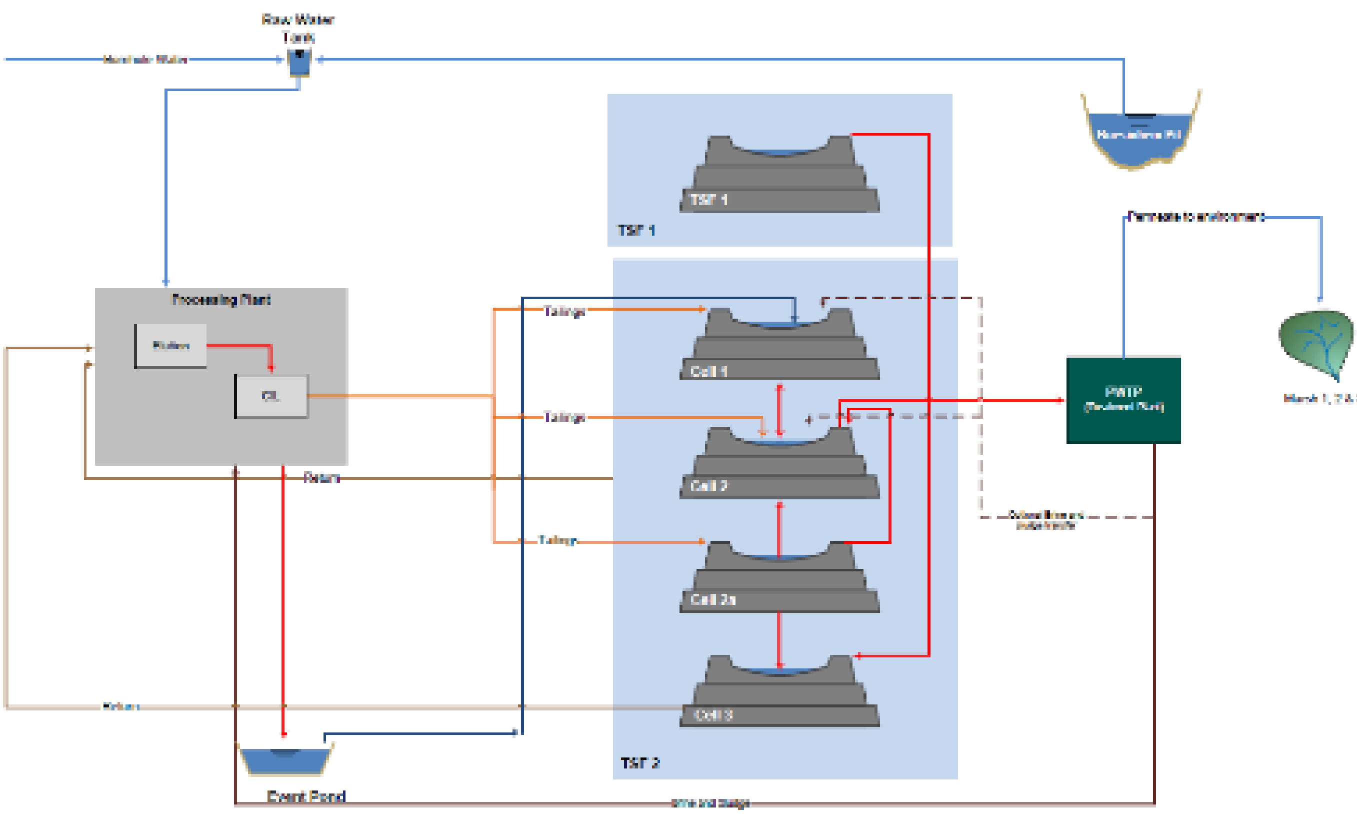


Figure 98 Bogoso Process Water Balance Schematic (Golder, 2015)

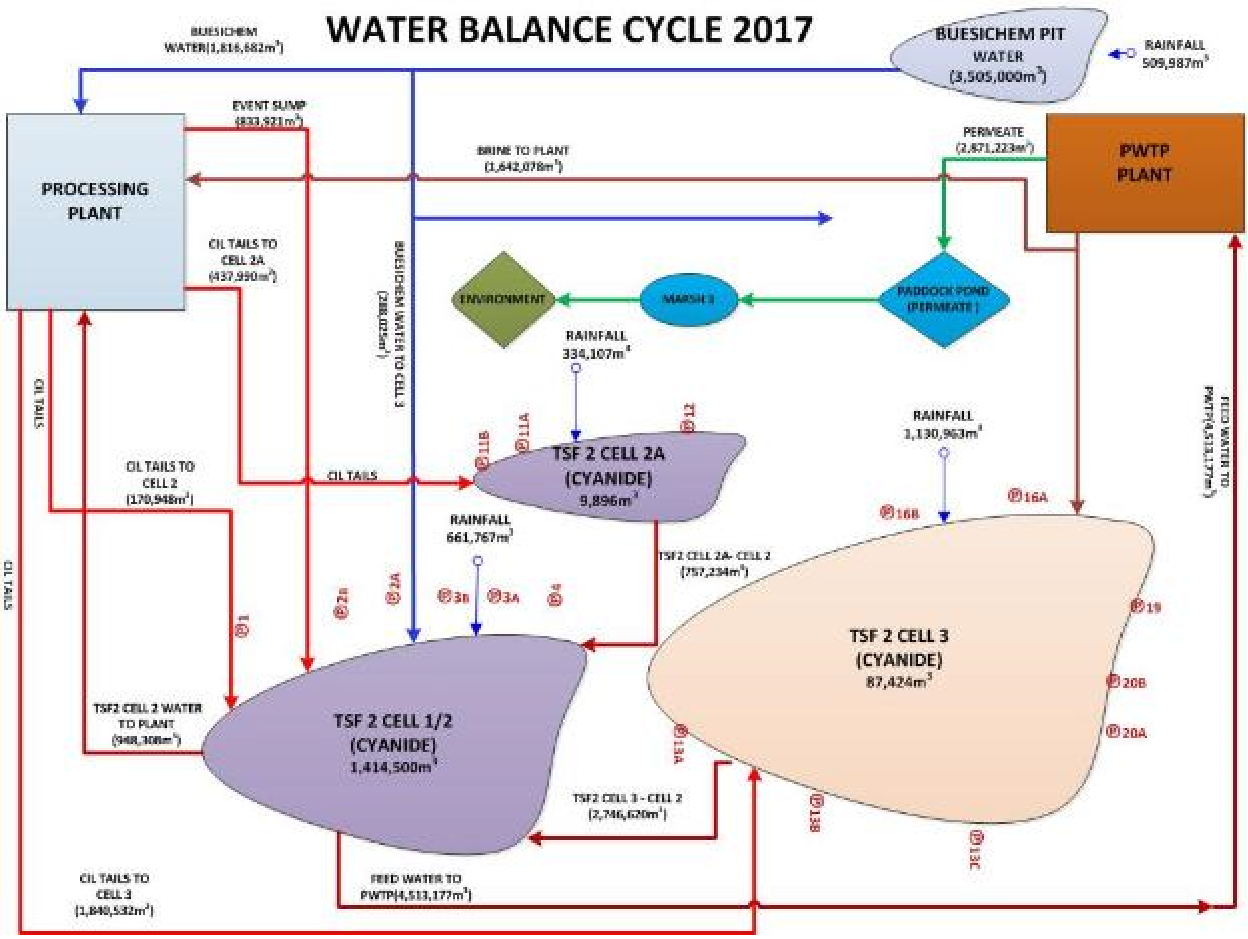


Figure 99 Bogoso Process Water Balance Actual 2017

18.9 Water Treatment

Impact assessment and technical studies for the PUG West Reef indicate that the mine will remain a dry mine for much of the mine life. Further, specific West Reef geochemical studies found that host rocks of the West Reef are non-acid generating, and mine drainage from the area is predicted to be near neutral to alkaline drainage, with all parameters expected to be within EPA guidelines.

In the event that mine water inflows from historic mine voids are not well managed and are allowed to impact the West Reef mine area, even under worst case conditions, as simulated by NAG Leach tests, the mine dewater quality would achieve EPA discharge criteria for most parameters, notably not for sulphate, and arsenic.

Whilst potential mine dewater volumes are low (conservatively calculated to be less than 300 m^3/day), in the event that mine dewatering is required, and worst case drainage conditions arise, GSBPL has undertaken studies on the potential design and cost of suitable water treatment systems for the possible West Reef dewater.

Studies included various treatment options and associated cost benefit analysis. Treatment options that were analysed are summarised as follows:

- Passive iron amendment for arsenic adsorption and co-precipitation;
- Passive packed bed sulphate reducing bacteria (SRB) biofilm reactor;
- Active water treatment using chemicals for settling and co-precipitation;
- Active water treatment using precipitation, membrane filtration and reverse osmosis; and
- Active water treatment using electrochemistry.

Other study findings include:

- Management of underground settling sumps will improve resulting discharge quality.
- A range of feasible water treatment technologies are available with varying capital and operating cost commitments.
- Plant designs would be progressed through further water analysis and design phases.

This technical report includes allowance for the design and construction of a water treatment plant should conditions in the PUG West Reef mining area require dewatering in future.

