NI 43-101 TECHNICAL REPORT ON A PRELIMINARY ECONOMIC ASSESSMENT OF THE WASSA OPEN PIT MINE AND UNDERGROUND PROJECT IN GHANA

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EXECUTIVE SUMMARY
NI 43-101 TECHNICAL REPORT ON A PRELIMINARY ECONOMIC ASSESSMENT OF THE WASSA OPEN PIT MINE AND UNDERGROUND PROJECT IN GHANA

EXECUTIVE SUMMARY (ITEM 1)

INTRODUCTION

This Technical Report is a Preliminary Economic Assessment ("PEA") of the combined Wassa open pit and underground gold project (the "Wassa Project") which has been prepared by SRK Consulting (UK) Limited ("SRK") for and on behalf of Golden Star Resources Ltd. ("GSR"). This report also incorporates an update of the Wassa Mineral Resources as at 31 July 2014.

GSR currently holds a 90% interest in the subsidiary company, Golden Star (Wassa) Limited ("GSWL") who operate the Wassa open pit gold mine located in the western region of Ghana, to the northeast of Tarkwa.

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").

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

In recent years the Wassa processing plant has received open pit feed from the Wassa Main deposit and a number of satellite pits (Benso and Hwini Butre) at a rate of 2.1 to 2.6 Mtpa. This PEA considers a combined open pit and underground feed sourced from the Wassa Main deposit at a rate of 2.6 Mtpa for the initial 5 years followed by a decreased rate of 1 Mtpa for the remaining underground only feed.

GSWL holds various exploration properties and mining leases, which incorporate the following key:

- Wassa mining lease: Wassa Main is an operating open pit gold mine comprising the following mineralization domains: F Shoot, 419, B Shoot, 242, Starter, South-East, Mid-East and Dead Man's Hill ("DMH").
- Benso mining lease: comprising the Subriso East ("SE"), Subriso West ("SW"), G-Zone and I-Zone deposits.
- Hwini Butre mining lease: comprising the Father Brown, Adoikrom and Dabokrom deposits, with only Father Brown having remaining open pit reserves.
- Chichiwelli exploration property: comprising two mineralized zones, Chichiwelli West ("West Domain") and Chichiwelli East ("East Domain").

- Manso exploration property, located adjacent to Benso, to the east.

The properties and leases are spread out along a line trending approximately 80 km southwest of the Wassa mine complex.

All monetary values shown in the report are US Dollars (“US$”) unless otherwise stated.

MINERAL RESOURCES

HISTORY LOCATION AND OWNERSHIP

The Wassa area has witnessed several periods of local small scale and colonial mining activity from the beginning of the 20th century. Mining of quartz veins and gold bearing structures are evident from the numerous pits and shafts covering the Wassa lease area.

The Wassa Mine was originally developed as a 3 Mtpa open pit heap leach operation with a forecasted Life of Mine (“LoM”) gold production of approximately 100,000 ounces per annum. The first material from the pit was mined in October 1998. After approximately one year of production, it became evident that the predicted heap leach gold recovery of 85% could not be achieved, mainly due to the high clay content of the resource and poor solution management.

In 2001, the project was put up for sale and GSR acquired the Wassa assets. As part of a final due diligence on the resources, GSR undertook a drilling program between December 2001 and April 2002. Upon completion of the acquisition of Wassa Mine by GSR, further exploration programs were undertaken. Both these exploration programs formed part of a Feasibility Study (“FS”) that was completed in July 2003, which demonstrated the economic viability of reopening and expanding the existing open pits, and processing the material through a conventional Carbon-in-Leach (“CIL”) circuit. The Wassa Mine has been operating as a conventional CIL milling operation since April 2005.

GEOLOGY AND MINERALIZATION

The Wassa property lies within the southern portion of the Ashanti Greenstone Belt along the eastern margin within a volcano-sedimentary assemblage located close to the Tarkwaian basin contact. The eastern contact between the Tarkwaian basin and the volcano-sedimentary rocks of the Sefwi group is faulted, but the fault is discrete as opposed to the western contact of the Ashanti belt where the Ashanti fault zone can be several hundred meters wide.

The Wassa lithological sequence is characterized by lithologies belonging to the Sefwi Group and consisting of intercalated meta-mafic volcanic and meta-diorite dykes with altered meta-mafic volcanic and meta-sediments which are locally characterized as magnetite rich, Banded Iron Formation (“BIF”) like horizons (Bourassa, 2003). The sequence is characterized by the presence of multiple ankerite-quartz veins which are sub-parallel to the main penetrative foliation. The lithological sequence is also characterized by Eoeburnean felsic porphyry intrusions on the south-western flank of the Wassa mine fold.

The Wassa mineralization is subdivided into a number of domains, namely; F Shoot, B Shoot, 242, South East, Starter, 419, Mid East and Dead Man’s Hill. Each of these represents discontinuous segments of the main mineralized system. The SAK deposits are located approximately 2 km to the southwest of the Wassa Main deposit on the northern end of a well-defined mineralized trend parallel to the Wassa Main trend. The mineralization is hosted in highly altered multi-phased greenstone-hosted quartz-carbonate veins interlaced with sedimentary pelitic units. The SAK mineralization is subdivided into a number of domains as well, SAK 1, 2 and 3.

Mineralization within the Wassa Mine is structurally controlled and related to vein densities and sulphide contents. In detail, the mineralization generally consists of broadly tabular zones containing dismembered and folded ribbon-like bodies of narrow quartz vein material. Three vein generations have been distinguished on the basis of structural evidence, vein mineralogy, textures and associated gold grades.

EXPLORATION AND DATA MANAGEMENT

Exploration drilling commenced in February 1994, and by March 1997, a total of 58,709 m of reverse circulation and diamond drilling had been completed.

In March 2002, GSR started an exploration program as part of a due diligence exercise following the ratification of a confidentiality agreement with the creditor of Satellite Goldfields, the mining lease was purchased later in the year. The exploration program consisted mainly of pit mapping and drilling below the pits to test the continuity of mineralization at depth. Exploration drilling resumed in November 2002 under GSR with the aim to increase mineral reserves and resources for the feasibility study, which was completed in 2003.

Simultaneously to the resource drilling program that targeted resource increases in the pit areas, GSR also undertook grass roots exploration along two previously identified mineralized trends. The 419 area was delineated south of the main pits and the South-Akyempim anomaly, a soil target that had never been previously drilled, and was discovered west of the main pits. Deep auger campaigns were also undertaken in the Subri forest reserve, which is located in the southern portion of the Wassa Mining lease.

In March and April 2004, a high resolution, aerial geophysical survey was carried out over the Wassa Mining Lease and surrounding Prospecting and Reconnaissance Licenses. The surveys consisted of 9,085 line kilometres covering a total area of 450 km^2. Flight lines were spaced at varying distances between 50 to 100 m depending on the survey type. The geophysical surveys identified several anomalies with targets being prioritized on the basis of supporting geochemical and geological evidences.

Drilling is carried out by a combination of diamond drilling ("DD"), reverse circulation ("RC") and reverse air blast ("RAB") techniques. In general the RAB method, which has a depth capability of 30 m, is used at early stages for follow up to soil geochemical sampling and during production for testing contacts and extensions of mineralization. To further test the prospective structures and anomalies defined from soil geochemistry and RAB drilling results RC drilling is used. RC drilling is typically carried out along drill lines spaced between 25 and 50 m apart with maximum drilled depths of 100 to 125 meters, depending on the ground water table. The DD method is used to provide more detailed geological data 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.

Sampling is typically carried out along the entire drilled length. For RC drilling, samples are collected every meter and then combined into 3m composites. Should any 3m composite samples return a significant gold grade assay, the individual 1m 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 1m lengths. The core is cut according to mineralization, alteration or lithology and is split into two equal parts along a median to the foliation plane. 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.

Sample assays are then performed at either SGS Laboratories in Tarkwa (SGS) or Transworld (now Intertek) Laboratories ("TWL") which is also based in Tarkwa. GSR has used both laboratories and regularly submits quality control samples to each for testing purposes. Both laboratories are currently in the process of accreditation for international certification for testing and analysis. Specific gravity (“SG”) determinations were carried out by GSR at the core facility using a water immersion method.

Quality control measures are typically set in place to ensure the reliability and trustworthiness of exploration data, and to ensure that it is of sufficient quality for inclusion in the subsequent Mineral Resource estimates. Quality control measures include written field procedures and independent verifications of aspects such as drilling, surveying, sampling and assaying, data management and database integrity.

The field procedures implemented by GSR are comprehensive and cover all aspects of the data collection process. At Wassa, each task is conducted by appropriately qualified personnel under the direct supervision of a qualified geologist. The measures implemented by GSR are considered to be consistent with industry best practice.

MINERAL RESOURCE ESTIMATE

The Mineral Resource Statement presented in Table ES 1 represents an estimate for the Wassa Main deposit as at 31 July 2014. Other satellite deposits which are part of the Wassa Mine were not included in this report.

Table ES 1: GSWL Mineral Resource Statement, 31 July 2014

Source	MEASURED			INDICATED			INFERRED		
	Quantity	Grade	Metal	Quantity	Grade	Metal	Quantity	Grade	Metal
	kt	g/t Au	koz Au	kt	g/t Au	koz Au	kt	g/t Au	koz Au
Wassa Open Pit	-	-	-	25,582	1.41	1,160	237	1.56	12
Wassa Underground	-	-	-	10,116	4.27	1,389	8,841	3.95	1,122
Total	**-**	**-**	**-**	**35,698**	**2.22**	**2,549**	**9,078**	**3.89**	**1,134**

The GSR exploration team was responsible for the modelling portion of the resource estimate exercise which included all topographic surfaces, weathering surfaces, geological and grade wireframes. SRK Consulting (Canada) was commissioned to construct a mineral resource model with estimated gold grades for the Wassa Main deposit. The mineral resource classification and statement was prepared by GSR under the supervision of Mitch Wasel, a Qualified Person pursuant to NI 43-101.

The Wassa database comprises four individual drillhole databases, namely: the GSR Wassa exploration database which contains exploration drilling conducted by GSR since 2002; the exploration database which contains historical exploration drill holes from the previous operator Satellite Goldfields; the Satellite Goldfields grade control database; and the GSR grade control database. The Satellite Goldfields grade control database was not included in the Mineral Resource estimate as the samples are considered to be of not a sufficient quality for inclusion.

A two-step approach was developed for modelling the auriferous zones at Wassa: wide low grade envelopes characterized by weak alteration were modelled around high grade zones with strong veining and sulphide content. A total of twenty-seven low grade wireframes were modelled by GSR surrounding four high-grade wireframes. SRK undertook an independent exercise to confirm the continuity of the high grade zones as modelled by GSR. Structural geology assistance was also provided by SRK to develop the high grade wireframes.

A 3D block model including rock type, gold, percent, density and class was built in Gems by GSR. The block size was set at 10 x 10 x 3 m in the northing, easting and elevation directions, respectively along the mine grid.

GSR has classified and reported the Wassa gold Mineral Resource in accordance with the NI 43-101 guidelines and the CIM standards on Mineral Resources.

The open pit Mineral Resource estimate is based on appropriate cut-off grades for the oxide and fresh material, and reported within a conceptual Whittle shell. Pit optimisation using industry standard software has been undertaken on the Mineral Resource models using appropriate slope angles, modifying factors (mining recovery and dilution), process recovery factors, costs and a long term gold price of US$ 1,400/oz (t.oz).

The underground portion of the Mineral Resource estimate is based on appropriate cut-off grades and reported below the conceptual Whittle shell.

The Mineral Resource models have been depleted using appropriate topographic surveys, to reflect mining until 31 July 2014.

Geological modelling of the mineralization was undertaken by GSR, using a cut-off grade of approximately 0.2 g/t Au for the low grade envelopes and 1.0 g/t Au for the high grade wireframes, assays less than 1 g/t Au were often included within the mineralized wireframes in order to model continuity both down dip and along strike. The Mineral Resource estimates are derived from a combination of diamond and reverse circulation drilling techniques, supported by an industry best practice QAQC programme. Drilling is typically carried out on sections spaced at 25 m.

The Mineral Resources were estimated using a block model with block sizes which typically reflect half the drillhole spacing within the Wassa Main deposit. The composite grades were capped for domains deemed necessary after statistical analysis. Local Ordinary Kriging was used to estimate the block grades.

The basis of the Mineral Resource classification included confidence in the geological continuity of the mineralized structures, the quality and quantity of the exploration data supporting the estimates, and the geostatistical confidence in the tonnage and grade estimates. Three-dimensional solids were modelled reflecting areas with the highest confidence, which were classified as Indicated Mineral Resources. All figures are rounded to reflect the relative accuracy of the estimate.

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

NOTE ON MINERAL RESERVES

No Mineral Reserves have been estimated for the purposes of the PEA for the Wassa combined open pit and underground project in Ghana.

Recent Mineral Reserve estimates have been prepared by:

- SRK Consulting (UK) Limited – NI 43-101 Technical Report with effective December 31, 2012, filed 21 March 2013; and
- GSR – Year-end reserve estimate effective December 31, 2013.

Both of the above recent reserve estimates have been related to open pit operations, while this PEA considers a combined open pit and underground operation.

MINE DESIGN AND SCHEDULE

The mine plan reflects continuing open pit production; underground production commencing in 2016; and combined open pit and underground production until 2020. In 2021, the underground is the only source of Run of Mine (“RoM”) feed at a production rate of 1 Mtpa until 2025. Total material mined amounts to 19.2 Mt at an average grade of 2.74 g/t Au estimated to produce some 1,574 koz gold. It should be noted that the underground grades are significantly higher than the open pit.

A conventional approach to open pit mining is currently being used at the Wassa mine employing excavators and trucks which are considered typical for this type and style of gold mineralization. The same approach to equipment and open pit mining are used for the purposes of the Wassa PEA. Drilling and blasting of rock is conducted over bench heights of 5 or 6 m and explosives are delivered to the hole by the manufacturer. Oxide or weathered material is generally only required to be lightly blasted or in some areas can be excavated without blasting. Hydraulic excavators are used to achieve good selectivity, and in conjunction with good blasting practice, mine to a 2.5 or 3.0 m flitch height. Broken rock is loaded to 95 t capacity off-highway haul trucks to a central stockpile or to the waste dump.

The underground design utilises a single decline access from the base of the MSN pit which reduces the access to the mineralization by 106 vertical metres from the surface topography. The selected mining method is sub-level open stoping utilizing a cemented rock backfill (“CRF”) to maintain support for mining the secondary stopes and increasing the overall extraction. A staged approach to ventilation is taken with raises to surface completed in line with the development and production schedule.

Mining is undertaken using trackless, diesel powered equipment including twin boom jumbos for development and longhole drills for production drilling. The approach to materials handling to surface is through a combination of 17 t capacity loaders and 40 t capacity trucks.

MINERAL PROCESSING

Gold recovery is achieved at Wassa through the use of conventional CIL technology, although the plant itself contains a few atypical features due to its history and development. The CIL plant has a nominal design capacity of 3.5 Mtpa that was historically achieved using a feed blend of 45% Fresh and 25% Oxide material and 30% reclaimed spent heap leach material. With the transition to largely fresh material feed, and the need to grind this material finer in order to maximise the gold recovery, the plant currently operates at a throughput rate of 2.7 Mtpa and has been operating on 100% Fresh rock since July 2014.

The historically-reported gold recoveries correspond very well with those predicted from the original metallurgical testwork. The gravity circuit contributes significantly to the total gold production; this is also consistent with the results of the original laboratory testwork.

The proposed Wassa underground operation will provide of the order of 2,500 tpd of high grade (4 to 6 g/t Au) feed to the mill. While the planned comminution circuit improvements are expected to provide scope to increase the plant capacity to 3.0 Mtpa with 100% fresh RoM feed, the projected plant feed rates shown are lower (not exceeding 2.7 Mtpa) in order to take into consideration the expected increase in mineralized material hardness with increasing depth.

The forecast gold production figures are lower than the historical production achieved to date and so the gold recovery section of the plant (i.e. leaching, adsorption, elution and electrowinning) will be capable of achieving the forecast levels of gold production.

PROJECT INFRASTRUCTURE - TAILINGS STORAGE

There are two tailings facilities that will accommodate the anticipated tailings production with the existing facility referred to as TSF1 and a new facility referred to as TSF2. SRK’s assessment of the geotechnical investigation and embankment stability and seepage analyses attests that they are relevant to the size and scope of the project and demonstrate satisfactory results.

The water management at the Wassa site has been such that discharges to the receiving environment from the tailings storage facility have not been required since 2010. The Wassa operation has an approved detoxification plant to treat elevated CN concentrations that is available for the treatment of water should a discharge be required. However, the water balance model for the current configuration of the site indicates that under normal conditions discharges should not be required. The current detoxification plant will be upgraded with the construction of the new tailings storage facility.

ENVIRONMENTAL AND SOCIAL FACTORS

The GSWL operational area is within the moist tropical rainforest area of the Western Region of Ghana. The mean annual rainfall is in the order of 1,750 mm. There are two rainy seasons, a major rainy season from April to June and then a minor rainy season in October and November. Afternoon thunderstorms are a frequent occurrence in the rainy seasons.

The Wassa Mine, and its associated processing plant and tailings storage facility, is in a rural area and there are no major urban settlements within 50 km by road. The villages of Akyempim, Akyempim New Site (formally Akosombo, which was resettled by the company), Kubekro, and Old Togbekrom (now resettled at Ateiku) are the closest to the mine. The total population of these communities is about 3,000.

An Environmental Certificate has been issued to GSWL and GSWL's Environmental Management Plan ("EMP") for 2010 to 2013 has been approved by the EPA. An EMP to cover the subsequent 3 years was submitted to the EPA within the required time. Comments were received from the EPA and the revised EMP submitted to the EPA in June 2014. It has yet to be approved although a recent audit by the EPA noted that the mine was in compliance for this aspect. The EMP is for the overall Wassa Project – covering the Wassa, Hwini Butre, and Benso Mines and all associated infrastructure, including the Hwini Butre Benso Access Road.

GSWL has received approval for the construction of TSF2. The environmental permit has been issued with the caveat that the facility be lined. Therefore, the facility is being re-designed to incorporate the change and the revised design drawings will then be submitted to the regulatory authorities. To make way for the TSF2 construction, the resettlement of the Togbekrom and surrounding hamlets was required. This has been completed in accordance with the approved Resettlement Action Plan ("RAP") and the people are now in their new accommodation adjacent to the community of Ateiku.

Given the ongoing discussions with the relevant authorities (primarily the EPA and also the Minerals Commission, the Forestry Commission and the Water Resources Commission), SRK does not believe any of the outstanding environmental authorisations noted above represents a material risk to the ongoing GSWL operations.

A preliminary closure plan and cost estimate have been prepared and updated as part of the recently-submitted EMP. The focus of closure planning is the removal of unwanted infrastructure and buildings, stabilization (and in some cases, backfilling) of pits, land-forming and revegetation of waste dumps, management of Acid Rock Drainage ("ARD") potential, removal of access routes, and the stabilization and revegetation of tailings storage facilities.

As required by its permitting conditions, a Reclamation Security Agreement was signed between the company and the EPA in 2005 and GSWL bonded US$3.0 million to cover future reclamation obligations at Wassa, comprised of US$0.15 million in cash and a US$2.85 million letter of credit. All bonds are up to date and GSWL is working with the EPA to meet the bonding requirements that were included in the Environmental Certificate issued in April 2011.

ECONOMIC ANALYSIS

SRK has derived an independent Technical Economic Model (“TEM”) for the PEA using the key assumptions as follows:

- All cost and revenues are in 2015 real terms;
- The start date of the valuation is January 2015;
- A base case discount rate of 5% has been applied for Net Present Value calculations;
- Gold price of US$1,300/oz;
- Royalty of 5% of revenue applied;
- Corporate income tax rate is 35%;
- SRK has relied on the Company’s advice that implementation of a Windfall Profits Tax (“WPT”) has not yet been ratified by the Ghanaian government, therefore WPT has not been provided for in the TEM; and,
- Capital investment is depreciated on an annual fixed percentage basis as per the fiscal regime of Ghana. It has been assumed that all capital items have been fully depreciated and at the end of the mine life there is no terminal value to consider.

The summary cash flow and unit cost results of the TEM are provided below in Table ES 2 and sensitivity analysis is shown in Figure ES 1.

Based on the results of the Discounted Cash Flow analysis the Wassa UG Project has a positive cash flow with a NPV of US$350M at a 5% discount rate.

With the on-going open pit operation contributing significantly to initial cash flows as the underground operations are established the project reflects a high IRR of 128.5%. The contribution from the open pit also results in a rapid payback being realized in year 3 of the Wassa Project.

The Project is most sensitive to sales price/revenue with a 32% reduction resulting in breakeven. Sensitivity to operating costs is low with a greater than 56% increase required for breakeven. The operation is least sensitive to changes in capital expenditure with a substantial increase required to reach breakeven.