19 Market Studies and Contracts

19.1 Market Studies

Gold is a freely traded commodity on the world market. The World Gold Council in its gold demand trends report (WGC 2015) stated that the global gold market is in an overall demand-supply balance. Top-line demand was broadly neutral despite substantial underlying differences across geographies and sectors among jewellery, technology, investment, central banks, and institutions. Total global supply was little changed year-on-year as lower recycling offset growth in mine supply.

For the Prestea Mine, all gold produced is shipped to a South African gold refinery in accordance with a long-term sales contract currently in place for GSR. The gold is shipped in the form of doré bars, which average approximately 90% gold by weight with the remaining portion being silver and other metals. The sale price is based on the London p.m. fix on the day of the shipment to the refinery.

19.2 Contracts

The following contracts will be part of the rehabilitation, mine development, and operations of the Prestea mine:

- Long-term doré bar sales contract is in-place with South Africa gold refinery for GSR's Bogoso mine.
- A general mining explosive supply agreement is in-place with AEL Mining Services, a South Africa based mining explosive supplier, for GSR's Ghana mine operations. Explosive price is subject to monthly adjustments based on raw material cost changes.
- Shaft hoist, headframe, and some surface infrastructure rental agreements with Ghana governments are in place.
- Alimak raise mining and stoping training contract with Manroc Developments Ltd of Ontario, Canada.

20 Environmental Studies, Permitting and Social or Community Impact

20.1 Relevant Legislation and Required Approvals

The Minerals and Mining Act, 2006 (Act 703) is the governing legislation for Ghana's minerals and mining sector. It requires that mines obtain environmental approvals from relevant environmental agencies as outlined in Table 102. Ghanaian environmental legislation is well developed and is enforced by the Environmental Protection Agency (EPA).

20.1.1 Environmental Assessment Requirements

The overarching Act that regulates the environmental regime of Ghana is the EPA Act, 1994 (Act 490). 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). These regulations cover requirements for environmental permitting, environmental impact assessment (EIA), the production of preliminary environmental reports (PERs), and subsequent Environmental Impact Statements (EIS), environmental certificates, Environmental Management Plans (EMPs), and reclamation bonding.

The EPA grants environmental approval to projects, in the form of an Environmental Permit. The decision on whether or not to grant the permit is based on the findings of an EIA, which also covers social aspects and is documented in an EIS. For a mine, an EIS must include a reclamation plan (Regulation 14 of LI 1652) and a provisional EMP. The EIS may be subject to a public exhibition period and public hearing before formal review by the EPA. Responses of regulators and community obtained through these processes are redirected to the proponent for incorporation into the Final EIS, before an Environmental Permit is granted.

Within 24 months of receipt of an Environmental Permit, mines are required to obtain an Environmental Certificate from the EPA (Regulation 22 of LI 1652). The Environmental Certificate is a follow-up mechanism that confirms: commencement of operations; acquisition of other permits and approvals where applicable; compliance with mitigation commitments made in the EIS or EMP; and submission of annual environmental reports to the EPA.

An EMP must be submitted within 18 months of commencement of operations and must be approved by the EPA. A provisional EMP is typically provided in an EIS, with the expectation that the new project EMP would be incorporated into the mine's overarching EMP when it is updated. Mines are required to update their EMPs every three years and have to submit the updated EMPs to the EPA for approval (Regulation 24 of LI 1652).

All mines in Ghana are required to have a reclamation plan (Regulation 14 of LI 1652). In addition, mining operations have to submit annual environmental reports (Regulation 25 of LI 1652) and monthly environmental returns of the environmental parameters monitored to EPA. Comments are also expected in cases where monitored values exceed limits and, as appropriate, a project is to provide the measures to prevent further occurrences.

Table 102 Primary Environmental Approvals Required for Mining Operations

Regulatory Institution	Approvals that Have to be Obtained	Reporting, Inspections and Enforcement
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.	**Environmental Permit** In accordance with Section 18 of the Mining Act, 2006 (Act 703), and the Environmental Assessment Regulations, 1999 (LI 1652), of the EPA Act, a holder of a mineral right requires an Environmental Permit from the EPA in order to undertake any mineral operations. **Approved 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 (Regulation 14 of LI 1652). **Reclamation bond** Mines must post a reclamation bond based on an approved reclamation plan (Regulation 22 of LI 1652).	**Reporting** Mines must submit monthly returns and 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.
Minerals Commission and Mines Inspectorate Division Established under the Minerals and Mining Act, 2006 (Act 703), the Minerals Commission administrate mineral rights in trust for the people of Ghana.	**Exploration and mining operating plans** A holder of a licence shall not commence operations unless an Operating Permit is issued by the Inspectorate Division for the operations. Modifications to operating plans are required to be approved by the Chief Inspector of Mines. **Emergency response plan** An emergency response plan must be submitted to the Inspectorate Division for approval. **Resettlement plan** LI 2175 defines specific requirements for compensation and resettlement, including approval of resettlement plans by the district planning authority. **Closure Plan** Regulations 273 to 277 provided detailed requirements for closure requirements and plans. **Other** An array of other permits and licences (e.g., explosives) are required to be obtained in support of mining operations, which incorporate environmental and social requirements.	**Reporting** Mines must submit monthly and quarterly returns. **Inspections** The Mines Inspectorate undertakes regular inspections to ensure that mineral right holders are compliant with regulations and laws generally. **Enforcement** Regulations 21 and 22 allow the Mines Inspectorate to issue improvement and/or prohibition notices for contraventions of the Regulations.

Regulatory Institution	Approvals that Have to be Obtained	Reporting, Inspections and Enforcement
Water Resources Commission Established under the Water Resources Commission Act, 1996 (Act 522), the Water Resources Commission is responsible for the regulation and management of the use of water resources.	**Approvals for water usage** Under Section 17 of the Mining Act, 2006 (Act 703), 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.	**Reporting** Holders of water use permits must submit quarterly and annual reports to the Water Resources Commission. Inspection The Water Resources Commission has power to inspect works and ascertain the amount of water abstracted. Enforcement Both Act 522 and LI 1692 prescribe sanctions for breaches.
Forestry Commission and Forestry Services Division	In accordance with Section 18 of the Mining Act, 2006 (Act 703), a holder of a mining right must obtain necessary approvals from the Forestry Commission.	

Guidelines and standards relevant to the mining industry have been made under the EPA Act. These include the Mining and Environmental Guidelines (1994), which provide guidance on the contents of EIS and EMP reports and of reclamation plans. They also include guidelines on EIA procedures, effluent and emission standards, ambient quality and noise levels and economic instruments.

The EPA conducts routine monitoring of environmental parameters for mining operations and the results obtained are cross-checked with the monthly return values submitted by operations and compared to relevant standards.

The EPA is empowered to suspend, cancel, or revoke Environmental Permits where the holder is in breach of LI 1652, the permit conditions, or the mitigation commitments in the EMP. Contravention of these regulations, failure to comply with directives of the EPA, and failure to submit annual environmental reports are offences that may result in fines or imprisonment.

Under the “Enforcement and Control” provision of Act 490, the EPA may, in the event of activities of any undertaking posing a serious threat to the environment or public health or simply non-complying with LI 1652, direct the immediate cessation of the activities or steps to be taken and the time within which to prevent or stop the activities. Where the EPA issues such an Enforcement Notice, all relevant institutions responsible for the issue of approvals for the operation are duly informed not to grant other approvals to the facility until notified otherwise by the EPA.

20.1.2 Minerals and Mining Requirements

The Minerals and Mining Act, 2006 (Act 703) establishes law on the process for obtaining mineral rights, and the administration and management of these rights and for the protection of the environment.

Supporting the Act, are the Minerals and Mining Regulations, 2012. These cover general aspects (LI 2173), matters relating to compensation and resettlement (LI 2175), explosives (LI 2177), support services (LI 2174), and health, safety and technical requirements (LI 2182).

The regulations listed below have particular relevance to environmental and social management:

- Minerals and Mining (Health, Safety and Technical) Regulations 2012 (LI 2182) – these regulations define requirements for approval of mine closure plans, hazard classes for tailings storage facilities, and set requirements for embankment design, factors of safety, impoundments, freeboard, discharge systems, safety arrangements, monitoring, planning, auditing, and closure.
- Mining General Regulations 2012 (LI 2173) – these promote preferential employment of Ghanaians and preferential procurement of goods and services from Ghanaian service providers. Mines are required to prepare localization plans to achieve this and to submit frequent reports (monthly, six-monthly and annual reports) that provide information on Ghanaian and expatriate staff numbers as well as information on payments of salaries and wages, royalty, and corporate tax.
- Mines (Support Services) Regulations, 2012 (LI 2174) – these extend the requirement to preferentially employ Ghanaians to providers of services to mines.
- Mines (Compensation and Resettlement) Regulations, 2012 (LI 2175) – these require that displaced people are resettled to suitable alternative land and that their livelihoods and living standards are improved. The resettlement plan must be approved by the district planning authority and then given effect by the Minister responsible for Mines.

GSBPL has a localization plan that has been approved by the Minerals Commission that covers expatriates. GSR is listed on the Ghana stock exchange and continues to submit its annual financial reports as required by the law.