Table ES 2: Summary of Cash Flow and Unit Costs

Description	Units	Value
Revenue	(US$M)	2,046
Operating Costs	(US$M)	(1,076)
Royalty	(US$M)	(102)
Operating Cash Flow	**(US$M)**	**868**
Tax	(US$M)	(207)
Capital Expenditure	(US$M)	(193)
Free Cash Flow	**(US$M)**	**468**
RoM Feed produced From U/G	(kt)	8,299
RoM Feed produced From O/P	(kt)	10,437
Feed From O/P Stockpile	(kt)	464
Total Feed	(kt)	19,200
Waste Mined U/G	(kt)	1,499
Waste Mined O/P	(kt)	52,515
Contained Au	(koz)	1,693
Recovered Au	(koz)	1,574
Mining cost (U/G & O/P)	(US$/t $_{\text{Total feed}}$)	(29.70)
Processing cost	(US$/t $_{\text{Total feed}}$)	(17.50)
G&A cost	(US$/t $_{\text{Total feed}}$)	(5.54)
Refining cost	(US$/t $_{\text{Total feed}}$)	(0.41)
Contingency on U/G Mining Cost	(US$/t $_{\text{Total feed}}$)	(2.92)
Royalty	(US$/t $_{\text{Total feed}}$)	(5.33)
Total	**(US$/t $_{\text{Total feed}}$)**	**(61.39)**
Revenue	(US$/oz)	1,300.00
Operating Costs	(US$/oz)	(748.82)
Cash Cost	**(US$/oz)**	**551.18**

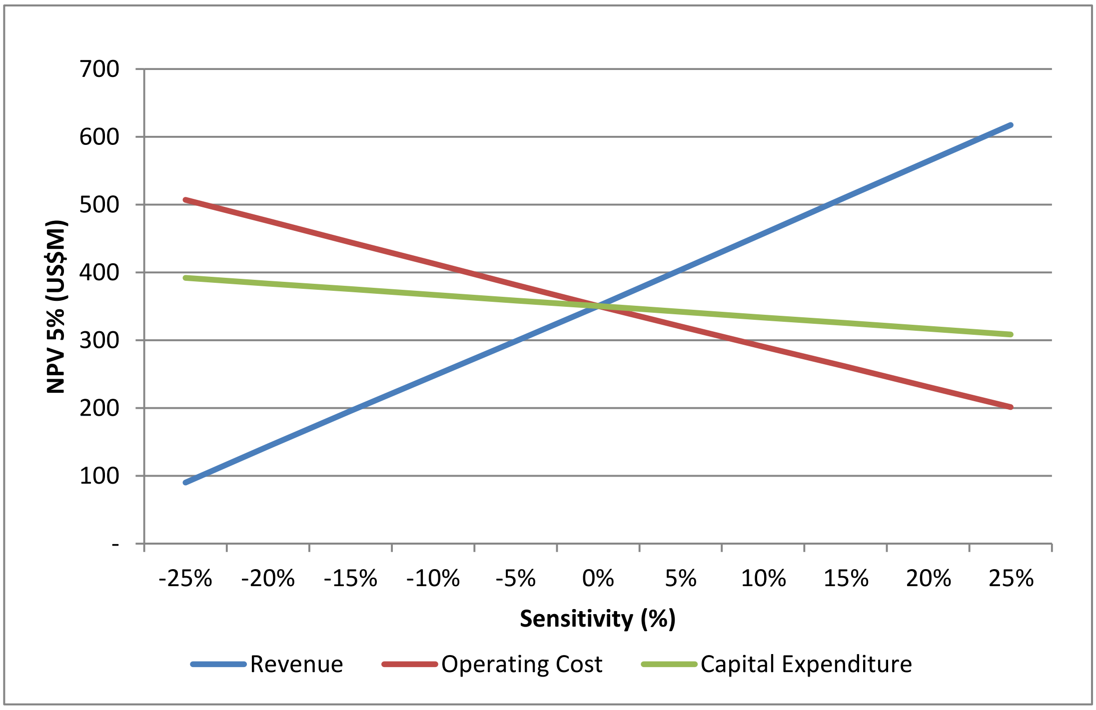

Figure ES 1: NPV Sensitivity Chart

CONCLUSIONS AND RECOMMENDATIONS

CONCLUSIONS

Based on the work carried out for this PEA, SRK concludes the results indicate that there is potential for a combined open pit and underground operation at Wassa. SRK notes that further detailed technical work and investigation is required to confirm the optimal approach and economic viability through improving the accuracy of mine planning and cost estimates.

In SRK's opinion an underground mine at Wassa is practically achievable and the PEA indicates that a sub-level open stoping method will be suitable together with a consolidated fill to maximise the extraction ratio. A single decline would appear to be the most suitable means for accessing the identified production areas using a trackless haulage for materials handling to surface. The study also indicates that an initial production rate of 0.75 Mtpa is achievable from the underground which can be increased to around 1 Mtpa over a number of years.

GSWL has a long history of mining at Wassa and has the required understanding of how to implement the Wassa Project into their current operations.

RECOMMENDATIONS

Based on the work carried out for this PEA, SRK recommends that consideration is given to advancing the Wassa Project to a feasibility level of study, using this PEA as a basis for development of the mining approach and technical detail. Further detailed investigation is required to confirm the following aspects:

- Location of underground development;
- Stope dimensions;
- Pastefill versus cemented rock fill
- Water balance;
- Detailed development and production scheduling; and
- Capital and operating cost estimation.

As the project advances to the next stage of its development, a dedicated programme of metallurgical testwork will be undertaken for the deeper mineralization to be accessed using underground mining methods.

This testwork should principally cover the expected increase in rock hardness with depth, and any potential variation in the recovery response with grind size. The other key parameter to be covered will be the settling and tailings characteristics properties of the future ores.

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Abbreviations

AAS	Atomic absorption spectroscopy
ALS	ALS Minerals in Ghana-Kumasi
ANCOLD	Australian Commission on Large Dams
ANFO	Ammonium Nitrate Fuel Oil
AP	Anchor Point
ARO	Asset Retirement Obligations
Au	Gold
BIF	Banded Iron Formation
Bwi	Bond Ball Mill Work Index
CIL	Carbon-in-Leach
CMCCs	Community Mine Consultative Committees
CRF	Cemented, or consolidated, rock fill
CRM	Certified Reference Material
CSA	Canadian Securities Administrators
DD	Diamond Drill
DIBK	Diisobutyl Ketone
East Domain	Chichiwelli East
EIA	Environmental Impact Assessment
EIS	Environmental Impact Statement
EMP	Environmental Management Plan
EOM	End of Month
EPA	Environmental Protection Agency
FOS	Factor of Safety
G&A	General and Administrative
GEMS	Gemcom Software
Geostats	Geostats Pty Ltd
Glencar	Glencar Exploration Limited
GRG	Gravity Recovered Gold
GSLib	Geostatistical Software Library
GSR	Golden Star Resources Limited
GSWL	Golden Star (Wassa) Limited
HDPE	High-density polyethylene
ICOLD	International Commission on Large Dams
IRS	Intact Rock Strength
ITCZ	Inter Tropical Convergence Zone
KP	Knight Piésold
LHD	Load Haul Dump
LoM	Life of Mine
MDM	Metallurgical Process Development Pty Ltd
MRE	Mineral Resource Estimate
MSO	mineable shape optimiser
NaCN	Sodium Cyanide
NI 43-101	National Instrument 43-101
NPV	Net Present Value
PEA	Preliminary Economic Assessment

PVC	Poly(vinyl chloride)
QAQC	Quality Assurance Quality Control
QP	Qualified Person
RAB	Rotary Air Blast
RC	Reverse Circulation
RL	Reduced Level
RoM	Run of Mine
RQD	Rock Quality Designation
SE	Subriso East
SEDAR	System for Electronic Document Analysis and Retrieval
SG	Specific Gravity
SGL	Satellite Goldfields Limited
SGS	SGS Laboratories in Tarkwa
SRK or SRK (UK)	SRK Consulting (UK) Limited
SSC	Single Shot Camera
SW	Subriso West
TEM	Technical Economic Model
TMM	Total Material Movement
TSF	Tailings Storage Facility
TSF1	Existing Tailings Storage Facility
TSF2	New Tailings Storage Facility
TTG	Tonalite–Trondhjemite–Granodiorite
TWL	Transworld (now Intertek) Laboratories
UDC Flow	undiscounted cash flow
VR	Vent Raise
Wassa Project	Wassa open pit and underground gold project
West Domain	Chichiwelli West
Whittle	Whittle Four-X software
WPT	Windfall Profits Tax
WRC	Water Resources Commission
WSL	Wassa Site Laboratory

Units

deg.	Degrees Celsius
g/t	Grams per tonne
kg	Kilogram
kW	Kilo Watt
kWh/t	Kilo Watt hour per tonne
l/s	litres per second
m	Metre
m/sec	Metres per second
m3	Cubic metre
Ma	Million years
ml	millilitre
mm	millimetre
Mt	Million tonnes
Mtpa	Million tonnes per annum
Pa	Pascal
t	Metric tonne
t.oz	troy ounce
US$	US Dollars

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NI 43-101 TECHNICAL REPORT ON A PRELIMINARY ECONOMIC ASSESSMENT OF THE WASSA OPEN PIT MINE AND UNDERGROUND PROJECT IN GHANA

1 INTRODUCTION (ITEM 2)

This Technical Report is a Preliminary Economic Assessment ("PEA") of the combined Wassa open pit and underground gold project (the "Wassa Project") which has been prepared by SRK Consulting (UK) Limited ("SRK") for and on behalf of Golden Star Resources Limited ("GSR"). This report also incorporates an update of the Wassa Mineral Resources as at 31 July 2014.

GSR currently holds a 90% interest in the subsidiary company, Golden Star (Wassa) Ltd. ("GSWL") who operate the Wassa open pit gold mine located in the western region of Ghana, to the northeast of Tarkwa. GSR is a gold exploration and producing company with operational mining interests in Ghana, West Africa and exploration interests throughout West Africa and in South America. This report deals exclusively with the mining operations at GSWL. The Ghanaian Government has a 10% ownership of GSWL and currently receives a 5% royalty on the gross revenue on all GSWL gold production.

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").

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

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

In recent years the Wassa Plant has received open pit feed from the Wassa Main deposit and a number of satellite pits (Benso and Hwini Butre) at a rate of 2.1 to 2.6 Mtpa. This PEA considers a combined open pit and underground feed sourced from the Wassa Main deposit at a rate of 2.6 Mtpa for the initial 5 years followed by a decreased rate of 1 Mtpa for the remaining underground only feed.

A NI 43-101 Technical Report published in March 2013, presented Mineral Resources and Mineral Reserves for GSWL under an open pit only mining scenario. This technical report presents a PEA with an updated mineral resource estimate, under the scenario of a limited size open pit and an underground operation mining the deeper mineralization.

All monetary values shown in the report are US Dollars ("US$") unless otherwise stated.

1.1 Terms of Reference

SRK completed an NI 43-101 Technical Report for Wassa Gold Mine for and on behalf of GSR in 2012, with an effective date of 31 December 2012. Following additional exploration drilling during 2013, the mineral resource models were updated by GSR to include the significant amounts of additional geological data. Mineral Resources were updated by GSR in Q4 2013 and, subsequent to that, the Mineral Reserves for Wassa were also updated by GSR, following the application of mining and processing modifying factors to the Mineral Resource, open pit optimisation, mine planning and scheduling.

SRK Consulting (Canada) Limited was engaged by GSR at the beginning of 2014 to assist GSR in the preparation of the updated Mineral Resource estimate. SRK (UK) was engaged in mid-2014 to undertake a PEA on a combined open pit and underground mining option based on the new resource estimate.

1.2 Data Sources

The data provided to SRK was largely provided by GSR staff based at the mine site, at the Toronto Head Office or at the Takoradi Exploration Office. The information provided comprised:

- Digitized exploration data containing drillhole and surface sampling information collected as part of either initial exploration works or subsequent mine exploration and grade control activities;
- Digital wireframe models produced by GSR staff;
- Digital mineral resource estimation block models containing grade, density, geological and quality information. A block model was completed by SRK (Canada) on behalf of GSR;
- Mining and production data, including monthly and annual reports, mining schedules and equipment schedules for the LoM plan, provided by GSR;
- Historical production and recovery statistics, current operating statistics and costs for the processing plant, provided by GSR;
- Geotechnical data and reports produced by SRK as part of previous commissions with GSR and on-going support with the geotechnical drilling program at Wassa;
- Tailings and waste management information, provided by Knight Piésold (“KP”) consultants; and
- Data relating to the social and environmental impact of the mining operations both historically and planned during the remaining Life of Mine (“LoM”), provided by GSR.

1.3 Qualified Persons

Table 1-1 provides a summary of the designated Qualified Persons (“QP”) and other key contributors for completion of this technical report.

Table 1-1: Qualified Persons and Contributors to this Technical Report

Qualified Persons Responsible for the Preparation of this Technical report						
Qualified Person	**Position**	**Employer**	**Independent of GSR**	**Date of Last Site Visit**	**Professional Designation**	**Sections of the Report**
Mike Beare	Corporate Consultant (Mining Engineer)	SRK Consulting (UK) Ltd	Yes	No visit [1]	CEng, BEng, ACSM, MIoM3	Sections 1 to 5, 14, 15 (Underground), 17 to 26 and overall responsibility for report
S. Mitchel Wasel	Vice President Exploration	Golden Star Resources Ltd	No	July 2014	BSc, MAusIMM(CP)	Section 6 to 13
Neil Marshall	Corporate Consultant (Geotechnical)	SRK Consulting (UK) Ltd	Yes	February, 2014	CEng, MSc(DIC), MIoM3	Section 15 (Geotechnical)
Chris Bray	Principal Consultant (Mining Engineer)	SRK Consulting (UK) Ltd	Yes	January, 2013	BEng, MAusIMM(CP)	Section 15 (Open Pit)
Dr John Willis	Principal Consultant (Mineral Processing)	SRK Consulting (UK) Ltd	Yes	January, 2013	BE, PhD, MAusIMM(CP)	Section 12, 16
Other Experts who assisted the Qualified Persons						
Expert	**Position**	**Employer**	**Independent of GSR**	**Visited Site**	**Sections of the Report**	
Martin Raffield	Senior Vice President Technical Services	Golden Star Resources Ltd	No	October 2014	All	
Dr Oy Leuangthong	Principal Consultant (Geostatistics)	SRK Consulting (Canada) Ltd	Yes	no visit	Mineral Resources	
Yan Bourassa	Director Business Development	Golden Star Resources Ltd	No	April 2014	Geology	
Kristina Huss	Consultant (Mining Engineer)	SRK Consulting (UK) Ltd	Yes	no visit	Mine Design & Scheduling	
Keith Joslin	Principal Consultant (Due Diligence)	SRK Consulting (UK) Ltd	Yes	no visit	Economic Analysis	
Mark Thorpe	Senior Vice President (Corporate Social Responsibility and Environmental Affairs)	Golden Star Resources Ltd	No	October 2014	Environmental Overview	
Jane Joughin	Corporate Consultant (Environmental & Social Management)	SRK Consulting (UK) Ltd	Yes	no visit	Environmental Review	
Tony Rex	Corporate Consultant (Hydrogeology)	SRK Consulting (UK) Ltd	Yes	February, 2014	Hydrogeology Review	
Kris Czajewski	Principal Consultant (Tailings)	SRK Consulting (UK) Ltd	Yes	no visit	Tailings Review	

Footnote 1: Although Mr Beare has not inspected the property the property in connection with this technical report due to other commitments, he has relied on other qualified persons from SRK who have inspected the property and he has assumed overall responsibility for the technical report.

1.4 Limitations, Reliance on SRK, Declaration, Consent, Copyright and Cautionary Statements

SRK's opinion contained herein and effective 30 October 2014, is based on information collected and completed by SRK throughout the course of the PEA, which in turn reflect various technical and economic conditions at the time of writing. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time. Consequently, actual results may be significantly more or less favourable.

This report may include technical information that requires subsequent calculations to derive sub-totals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK does not consider them to be material.

SRK is not an insider, associate or an affiliate of GSR, and neither SRK nor any affiliate has acted as advisor to GSR, its subsidiaries or its affiliates in connection with this project. The results of the technical review by SRK are not dependent on any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings.

2 RELIANCE ON OTHER EXPERTS (ITEM 3)

The information reviewed in this report has largely been provided directly by GSR and has been produced either by GSR or by its sub-consultants. In addition to SRK (Canada)'s contribution to the Mineral Resource Estimate ("MRE") and geotechnical areas of the Wassa project, the other major contributor has been KP consultants, which has undertaken the bulk of the surface geotechnical investigations and design for the tailings management. Environmental and social information was provided by GSR staff.

SRK has reviewed reports prepared by GSR and has undertaken sufficient verification work to give its independent opinion on these.

SRK has confirmed that the Mineral Resources reported herein are within the licence boundaries given below. SRK has not performed an independent verification of land title and tenure as summarized in Section 3 of this report. SRK did not verify the legality of any underlying agreement(s) that may exist concerning the permits or other agreement(s) between third parties.

3 PROPERTY DESCRIPTION AND LOCATION (ITEM 4)

3.1 Location and Land Tenure

The Wassa Gold Mine is located near the village of Akyempim in the Wassa East District, in the Western Region of Ghana. It is located 80 km north of Cape Coast and 150 km west of the capital Accra. The property lies between latitudes 5°25' and 5°30' north and between longitudes 1°42' and 1°46' east. The location of the Wassa mine is shown on Figure 3-1 and Figure 3-2.

GSWL currently holds three mining leases, namely Wassa, Hwini Butre, and Benso. In addition, the company holds several prospecting leases in the region. The mining and prospecting lease details are summarized in Table 3-1 and Table 3-2.

Figure 3-1: Location of Wassa (red star) in the regional context of Ghana and West Africa showing principal infrastructure and population centres

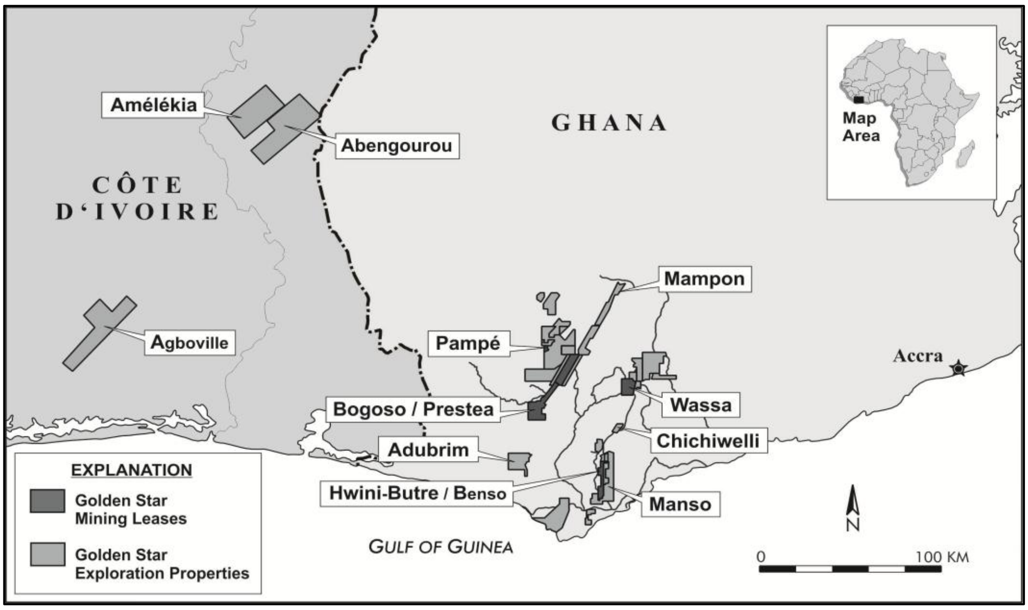


Figure 3-2: GSWL and GSR Mining and Exploration Concessions in Ghana (GSR, 2013)

Table 3-1: GSWL Mining Leases, Prospecting Leases and Mining Permits

Permit / Lease	Permit No.	Agency	Date of Issue	Expiry Date	Comments
Wassa Mining Lease	LVB 7618/94	Minerals Commission	17/09/1992	16/09/2022	
Benso Mining Lease	LVB26871/07	Minerals Commission	27/09/2007	26/09/2019	
Hwini Butre Mining Lease	LVB1714/08	Minerals Commission	11/01/2008	01/10/2018	
Accra New Town Prospecting	LVB 15568/07 (RL.2/99)	Minerals Commission	11/10/2004	13/11/2012	Extension under application
Dwaben (Safric) Reconnaissance	LVB1624/06	Minerals Commission	02/02/2006	01/10/2012	Renewal under application
Wassa Prospecting /Exploration permit in the Subri River Forest Reserve	N/A	Forestry Commission	14/12/2006	30/06/2007	Applied for Renewal – only required when work is to be done in forest
Benso (Chichiwilli-Amantin) Prospecting	LVB 9113094 (PL.3/61)	Minerals Commission	27/09/2007	18/11/2010	Renewal under application
Oseneso 1 & 2 Prospecting	LVB 13975/06	Minerals Commission	08/09/2006	01/03/2011	Extension under application
Mining Permit	#0014759/14	Inspectorate Division	17/01/2014	31/12/2014	Renewal
Permit to store explosives	#0004089/14	Inspectorate Division	17/01/2014	31/12/2014	
Explosive Purchasing Permit (Mine Form 'A')	008259/14	Inspectorate Division	17/01/2014	31/12/2014	
Benso Mining Permit	#0014757/14	Inspectorate Division	17/01/2014	31/12/2014	
Hwini Butre Mining Permit	#0014758/14	Inspectorate Division	17/01/2014	31/12/2014	

Table 3-2: Additional detail on GSWL Mining Leases

Property	Tenement Name	Tenement No.	Area (km^2)	Granted	Renewal
Wassa	Wassa Mining Lease	LVB 7618/94	52.89	17/09/92	16th September, 2022
Wassa	Benso Mining Lease	LVB 26871/07	20.38	27/09/07	26/09/11 Renewed ML document signed by GM & Secretary with company seal. Document returned to Mincom on 20th June 2012 for Minister's signature. Renewal for 7 years from date of ministers signature
Wassa	Hwini Butre Mining Lease	LVB1714/08	40.00	11/01/08	10/01/12 Renewed ML document received from Mincom on 20th June 2012 for Secretary/GM's signatures and company seal. Renewal for 5 years from date of ministers signature

The map in Figure 3-3 shows the location of the various GSR/GSWL exploration properties and mining leases, which incorporate the deposits listed as follows:

- Wassa mining lease: Wassa Main is an operating open pit gold mine comprising the following mineralization domains: F Shoot, 419, B Shoot, 242, Starter, South-East, Mid-East and Dead Man's Hill ("DMH"). SAK comprises a number of deposits to the West of Wassa Main, which are no longer mined.
- Benso mining lease: comprising the Subriso East ("SE"), Subriso West ("SW"), G-Zone and I-Zone deposits.
- Hwini Butre mining lease: comprising the Father Brown, Adoikrom and Dabokrom deposits, with only Father Brown having remaining open pit reserves.
- Chichiwelli exploration property: comprising two mineralized zones, Chichiwelli West ("West Domain") and Chichiwelli East ("East Domain").
- Manso exploration property, located adjacent to Benso, to the east.

The properties and leases are spread out along a line trending approximately 80 km southwest of the Wassa mine complex.

Figure 3-3 shows the location of the mining lease boundaries in relation to the location of the main GSWL mine workings at Wassa, Hwini Butre and Benso. The mine infrastructure, including waste dumps and tailings storage facility, lies within the limits of the current lease boundaries.

SRK notes that the mining leases are subject to royalties in accordance with the laws of the Republic of Ghana which applies a levy on revenue of 5%. Further detail is provided in Section 21.2.

It is further noted that GSR requires an Environmental Permit from the Environmental Protection Agency ("EPA") and an extension to the current surface mining permit to cover underground operations from the Minerals Commission. Preparations are currently being made to obtain these permits.

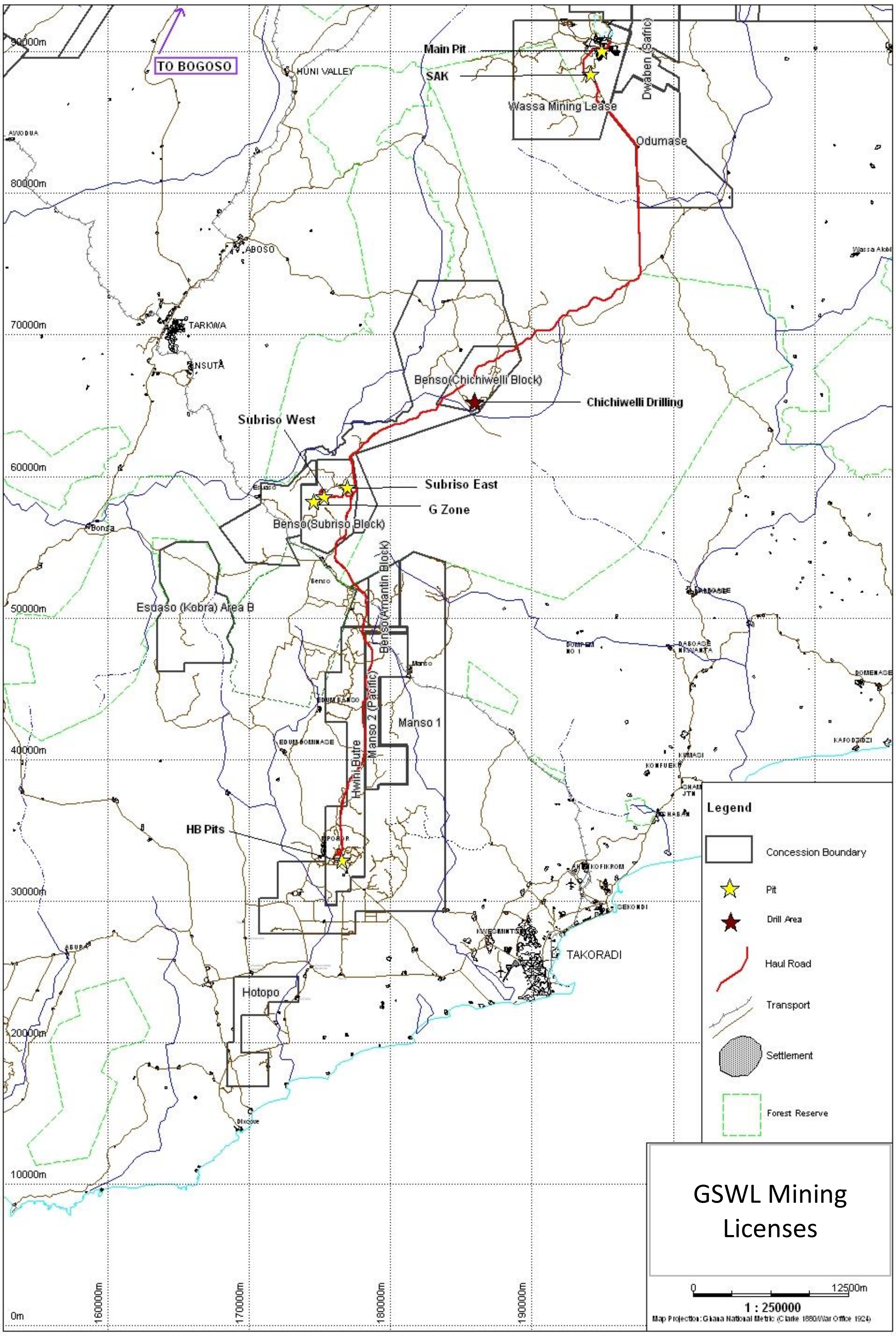


Figure 3-3: Location of principal operations and local infrastructure in relation to mining licence boundaries at GSWL (GS Exploration, 2012)

3.2 Environmental Approvals and Liabilities

Information on environmental approvals and environmental liabilities is presented in Sections 19.1 and 19.4.

4 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY (ITEM 5)

4.1 Access to Property

The Wassa Gold Mine is located near the village of Akyempim in the Wassa East District in the Western Region of Ghana. It is 62 km north of the district capital, Daboase, and 40 km east of Bogoso. It is located 80 km north of Cape Coast and 150 km west of the capital Accra. The main access to the site is from the east, via the Cape Coast to Twifo-Praso road, then over the combined road-rail bridge on the Pra River. There is also an access road from Takoradi in the south via Mpohor.

4.2 Local Resources and Infrastructure

There are four other mines in the vicinity of Tarkwa; namely, Ghana Manganese Company – Nsuta Mine, Anglo Gold Ashanti Iduapriem Gold Mine and Goldfields Ghana Limited - Damang and Tarkwa mines.

The Wassa Mine itself is located in the Wassa Mining Lease within the Subri-Akyempim Concession, which covers an area of 57 km^2.

Wassa Mine is currently an operating open pit mining approximately 1.2 Mt of total material per month, therefore, the required services, infrastructure, and community support are already in place. The following are relevant to the assessment of resources and infrastructure:

- Access to the Project is via the public road network that extends on to the site;
- Electricity and water are available;
- Surface infrastructure in the area consists of a variety of government, municipal, and other roads with good overall access;
- Processing will be carried out at the existing GSWL processing plant;
- Tailings will be stored in the existing GSWL tailings storage facility and the new tailings storage facility, which is currently permitted;
- Waste rock generated at the site will be placed in existing waste dumps, adjacent to the Wassa open pit; and
- The extensive history of mining in Ghana provides opportunities to obtain skilled underground mine workers.

4.3 Climate and Length of Operating Season

The climate in the project area is classified as wet semi-equatorial. The Inter Tropical Convergence Zone (“ITCZ”) crosses the area twice a year, resulting in a bi-modal rainfall pattern, with peaks in March to July and September to October. During the dry season months of November to February, the climate is heavily influenced by the dry, dust-laden, northwest trade wind, known locally as the Harmattan, which blows from the Sahara desert.

Analysis of available rainfall data, obtained from the Ateiku Meteorological survey (1944 to 2009) indicates that the average annual rainfall is 1,996 ± 293 mm. The wettest month of the year is generally June, with an average rainfall of about 241 ± 85 mm, whilst January is the driest month of the year with an average rainfall of about 31 ± 35 mm. The wettest month on record is June 2009, when 475 mm of precipitation was recorded. Rainfall is mainly influenced by south-west monsoon winds, which blow from the south-western part of the country towards the north-east.

Using data from the GSWL weather station, the average annual rainfall has been estimated at about 1,750 mm. A drier period, which is influenced predominantly by a sweep of the North-east traded winds, is experienced between the month of November and February.

Annual potential evapotranspiration is estimated to be approximately 1,337 mm/yr, indicating a minimum precipitation excess of 288 mm/yr. Rainfall exceeds potential evapotranspiration from March to July and September to October, and groundwater recharge is most likely to be prevalent during these periods.

Under such climatic conditions surface mining operations can continue year round with short breaks during storms, most of which are short-lived and may be experienced throughout most of the year. Underground mining operations will not be directly affected by storms as long as effective storm water management infrastructure is in place at surface to divert runoff from mine accesses.

4.4 Topogrophy and Vegetation

The project area is characterized by gently rolling hills with elevations up to 1000 and 1100 m RL, incised by an extensive drainage network. The area comprises tropical rainforest and is relatively wet, with many low lying swampy areas. Extensive subsistence farming occurs throughout the area, with plantain, cassava, pineapple, maize, and cocoyam being the principal crops. Some small scale cultivation of commercial crops is also carried out, with cocoa, teak, coconuts and oil palm being the most common. Wet and moist evergreen forest types of Western Ghana are the natural vegetation types in the concession area. The natural vegetation has been degraded by earlier logging activities, and past and present farming activities, and now largely comprises broken forest, secondary-forest, farmland and abandoned farmland, with upland type re-growth with swamps in some valley areas. Forests patches are present on the steep slopes and in areas unsuitable for agriculture.

5 HISTORY (ITEM 6)

5.1 Ownership

The Wassa area has witnessed several periods of local small scale and colonial mining activity from the beginning of the 20th century and mining of quartz veins and gold bearing structures are evident from the numerous pits and adits covering the Wassa lease area.

From 1988, the property was operated as a small scale mining operation with a gravity gold recovery circuit by a Ghanaian company, Wassa Mineral Resources Limited. In 1993, Wassa Mineral Resources was looking for a capital partner to further develop the mining lease, and invited the Irish companies Glencar Exploration Limited (“Glencar”) and Moydow Ltd to visit the concession. Following this visit, Satellite Goldfields Limited (“SGL”) was formed between Wassa Mineral Resources, Glencar and Moydow Ltd. The mining lease, which is valid for a 30 year period expiring in 2022, was assigned by Wassa Mineral Resources Limited to SGL.

Extensive satellite imagery and geophysical interpretations were carried out, which identified a strong gold target (>1 g/t Au). Exploration drilling commenced in February 1994, and by March 1997 a total of 58,709 m of reverse circulation and diamond drilling had been completed. In September 1997, consulting engineers Pincock, Allen and Holt completed a Feasibility Study (“FS”), which determined a proven and probable mineable reserve of 17.6 Mt at 1.7 g/t Au, for a total of 932,000 contained ounces of gold. The reporting code, and key assumptions and parameters used to report this historical MRE are not known and a Qualified Person 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. Construction of the Wassa Mine was initiated in September 1998 after Glencar secured a US$42.5 million debt-financing package from a consortium of banks and institutions.

The Wassa Mine was originally developed as a 3 Mtpa open pit heap leach operation with a forecasted LoM gold production of approximately 100,000 ounces per annum. The first material from the pit was mined in October 1998. After approximately one year of production, it became evident that the predicted heap leach gold recovery of 85% could not be achieved, mainly due to the high clay content of the resource and poor solution management. After a number of attempts to improve the recovery, including increased agglomeration and doubling the leach solution application rate, it was concluded that the achievable gold recovery by heap leach was between 55% and 60%. The combined effect of the lower than planned gold recovery and lull in the gold price at the time resulted in the company not being able to service its debt to the creditors. In early 2001, the creditors together with Glencar decided to sell the project to recover some of the accumulated debt. Mining was stopped at the end of October 2001 and irrigation of the heap leach with cyanide solution continued until March 2002, after which rinsing of the heaps with barren solution continued until August 2002.

When the secured senior creditors exercised security over the project in 2001, the project was put up for sale and GSR was invited, amongst other parties, to conduct a due diligence on the operation. In November 2001, negotiations were started to acquire the Wassa assets. As part of a final due diligence on the resources, GSR undertook a structural evaluation and drilling program between December 2001 and April 2002. Upon completion of the acquisition of Wassa Mine by GSR, a further exploration program was undertaken. Both these exploration programs formed part of a FS that was completed in July 2003, which demonstrated the economic viability of reopening and expanding the existing open pits, and processing the material through a conventional Carbon-in-Leach (“CIL”) circuit. Wassa has been operating as a conventional CIL milling operation since late April 2005. A summary of production form the

Wassa mine is presented in Table 15-1 in Section 15.2.1 of this report.

6 GEOLOGICAL SETTING AND MINERALIZATION (ITEM 7)

6.1 Regional Geology

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

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

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

The Birimian series was first described by Kitson (1918) based on outcrops located in the Birim River (around 80 km east of the Ashanti Belt). Since this early interpretation, the Birimian stratigraphic column has been revised significantly. Before the application of geochronology, the Birimian super group was divided in an Upper Birimian group composed mainly of metavolcanics and a Lower Birimian group corresponding to metasedimentary basins. Subsequent authors have proposed synchronous deposition of Birimian metavolcanics. Most recently, Sm/Nd and U/Pb analyses have reversed the earlier stratigraphic interpretation with the younger metasediments overlying the older metavolcanics. Proposed ages for the metavolcanics vary between 2,162 ± 6 Ma and 2,266 ± 2 Ma. Detrital zircons in the metasediments indicate the initiation of their deposition between 2,142 ± 24 Ma 2,154 ± 2 Ma. The Kumasi Group was intruded by the late sedimentary Suhuma granodiorite at 2,136 ± 19 Ma (U/Pb on zircon, Adadey et al., 2009).

The Tarkwa super group was first recognized by Kitson (1928) and consists of a succession of clastic sedimentary units, which have been divided in four groups by Whitelaw (1929) and Junner (1940). The Kawere Group located at the base of the Tarkwaian super group is composed of conglomerates and sandstones with a thickness varying between 250 m and 700 m. The unit is stratigraphically overlain by the Banket Formation, which is characterized by sequences of conglomerates interbedded with cross-bedded sandstone layers, the maximum thickness of this group being 400 m. The conglomerates are principally composed of Birimian quartz pebbles (>90%) and volcanic clasts (Hirdes and Nunoo, 1994) that host the Tarkwa Placer deposits. The Banket formation is overlain by approximately 400 m of Tarkwa Phyllites. The uppermost unit of the Tarkwa super group is the Huni Sandstone, comprised of alternating beds of quartzite and phyllite intruded by minor dolerite sills that form a package up to 1,300 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 2,133 ± 3.4 Ma for the Banket formation (Davis et al., 1994; Hirdes and Nunoo, 1994). These ages agree with the study by Pigois et al. (2003) that yielded maximum depositional age of 2,133 ± 4 Ma from 71 concordant zircons of the Banket formation. According to all concordant zircon histograms (161 grains) and their uncertainties, a reasonable estimation for the start of the Tarkwaian sedimentation could be as young as 2,107 Ma.

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

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

With the exception of some late Eburnean granitoids, dolerite dykes and Phanerozoic sediments, all other lithologies have undergone metamorphism that generally does not exceed upper greenschist facies. Studies on amphibole/plagioclase assemblages suggest the peak temperature and pressure was 500 to 650 ◦C and 5 to 6 kbar (John et al., 1999), dated at 2092 ± 3 Ma (Oberthür et al., 1998).

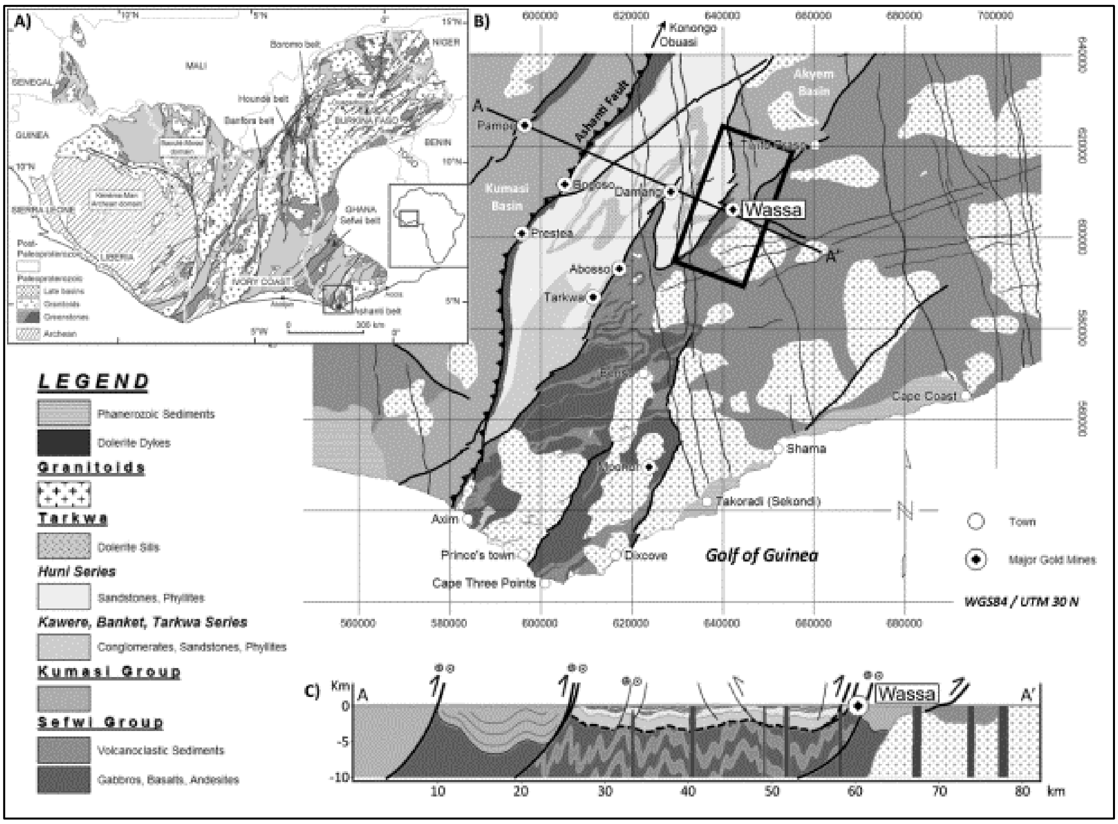


Figure 6-1: Location of the Wassa Mine with the geology of the Ashanti Belt (Perrouty et al., 2012).

6.2 Local Geology

6.2.1 Introduction

The Wassa property lies within the southern portion of the Ashanti Greenstone Belt along the eastern margin of the belt within a volcano-sedimentary assemblage located at proximity to the Tarkwaian basin contact. The eastern contact between the Tarkwaian basin and the volcano-sedimentary rocks of the Sefwi group is faulted, but the fault is discrete as opposed to the western contact of the Ashanti belt where the Ashanti fault zone can be several hundred meters wide. Deposition of the Tarkwaian sediments was followed by a period of dilation and the intrusion of late mafic dykes and sills.

The lithologies of the Wassa assemblage are predominantly comprised of mafic to intermediate volcanic flows which are interbedbed with minor horizons of volcaniclastics, clastic sediments such as wackes and magnetite rich sedimentary layers, most likely banded iron formations. The volcano-sedimentary sequence is intruded by syn-volcanic mafic intrusives and felsic porphyries.

The magnetic signature of the Ashanti belt is relatively high in comparison to the surrounding Birimian sedimentary basins such as the Kumasi basin to the west of the Ashanti belt and the Akyem Basin to the East as illustrated in Figure 6-2.