20.1.3 Water Resources Legislation Requirements

The Water Resources Commission Act, 1996 (Act 552) establishes the Water Resources Commission and provides for its composition and functions in the regulation and management of the utilisation of water resources in Ghana, and for related matters. Specifically it sets requirements regulating the use of water resources, including the requirement to obtain authority for the use of water resources for commercial and domestic purposes, and granting of water rights.

The Water Use Regulations, 2001 (LI 1692), and Drilling Licence and Groundwater Development Regulations, 2006 (LI 1827), complement the Act by specifying: the requirements for obtaining permits for water use, water rights, and priorities for water use; and water drilling licences, and well construction requirements; respectively.

20.1.4 Existing Environmental Approvals

A summary of environmental approvals held by GSBPL is provided in Table 103.

Prestea

The Prestea area has a long gold mining history, starting in the 18th century, with official mining underway by about 1887. Early mechanized operations at Brumasi, Prestea Central, and Bondaye included the use of steam-powered drills, explosives for tunnelling, and timbering. By 1929, the

Ariston and Gold Coast Main Reef mines were operational on the area covered by the current Prestea concession.

By 1965, the gold mines at Prestea had been acquired by the Government of Ghana through the State Gold Mining Company Limited (SGMC) and the local mines were held by Prestea Goldfields Limited. Mining was predominantly underground; the shafts, headgear, tailings, rock waste dumps, roads, and other infrastructure remain as evidence of these activities. Prestea Goldfields Limited continued operation until the 1990s, when the operations were privatized and eventually awarded to Prestea Gold Resources (PGR) in 2000. PGR operated for a short period and ceased operations in 2002.

Immediately after Prestea Gold Resources' operations ceased, New Century Mines – a three partner joint venture, consisting of the Ghana Government, GSBPL and Prestea Gold Resources – was formed to take over the rights held by Prestea Gold Resources on the Prestea concession. The underground mine was placed on care and maintenance by New Century Mines, while GSBPL commenced studies on the potential for the re-opening of the underground operations.

Mining activities that have taken place on the Prestea concession since that time include:

- Re-processing of historic tailings and dumps by Prestea Sankofa Gold Limited (PSGL)
- GSBPL surface mining operations at the Buesichem (now used for water storage), Brumasi and Plant North pits (now backfilled)
- Unauthorized small scale mining activities (locally called Galamsey)
- Unauthorized mining activities using mechanized equipment
- Recommencement of the Prestea underground mine (PUG) – Phase 1 operations
- Commencement of the Prestea underground mine (PUG) – West Reef operations (formerly known as Phase 2 operations).

The PUG Phase 1 operations involved recovery of relatively small quantities of ore from historical stopes. From March 2002 to late 2012, PUG was under care and maintenance. In December 2012, following EPA and Mineral Commission permitting (EPA/EIA/804), GSBPL commenced PUG Phase 1 (remnant) mining operations from a permit perspective view.

GSBPL has subsequently permitted the West Reef (Phase 2) operations of the PUG and commercial production was announced in February 2018. These operations incorporate a range of new infrastructure including surface waste backfilling, underground raisebore development of ventilation raises, water treatment and increased ore transportation. The status of the environmental permitting process for this project is explained in Section 20.1.6.

Bogoso

Mining in the Bogoso area has a relatively long history, although there is no official record of any workings prior to 1882. Small scale mining activities began near Bogoso in 1906 on a commercial basis and continued to the early 1930s.

In the early 1930s, several large low-grade gold deposits were recognized and developed as open pit operations by Marlu Gold Mining Areas Ltd. When the Marlu and Bogoso North deposits were mined out, other prospects were developed further south, i.e., Chujah, Dumasi, Ablifa and Buesichem. Gold ore production peaked in the period from 1936 to 1942 at nearly 800,000 tonnes per annum (t/a). The operations then closed as a result of declining grades and reserves of the oxide ores and recovery problems with the sulphide ores.

The State Gold Mining Corporation examined and sampled the underground adits during the early 1970s and the United Nations Development Programme financed a two year drilling program in 1976-78, which concentrated on the Chujah-New Dumasi deposit. This work indicated substantial low-grade reserves. In 1981, a United Nations Development programme sponsored study evaluated the available information on the old workings at Bogoso.

The Bogoso prospecting concession was awarded to Denison Mines Limited in May 1986. Denison entered into a joint venture agreement with Sikaman Gold Resources Limited. The joint venture then formed a Ghanaian company, Canadian Bogoso Resources Limited, with shareholdings held by Denison Mines Limited, Sikaman Resources Limited, the Republic of Ghana and the International Finance Corporation (IFC). Within the same year, a Canadian consulting group conducted preliminary exploration and concluded that there was considerable merit in further exploration and in early 1987 detailed studies were conducted to provide a prefeasibility study on technical aspects of a potential mining operation at Bogoso.

In April 1988, Billiton International Metals BV, a subsidiary of the Royal Dutch Shell Group, acquired Denison Mines' share and, in 1992, the company's name was changed to Billiton Bogoso Gold Limited. Further study indicated that, due to the apparent inadequate supply of oxide ores to sustain a viable project, emphasis should be directed towards providing a sustainable project based on the refractory (sulphide) gold resources. It was concluded that an operation based on the mining and processing of sulphide ores, using open pit mining methods and the flotation and roasting of sulphide concentrates, was viable. Design and construction of the treatment plant and associated facilities were completed in 1991, when mining and processing activities commenced.

In early 1994, following a lengthy period of technical and ore grade problems, it was recognized that the roasting process was unsuitable for the ore being mined due to the low sulphur content. The sulphide flotation and roasting sections were shut down in February 1994 and the mining and processing of oxide ores commenced. In November 1994, Gencor Limited purchased the majority of Shell's exploration, mining and processing interests.

Gencor disposed of its interests in the project in March 1998 to the senior lenders and other banks, so that in 1999 the company's shareholders were IFC (28%), Deutsche Investitions und Entwicklungsgesellschaft mbH (DEG) (20%), the Republic of Ghana (10%), and the remainder owned by several commercial banks.

In September 1999, GSR purchased its interest in the Bogoso mine from the consortium of banks led by the IFC. A series of new pits were permitted to supplement existing oxide pit sources, including the October 2001 permitting of the Buesichem and North deposits (EPA/EIA/044), in May 2002 the Brumasi pit (EPA/EIA/059), and in November 2002, the Prestea Plant North pit (EPA/EIA/069), all on the Prestea concession.

When Bogoso Gold Limited was acquired by GSR, a feasibility study was completed into the processing possibilities for the sulphide ore. The feasibility work resulted in the design and construction of a processing plant expansion to process the sulphide ore using bio-oxidation (BIOX®). This and the associated mining of several refractory pits were permitted by the EPA in June 2005 (EPA/EIA/147).

In May 2006, an extension to the TSF 2 was permitted (EPA/EIA/188), and mining at Pampe was permitted in November 2006 (EPA/EIA/219).

Re-mining and re-processing of tailings from the existing TSF 1 was permitted in October 2011.

Following the submission of an EMP in April 2005, GSBPL received its first Environmental Certificate for the period January 16, 2006 to January 15, 2009 (EPA/EMP/045). The certificate renewal process was initiated with the submission of a new EMP to the EPA that was approved for certification in February 2009. In November 2011, the subsequent EMP was submitted to the EPA, and in March 2013 GSBPL was invoiced for certification. In December 2015, GSBPL submitted an updated EMP for its operations which will remain in force until the end of 2018.

Table 103 Environmental Approvals Obtained for the Bogoso and Prestea Operations

Approval	Permit No.	Date of Issue	Expiry Date	Comments
Environmental Protection Agency and Minerals Commission				
Bogoso Gold Project		1988		Canadian Bogosu Resources Limited
Bogoso North East pit		1993		Billiton Bogosu Gold Limited
Buesichem Main and North deposits	EPA/EIA/044	15-Oct-2001	N/A	Based on Mining of Buesichem Main and North Deposits SEIS (2001)
Brumasi	EPA/EIA/059	3-May-2002	N/A	Based on Brumasi SEIS (2002)
Prestea Plant North	EPA/EIA/069	7-Nov-2002	N/A	Based on Prestea Mining Project Plant North EIS (2003)
Bogoso Sulphide Project (refractory plant and mining/expansion of several pits)	EPA/EIA/147	7-Jun-2005	N/A	Based on Bogoso Sulphide Project EIS (2005)
Bogoso EMP	CM 81/02	30-Jun-2005	N/A	Bogoso Gold Limited
Tailings storage facility II extension	EPA/EIA/188	5-May-2006	N/A	Based on Tailings Storage Facility (TSF) II Extension EIS (2006)
Pampe	EPA/EIA/219	29-Nov-2006	N/A	Based on Pampe Project EIS (2010)
GSBPL EMP	EPA/EMP/045	16-Jan-2006	15-Jan-2009	
TSF 2 Water dilution and discharge	EPA/EIA/317	21-Oct-2010	20-Feb-2011	Based on EIS for Proposed TSF II Water Dilution Project (2010)
Marlu Tailings Re-Mining				Draft EIS submitted 7-Aug-2013
TSF 1 Re-Mining	EPA/EIA/340	31-Oct-2011	N/A	Based on EIS for Reprocessing of Tailings from TSF 1 (2011)
Process water treatment plant				Submitted 22 June 2011 with verbal approval Nov-2012
Transfer to Buesichem process water management facility	Letter CM 8/7	23-Sep-2011	22-Mar-2012	
Transfer to Buesichem process water management facility	Letter CM 81/8	10-Oct-2012	10-Jan-2013	
Prestea Underground Gold (PUG) mining project Phase 1	EPA/EIA/804	17-Dec-2012	N/A	Based on EIS for Phase 1 of Prestea Underground Project (2012)
GSBPL EMP				Submitted Dec-2015. Valid through 31-Dec-18.
Process water treatment plant EMP	EPA/EMP/157	15-Dec-2015	14-Dec-2018	Final submitted Dec-2015
Prestea South Mbease Nsuta	EPA/EIA/425	12-Jun-2015	N/A	Based on EIS for Prestea South Mbease Nsuta project (2015)
Mampon –Abronye	EPA/EIA/469	24-Jan-2017	N/A	Based on EIS for Mampon-Abronye project (2016)
Prestea (West Reef)				Based on EIS for Prestea (West Reef). Permit invoiced 29-Jan-2018. Pending permit issuance.
Prestea Underground Mining Operating Plan				Submitted and approved annually.
Water Resources Commission				
Water Use Permit (groundwater for domestic use)	GSBPL ID 288/1/17	01-Jan-2017	31-Dec-2019	
Water Use Permit (12 bores, pits and ponds)	GSBPL ID 288/2/17	01-Jan-2017	31-Dec-2019	
District Assembly				
Prestea Projects Resettlement Action Plan	PHDA/ADMIN/ MC's/39	Dated 6-Nov-2013,	N/A	Prestea Huni-Valley District Assembly

Approval	Permit No.	Date of Issue	Expiry Date	Comments
		received 1-May-2014		
Mampon-Abronye Resettlement Action Plan	WAEDA/BGL.3/Vol.I/79	Dated 16-Nov-2016	N/A	Wassa Amenfi East District Assembly

The Environmental Certificate and the EMP are for the overall GSR Bogoso/Prestea operations, incorporating operations on the Bogoso, Prestea, Asikuma and Pampe concessions.

Requirements under the schedule to the Environmental Permit and in the EISs pertaining to the operations include:

- The Company must post a reclamation bond within one year of commencement of operations – GSBPL posted its initial reclamation bond in December 2005. The bond is updated periodically to reflect approval of new/expansion projects. As at the end of the year 2017, the GSBPL bond was US$ 8,100,000.
- Compliance with other applicable legislation.

The mining leases listed in Section 4.6 also contain conditions relevant to environmental management. The Bogoso Mining Lease LVB 570B-87 and 272-80 (WR348B-87 and WR366-88), Prestea surface Mining Lease LVB 2876-01 (WR3218/01), Prestea underground Mining Lease LVB 2799-01 (WR 3218/01), Pampe Mining Lease LVB 14181B-2007, and Asikuma Mining Lease PL. 3/32/Vol 2 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.

20.1.5 Historic Environmental Liability and Indemnity

In June 2001, Bogoso Gold Limited acquired the surface and mineral rights to gold and other associated mineral substances within the Prestea Mining Lease. This area had been subjected to over 100 years of mining activity and Bogoso Gold Limited’s acquisition agreement incorporates an indemnity agreement with the Government of Ghana for the environmental liabilities emanating from these activities (dated December 21, 2001). GSR later renamed Bogoso Gold Limited to GSBPL. At the same time as establishing the Bogoso Gold Limited indemnity, Prestea Gold Resources also entered into an indemnity agreement with the Government of Ghana for the Prestea underground concession (see Section 20.1.4 for history on Prestea Gold Resources and New Century Mines joint venture).

In order to quantify the nature of the existing liability at the time of the agreements of indemnity with the Government of Ghana, audits were completed (referred to as ‘Baseline Study’ in the Agreements) and documented as registers of environmental liabilities. GSBPL continues to actively monitor the nature, scale, and impact of these liabilities to fully understand the pre-existing conditions.

GSBPL recognizes the pre-existing extensive environmental impact of both historic mining and modern day unauthorized small-scale mining and illegal medium-scale mining activities throughout the Prestea concession. GSBPL has predominantly located the PUG West Reef mining activities in specific preselected areas in order to isolate the West Reef underground mining activities from those of its predecessors, especially underground water management and dewatering system.

While there are pre-existing impacts across the concession for which GSBPL is indemnified, GSBPL will be in a unique position, at reclamation of its operations, to make an overall improvement in environmental conditions within the Prestea concession. Given the heavily urbanized nature of the Prestea community, and to a lesser extent the area around the Bondaye community, GSBPL also has the potential for a broader range of next (post-mining) land uses than is typical for most mining areas, such as establishment of recreational areas.

20.1.6 Primary Approvals Required for the PUG West Reef Project

For the PUG West Reef, an EIA was completed and an EIS was subsequently submitted to the EPA to obtain an Environmental Permit for the operations. As required, the EIS contains a provisional EMP and closure plan for the project. GSBPL has also updated the Mining Operating Plan with the Mines Inspectorate to reflect the underground mining areas. The Mining Operating Plan has also addressed modifications to the existing surface operations.

The status of the permitting, land acquisition and stakeholder engagement for the PUG West Reef are explained in Sections 20.3.5 and 20.3.6.

20.2 International Requirements

20.2.1 Environment and Conservation

The Government of Ghana is a party to a number of international treaties relating to the environment, notably:

- Ramsar Convention on Wetlands of International Importance - regulated under the Wetland Management (Ramsar Sites) Regulations, 1999 of the Wild Animals Preservation Act, 1961 (Act 43) providing for the establishment of Ramsar sites within Ghana. There are five designated Ramsar sites along the coast of Ghana. There are no Ramsar sites in the project area.
- Convention of International Trade in Endangered Species (CITES).
- United Nations Framework Convention on Climate Change.

In regards to protected areas, Ghana has one UN Biosphere Reserve and two World Heritage Convention Sites (UNESCO, 2009). These are not in or near the project area. According to the World Resources Institute, EarthTrends (2003), in a country profile for Ghana, it was reported that Ghana has more than 1,000 IUCN-management protected areas including 317 Forest Reserves. There are no forest reserves in the vicinity of the project area.

20.2.2 Human Rights

In 2005 GSR, with the full support of its Board of Directors wrote to the UN Secretary General as a statement of commitment to adoption of the United Nations Global Compact. GSR’s 2017 Corporate Responsibility Report will be its twelfth report on progress of implementation of the UN Global Compact, and GSR continues to integrate the UN Global Compact principles into its business activities (www.unglobalcompact.org). The UN Global Compact's ten principles in the areas of human rights, labour, the environment are derived from:

- The Universal Declaration of Human Rights
- The International Labour Organisation's Declaration on Fundamental Principles and Rights at Work
- The Rio Declaration on Environment and Development
- The United Nations Convention Against Corruption

Through its annual public Corporate Responsibility Report GSR details ways in which the company is contributing to advance Ghana's performance in regards to the Sustainable Development Goals.

20.2.3 Anti-Corruption

The Government of Ghana was designated as Extractive Industries Transparency Initiative compliant in 2010. In support of this, GSR publically reports on an annual basis on the payments made by the company to the Government of Ghana. As at the end of 2017, GSR businesses have made significant contributions to the people of Ghana through government payments:

- GSBPL Life to date: over US$174 million
- In 2017 the Office of the Administrator of Stool Lands, Traditional Authorities, Stool Lands, and District Assemblies had expected royalty distributions from GSR's operations of over US$1.7 million

GSBPL's parent company GSR being registered in the US and Canada is subject to the US Corruption of Foreign Officials Act, and the Canadian Corruption of Foreign Public Officials Act. Internal GSR policies address these items for GSR management.

20.2.4 Voluntary Codes

GSR has adopted a number of voluntary international codes and standards of practice pertaining to corporate responsibility at the GSBPL operations:

- Tailings storage facilities – current TSF 2 designs align with the requirements of the International Committee on Large Dams (ICOLD), and Canadian Dam Association (CDA).
- Gold mining and processing – as a member of the World Gold Council, GSR ascribes to the Responsible Gold Standard.
- Extractive Industries Transparency Initiative (EITI) – GSR works against corruption in all its forms and continues to participate in the EITI by reporting on payments made to Government.
- Resettlement, land acquisition, and compensation – GSR has since 2009 ensured all resettlement projects conform to the International Finance Corporation (IFC) Performance Standard 5 on Land Acquisition and Involuntary Resettlement.
- Voluntary Principles on Security and Human Rights – as a signatory to the UN Global Compact, GSR supports the Voluntary Principles by ensuring that all private security personnel are inducted to the standards. Additionally, GSR's private security contracts and Memoranda of Understanding with public security require adherence to the voluntary principles.

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.