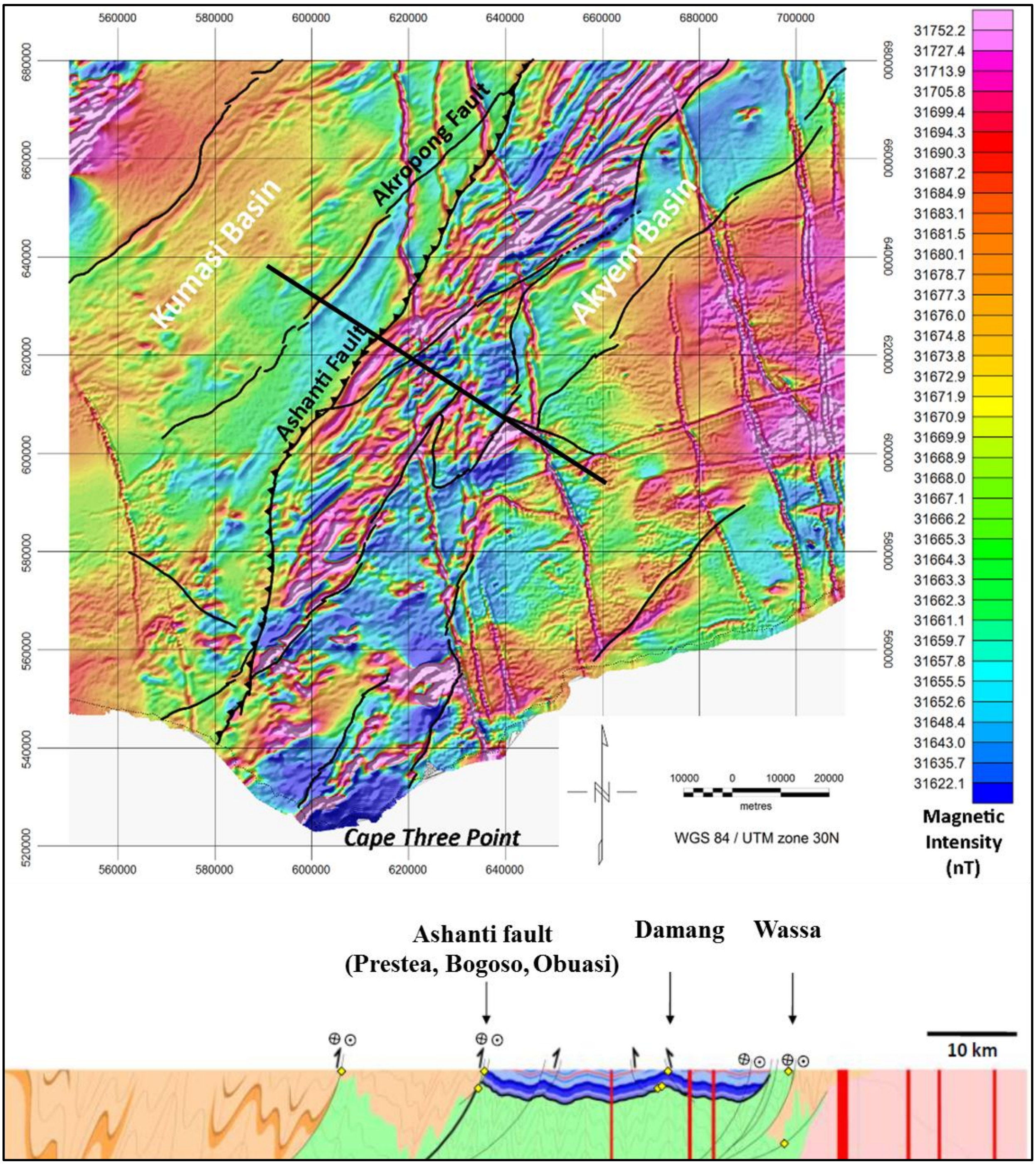


Figure 6-2: Total magnetic intensity reduced to pole of the Ashanti Belt (Modified from Perrouty et al, 2012)

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 6-4, 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 event. Structural evidences and relationships observed at in drill core and pits at Wassa would suggest the mineralization to be of Eoeburnean timing while other known deposits in the southern portion of the Ashanti belt such as Chichiwelli, Benso and Hwini Butre are considered to be of Eburnean age.

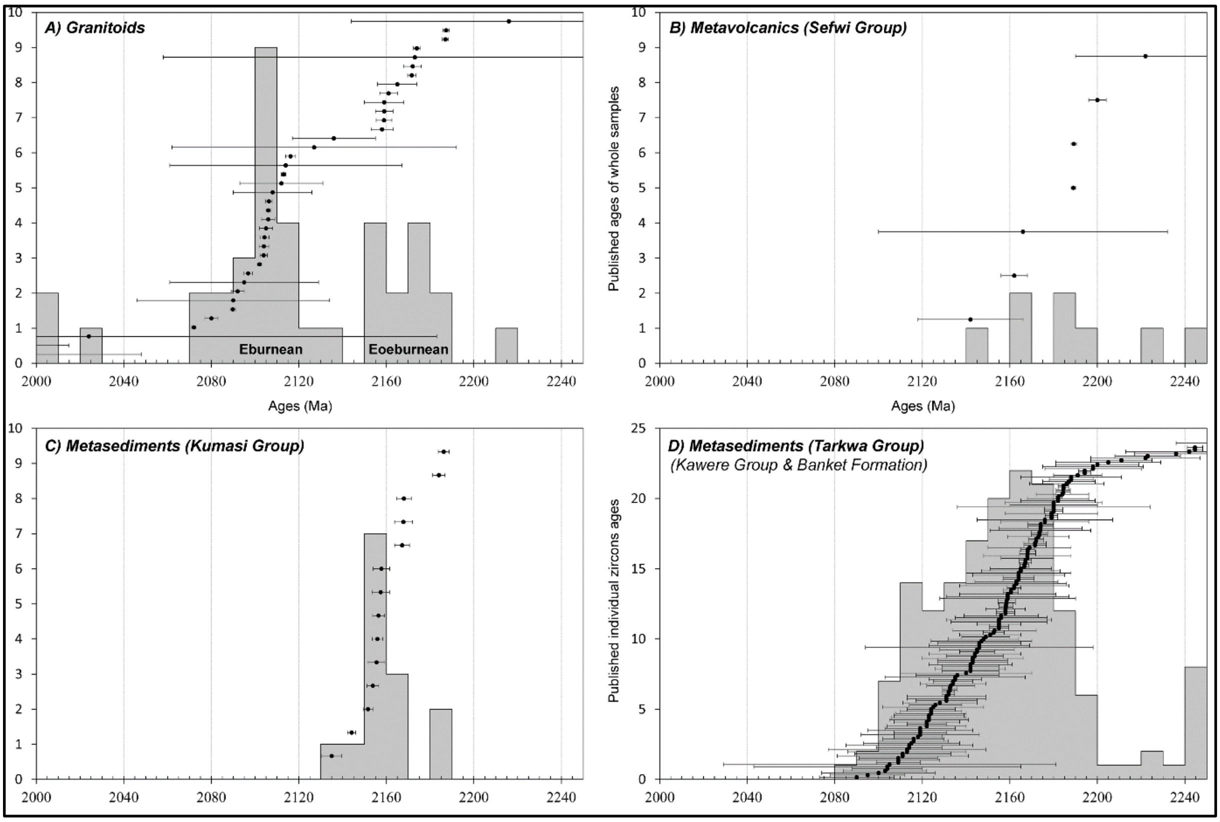


Figure 6-3: Compilation of geochronology dating from the Ashanti Belt (Perrouty et al, 2012)

The Eoeburnean deformation is best observed at Wassa where the deformational event has produced a penetrative foliation with an associated lineation which is defined by mineral alignments. A period of extension occurred between the Eoeburnean and Eburnean deformational events which resulted in the formation of the Akyem Basin (Kumasi Group) to the northeast of the Wassa Mine and the Tarkwa group to the west of the Wassa concession. Both metasedimentary sequences of the Takwa and Kumasi group have not been affected by the penetrative foliation observed at Wassa.

The Eburnean deformation is divided in multiple events which vary in number depending on the authors as summarized in Figure 6-4. All deposits underlying the Wassa concession have been affected by the Eburnean deformational events, the main penetrative foliation has been affected by at least three Eburnean folding events which have resulted in a large scale refolded synform. The main foliation is sub-vertical and oriented northeast to southeast on the southeastern flank of the Wassa mine fold whereas it is dipping at around 45° to the south-southeast on the northwest flank of the Wassa mine fold.

<table>
<tr><th colspan="2">Regional Interpretation
(This Study)</th><th>In Birimian
Obuasi / Bogoso
(Allibone et al., 2002a, b)</th><th>In Tarkwaian
Damang
(Tunks et al., 2004)</th><th>Regional
(Eisenlhor et al., 1992)</th><th>Regional
(Feybesse et al., 2006)
(Milesi et al., 1992)</th></tr>
<tr><td rowspan="2">Eburnean 1
> 2150 Ma</td><td>Early Birimian volcanism and sedimentation</td><td rowspan="2">Volcanism
Granitoids intrusion
Regional metamorphism</td><td rowspan="3"></td><td>Birimian sediments and volcanics penecontemporaneous
Plutonism (Dixcove type granitoids)</td><td rowspan="3">Magmatic accretion
Plutonism
Birimian sedimentation</td></tr>
<tr><td>D1, N-S shortening
Regional scale folding in the Early Birimian unit
Syn-tectonic plutonism before 2170 Ma
Possible gold mineralization</td><td rowspan="2">Onset of deformation in a "foreland thrust" and Tarkwaian deposition</td></tr>
<tr><td colspan="2">D2, Extension Phase
Late Birimian sedimentation
S2 parallel to bedding (S0) in Birimian sediments
Tarkwaian basin formation</td><td>D1
S1 parallel to bedding
Flat-lying bedding parallel shearing</td></tr>
<tr><td rowspan="5">Eburnean 2
2120 - 2060 Ma</td><td rowspan="2">D3, NW-SE shortening
Km scale folds in Birimian and Tarkwaian
S3 subvertical crenulation cleavage NE-SE
Thrust faults (Ashanti, Damang, …)
Peak of metamorphism (Low Amphibolite)</td><td>D2, NW-SE shortening
Isoclinal folds with axial surface parallel to the regional faults and shear zones
Ashanti thrust fault</td><td>D1, NW-SE shortening
Km scale folds (with subvertical axial surface (S1))
Damang thrust fault</td><td>D1, NW-SE shortening
S1 (NE-SE) subvertical and subparallel to bedding in both Birimian and Tarkwaian
Regional folds (tight to isoclinal)</td><td>D1, NW - SE shortening
Thrust faults
Tarkwaian sediments deposition (Syn D1)
Metamorphism (6 kbar / 550 - 650 °C)</td></tr>
<tr><td>D3
Low dip axial surface fold at Obuasi
S3 crenulation cleavage overprinting S2
Final stage of D2 ?</td><td></td><td rowspan="3">D2, Continuing compression
S2 (NE-SE) fabrics overprint S1 foliation
S2 is defined by aligned muscovite and elongate recrystallised quartz grains

Metamorphism

Syn-tectonic plutonism
(Cape-Coast type granitoids)</td><td rowspan="3">D2/D3, NW-SE shortening
Tarkwaian folds
Strike-slip faults and shearing
Gold mineralizations
Metamorphism (2 - 3 kbar / 200 - 300 °C)</td></tr>
<tr><td rowspan="2">D4, NNW-SSE shortening
Sinistral shear reactivation of D3 thrust
S4 crenulation cleavage ENE-WSW
Greenschist retrograde metamorphism
Remobilization and concentration of gold particle along the shear zones and at the base of Tarkwaian</td><td>D4, NNW-SSE shortening
Hm scale fold at Obuasi</td><td rowspan="2">D2, NNW-SSE shortening
Thrust faults and minor folds</td></tr>
<tr><td>D5 or syn-D4
Sinistral strike-slip faults and shearing
Gold mineralization</td></tr>
<tr><td>D5
Recumbant folds (< m)
Subhorizontal crenulation cleavage
Last pyrite/gold mineralization associated with quartz vein</td><td rowspan="2"></td><td>D3, ESE-WNW shortening
Folds with shallowly dipping axial surfaces and mineralized quartz veins, post-dating the peak of metamorphism</td><td rowspan="2">K-rich plutonism
(cross-cutting all previous structures)</td><td rowspan="2">Late plutonism</td></tr>
<tr><td colspan="2">D6, NE-SW shortening, Panafrican (600 Ma) ??
Low amplitude folds + crenulation cleavage ≈ N320 / 70 (RH)
Reverse faults oriented NW-SE</td><td>D4
Faults oriented NW-SE</td></tr>
</table>

Figure 6-4: Compilation of deformational history of the Ashanti Belt (Perrouty et al, 2012)

6.2.2 Wassa Mine Geology and Mineralization

The Wassa lithological sequence is characterized by lithologies belonging to the Sefwi Group and consisting of intercalated meta-mafic volcanic and meta-diorite dykes with altered meta-mafic volcanic and meta-sediments which are locally characterized as magnetite rich, Banded Iron Formation (“BIF”) like horizons (Bourassa, 2003). The sequence is characterized by the presence of multiple ankerite-quartz veins which are sub-parallel to the main penetrative foliation. The lithological sequence is also characterized by Eoeburnean felsic porphyry intrusions on the south-western flank of the Wassa mine fold.

The first deformational event (D1) at Wassa is of Eoeburnean timing and consists of North-South Shortening. This pre-Tarkwaian event resulted in a penetrative foliation which transposed lithological contacts along this main foliation. Early, gold bearing, synD1 quartz-ankerite veins were also formed during the Eoeburnean event.

The second event of deformation (D2) is an extension period with no local deformation at the mine scale at Wassa. Regionally, this event separates the Eoeburnean and Eburnean orogeny by an extension period of approximately 40 Ma which resulted in the sedimentation of the Birimian and Tarkwaian basins.

The Eburnean orogeny is divided in three distinct deformational events, D3 is a Northwest-Southeast shortening event which resulted in the inversion of regional detachment faults into thrust faults. At the mine scale, this event generated a second penetrative foliation at Wassa and a first phase of Eburnean folding. The D4 deformational event, a North Northwest-South Southeast shortening event resulted in the sinistral reactivation of earlier faults at the regional scale and severely buckled the Wassa stratigraphic sequence into moderately steeply dipping, tight fold patterns (F4 Fold) and a third penetrative foliation (S4).The last deformational event, D5, is the result of sub-vertical compression which resulted in open recumbent folds at Wassa and a fourth foliation located in the axial plane of the F5 folds and is generally sub-horizontal, shallowly plunging to the South. The various phases of Eburnean deformations are illustrated in Figure 6-6.

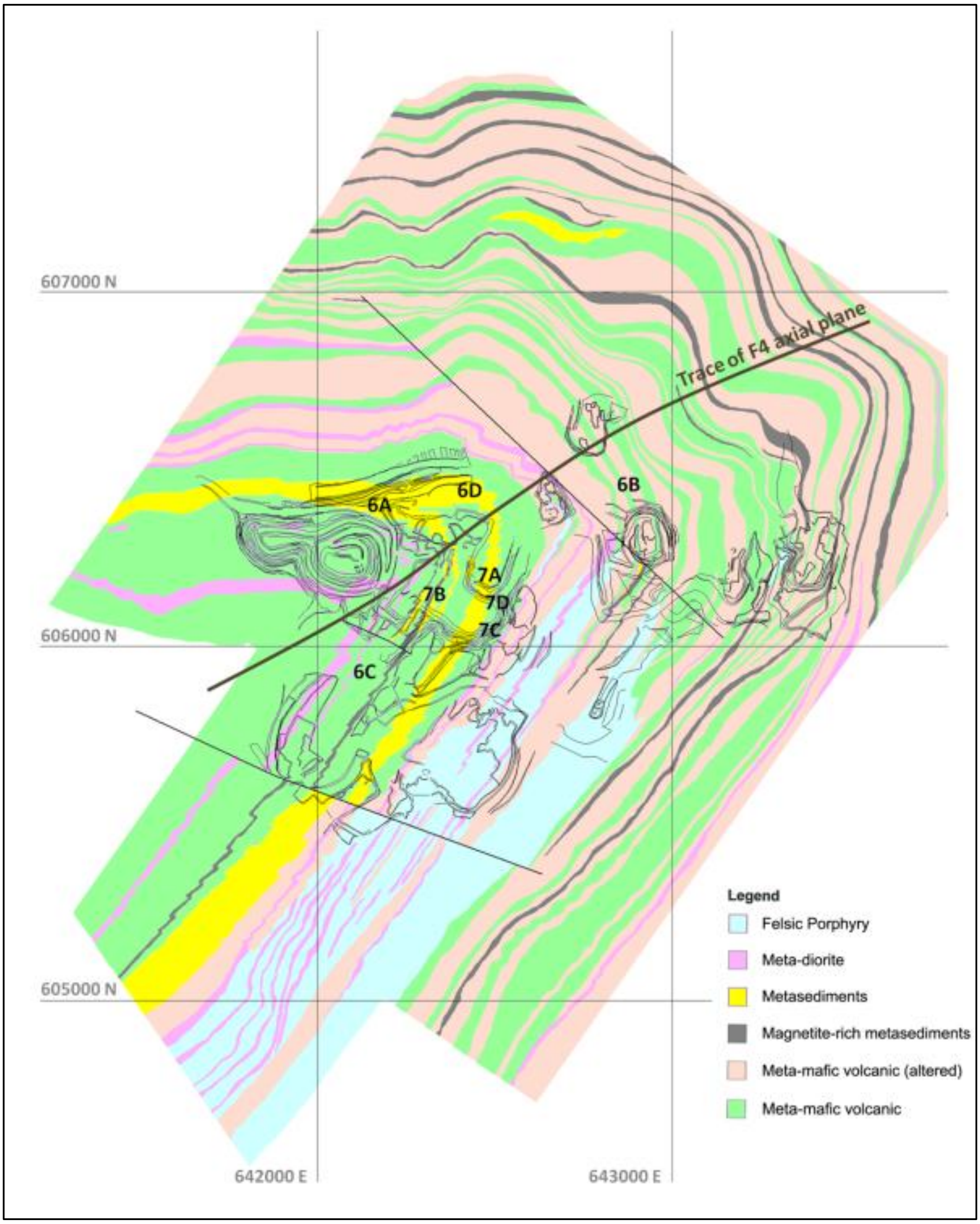


Figure 6-5: Wassa Mine geology (Modified from Bourassa, 2003 and Perrouty et al 2013)

The Wassa mineralization is subdivided into a number of domains, namely; F Shoot, B Shoot, 242, South East, Starter, 419, Mid East and Dead Man's Hill. Each of these represents discontinuous segments of the main mineralized system which extends for approximately 3.5 km along strike from surface and is still open at depth. The SAK deposits are located approximately 2 km to the southwest of the Wassa Main deposit on the northern end of a well-defined mineralized trend parallel to the Wassa Main trend. The mineralization is hosted in highly altered multi-phased greenstone-hosted quartz-carbonate veins interlaced with sedimentary pelitic units. The SAK mineralization is subdivided into a number of domains as well, SAK 1, 2 and 3.

Mineralization within the Wassa Mine is structurally controlled and related to vein densities and sulphide contents. In detail, the mineralization generally consists of broadly tabular zones containing dismembered and folded ribbon-like bodies of narrow quartz vein material, zones are typically 10 m to 50 m wide within a 900 m mineralized corridor. Three vein generations have been distinguished on the basis of structural evidence, vein mineralogy, textures and associated gold grades. Evidence further relates the majority of gold mineralization to the earliest recognized vein generation which is believed to be syn-Eoeburnean. Gold grades broadly correlate with the presence of quartz-dolomite/ankerite-tourmaline bearing quartz veins and the presence of sulphide minerals (predominantly pyrite) within and around the quartz veins. Gold grades appear to be spatially restricted to the quartz veins, vein selvages and the immediate wall rocks. The alteration haloes developed around the veins and pervasively developed within the core of the Wassa Fold contain lower grade mineralization. The combined and overprinted Eburnean deformational events (D3 to D5) render precise prediction of the vein geometries and localities difficult in areas remote from drillhole data. While the general peripheries of the mineralized zones can be defined reasonably well with the drillhole data, the internal geometry is difficult to resolve with confidence.

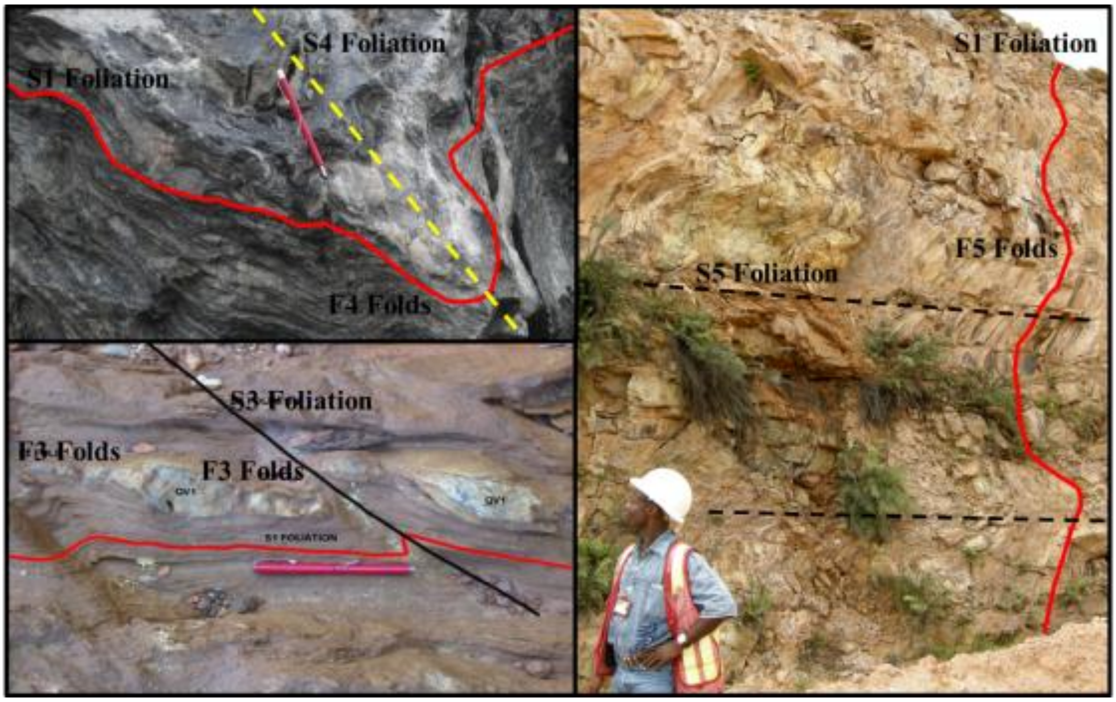


Figure 6-6: Various Eburnean fold and foliation generations from the Wassa Mine, top left are syn-D4 structural features from the Starter pit, bottom left are syn-D3 structural features from Dead Man Hill pit and on the right, syn-D5 structural features from the South-East pit

7 DEPOSIT TYPES (ITEM 8)

The Wassa deposit is located on the eastern flank of the northeast trending Ashanti Belt, a Paleoproterozoic greenstone belt which was formed and deformed, along with the dividing Birimian and Tarkwaian sedimentary basins during the Eoeburnean and Eburnean orogeny. Most deposits found within the Ashanti belt can be classified as lode gold deposits or orogenic mesothermal gold deposits, with the exception of the Tarkwaian paleoplacer deposits which have a sedimentary origin. Orogenic gold deposits are the most common gold systems found within Archean and Paleoproterozoic terrains, in the West African shield, these deposits are typically underlain by geology considered to be of Eburnean age and are generally hosted by volcano-sedimentary sequences.