20.3 Environmental and Social Management

20.3.1 Golden Star Corporate Commitment

GSR has policies pertaining to the environment, community relations and human rights, health, safety, and wellbeing and community development and support. These policies enunciate the commitment of the company to appropriate corporate governance, are reviewed annually, and are endorsed by the company President / Chief Executive Officer.

Policy on the Environment

GSR is committed to meeting or surpassing regulatory requirements in all of its exploration, development, mining and closure activities while safeguarding the local environment for stakeholder communities and future generations.

Policy on Community Relations and Human Rights

GSR is committed to being a part of the community in which it operates by maintaining and building strong relationships with other members of the community based on mutual respect and recognition of each other's rights, together with an active partnership and long term commitment to the betterment of the community and local economic development. GSR supports and respects the protection of international human rights.

Policy on Safety, Health and Wellbeing

GSR values and is committed to safety and employee wellbeing. GSR believes that job-related injuries and illnesses are unacceptable.

In support of the company policies, GSR demonstrates its management commitment through provision of appropriate and dedicated specialist human resources in the disciplines of environment, safety, health, community affairs and resettlement. In 2017, GSBPL employed 63 dedicated personnel in the disciplines of environment, communities, safety, health, and security, representing over 9 % of the total employees.

GSR supports achievement of its corporate policies by providing training and development for its workforce and members of the community.

20.3.2 Social Investment

Golden Star Development Foundation

The primary vehicle for GSR's social investments is the community-led Golden Star (Bogoso/Prestea) Development Foundation, which is funded annually with US$1.0 per ounce gold produced and 0.1% of pre-tax profit. Under the foundation umbrella, GSBPL works with local Community Mine Consultative Committees, government bodies, and third-party non-governmental organizations (among others), to strategize and implement a variety of community development projects and programs.

In 2017, GSR contributed over US$0.26 million to the foundation, bringing contributions to date to over US$2.6 million.

Golden Star Oil Palm Plantation

Golden Star Oil Palm Plantation (GSOPP) is a community-based oil palm plantation company established in 2006 as a non-profit subsidiary of GSR. The program adopts the small-holder concept of sustainable agribusiness, which addresses environmental, food access, and community concerns. Initially, development is sponsored by GSR as part of its local economic development program. The plantations are later able to become self-supporting and the small-holder farmers pay back the start-up loans to GSOPP to allow for further development. GSR commits US$1.0 per ounce gold produced to the program, resulting in over US$6.2 million in funding as at year end 2017. To date, GSOPP has established 1,133 ha of plantations including 100 ha of out-grower plantations. In 2017, GSOPP produced and sold over 11,900 tonnes of oil palm fruit. In 2017 GSR commenced expansion of GSOPP into two former TSF cells as part of its next land use activities.

Capacity Building and Livelihoods Enhancement

Employment, particularly for the youth continues to be of the foremost concern to GSR's catchment communities. Education and training initiatives are extended to GSR's community out-reach programs, with a view of imparting lasting educational benefits to stakeholder communities.

The Golden Star Skills Training and Employability Program provides training to young people in practical and technical skills in sectors unrelated to mining, contributing to the diversification of the local economy's employment base. This program has also been integrated into many of the negotiated resettlement agreements that conform to the IFC Performance Standard 5 on involuntary resettlement.

Inaugurated in 2009, as at the end of 2017, 14 courses had been run as part of the program, providing skills training to over 600 youth in trades such as masonry, commercial cookery, carpentry, mobile phone repairs, building electrical, beads and jewellery making, hair dressing, local fabric bags and sandal making, and others.

GSR also provides scholarships for needy students attending secondary school. Since 2008, the company has provided scholarships for over 890 children. A further 3,000 registered dependents of our employees are also supported through educational subsidies on an annual basis.

20.3.3 Corporate Responsibility

In accordance with its commitment to the UN Global Compact, GSR supports and respects internationally proclaimed human rights within their sphere of influence. As per GSR's policy on Community Relations and Human Rights, GSR works to create a culture that makes the protection of human rights an integral part of the short-term and long-term operations, including the performance management systems.

Since 2011, GSR has conducted periodic human rights desktop reviews in conjunction with its top suppliers with results reported to the GSR Sustainability Committee. This will help to further ensure that GSR is not complicit in any human rights abuses – directly or indirectly.

Building on training covering human rights matters for our Human Resources personnel – and later our wider workforce – GSR developed a similar program covering matters related to harassment and discrimination awareness and prevention.

In 2016 GSR began implementing a number of major underground safety management programs to further embed safety management into its increasingly underground operations. These programs were further embedded into GSR's operations including:

- Upgraded emergency and crisis management systems;
- Emergency response capability and capacity enhancement;
- Underground mine focussed safety standards development; and in 2018
- Safety cultural reviews and safety leadership development initiatives, amongst others.

GSR is dedicated to engaging in accurate, transparent, and timely two-way consultation with local stakeholders in order to communicate on the business, and address the needs of local partners. Regular dialogue with stakeholders – including but not limited to public meetings, open houses, and sensitization forums – is central to understanding key issues and concerns related to the operations, and, in turn, helps to realize sustainable solutions suitable to the stakeholders.

GSR assumed the role as a catalyst for sustainable economic development in the communities in which operations are situated. Doing so enhances relationships with partners by maximizing the benefits that accrue to the stakeholder communities. Accordingly, GSR makes regular investments in local communities that go beyond traditional philanthropy, namely by adopting a strategic approach to social investment. This helps to create lasting, meaningful benefits for local communities and contributes to a positive long-term legacy surrounding the operations.

In the area of security and human rights, in 2014, GSR commenced a program of training and awareness with its security personnel, and public security personnel engaged through the Ghana Chamber of Mines, in the Voluntary Principles on Security and Human Rights. All security personnel are required to ascribe to the principles through this program. In 2015 this program continued, and GSR's private security contracts and Memoranda of Understanding with public security had been upgraded to require adherence to the voluntary principles. Further the principals were incorporated as a standard part of induction for new personnel to the security team.

20.3.4 Environmental and Social Management System

For existing operations, environmental management is addressed through an Environmental and Social Management System (EMS) developed along the lines of an ISO 14001 EMS. This allows the operation to provide a program addressing the legal and corporate needs for monitoring and reporting. The EMP and the associated Environmental Certificate provide legal framework for GSBPL environmental management, while EIS and associated Environmental Permits provide the legal framework for project developments.

Community management at GSBPL is carried out by dedicated and professional personnel in the fields of environment and communities. GSBPL has established a series of Community Mine Consultative Committees within the local stakeholder communities. These committees collect the recommendations and then report them to the corporate and company entities (such as the Golden Star Development Foundation) on behalf of the communities.

The Community Mine Consultative Committees are responsible for selecting development projects and assisting the operations understanding of community concerns and needs. Development opportunities for the stakeholder communities are funded by the Golden Star Development Foundation, and through direct mine-funding.

GSBPL maintains a grievance mechanism enabling catchment communities to document concerns and grievances for investigation / action. The mechanism is well publicized 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.

20.3.5 Status of PUG West Reef permitting

The EIA process for the PUG West Reef project was carried out in two phases. The first baseline data collection and impact assessment processes occurred for the then proposed mechanized mining of the West Reef deposit as described in the NI 43-101 technical report for the Prestea West Reef feasibility study (SRK UK, 2013). Subsequently, a further process of EIA was conducted to reflect the modified project scope that is for the non-mechanized shrinkage mining of the deposit as described in the NI 43-101 technical report on the preliminary economic assessment (GSR, 2013).

The baseline data collection for the EIA is completed, however GSBPL continues to monitor baseline conditions for its due diligence. With the completion of the EIA, an EIS was compiled and submitted to regulatory agencies in March 2017. The EPA issued advice that the EIS could proceed to pubic exhibition in June 2017 and the EIS was then publically advertised in July 2017. After the EPA set a Public hearing date for December 2017 an appeal was made by Traditional leadership that a public hearing was not required given the extensive historic project consultation. The EPA agreed to undertake alternative validation consultations and a stakeholder engagement in lieu of public hearing was held in January 2018. In late January 2018 the EPA invoiced GSBPL for the Environmental Permit, an effective project approval. The permit issuance is now pending.

20.3.6 Status of Land Acquisition for the Project

GSBPL has, wherever possible, sought to minimize or avoid the displacement of people or assets. As intended by the national regulations, and GSR's commitment to the IFC PS 5, GSBPL has incorporated a number of project design features to prevent or minimize potentially adverse impacts on the surrounding local communities and assets.

Where land used for farming or habitation (whole or partial) will be required, GSBPL will adhere to all the applicable laws governing land acquisition and compensation. The PUG West Reef surface land requirements are largely confined to existing infrastructure locations.

Since 2005, a substantial crop compensation program has been carried out across the Prestea area in preparation for GSBPL's Prestea projects. To date in excess of GH¢1,040,000 have been paid for crops covering an area of some 472 ha, the majority of which relates to the GSBPL surface mining projects (Prestea South Mbease Nsuta), but which also include the transportation route for the PUG West Reef. In total, over 733 farmers have been compensated. The approved Prestea Projects Resettlement Action Plan provides summary details of compensation paid. The Prestea Projects Resettlement Action Plan (RAP) was approved by the Prestea Huni-Valley District Assembly on May 1, 2014. The finalized RAP was released on September 29, 2014.

As it is possible that further areas of compensation may be required for localized areas of the West Reef mine development, any further compensation associated with the Prestea projects will be carried out in accordance with the GSR procedure as described in the approved Prestea Projects Resettlement Action Plan.

20.4 Environmental and Social Issues

This section highlights environmental and social issues that could affect the project permitting, operations or maintenance of approvals, issues that are of concern to local stakeholder communities and/or issues with management costs that may affect the value of the assets. Environmental and social impacts that can be managed readily without remarkable cost are not discussed here.

20.4.1 Community Expectations and Sensitivities

Employment

The main socioeconomic concern for most stakeholders is employment. The local community around the Bogoso and Prestea operations see working at the mine as a preferred occupation. The extension of the mine life with the development of the underground mine has received wide-spread local support. Community expectations will be managed via the normal community consultative methods.

Although GSBPL is unable to employ all the people seeking work, there is a Corporate Responsibility Agreement in place addressing local employment that provides affirmative action for people from local stakeholder communities. All vacant positions are advertised locally first and then nationally. Local people are recruited exclusively for unskilled positions and as much as possible for all other positions within the operation.

GSR operates a local training program (Golden Star Skills Training and Employability program) where people from the local community are offered the opportunity to train in work areas to broaden and enhance livelihoods. These programs are complemented by an array of other alternative livelihoods initiatives.

Access to Land, Noise and Blasting Effects

Other community concerns include access to land, and noise and blasting effects. Given the underground operations are on a small scale when compared to former underground operations, and the existing widespread galamsey operations in the area, the cumulative impacts of the operations are expected to have negligible impact. The EIA studies and EIS incorporated predictive modelling for air, noise and blasting impacts, to confirm the required controls for application at the 'combined' operations.

20.4.2 Unauthorized Small Scale Mining

Galamsey is the local name for unauthorized small-scale mining. It is often associated with environmental degradation, safety hazards, and general community and social concerns. The name is something of a misnomer, as the galamsey operators are often well resourced, with excavators, dredges and normal scale processing equipment in use. Galamsey has been widespread in the Prestea and Bondaye communities, but has declined somewhat with the decline in the gold price, and various Government programs to reduce illegal mining.

To assist the Government and community in the regularization of galamsey, GSBPL, in January 2012 held a meeting with representatives of the EPA, District Assembly, Minerals Commission, Traditional Authorities, and small scale miners, to inform stakeholders of GSBPL's preparedness to cede a part of its inactive, but prospective land on the Prestea concession to authorized small scale miners in the catchment through the Minerals Commission.

This program had the aim to provide support for legal small scale mining, assist in providing greater livelihood opportunities, and to therefore reduce illegal mining activities within GSBPL's active mining areas. This program was highly embraced and the process involved was clearly described to the groups concerned. Stakeholders were notified that the exact process for ceding of an area of GSBPL's Concession, would require GSBPL to cede the land back to the government. It is then the role of government to allocate this land. GSBPL does not retain the legal right to select to which parties ceded portions of concessions would be allocated.

On the August 28, 2012, GSBPL was officially notified by the Minerals Commission of the acceptance of GSBPL's proposed ceding of a part of the Prestea Mining Lease for small scale mining and the concomitant division of the Prestea Lease into two separate leases as a result. The Minerals Commission retains the authority in relation to issuance of small-scale mining permits over the area of land that was ceded.

GSR expects that galamsey in the area of the project has little potential to affect the West Reef operations. The main Central shaft and Bondaye shaft sites are well secured. Accesses to other shafts on the Prestea concession have been backfilled with the support of Government and key stakeholders.

GSR has contracts with private security providers, and Memoranda of Understanding with public security agencies that require adherence to the Voluntary Principles on Security and Human Rights (see Section 20.2.2).

20.4.3 Cumulative Impacts to Areas of Historic Liability

Indemnity agreements that exonerate GSBPL from environmental liabilities emanating from historical mining activities are in place, these agreements with the Government of Ghana were signed in 2001 (Section 20.1.5). In recognition of the requirements of the environmental indemnity and for the purposes of due diligence:

- GSBPL has completed an extensive program of baseline assessments to understand the prevailing environmental and social conditions of the project area.
- GSBPL has designed key elements of the PUG West Reef, such as the completely dedicated mine dewatering system, so that it can isolate its underground mining activities from those of the predecessors and avoid posing any cumulative burden to the already heavily impacted systems.

20.4.4 Discharges to the Environment

Discharges from the Process Water Circuit

The historic process water balance for the Bogoso/Prestea operations was positive and as a result GSBPL constructed and operated an 11 ML/day process water treatment plant that produces a permeate (clean water) product, that achieves discharge requirements, which is discharged to the environment.

With the suspension of refractory processing operations in Q3 2015 and the resulting net negative water balance, as well as continued programs of process water treatment and catchment management, the process water inventory has significantly reduced and remaining volumes in inventory are no longer in excess, only representing required raw water inputs.

Discharges from the Mine Workings

The mine water generated in the PUG West Reef mining area will be handled via a dedicated mine dewatering system (Section 16.2 and 16.10.3). This water will be removed via the Bondaye shaft to an appropriate treatment system to remove mine sediments, oil and grease, and trace metals. Treated waters that achieve the EPA Eluent Discharge Guidelines will be released to the Anobaka Stream. The natural stream and creek systems contain seasonal flood flows, as well as water dewatered via the Bondaye shaft from the historic mining operations, and studies have shown that any additional dewatering will not impact flow regimes.

20.4.5 Acid Rock Generation

It has been identified by historic studies that transitional rock and fresh rock in the geological region are potentially acid generating. Mine waters from the historic Prestea underground mines show evidence of acid rock drainage, neutral leaching, as well as saline drainage.

Dedicated studies for the PUG West Reef demonstrate the West Reef rock has limited potential for acid generation and leaching tests have shown that the geochemical impact of mining the West Reef deposit is expected to be low.

Taking the precautionary approach, GSBPL has determined to mitigate for potential impacts by ensuring that where any rock is mined that has the potential for acid generation, it will be backfilled or placed into underground mining voids as backfill or existing mine voids on the Prestea concession.

20.5 Closure Planning and Cost Estimate

An Environmental Permit has been invoiced for the PUG West Reef and a Mining Operating Plan has been obtained (Section 20.1.6). The EIS submitted to the EPA to obtain the Environmental Permit contained a provisional closure plan and cost estimate. The Mining Operating Plan also contains details relating to closure and reclamation.

The rehabilitation and closure of the existing operations (e.g., processing plant, tailings storage facility, pits, waste dumps, and transportation corridor) are covered under the EMP, Reclamation Security Agreement, and associated bond. With each successive expansion or new development, GSBPL develops a conceptual closure plan that is incorporated into the applicable EIS. For the PUG West Reef, a closure cost estimate has been incorporated in the project EIS and is updated through annual updates of the asset retirement obligations.

The closure cost estimate incorporated into this Technical Report is based on the following principles:

- Closure occurs as per the Closure Plan.
- No allowance for scrap value.
- Progressive closure integrated with on-going operations.
- Costs based on a mix of current contractor rates and work being undertaken directly by the operation.

- No provision for ongoing treatment of water. The underground mine will be allowed to flood to the natural level, which is the current closure plan for the Prestea Phase 1 underground operations.
- Expected hand-over of nominated infrastructure and equipment to community and Government.
- No allowance for preliminaries and generals or contingencies.

GSR expects the EPA to request either a reclamation bond for the underground mine or a modification to the existing GSBPL reclamation bond through the Reclamation Security Agreement. The current bond over the combined Bogoso/Prestea concessions of GSBPL is US$8.1 million (Section 20.1.4).

21 Capital and Operating Costs

21.1 Capital Costs

Table 104 presents the capital expenditure schedule for the Prestea Mine.

Sustaining capital consists of the following items:

- Surface – this item is for surface plant and infrastructure maintenance and includes the processing plant, Bogoso site and tailings impoundments. The detail of this estimate is derived from the operations budgeting and LOM planning process.
- Electrical – extension of the electrical backbone into new mining areas on 21 and 27L. This estimate is derived from a 2016 design and budget for the Alimak mining project. The original estimate has been updated in the 2018 Budget to take into account the current situation.
- Mining equipment – purchase of haul trucks for 27 to 24L haul. Derived from 2016 internal feasibility study
- Ventilation raises – 24 to 11L system in 2018/19 and 27 to 24L in 2020. This estimate is derived from a 2016 design and budget for the Alimak mining project. The original estimate has been updated in the 2018 Budget to take into account the current situation.
- Miscellaneous – maintaining a level of annual capital sufficient to replace equipment and provide services as required.

Some delayed project capital items are still outstanding from the construction phase:

- Plant modifications – due to the extension of the open pit mining, it was not necessary to complete the plant modifications in 2017 as originally planned. This expense is delayed capital, not an over-run. Estimate based on 2018 Budget.
- Shafts and winders – completion of rehabilitation works. Estimate based on 2018 Budget.