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 sulphide-rich replacement zones.

The Ashanti belt is considered prospective for orogenic mesothermal gold deposits and hosts numerous lode gold deposits and paleoplacer deposits. As illustrated by Figure 7-1, several major gold deposits are found within the Ashanti belt which can be classified into six different deposit types:

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

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

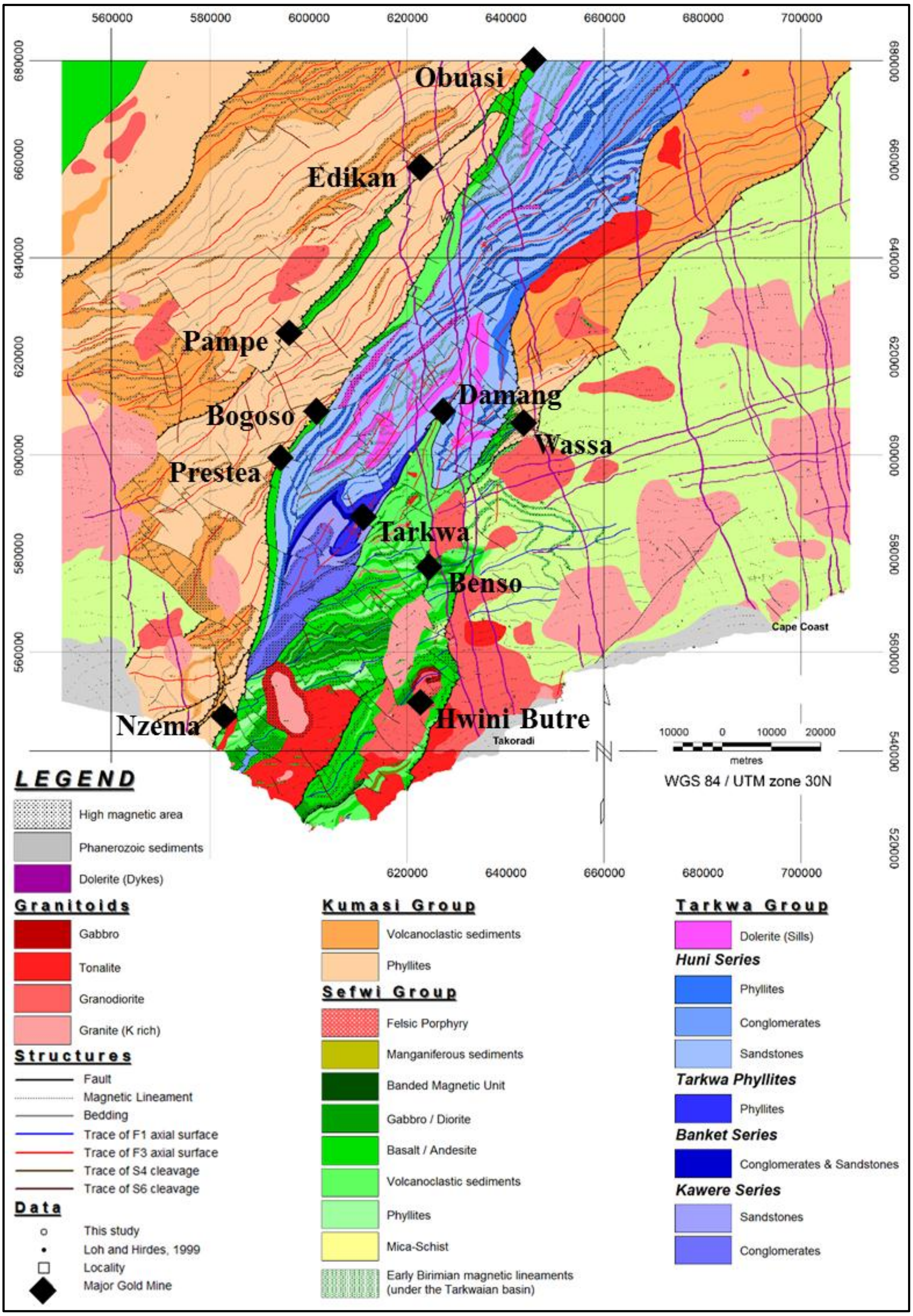


Figure 7-1: Geology of the Ashanti belt with location of major gold deposits. (Modified from Perrouty et al, 2012)

The second type of deposit found within the Ashanti belt are laminated quartz vein deposits containing free gold. Fault filled quartz vein deposits also occur along the Ashanti trend but are only present at Obuasi and Prestea. The third type of deposit are paleo-placer deposits within the Tarkwaian sedimentary basin which are hosted within narrow conglomerate horizons intercalated with sandstone units characterized by iron oxides cross beddings. Paleoplacer deposits occur in the southern portion of the Tarkwa basin and examples include Tarkwa, Teberebie and Iduaprim. The fourth type of deposit found within the Ashanti belt are intrusive hosted deposits which occur along second order structures such as the Akropong trend in the Kumasi basin and the Manso trend in the Southern portion of the Ashanti belt. These deposits can be hosted both within felsic and mafic intrusives and are characterized by a penetrative fabric where gold is associated with pyrite and arsenopyrite. Examples of such deposits include Edikan and Pampe along the Akropong trend and Benso and Hwini Butre along the Manso trend. The fifth type of deposit found within the Ashanti belt is late thrust fault associated quartz vein deposits. The Damang mine which is located just west of Wassa is the only known thrust fault related deposit in the Ashanti belt. The deposit is characterized by low angle; undeformed extensional and tensional veins associated with low angle thrust Faults. This type of deposit contrasts with the last type of deposit found with the belt, the multi-phase folded Wassa vein deposit. The Wassa mineralization consists of greenstone-hosted, low sulphide hydrothermal deposits where gold mineralization occurs within folded quartz-carbonate veins, as illustrated in Figure 7-2. The Wassa deposit can therefore be classified as an Eoeburnean folded vein system and is the only such deposit recognized to date within the Ashanti belt.

Host rocks in the Wassa mine area have been affected by at least four phases of ductile deformation, producing a polyphase fold pattern at the mine scale. Discrete high-strain zones locally dissect this fold system. The structural history of the Wassa area is important in that the various deformational events have been responsible for the emplacement of the gold mineralization as well as the geometry of the zones themselves. Mineralized zones at the Wassa mine are related to vein swarms and associated sulphides that formed during the Eoeburnean deformational event. All rock types underlying the Wassa Mine appear to be altered to variable degrees with the most common alteration consisting of a carbonate-silica-sulphide assemblage.

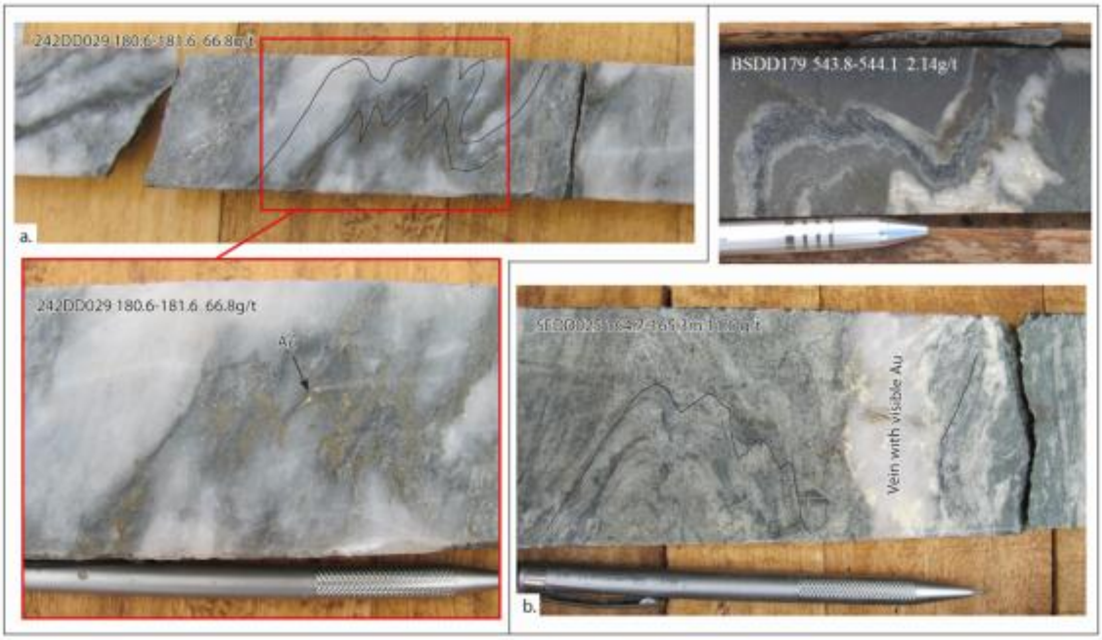


Figure 7-2: Syn-Eoeburnean veins from the B-Shoot, 242 and South-East zones at the Wassa Mine refolded by Eburnean deformational events. (Modified from Perrouty et al, 2013)

8 EXPLORATION (ITEM 9)

8.1 Introduction

In addition to the drilling described in Section 9, extensive exploration work has been conducted on and around the Wassa concession. Previously, several airborne and ground geophysical surveys consisting of aero-magnetics, radiometrics and Induced Polarization ("IP") were conducted on the properties. The geophysical surveys targeted geochemical anomalies, which had previously been identified following multiple stream and soil geochemical sampling programs.

8.2 Wassa

Modern exploration programs on the Wassa concession began in the early 1990s with satellite imagery and geophysical surveys which identified geophysical lineaments and anomalies over small scale and colonial mining areas. Stream and soil geochemistry sampling programs were conducted over the geophysical anomalies and identified two linear gold in-soil anomalies as illustrated in Figure 8-1.

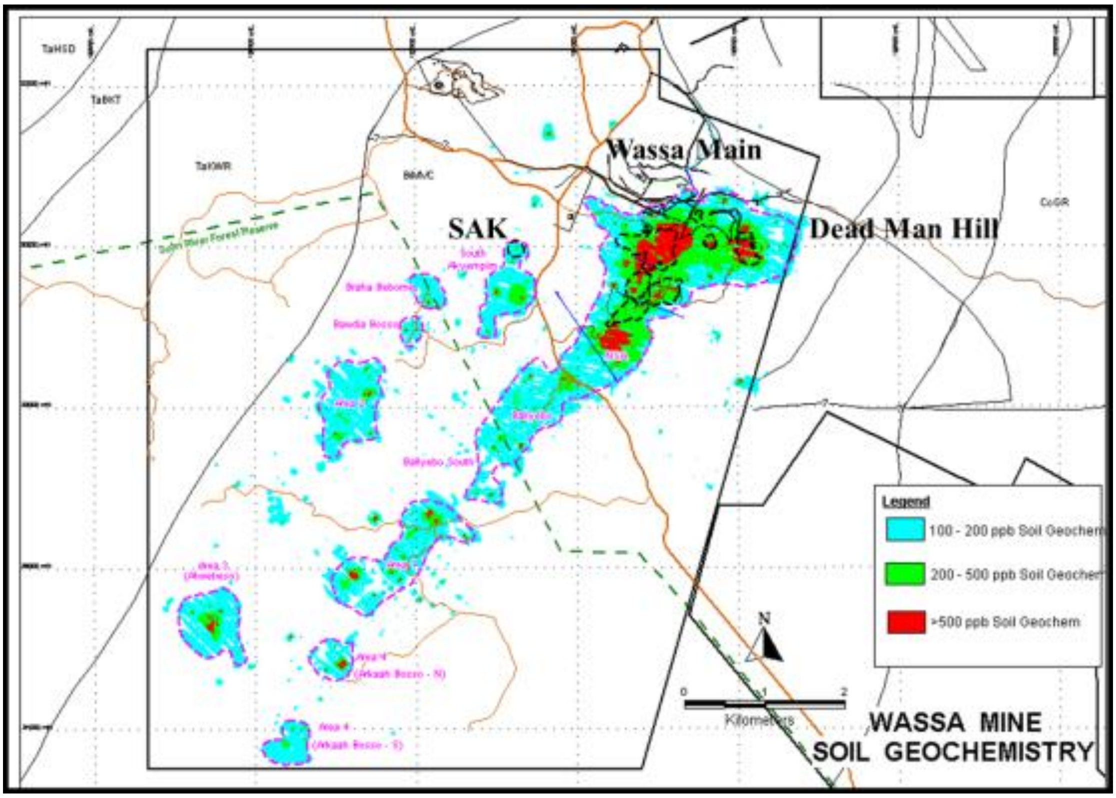


Figure 8-1: Wassa Mine soil geochemistry anomalies.

Exploration drilling commenced in February 1994, and by March 1997, a total of 58,709 m of reverse circulation and diamond drilling had been completed. In September 1997, consulting engineers Pincock, Allen and Holt completed a FS. Only minimal exploration work was conducted by Satellite Goldfields between the completion of the FS in 1997 and the 2001 bankruptcy.

In March 2002, GSR started an exploration program as part of a due diligence exercise following the ratification of a confidentiality agreement with the creditor of Satellite Goldfields. The exploration program consisted mainly of pit mapping and drilling below the pits to test the continuity of mineralization at depth The concession was acquired later that year by GSR following the completion of the due diligence exercise. Exploration drilling resumed in

November 2002 under GSR with the aim to increase the quoted reserves and resources for the feasibility study, which was completed in 2003.

Simultaneously to the resource drilling program that targeted resource increases in the pit areas, GSR also undertook grass roots exploration along two previously identified mineralized trends. The 419 area was located south of the main pits and the South-Akyempim anomaly was a soil target that had never been previously drilled and was located west of the main pits. Deep auger campaigns were also undertaken in the Subri forest reserve, which is located in the southern portion of the Wassa Mining lease.

In March and April 2004, a high resolution, helicopter geophysical survey was carried out over the Wassa Mining Lease and surrounding Prospecting and Reconnaissance Licenses (Figure 8-2). Five different survey types were conducted, namely: Electromagnetic, Resistivity, Magnetic, Radiometric and Magnetic Horizontal Gradient. The surveys consisted of 9,085 km of flown lines covering a total areal of 450 km^2. Flight lines were flown at various line spacing varying between 50 to 100 m depending on the survey type. The geophysical surveys identified several anomalies with targets being prioritized on the basis of supporting geochemical and geological evidences.

The exploration program in 2005 continued to focus on drill testing anomalies identified by the airborne geophysical survey as well as infill drilling within the pit area to expand the reserve and resource base. The resource definition drilling program focused mainly on South-Akyempim, South-East and the 419 area. The following years were subject to more infill and resource definition drilling in the pit areas at Wassa until the 2011 exploration program was undertaken, at which point a shift toward drilling deep high grade targets below the pits became the main focus of the exploration programs and remain the priority for the 2013 and 2014 programs.

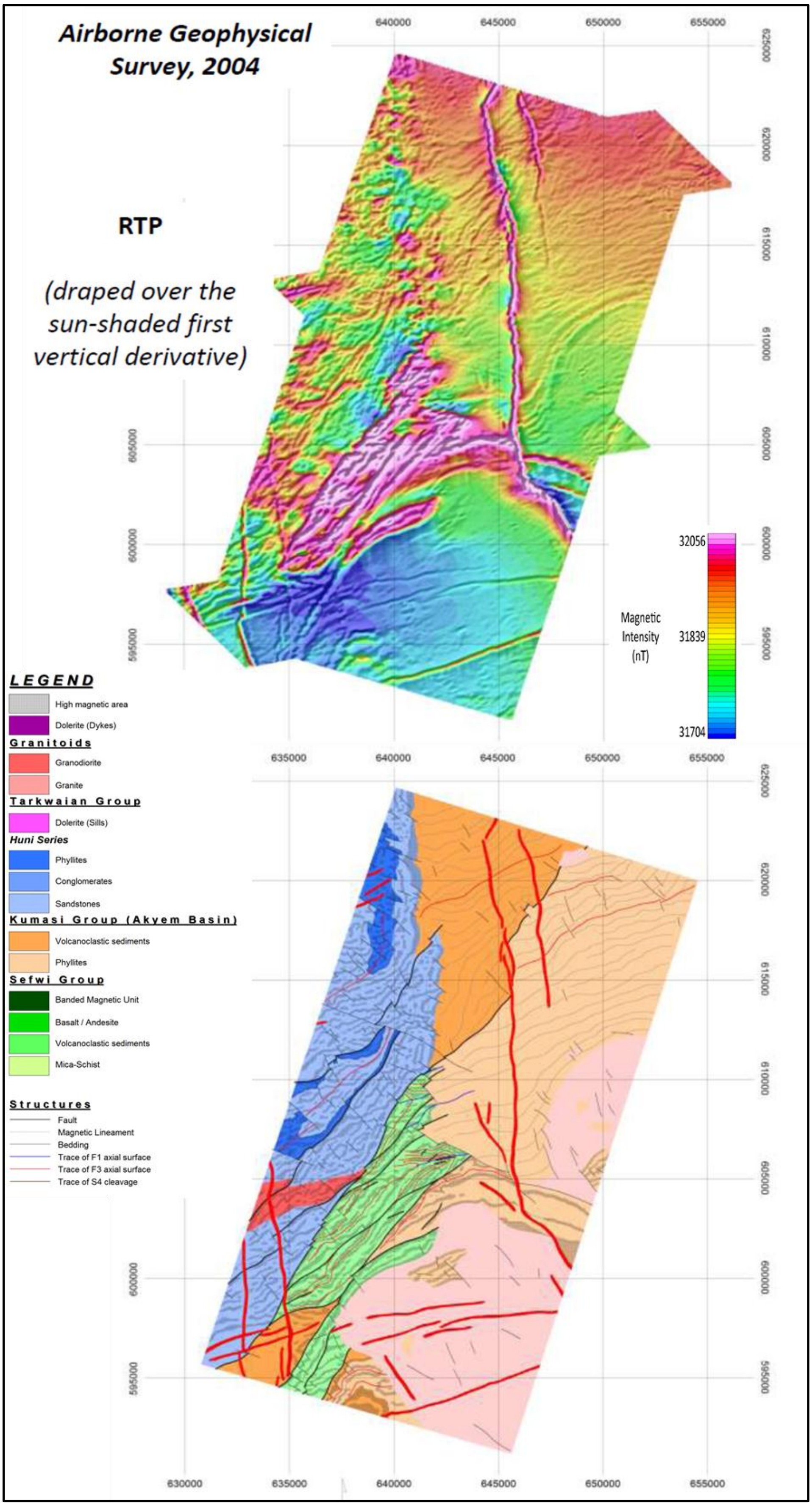


Figure 8-2: Wassa airborne magnetic interpretation. (Modified from Perrouty et al, 2014, unpublished draft)

9 DRILLING (ITEM 10)

9.1 Drilling

Drilling is carried out by a combination of Diamond Drill (“DD”), Reverse Circulation (“RC”) and Rotary Air Blast (“RAB”) techniques. In general the RAB method is used at early stages for follow up to soil geochemical sampling and during production for testing contacts and mineralization extensions around the production areas and has a maximum drilling depth of around 30 m. The 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 100 to 125 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.

RC and DD drilling were carried out in double shifts and during every shift a GSR geologist was on site to align the drill rig and check the drill head dip and azimuth. Downhole surveying was conducted using a Single Shot Camera (“SSC”), for RC and DD holes at the bottom of holes exceeding 30 m depths and then taken progressively every 30 m up hole. The SSC recorded the dip and azimuth for each of the surveys on a film image, this image was validated and recorded by the GSR geologists or was recorded by a Reflex survey instrument and captured in the database as well as being filed in the respective drillhole file folders on site.

Drilling depths at Wassa Main have generally been less than 250 m but with the discovery of higher grades below the Wassa Main Pit in late 2011 hole depths have increased. In the 1st half of 2014, two gyro survey instruments were utilized to resurvey several of the deeper holes. In total 153 holes, drilled during 2012 to 2014 were resurveyed. The gyro survey readings were conducted every 10 m both in and out of the hole and the values were then averaged. The 153 gyro surveyed holes were updated in the database and subsequently used for the resource estimates. The gyro surveys showed that there was some deviation in the holes below 250 m drilled depth. Deviations varied from location to location depending on drill orientation with a general tendency for the hole to steepen and swing to the north.

A summary of the exploration data used in the Mineral Resource models is given in Table 9-1.

Table 9-1: Summary of exploration data used for the Mineral Resource models

Location	Type	Number of Holes	Meterage (m)
Wassa	RC	2,398	208,756
	DD	814	212,772

All of the drillhole collars were surveyed using a Nikon Total Station (DTM-332) or Sokkia Total Station by a GSR surveyor. Individual RC and DD holes have been identified and marked in the field with Polyvinyl chloride (“PVC”) pipes. RAB drill holes have been also surveyed in the field and identified and marked with wooden pegs.

9.2 Sampling

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 1m lengths. The core is cut according to mineralization, 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 established to allow a minimum of at least two composites per drillhole 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 mineralization. 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);
- The composite samples are taken using tube sampling, which uses a 50 mm diameter PVC tube which has been 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. The “shuffling” approach ensures 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 mineralized 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.

10 SAMPLE PREPARATION, ANALYSES, AND SECURITY (ITEM 11)

10.1 Sample Preparation

Sample preparation on site is restricted to core logging and splitting. The facilities consist of enclosed core and coarse reject storage facilities, covered logging sheds and areas for the splitting of RC and RAB samples. Sub-sampling of RC and RAB samples is carried out using a Jones Riffle splitter.

10.2 Sample Despatch and Security

Samples are collated at the mine site after splitting and then transported to the primary laboratory for the completion of the sample preparation and chemical analysis. Exploration samples are trucked by road to the laboratories in Tarkwa.

Sample security involves two aspects, namely maintaining the chain of custody of samples to prevent inadvertent contamination or mixing of samples, and rendering active tampering of samples as difficult as possible.

The transport of samples from site to the laboratory is by road using a truck dispatched from the laboratory. As the samples are loaded they are checked and the sample numbers are validated. The sample dispatch forms are signed off by the driver and a company representative. The sample dispatch dates are recorded in the sample database as well as the date when results are received.

No specific security safeguards have been put in place by GSR to maintain the chain of custody during the transfer of core between drilling sites, the core library, and sample preparation and assaying facilities. Core and rejects from the sample preparation are archived in secure facilities at the core yard and remain available for future testing.

10.3 Laboratory Procedures

Sample assays are then performed at either SGS Laboratories in Tarkwa (“SGS”) or Transworld (now Intertek) Laboratories (“TWL”) which is also based in Tarkwa. GSR has used both laboratories and regularly submits quality control samples to each for testing purposes. Both laboratories are independent of GSR and are currently in the process of accreditation for international certification for testing and analysis.

The sample preparation and analysis processes at Wassa Site Laboratory (“WSL”), TWL and SGS differ slightly. WSL was used as the primary laboratory for 3 m composite and grade control RC drill samples from July 2007 onwards. The laboratory had previously operated as a metallurgical sample processing laboratory at the Wassa mine site.

The sample preparation and analysis process at WSL is as follows:

- Sample reception, sorting, labelling and loading;
- Dry entire sample (3 kg) at 110 to 140°C for between 4 and 8 hours;
- Jaw crush entire sample to 3 mm, and secondary keegor crusher to 1 mm;
- Split 3 kg sample and pulverize for 3 to 8 minutes to 95% passing 75 µm;
- Sample homogenisation using a mat rolling technique, and sub-sample 1kg into BLEG roll bottle;
- Bottle roll for 6 hours with LeachWell™ accelerant. Allow to settle for 30 to 60 minutes;
- Filter 20 ml aliquot from bottle;
- DIBK extraction and AAS determination of gold content; and
- 1 in 10 residue samples are retained for gold determination using fire assay.

TWL was the primary laboratory for samples until July 2007, when it was discontinued due to the following issues:

- Contamination due to poor dust control in pulverizing area of the laboratory. Use of dust attracting cloth gloves for sample handling. BLEG aliquot preparation area containing dirt and liquids, which may result in sample cross-contamination.
- Large fluctuation in employee numbers (60 to 180), which resulted in a risk of training and quality control issues when increasing employment numbers over a short period of time.
- The use of a manual data tracking and capture system, which increased risk of data entry errors. GSR considered this to be a sub-optimal process for a commercial laboratory.
- The sample preparation and analysis process used for all samples submitted to TWL is illustrated in Figure 10-1.

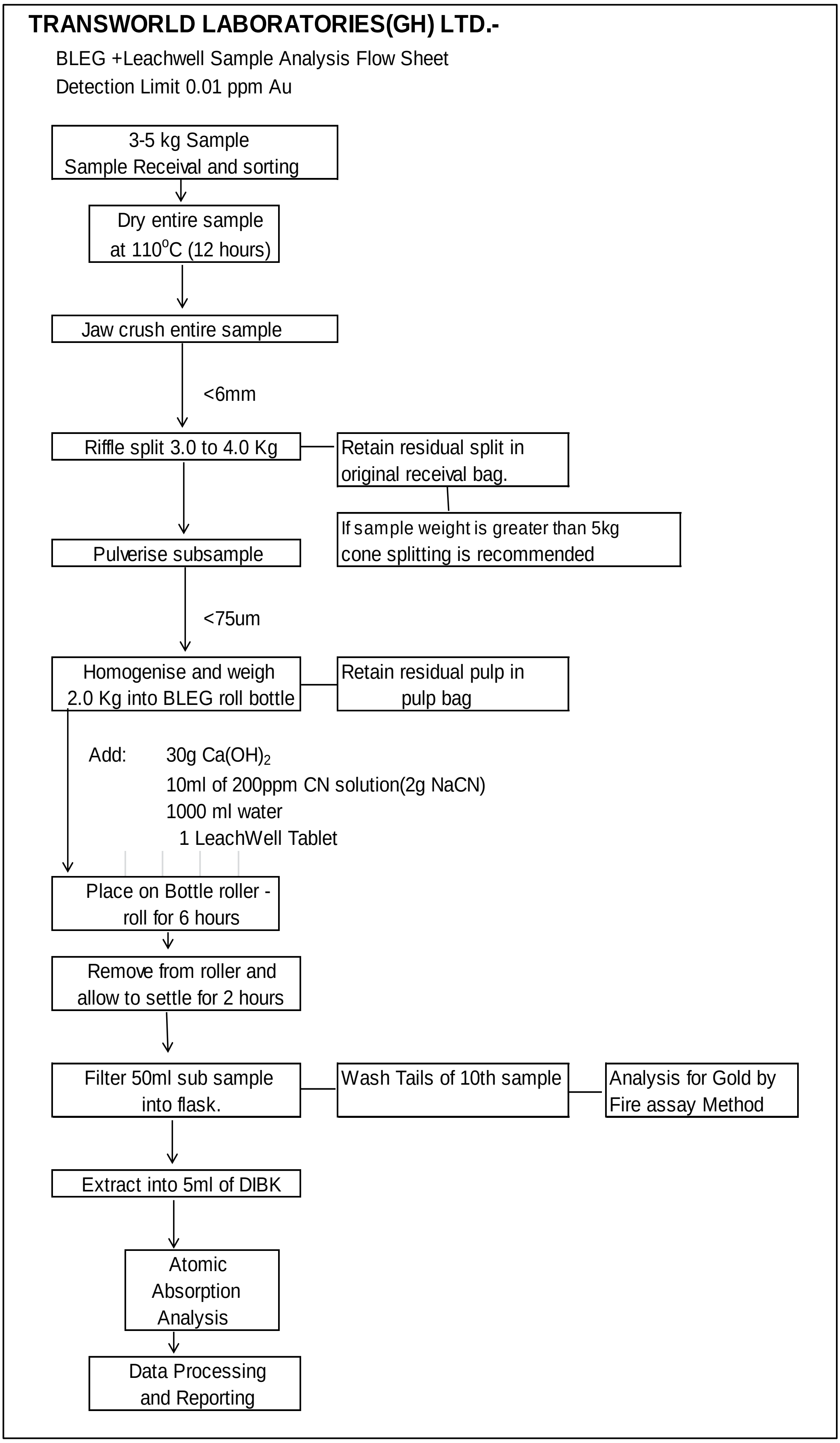


Figure 10-1: TWL sample processing flowsheet

The SGS laboratory in Tarkwa has been used for exploration samples since July 2007 with the sample preparation and analysis process as follows:

- Sample received, entered in LIMS, worksheets printed and samples sorted;
- Samples emptied into aluminium dishes;
- Dry entire sample at between 105 and 110°C for 8 hours;
- Jaw crush entire sample to 6 mm;
- Split sample using a single stage riffle splitter, to result in a 1.5 kg sub-sample;
- Pulverise sub-sample for 3 to 5 minutes, to give 90% passing 75 µm;
- Sample homogenisation using a mat rolling technique, and put 1 kg of sample into the BLEG roll bottle;
- The remainder of the sample is retained as pulp and crushed sample duplicates;
- Bottle roll for 12 hours with LeachWell™ accelerant. Allow to settle for 2 hours;
- Filter 50 ml of aliquot; and
- Diisobutyl Ketone (“DIBK”) and Atomic absorption spectroscopy (“AAS”) for gold grade determination.

The measures implemented by GSR are considered to be consistent with industry best practice.

10.4 Quality Control and Quality Assurance (“QAQC”) Procedures

Quality control measures are typically set in place to ensure the reliability and trustworthiness of exploration data, and to ensure that it is of sufficient quality for inclusion in the subsequent Mineral Resource estimates. Quality control measures include written field procedures and independent verifications of aspects such as drilling, surveying, sampling and assaying, data management and database integrity. Appropriate documentation of quality control measures and analysis of quality control data are an integral component of a comprehensive quality assurance program and an important safeguard of project data.

The field procedures implemented by GSR are comprehensive and cover all aspects of the data collection process such as surveying, drilling, core and reverse circulation cuttings handling, description, sampling and database creation and management. At Wassa, each task is conducted by appropriately qualified personnel under the direct supervision of a qualified geologist. The measures implemented by GSR are considered to be consistent with industry best practice.

The quality control employed by GSR to verify the results obtained from the laboratories takes the form of the following types of check sample:

- Field Duplicates to check sampling precision and deposit variability. Two separate samples are collected at the drill site and bagged separately from which two individual samples are produced. The results of these checks can be useful in highlighting natural variability of the grade distribution.
- Pulp Duplicates as a check of sampling precision and coarse gold in pulps. Two separate pulp samples are prepared from a single coarse reject after sample splitting and on site preparation. The results are useful in indicating problems with sample preparation and splitting.
- Repeats as a check of analytical precision and coarse gold. Two separate aliquots are prepared from separate samples taken from the original coarse reject and the two samples are then checked against one another.
- Blanks for highlighting contamination problems and also cross labelling when samples

are mislabelled in the laboratory.

- Standards as a check of analytical precision and accuracy.

10.5 Specific Gravity Data

Specific gravity ("SG") determinations were carried out by GSR. SG is measured on representative core samples from each drill run. This ensures representative specific gravity data across all rock types irrespective of gold grade.

SG is measured at the core facility using a water immersion method. Each sample is weighed in air, then coated in wax and weighed in air and immersed in water. A total of 606 determinations were collected on core samples.

The water immersion methodology is considered by SRK to provide accurate estimates of variations in bulk specific gravity throughout the Wassa gold deposits. After testing each sample is carefully replaced at its original location in the core box. SRK examined core from several boreholes and no misplaced core pieces were identified.

Samples were selected from all the different lithologies intersected in the core of all the available drill holes. The sampling procedure was guided by pit location, lithology, depth, quartz contents (in oxide) and the oxidation state. A total of nineteen holes from Deadman's Hill, South East; Starter; 419, 242, B-shoot and F-Shoot were selected with the results presented in Table 10-1.

Table 10-1: Results of Specific Gravity

Material	# Samples	SG Value (g/cm^3)	Standard Error
Oxide	213	1.8	2%
Transition	42	2.19	3%
Fresh	327	2.7	1%
Quartz Vein	24	2.56	1%

Another 13 samples consisting of oxide (9), trans (1), fresh (2) and quartz (1) were sent to the Western University College (WUC, Tarkwa) as independent checks. The average results were 1.76, 2.29, 2.73 and 2.59g/cm^3 respectively.

The specific gravity determinations are considered accurate as the reconciliations between the mined tonnages and those estimated from the resource models reconcile well.

11 DATA VERIFICATION (ITEM 12)

11.1 Introduction

SRK has not carried out any independent collection and verification of individual samples or assay results. SRK has, however, obtained and reviewed the Quality Assurance Quality Control ("QAQC") results produced by GSR, its consultants and the laboratories themselves.

SRK has reviewed the core and samples available on site and cross-checked them against the geological logs and assay records.

The quality of the results is generally considered good. GSR frequently sends "blind" test samples to the laboratory and monthly batch results are analysed and any anomalous results are queried immediately. A small number of anomalous and/or poor results have been noted over the years, but these have been identified and the reasons fall into two main categories, namely:

- Mislabelling of individual samples, standards and blanks.
- Individual batch issues corresponding to changes in the laboratory setup or calibration, in these cases re-assay has been carried out.

11.2 Data verification by GSR

The field procedures implemented by GSR involve several steps designed to verify the collection of exploration data and minimize the potential for inadvertent data entry errors. The data entry and database management involves two steps punctuated by validation steps by the logging geologist. All data is thoroughly checked prior to the incorporation into the project GEMS database.

Analytical data is also routinely checked for consistency by GSR personnel. Upon reception of digital assay certificates, assay results together with the control samples are extracted from the certificates and imported into the Acquire database. Failures and potential failures are examined and depending on the nature of the failure, re-assaying was requested from the primary laboratory. Analysis of quality control data is documented along with relevant comments or actions undertaken to either investigate or mitigate problematic control samples.

11.3 Analytical QAQC

11.3.1 Introduction

SRK has reviewed the supplied QAQC reports, and a summary of the historical and current QAQC results is included here.

QAQC data for the first half of 2014 and all of 2013 have also been reviewed as this data has been included in the recent resource model update used for this study.

11.3.2 Comparison of assay methodologies

In the February 2003, SRK demonstrated that sample results prior to this date, when assayed using a 50 g fire assay resulted in poor reproducibility between field duplicates. This effect was also evident between pulp duplicates, although not as marked. The conclusion of the analyses of the quality control data available then was that a component of coarse gold present in the samples was contributing to poor reproducibility and that an analytical process that makes use of significantly larger aliquots, such as LeachWell™ assays should be considered.

To address this, GSR now assays using a 1 kg BLEG assay, with a LeachWell™ accelerant. The gold grade is determined using an AAS finish. Initially, the sample splitting was completed using a rotary splitter and a 6 hour leach was used. Following analysis of the leach tailings, the leach time has been extended from 6 to 12 hours. SRK also understand that due to time constraints, the use of the rotary splitter has been discontinued and a Jones Riffle has been used to split sub-samples from the larger RC drillhole samples. The difference between the fire assay and larger BLEG assays are illustrated in Figure 11-1.

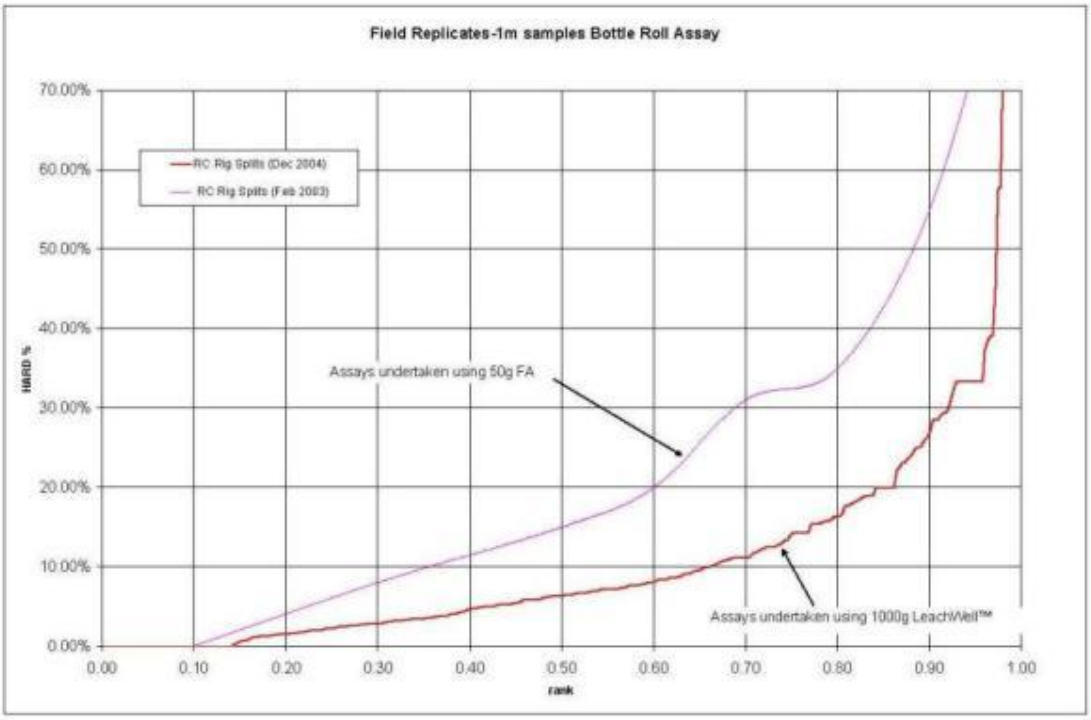


Figure 11-1: HARD plot comparing fire assay and BLEG, for field duplicates

Figure 11-1 shows a significant improvement with respect to sample reproducibility between the fire assay and BLEG methodologies. Using BLEG, 80% of pairs report HARD precisions of less than 17%, compared to the 35% precision attributable to the fire assay method. SRK recommend that GSR continue to monitor the reproducibility of the sample grades from paired data analysis.

11.3.3 Field Duplicates

Historical data on field duplicates, from between 2003 and 2007 were poorly correlated. Supplied documentation from the time indicates that the field sampling techniques were identified as the likely cause. Improvements since 2008 are thought to be due to increased sample splitting training and awareness amongst GSR sampling crew members, rather than improvements at the laboratory. The correlation plots for both periods, including Q1 and Q2, 2013, are included as Figure 11-2 to Figure 11-4.