Table 104 Capital cost schedule

GSBPL Capital Expenditure				TOTAL	2018	2019	2020	2021	2022
Sustaining Capital	Surface		$M	**8.2**	2.2	2.0	1.5	1.5	1.0
	Underground	Electrical	$M	**1.2**	0.8	0.4	-	-	-
		Mechanical services	$M	**2.0**	0.9	1.1	-	-	-
		Mining equipment	$M	**1.2**	-	1.2	-	-	-
		Vent raises	$M	**4.3**	2.2	0.6	1.5	-	-
		Miscellaneous	$M	**10.2**	0.7	1.5	3.0	3.0	2.0
	Total sustaining capital		*$M*	***27.1***	*6.8*	*6.8*	*6.0*	*4.5*	*3.0*
Development Capital	Plant modification		$M	**1.2**	1.2	-	-	-	-
	Shafts and winders		$M	**0.8**	0.8	-	-	-	-
	Total development capital		*$M*	***2.0***	*2.0*	*-*	*-*	*-*	*-*
Total Capital			$M	**29.1**	8.8	6.8	6.0	4.5	3.0

21.2 Operating Costs

Table 105 shows the annual total operating costs for the Prestea Mine.

Underground mining costs are estimated from a zero-based approach including manpower, consumables, maintenance, electrical, Alimak training contract and services.

Open pit mining costs are estimated based on historical experience. This pit mining will be completed in Q1 2018.

Processing costs in the first half of 2018 are based on the oxide plant running at full capacity processing open pit oxides and underground ore. In the middle of 2018 the plant will reduce throughput to 650tpd of underground material only and the process unit cost will rise.

G&A costs are estimated based on historic performance in addition to plans to optimize our cost structure over the next 12 months.

Closure costs are estimated based on our total rehabilitation requirements at Prestea and Bogoso sites.

Table 105 Operating costs

GSBPL Operating Cost					TOTAL	2018	2019	2020	2021	2022	2023	2024	2025
Costs	UG mining	111		$M	**129**	30	29	23	23	20	4	-	-
	OP mining	4.10		$M	**4**	4	-	-	-	-	-	-	-
	Processing	25	75	$M	**101**	26	17	18	18	18	5	-	-
	G&A	11	45	$M	**58**	13	12	11	10	10	2	-	-
	Cash operating cost			*$M*	***292***	*73*	*58*	*52*	*51*	*48*	*11*	-	-
	GOG royalty	5.0%		$M	**30**	6	5	6	6	5	1	-	-
	Closure			$M	**41.8**	4.1	3.9	7.1	7.1	5.6	5.4	4.4	4
	Total operating cost			*$M*	***364***	*83*	*67*	*65*	*64*	*59*	*18*	*4*	*4*

22 Economic Analysis

22.1 Inputs and Assumptions

Inputs to the cash flow model include:

- A 5 year production period with a nominal production rate of 650 t/d
- Total LoM production of 1,920 kt at an average grade of 8.1 g/t containing 497 thousand ounces of gold
- Underground mining production of 1,165 kt at an average grade of 12.35 g/t containing 463 thousand ounces of gold
- Remaining open pit mining production and stockpiles containing 35 thousand ounces of gold
- A metallurgical process recovery of 94% yielding 468 thousand recovered ounces
- Revenue based on a gold price of US$1,300/ounce
- A mineral royalty of 5% of gross revenue
- A stream payment that is the equivalent of an 8.4% royalty of gross revenue
- Total development capital cost estimate of $2.0 million
- Total sustaining capital cost estimate of $27.1 million
- All expenditures on the project prior to January 2018 are considered as sunk costs
- Project economics are evaluated from January 2018

22.2 Taxes and Royalties

Golden Star Resources Ltd (GSR) holds a 90% interest in the Prestea underground mine with the government of Ghana holding a 10% ownership interest. The Government of Ghana receives a gross revenue Mineral Royalty of 5% on all gold production.

The corporate tax rate on mining companies in Ghana is 35%. GSR has an opening assessed tax loss of US$593 million which will apply to the economic assessment of the project. This figure represents the full losses incurred at both Prestea and Bogoso operations. This results in no corporation tax being payable for the PUG West Reef project.

22.3 Cash Flow Model and Project Economic Results

The Prestea West Reef annual economic model is shown in Table 106.

The following pre-tax economic indicators were calculated:

- Free cash flow $164 million
- NPV at 5% discount rate $144 million
- LOM Cash operating cost $624/oz
- LOM Mine-site AISC $754/oz

Although these results are quoted on a pre-tax basis, GSR will not be subject to the impact of the Ghana 35% mining corporate tax rate due to previous tax losses.

Table 106 Economic Model

GSBPL Economic Model					TOTAL	2018	2019	2020	2021	2022	2023	2024	2025
Mining	Underground	Ore		T	**1,165,196**	161,569	229,570	235,042	239,684	236,962	62,369	-	-
				g/t	**12.35**	13.30	12.09	13.99	12.24	11.10	9.86	-	-
	Open Pit	Ore		T	**207,345**	207,345	-	-	-	-	-	-	-
				g/t	**2.02**	2.02	-	-	-	-	-	-	-
		Ore+waste		T	**518,363**	518,363	-	-	-	-	-	-	-
	Stockpile			T	**547,416**	547,416	-	-	-	-	-	-	-
				g/t	**1.21**	1.21	-	-	-	-	-	-	-
Processing				T	**1,919,957**	916,330	229,570	235,042	239,684	236,962	62,369	-	-
				g/t	**8.06**	3.52	12.09	13.99	12.24	11.10	9.86	-	-
Contained gold				oz	**497,360**	103,825	89,226	105,708	94,287	84,552	19,762	-	-
Recovered gold		94%		oz	**467,518**	97,595	83,872	99,366	88,630	79,479	18,576	-	-
Gross revenue		1300		$M	**608**	127	109	129	115	103	24	-	-
RG stream		8.4%		$M	**51**	11	9	11	10	9	2	-	-
Net revenue				$M	**557**	116	100	118	106	95	22	-	-
Costs	UG mining	111		$M	**129**	30	29	23	23	20	4	-	-
	OP mining	4.10		$M	**4**	4	-	-	-	-	-	-	-
	Processing	25	75	$M	**101**	26	17	18	18	18	5	-	-
	G&A	11	45	$M	**58**	13	12	11	10	10	2	-	-
	Cash operating cost			*$M*	***292***	*73*	*58*	*52*	*51*	*48*	*11*	*-*	*-*
	GOG royalty	5.0%		$M	**30**	6	5	6	6	5	1	-	-
	Closure			$M	**41.8**	4.1	3.9	7.1	7.1	5.6	5.4	4.4	4
	Total operating cost			*$M*	***364***	*83*	*67*	*65*	*64*	*59*	*18*	*4*	*4*
Operating cash flow				$M	**193**	33	33	53	42	36	4	(4)	(4)
Sustaining Capital	Surface			$M	**8.2**	2.2	2.0	1.5	1.5	1.0	-	-	-
	Underground	Electrical		$M	**1.2**	0.8	0.4	-	-	-	-	-	-
		Mechanical services		$M	**2.0**	0.9	1.1	-	-	-	-	-	-
		Mining equipment		$M	**1.2**	-	1.2	-	-	-	-	-	-
		Vent raises		$M	**4.3**	2.2	0.6	1.5	-	-	-	-	-
		Miscellaneous		$M	**10.2**	0.7	1.5	3.0	3.0	2.0	-	-	-
	Total sustaining capital			*$M*	***27.1***	*6.8*	*6.8*	*6.0*	*4.5*	*3.0*	*-*	*-*	*-*
Development Capital	Plant modification			$M	**1.2**	1.2	-	-	-	-	-	-	-
	Shafts and winders			$M	**0.8**	0.8	-	-	-	-	-	-	-
	Total development capital			*$M*	***2.0***	*2.0*	*-*	*-*	*-*	*-*	*-*	*-*	*-*
Total Capital				$M	**29.1**	8.8	6.8	6.0	4.5	3.0	-	-	-
Free Cash Flow				$M	**164**	**24**	**26**	**47**	**37**	**33**	**4**	**(4)**	**(4)**

Financial Metrics	NPV	0%	**164**							
		5%	**144**							
		10%	**128**							
	Cash Operating Cost		**624**	746	686	519	573	606	592	-	-
	Minesite AISC		**754**	885	836	648	693	714	680	-	-
	Minesite Total cost		**759**	906	836	648	693	714	680	-	-

22.4 Sensitivity Analysis

The base case results are based on:

- Gold price $1,300/ounce
- Plant feed gold grade 12.4 g/t gold (note open pit material excluded)
- Total capital cost $29.1 million
- LOM site operating cost $364 million (includes $42M closure costs)

A sensitivity analysis has been prepared varying these four inputs. Table 107 shows the impact of varying input values on the base case pre-tax economic indicators – $NPV_{5\%}$ in millions of dollars.

Figure 100 presents these sensitivities in graphical format.

Of these parameters, the project economics are most sensitive to gold price and gold grade and least sensitive to capital expenditure changes.

Table 107 NPV5% Sensitivity

Variable	$NPV_{5\%}$ Sensitivity ($M)						
	-30%	-20%	-10%	Base	10%	20%	30%
Capex	136	139	142	144	147	149	152
Opex	66	92	118		170	196	222
Gold Price or grade	16	59	101		187	230	273

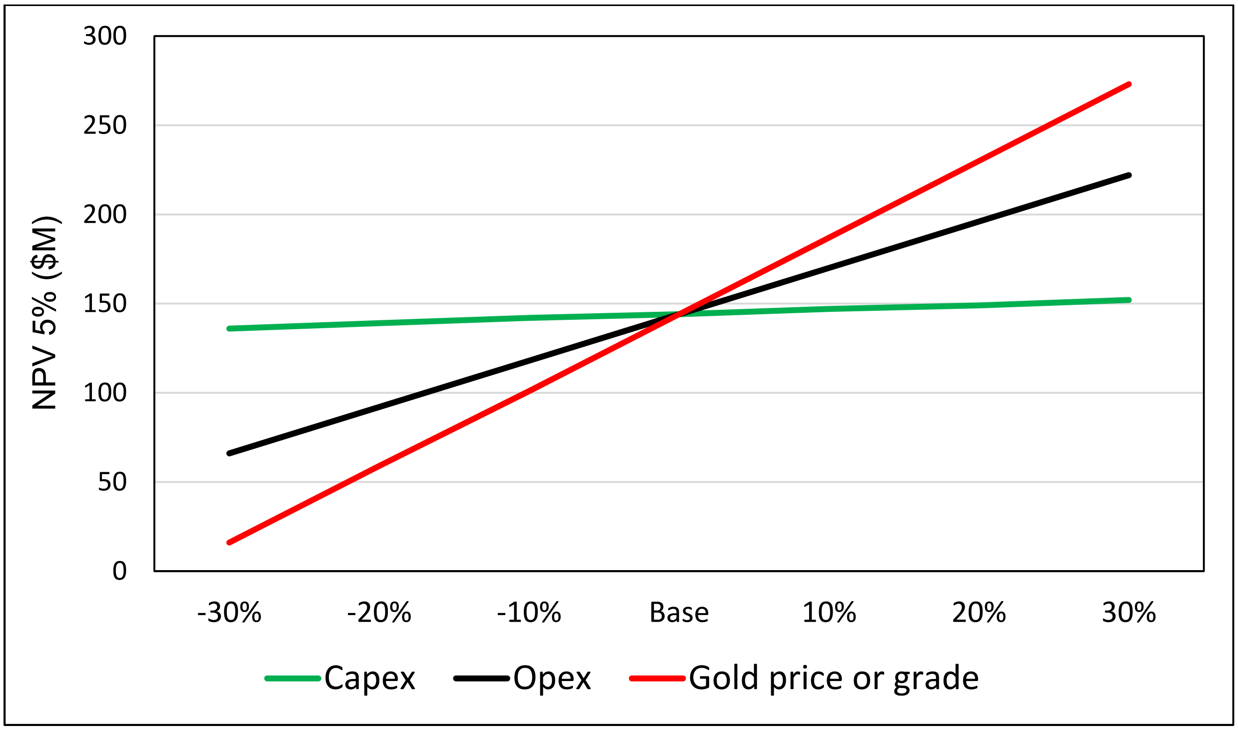


Figure 100 NPV5% Sensitivity

23 Adjacent Properties

There is no relevant data regarding adjacent properties.

24 Other Relevant Data and Information

There is no other relevant data available.

25 Conclusions and Recommendations

Geology

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)* by S.Mitchel Wasel, MAusIMM(CP), of GSR, qualified person (not independent) as this term is defined in NI 43-101. 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.

Further definition drilling and sill development could convert some of the existing underground Inferred mineral resources to Indicated category. This will be a benefit for extending the mine life.

The underground deposit remains open for possible expansion at depth below the current planned mine bottom, especially following the high grade down plunge trend, which could increase the project mineral resource base.

The Main Reef at Prestea Underground could contain opportunities to expand and/or extend underground production in the future.

Surface drilling to identify new open pit and underground opportunities should be continued.

Mining

The Alimak stoping process is currently in early stages with one stope complete, one mining and six raises complete or in development. GSR is evaluating options to optimize the underground ore production through drilling, blasting and material movement improvements.

Metallurgy and Mineral Processing

The metallurgical testwork programs conducted on ore from the Prestea West Reef have all indicated that the ore is free milling, with a relatively high gravity-recoverable component, and demonstrating a degree, although relatively low, of preg-robbing behaviour.

This is borne out by the experience over the past six months with processing the high grade underground ore.

By Q3 2018 the process plant will be operating on only the high-grade underground material with a throughput of 650 tpd.

Infrastructure

The new Prestea Underground Mine infrastructure upgrade is 90% complete with only some final winder items and the new pump system outstanding. All the upgraded areas have worked to specification.

The (original) base case tailings management strategy involves construction of an additional 1.2 m raise around Cell 1/2 for storage of 656 kt of tailings produced at the Bogoso low throughput plant. There is potential to store the remaining 385 kt in other cells within the TSF2 impoundment or potentially in the TSF1 location (pending EPA approval).

Environmental and Social Management

The PUG West Reef involves new underground development and is generally isolated from previous underground workings. Surface facilities include expanded electrical substation, and use of existing ventilation shaft collars. Surface activity will include water treatment if necessary and ore transportation by truck. The project extend 9 km along strike in a north-south direction beneath the Prestea township (including related existing mine development).

The primary environmental approvals for the PUG West Reef are the EPA Environmental Permit and Mine Operating Plan with the Minerals Commission. The Mine Operating Plan has been submitted and approved and the EPA Environmental Permit has been invoiced (January 2018) and is pending permit issuance.

GSR has an environmental and social management system developed along the lines of an ISO 14001 management system. The management is carried out by the Environment Services and the Community Relations and Social Responsibility Departments. GSR has also established a series of Community Mine Consultative Committees for on-going engagement of local communities.

Dedicated studies for the PUG West Reef demonstrate low potential for acid drainage generation and overall the geochemical impact of mining the West Reef stopes is expected to be low. Mine water leachate is predicted to achieve EPA discharge criteria.

Capital and Operating Costs

Total capital of $29.1 million is comprised of $2.0 million project capital and $27.1 million sustaining capital.

Mine operating costs include:

- $111/t underground mining operating cost
- $4.10/t open pit mining costs only in Q1 2018
- $25/t processing costs in H1 2018 when open pit material is being processed
- $75/t processing costs with underground feed of 650 tpd
- $45/t general and administration cost, covering both the Prestea and Bogoso sites

Life of mine cash operating cost was estimated at $624 per ounce and all-in sustaining cost at $754 per ounce during the production period.

Economics

The mine has been evaluated on a discounted cash flow basis. The cash flow analysis was prepared on a constant 2018 US dollar basis. No inflation or escalation of revenue or costs has been incorporated. With an applicable $593 million tax loss pool the pre-tax and post-tax present value of the net cash flow with a 5% discount rate ($NPV_{5\%}$) is $144 million using a base case gold price of $1,300/oz.

The $NPV_{5\%}$ is most sensitive to changes in gold price and plant head grade. The project is least sensitive to changes in capital costs.

Risks

The most significant risks to the operation are as follows:

- **Unplanned dilution resulting in overall dilution greater than the 33% currently planned.** Moderate increases in dilution, up to around 50%, have a low impact on the mine economics, particularly if the production rate can be increased above the planned 650 tpd.

- **Not achieving planned stoping productivities.** If Alimak raise development, longhole drilling, blasting and mucking productivities cannot be achieved, the economic value of the mine will reduce. Although the overall reserve would remain intact, the slower production rates would lead to reduced annual cash flow.

- **Pump failures and mine flooding.** This situation occurred in 2017 when the pressure of the mine startup caused multiple failures in the pumping system. Working levels were not affected, however, water levels in the shaft stopped hoisting for a two month period. Significant improvements have been made to the pumping system and further upgrades are planned. This risk is considered to be low but with high potential impact on economics.

- **Mechanical failure in shaft or winder system.** This potential is always a risk in shaft access underground mines. GSR has invested heavily in upgrading shaft and winder infrastructure and safety systems. This risk is considered to be low but with high potential impact on economics.

- **Illegal miner activity affecting underground mine operations.** Over the past three years there have been a couple of instances of illegal miners accessing the Prestea underground through old disused shafts and workings. However, these instances have decreased as activity at Prestea has ramped up.

- **Adverse changes to regulatory conditions**. The economic assessment is based on the current regulatory environment in Ghana. Changes to the Ghana tax regime; environmental regulations; mining regulations; and treatment/interpretation of the existing agreements between the Company and the Government may have a significant adverse effect on the economics of the Bogoso/Prestea operation.

Recommended Work Programs

- **Exploration** – exploration drilling continues underground to define additional resources and to upgrade Inferred resources to Indicated status. The drilling program will include underground drilling in the West and Main Reef areas between 17 and 24L and surface drilling of various targets on the Prestea and Bogoso concessions. The cost of the drilling program in 2018 is expected to be $4.2 million

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27 Date and Signatures

The effective date of this technical report is December 31, 2017.

[“Signed and sealed”]

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Dated March 29, 2018