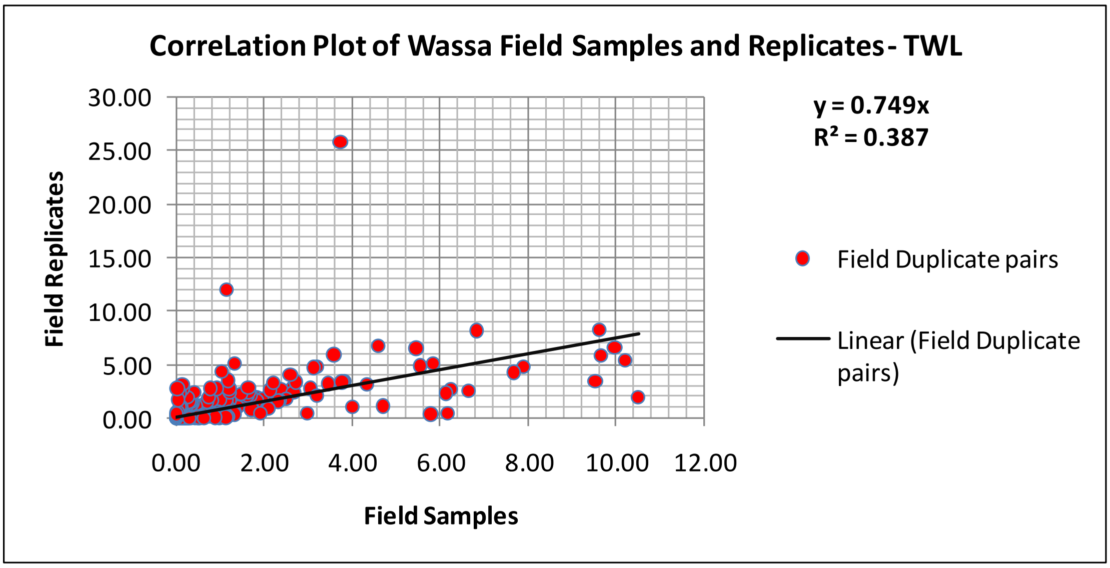


Figure 11-2: Correlation of field duplicates (2003 to 2007) from TWL

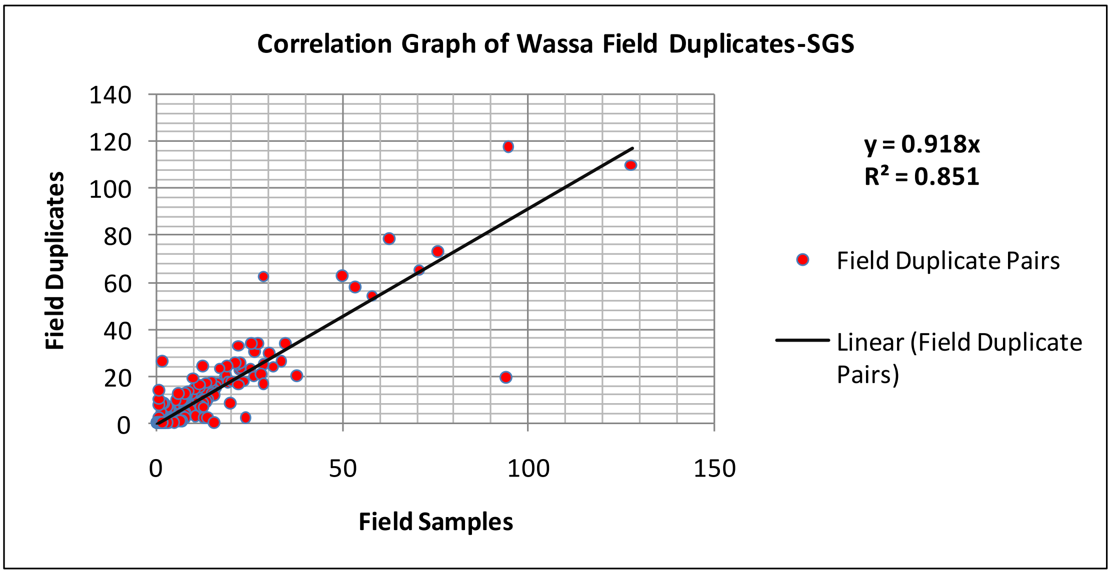


Figure 11-3: Correlation of field duplicates (2008 to 2012) from SGS

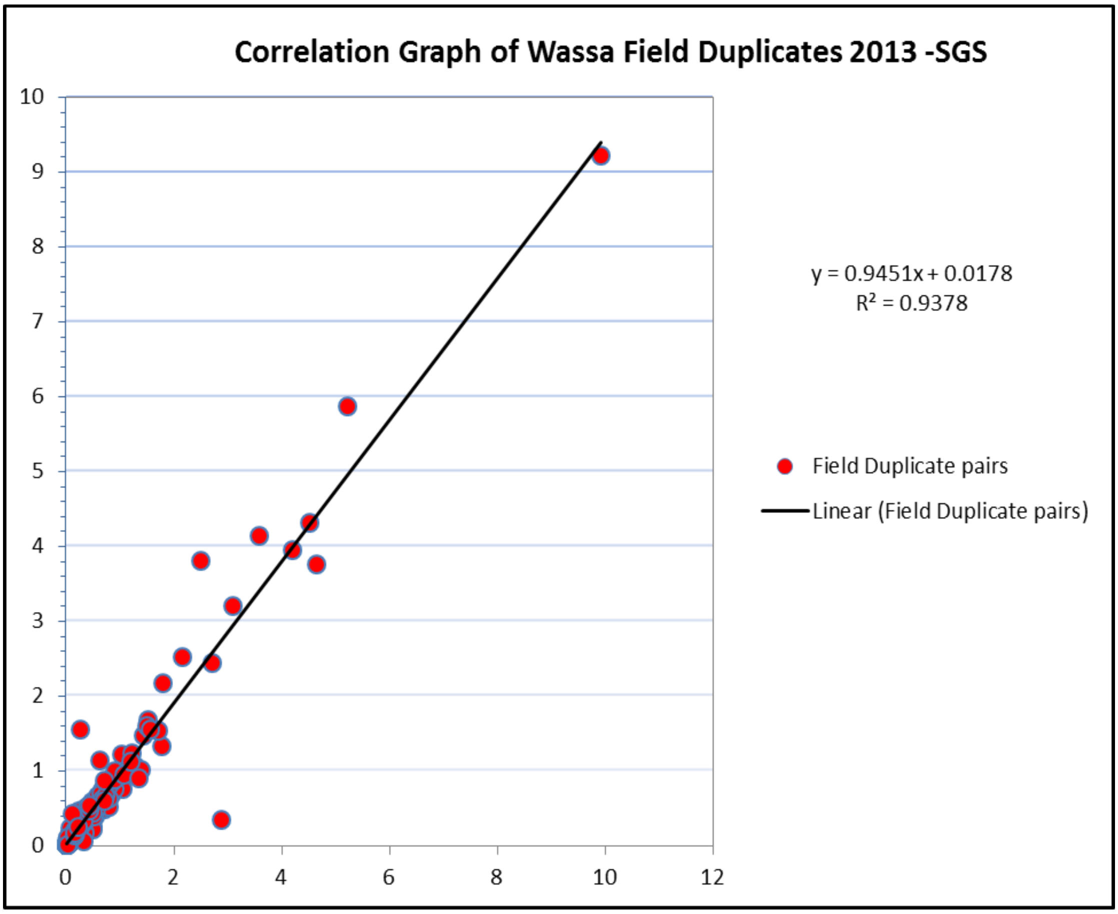


Figure 11-4: Correlation of field duplicates (2013) from SGS

11.3.4 Laboratory (Pulp) Duplicates

Laboratory duplicates pair analysis for all samples generated between 2003 and 2007, from TWL, and from 2008 to 2012 for SGS are shown in Figure 11-5 and Figure 11-6 respectively. Assay results from the earlier samples submitted to TWL show relatively low gold grade reproducibility. There was a significant improvement in assay grade reproducibility after the move to SGS.

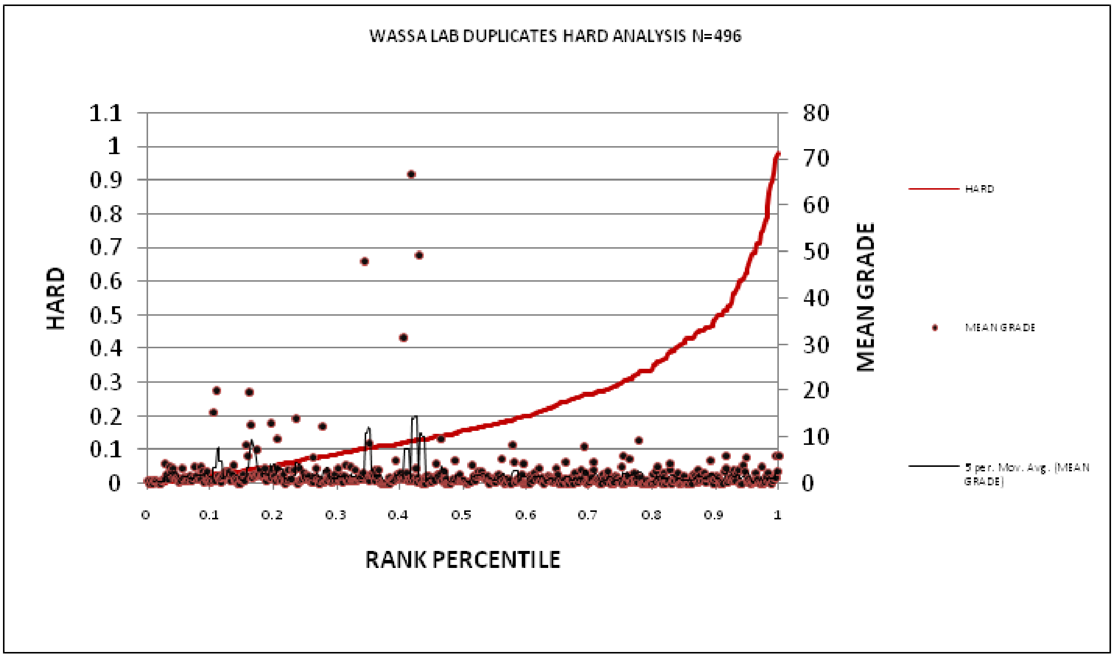


Figure 11-5: HARD plot of laboratory duplicates (2003 to 2007) from TWL

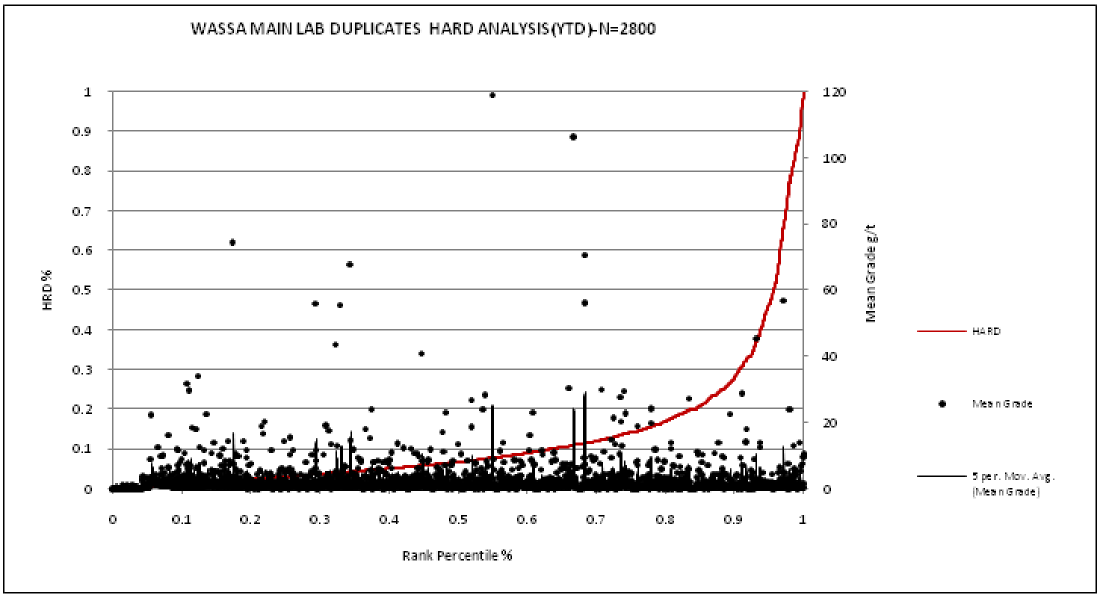


Figure 11-6: HARD plot of laboratory duplicates (2008 to 2012) from SGS

11.3.5 Laboratory (Coarse) Duplicates

SRK understands that the Company is no longer submitting laboratory pulp duplicates for analysis. Instead, coarse reject sample (from the laboratory sample split) is re-numbered and re-submitted to the laboratory for repeat analysis. The coarse duplicates are intended to monitor the sample preparation section of the laboratory.

The HARD plot of all coarse rejects for 2013, is presented in Figure 11-7. The results of this HARD analysis show that approximately 80% of the 1,264 coarse duplicate samples fall with approximately 20% error. This is considered by SRK to be acceptable for a gold deposit.

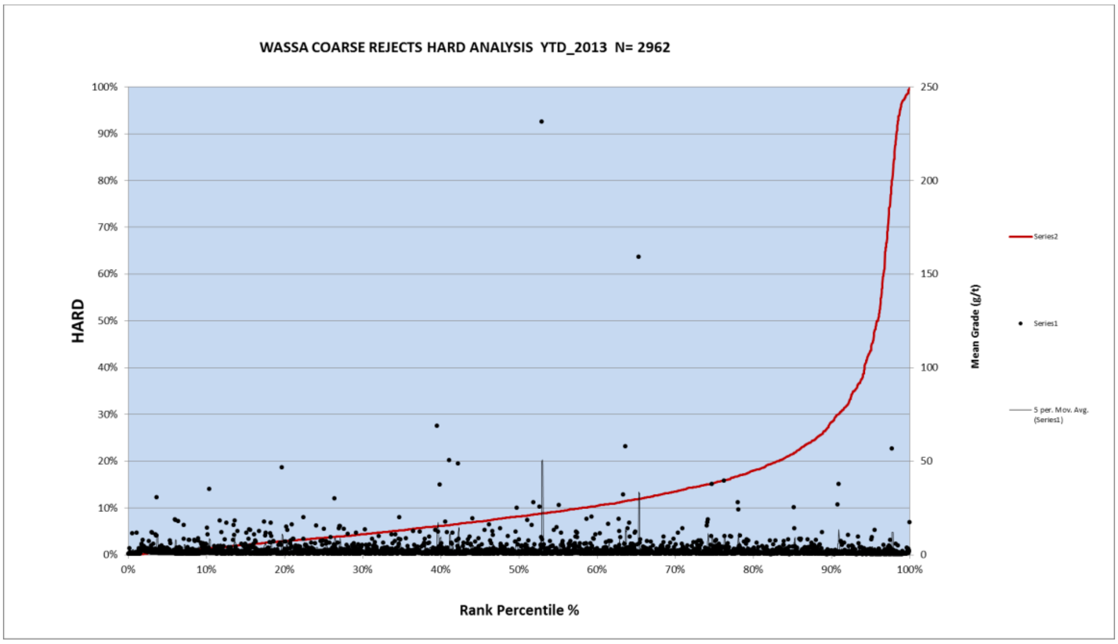


Figure 11-7: HARD plot of all coarse rejects (2013) from SGS

The HARD plot of all coarse rejects for Q1 and Q2 (2014) are presented in Figure 11-8. The results of this HARD analysis show that approximately 80% of the 1,878 coarse duplicate samples fall with approximately 20% error. This is considered by SRK to be acceptable for a gold deposit.

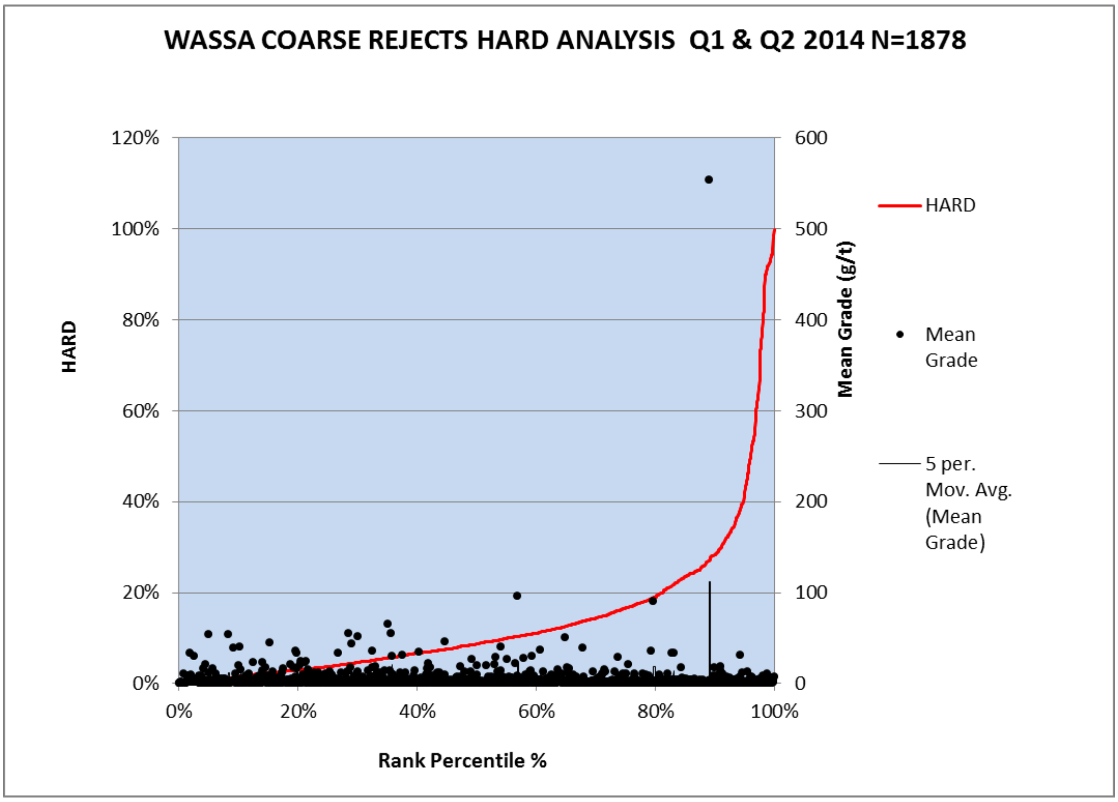


Figure 11-8: HARD plot of all coarse rejects (Q1 and Q2, 2014) from SGS

11.3.6 Certified Reference Material ("CRM")

CRM material was introduced by GSR into the sample stream to monitor the accuracy, precision and reproducibility of the assay results. CRM materials were sourced from Geostats Pty Ltd ("Geostats"), and from Gannet. Although the CRM material could be easily identified by the laboratory, the actual grade of the standard would be difficult to determine due to the large number of different standards used. Standards in use between 2003 and 2007 are shown in Table 11-1.

Table 11-1: CRM for 2003 to 2007 time period (TWL)

Standard	Certified Mean (g/t Au)	Number Samples Submitted	Mean Assay Grade (g/t Au)	Laboratory Bias (%)
Gannet A	0.22	196	0.22	0%
Gannet B	2.52	185	2.57	2%
Gannet C	3.46	21	3.53	2%
Gannet D	3.40	75	3.40	0%
Gannet E	2.36	77	2.45	4%
Gannet F	0.78	47	0.75	-4%
Gannet G	3.22	82	3.02	-6%
Gannet M	1.18	159	1.28	+2%
Gannet N	0.50	171	0.49	-2%

CRM material in use between 2008 and the end of Quarter 2, 2014 were supplied by both Geostats and Gannet, and are shown in Table 11-2 to Table 11-5.

Table 11-2: Geostats CRM for 2008 to 2012 time period (SGS)

Standard	Certified Mean (g/t Au)	Number Samples Submitted	Mean Assay Grade (g/t Au)	Laboratory Bias (%)
G901-10	0.48	82	0.51	6%
G305-3	0.71	14	0.66	-7%
G901-2	1.70	32	1.54	-9%
G906-4	1.90	137	1.99	5%
G999-4	2.30	36	2.40	4%
G302-2	2.44	70	2.50	2%
G901-1	2.50	38	2.38	-5%
G396-9	2.60	29	2.39	-8%
G900-7	3.19	193	3.22	1%

Table 11-3: Gannet CRM for 2008 to 2012 time period (SGS)

Standard	Certified Mean (g/t Au)	Number Samples Submitted	Mean Assay Grade (g/t Au)	Laboratory Bias (%)
ST 07/9453	0.21	476	0.21	2%
ST 14/9501	0.43	447	0.42	-3%
ST 16/9487	0.49	110	0.51	3%
ST 16/5357	0.52	654	0.52	0%
ST 486	0.57	124	0.54	-5%
ST 17/2290	0.78	14	0.79	2%
ST 481	1.02	32	1.05	3%
ST 06/5356	1.04	115	1.06	2%
ST322	1.04	18	1.07	3%
ST 06/7384	1.08	1881	1.04	-4%
ST 384	1.08	173	1.06	-2%
ST 39/6373	1.67	117	1.74	4%
ST 09/7382	1.93	205	1.87	-3%
ST 482	1.94	695	1.98	2%
ST 5355	2.37	145	2.39	1%
ST 05/9451	2.45	538	2.53	3%
ST 05/6372	2.46	168	2.44	-1%
ST 05/2297	2.56	78	2.49	-3%
ST 486	2.63	49	2.59	-5%
ST 10/9298	3.22	132	3.30	3%
ST 37/6374	3.33	129	3.08	-7%
ST 43/7370	3.37	834	3.33	1%
ST 5359	3.91	131	3.97	1%
ST 359	3.93	87	3.96	1%
ST 48/8462	4.82	508	4.89	1%

Table 11-4: Gannet CRM for 2013 (SGS)

Standard	Certified Mean (g/t Au)	Number Samples Submitted	Mean Assay Grade (g/t Au)	Laboratory Bias (%)
ST07/9453	0.21	645	0.22	4%
ST14/9501	0.43	402	0.50	17%
ST06/7384	1.08	39	1.05	-3%
ST482	1.94	528	1.99	2%
ST05/6372	2.46	665	2.48	1%
ST37/6374	3.33	579	3.29	-1%
ST48/8462	4.82	187	4.89	1%

Table 11-5: Gannet CRM for Q1 and Q2, 2014 (SGS)

Standard	Certified Mean (g/t Au)	Number Samples Submitted	Mean Assay Grade (g/t Au)	Laboratory Bias (%)
ST07/9453	0.21	474	0.21	-2%
ST482	1.94	489	2.00	3%
ST596	2.51	61	2.51	0%
ST48/8462	4.82	233	5.00	4%
ST517	5.23	208	5.10	-2%

Results from the CRM analysis between 2008 and Quarter 2, 2014 indicate that SGS reports both higher and lower than expected values with some variation to the detection limit. SGS returned assays of Geostats standards with laboratory bias ranging between -9% to +5% with 94% of the determinations falling between 2 standard deviations of the certified mean values. Assays of Gannet standards also reported Au values with a laboratory bias of between -7% and +3% with 94% of the determinations falling between 2 standard deviations of the certified mean.

Assay results of Gannet standards for the first half of 2014 and all of 2013 reported Au values with a laboratory bias of between -3% and +4% with 98% of the determinations falling between 2 standard deviations of the certified mean.

11.3.7 Blanks

Blank samples are routinely inserted into the sample stream to check for possible sample contamination during the preparation and assaying process. Typically, blanks are inserted prior to the delivery of samples for preparation and analyses. For RC samples, a blank is inserted before the splitting process to monitor possible contamination occurring during the splitting of the original sample collected from the drill cyclone. The blank sample consists of barren coarse sand. The blank assay data include 559 assays for reverse circulation and core samples, all assayed by SGS. Summary statistics for the assays of blanks returned by SGS are shown in Table 11-6.

For Q1 and Q2, 2013, the blank assay data include 201 assays for reverse circulation and core samples, all assayed by SGS. Summary statistics for the assays of blanks returned by SGS are shown in Table 11-7.

Table 11-6: Blank sample summary statistics

Sample type	Count	Minimum (g/t)	Maximum (g/t)	Median (g/t)	Mean (g/t)
Blanks	559	0.01	0.05	0.01	0.01

Table 11-7: Blank sample summary statistics – Q1 and Q2, 2013

Sample type	Count	Minimum (g/t)	Maximum (g/t)	Median (g/t)	Mean (g/t)
Blanks	201	0.01	0.11	0.01	0.01

11.3.8 Umpire Laboratory Performance (Round Robin)

Laboratory checks are performed semi-annually to check on the reliability of the primary laboratory, in this case SGS, Tarkwa. In 2013 and 2014 “round robin” sample check studies were conducted using SGS, Intertek and the Wassa site laboratory.

In 2014, 252 quarter core samples were selected from drilling conducted from 2012 to mid-2014. The intersections selected were high grade intervals which averaged approximately 17 g/t Au. As the coarse sample reject was no longer available for these intervals a new sample was collected by cutting the half core on file in the core yard. Golden Star has conducted similar quarter core sampling studies on other deposits and repeatability of the original results is often not good due to the change in sample size going to half the volume from the original sample. The Wassa quarter core sampling study concluded the same as previous studies with good repeatability between the original sample and it's coarse sample reject and much poorer repeatability with the quarter core sample. The average grade for both the original assay and the coarse sample reject duplicate compare well at 17 g/t Au and the quarter core sample was less at 12 g/t Au. However control sample standards that were submitted with these sample batches consistently came up with a negative bias, Table 11-8 so this can partially account for the lower average. The HARD plots shown in Table 11-9 show the good correlation between the original assay value and the coarse sample reject duplicate but these do not repeat well when using the quarter core samples analysed at Intertek Laboratory. Although the negative lab biais and the smaller sample volume attributes to poor repeatability the Wassa deposit has a high nugget gold distribution which alone will result in poor repeatability. The variability of the gold distribution was recognized and GSR has put in sample protocols to help reduce the variability, ie larger sample volumes, BLEG leach well analysis.

Table 11-8: Gannet CRM for quarter core sample analysis (Intertek)

Standard	Certified Mean (g/t Au)	Number Samples Submitted	Mean Assay Grade (g/t Au)	Laboratory Bias (%)
ST517	5.23	5	4.98	-5%
ST482	1.94	9	1.78	-8%
ST16/9487	0.49	14	0.46	-6%

Table 11-9: Summary HARD plot results for quarter core sample analysis

Laboratory	No of Samples	<10% HARD	<15% HARD	<20% HARD	CORRELATION COEFF ®
Orig SGS vrs Check SGS	252	65%	81%	90%	0.94
Orig SGS vrs Check Intertek	252	32%	45%	57%	0.60
SGS _Check vrs Intertek Check	252	29%	44%	55%	0.45

In 2013, 120 RC samples were split into three samples which were sent to each of the laboratories for gold analysis. The sample batches also contained control samples to monitor the precision of the individual laboratories.

The three laboratories all performed well with the best correlation being between SGS and the Wassa site laboratory. The HARD plots for the laboratory comparisons are shown below in Table 11-10.

Table 11-10: Summary HARD plots 2013 Round Robin Results

LAB	<10% HARD	<15% HARD	<20% HARD	CORRELATION COEFF
SGS vrs Wassa	65%	84%	92%	97%
SGS vrs Intertex	68%	84%	88%	97%
Wassa vrs Intertek	71%	84%	90%	98%

The high correlation at 20% Half Absolute Relative Deviation for all the labs demonstrates how the larger RC chip samples give a better representation of grade. Approximately 90% of the 120 RC samples submitted for this study show a 20% error, compared to the coarse reject core samples submitted in 2013 and 2014 which show a correlation of approximately 80% of the data set with 20% error.

In 2012, a "round robin" exercise was undertaken to have an independent check on the reliability of Au assay results from the primary laboratory, SGS. A total of 10% of all assays from the 1 m samples received each month were randomly picked from the data set. The data is grouped into six separate ranges, namely 0.00 to 0.50 g/t, 0.50 to 0.90 g/t, 0.90 to 1.20 g/t, 1.20 to 2.00 g/t, 2.00 to 2.50g/t and greater than 2.50g/t. The selection in each range is manipulated until the 10% is achieved with a bias towards the mineralized intervals.

Three samples, each weighing about 3 kg were prepared from each original sample bag using the one stage riffle splitter. Four batches of 175 samples including duplicates and standards were dispatched to SGS, WSL, TWL, and ALS Minerals in Ghana-Kumasi ("ALS"). All samples were labelled with the same identification numbers. A total of 157 assays were returned by each laboratory for analysis.

Statistical comparison of the data indicates that ALS returned lower grades and variance than SGS, WSL and TWL. SGS and TWL correlated well with similar minimum and maximum grades, and standard deviation population distribution. The descriptive statistics from the round robin exercise are included in Table 11-11.

Table 11-11: Round-robin descriptive statistics

Laboratory	Count	Minimum (g/t)	Maximum (g/t)	Mean (g/t)	Variance	Std Dev
SGS	157	0.01	12.0	1.33	1.75	1.32
WSL	157	0.01	8.9	1.09	1.47	1.21
TWL	157	0.01	11.68	1.15	1.68	1.30
ALS	157	0.01	9.32	1.02	1.31	1.15

12 MINERAL PROCESSING AND METALLURGICAL TESTING (ITEM 13)

12.1 Overview

On obtaining ownership of the project in 2002, GSR commissioned a FS for a CIL operation with the process engineering component undertaken by Metallurgical Process Development Pty Ltd ("MDM"). The FS was completed in 2003.

The metallurgical testwork conducted in support of the MDM FS was conducted on samples from the Wassa area. Samples were originally sent to SGS Lakefield in Johannesburg for both variability and bulk sample testwork. Further variability testwork was conducted at AMMTEC in Perth.

A total of 24 variability samples were tested; 10 of fresh mineralized material, 6 of oxide and 8 samples taken from the existing (now decommissioned and reclaimed) heap leach operation. Four bulk samples were also tested, representing fresh, oxide, heap leach phase 1 and heap leach phase 2. The samples were from the Dead Man's Hill, Main Pit, Starter and B Shoot mineralization. Wassa area mineralization not included in this testwork included 242, South East, 419 and the SAK pits.

At a grind size of 75% -75 μm and a 24 hour leach time, the fresh bulk sample achieved a leach recovery of 92%. The Bond Ball Mill Work Index ("BWi") for this sample was 14.8 kWh/t. Under the same conditions, the oxide bulk sample achieved a leach recovery of 93%. The BWi for this sample was reported as 8 kWh/t. Minor preg-robbing behaviour was noted, and gravity recovery testwork indicated that plant recoveries of 30 to 40% could be expected from a gravity circuit.

For ongoing "future ores" testwork, the Wassa metallurgical laboratory has the capability to conduct bottle roll leach tests. For other testwork, samples are sent to SGS Lakefield laboratories in either Tarkwa or Tema or to the University of Mines and Technology in Tarkwa.

In order to quantify the expected increase in rock hardness with depth in the Wassa mineralization, BWi tests were conducted on material from two diamond drillholes, one drilled in the SE area and the other into the MSN area. BWi tests were conducted on samples representing 5 m intervals of vertical depth.

The results of this investigation are shown in Figure 12-1, which shows a consistent trend of increasing hardness (BWi) with depth, from a value of approximately 16 kWh/t at 950 mRL to approximately 19 kWh/t at 825 mRL which represents the bottom of the current LoM open pit designs.

The proposed underground operation would extend from approximately 800 mRL down to approximately 200 mRL. While the hardness below 800 mRL has not been investigated, on the basis of the data shown in Figure 12-1, the underground feed is expected to be harder than the open pit feed, increasing further with depth.

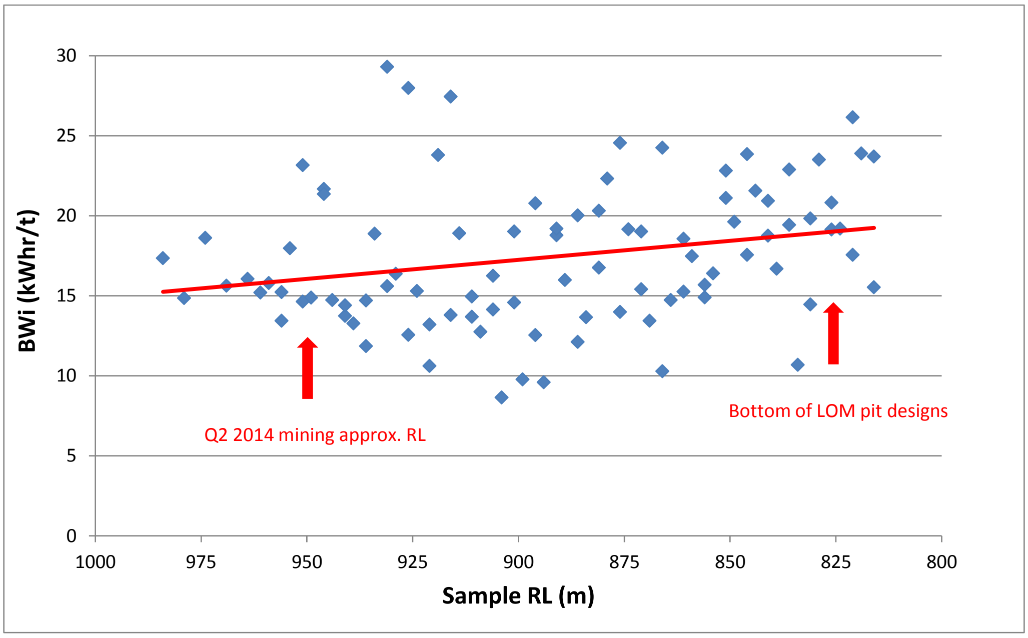


Figure 12-1: Bond Ball Mill Work Index with Mineralization Depth

While no other metallurgical testwork, e.g. leaching, has been undertaken in support of this PEA, monthly operating data from the Wassa plant over the period January to July 2014, when the plant has been processing Wassa Fresh open pit material at a portion of 85 to 100% of the plant feed, shows that gold recoveries have ranged, on a monthly basis, from 91.8 to 93.8%, averaging 92.6%, for head grades varying between 1.22 and 1.91 g/t Au.

12.2 Future Metallurgical Testwork

As the project advances to the next stage of its development, a dedicated programme of metallurgical testwork will be undertaken for the deeper mineralization to be accessed using underground mining methods when the exploration drilling is completed.

This testwork should principally cover the expected increase in rock hardness with depth, and any potential variation in the recovery response with grind size. The other key parameter to be covered will be the settling and tailings characteristics properties of the future ores.

13 MINERAL RESOURCE ESTIMATES (ITEM 14)

13.1 Introduction

The Mineral Resource Statement presented herein represents an estimate for the Wassa Main project, other satellite deposits which are part of the Wassa Mine were not included in this report. The Mineral Resource Statement is presented in accordance with the guidelines of the Canadian Securities Administrators' National Instrument 43-101.

The GSR exploration team was responsible for the modelling portion of the resource estimate exercise which included all topographic surfaces, weathering surfaces, geological and grade wireframes. SRK Consulting (Canada) Inc. was commissioned to construct a mineral resource model with estimated gold grades for the Wassa Main deposit, the methodology of the estimate was described in a memo dated 12 September, 2014 and authored by Dr Oy Leuangthong and Dr David Machuca. The mineral resource classification and statement was conducted by GSR under the supervision of Mitch Wasel, a Qualified Person pursuant to NI 43-101.

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

The databases used to estimate the Mineral Resources were audited internally by GSR. SRK is of the opinion that the current drilling information is sufficiently reliable to interpret with confidence the boundaries for gold mineralization and that the assay data are sufficiently reliable to support mineral resource estimation.

13.2 Resource Estimation Procedures

The resource evaluation methodology involved a database compilation and internal validation exercise by GSR. All geological surfaces and wireframes were modelled by GSR, including the weathering surfaces and mineralization boundaries. GSR provided a Gems project to SRK that consisted of two borehole databases (AW and 2002), two grade control databases (historic and current), solid wireframes, topographic surfaces as well as weathering surfaces. A block model within the Gems project was also provided to SRK, the block model was coded by GSR with rock codes, percent model and density.

Prior to initiating the resource estimation process, SRK reviewed the wireframes defining the gold mineralization for the Wassa project modelled by GSR. There are two sets of envelopes that separate low and high grade gold distribution. No major problems were identified during the review. There are, however, some minor concerns that should be addressed in future updates of the geology model. These include avoiding overlaps in geology wireframes and ensuring that interior high grade zones are entirely encompassed by their corresponding low grade zones. SRK reviewed these concerns with GSR, as these concerns are minor and immaterial to the overall resource, GSR and SRK chose to proceed with the resource estimation using the mineralized wireframes as provided in the initial data transfer.

After evaluating the available database, SRK proceeded with data conditioning (compositing and capping) for geostatistical analysis and variography. The grade interpolation methodology was discussed between GSR and SRK, it was decided to use ordinary kriging with local varying angles and local variograms for the estimation of gold grades based on the structural complexity and folded nature of the deposit.

The classification and preparation of the Mineral Resource Statement were conducted by GSR under the supervision of GSR's Qualified Person, Mr Mitch Wasel.

13.3 Resource Database

The Wassa database is made of four individual drillhole databases, namely: the GSR Wassa exploration database which contains exploration drilling conducted by GSR since 2002; the AW exploration database which contains historical exploration drill holes from previous operator Satellite Goldfields; the Satellite grade control database; and the GSR grade control database. The Satellite grade control database was not included in the Mineral Resource estimate as the blast holes samples are considered to be of not a sufficient quality for inclusion into the Mineral Resource estimate.

A July 7th 2014 cut-off was applied to the GSR exploration database. At this date, only drillholes with a complete assay table were retained for the subsequent Mineral Resource estimate. The final drillholes for each respective domains were BSDD327B and BSRC071 for the Bshoot domain, STDD021, STRC026, 242DD080 and 242RC078 for the 242-Starter domain, 419DD027, 419RC069, FSDD008 and FSRC051 for the Fshoot-419 domain, SEDD092, SERC198, NSADD009 and NSARC271 for the South-East domain.

The Gems project that was provided to SRK also contained geological and mineralization wireframes, topographic surfaces, weathering surfaces and a coded block model.

13.4 Modelling

A two-step approach was developed for modelling the auriferous zones at Wassa, wide low grade envelopes characterized by weak alteration were modelled around high grade zones with strong veining and sulphides contents. A total of twenty-seven low grade wireframes were modelled by GSR surrounding four high-grade wireframes. SRK (Canada) undertook an independent exercise to confirm the continuity of the high grade zones as modelled by GSR, these high grade wireframes were developed with structural geology assistance provided by SRK (UK).

13.4.1 Mineralization Wireframe

The wireframe modelling was carried out by GSR under the supervision of Mr. Mitch Wasel. Mineralization wireframes at Wassa are modelled using a 0.2 g/t cut-off as a guide for the low grade envelopes while a cut-off of 1g/t is used as a guide for high grade zones depending on the grade distribution of each domains. Visual inspection of assay data suggests that these respective lower cut-off levels are reasonable to separate barren from auriferous sections intersected by each borehole.

The modelling is based on the original samples prior to any data cutting or compositing, during the grade modelling exercise. Lithological contacts along with structural measurements are also used as guides. The modelling is conducted on 25m spaced vertical sections for all deposits, with the two-dimensional sectional interpretations being snapped to intersections along the drillholes trace.

For the construction of three dimensional wireframes, the two dimensional interpretations are linked with tie lines and the resulting wireframes are validated prior to compositing and grade interpolation.

The mineralized zones of Wassa are structurally controlled, with gold emplacement related to the density of quartz veining and sulphide content. The mineralized veins have been deformed and are folded around a large synform commonly referred to as the Wassa fold. The Wassa mineralized zones have been subdivided into a number of domains, namely: F Shoot-419, B Shoot, 242-Starter, South East, Mid-East and Dead Man’s Hill. Each domain represents segments of the mineralized system which has been folded around the synform.

All domains were updated with recent drilling except for the Mid-East and Dead Man Hill domains, which have not been drilled since the 2010 exploration campaign. Overall, a total of thirty-one auriferous zones were interpreted for this resource estimate along four domains using the following nomenclature: 8850s (242-Starter domain), 8870s (F Shoot-419), 8880s (Southeast zone), and 8890s (B Shoot). All zones were assigned individual rock codes: 242-Starter (8850 to 8857), Fshoot-419 (8870 to 8876), South-East (8880 to 8887) and B-Shoot (8890 to 8894). An additional four high grade wireframes were also modelled and included in this modelling exercise, one of the four high grade wireframes lays in the 8870s series, and the remaining three reside in the 8880s series. These high grade zones are located within low grade envelopes and were coded in the 4800’s series (4872, 4881 to 4883).

As portrayed in Figure 13-1, the 242-starter domain is located on the western flank of the large scale synform while the remaining domains are located on the eastern flank. Domains along the western flank generally dip to the south-east at an angle of approximately 50 degrees, while domains located along the eastern flank are steeply dipping to the west near the surface and progressively flattened with depth towards the fold closure of the Wassa synform. As a result of the complex nature of mineralization at Wassa, several domaining methods have been trialled over the years.

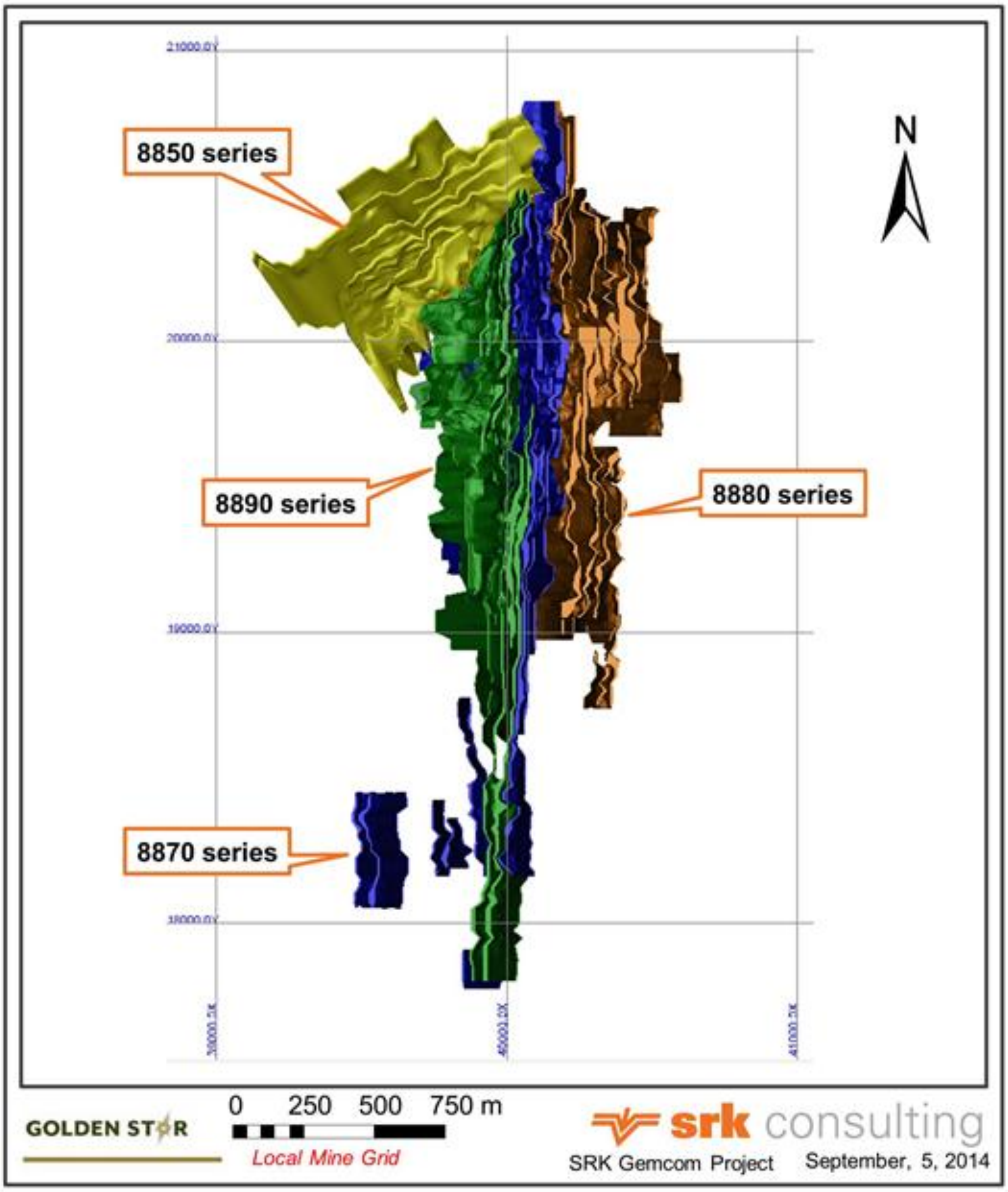


Figure 13-1: Plan View of Available Wireframes for the Wassa Deposit

13.4.2 Multi-Gaussian Kriging Modelling

The focus of this exercise was to independently confirm the continuity of the high grade zones as modelled by GSR. SRK (Canada) understands that these high grade wireframes were developed with structural geology assistance provided by SRK (UK).

Only the assays in the region of the HG zone were required for this phase. SRK extracted all (mineralized and un-mineralized) borehole assays within the following bounding box:

- X: 39,700 to 40,300 m Easting
- Y: 19,000 to 20,250 m Northing
- Z: 100 to 1,000 m Elevation

This resulted in a subset database, which comprised 124,915 assay intervals that were available for Phase 1 of this study.

13.4.2.1 Methodology

SRK used Multi-Gaussian kriging for grade domaining (Leuangthong and Srivastava, 2012). This method involves the following steps in the workflow:

1. Normal score transform the gold grades;
2. Calculate and model the variogram of the normal scores of gold;
3. Perform simple kriging simulation of the normal scores and back transform the kriged mean and variance to determine the local distribution of uncertainty;
4. Specify possible grade thresholds and calculate the probability to be above the specified grades;
5. Generate iso-probability contours, usually in a general mine planning package (GMP); and
6. Select the grade threshold and probability thresholds associated with reasonable grade shells.

The software used for this methodology is a combination of the Geostatistical Software Library (“GSLib”) (Deutsch and Journel, 1998), GSLib-compatible prototype program for Step 4 above and GOCAD® for Steps 5 and 6.

13.4.2.2 Normal Scores Variogram

SRK calculated the normal scores variogram of gold grades within this subset of the assay database. Various orientations were analyzed to identify principle continuity directions. Table 13-1 and Figure 13-2 gives the normal scores variogram modelled and subsequently used in MG kriging.

Table 13-1: Normal Score Variogram Models*

Data	GSLib Angles			Nugget Effect	Structure 1					Structure 2				
	ANG1	ANG2	ANG3		Type	CC1	Ah1	Ah2	Avert	Type	CC1	Ah1	Ah2	Avert
HG Subset	180	-20	50	0.2	Exp	0.6	50	30	20	Sph	0.2	330	160	90

* Ranges in the table are denoted by Ah1, Ah2, and Avert, where Ah1 corresponds to an azimuth/dip direction of 180/-20 degrees, Ah2 corresponds to azimuth/dip direction of 112/46 degrees, and Avert corresponds to azimuth/dip direction of -106/37 degrees.

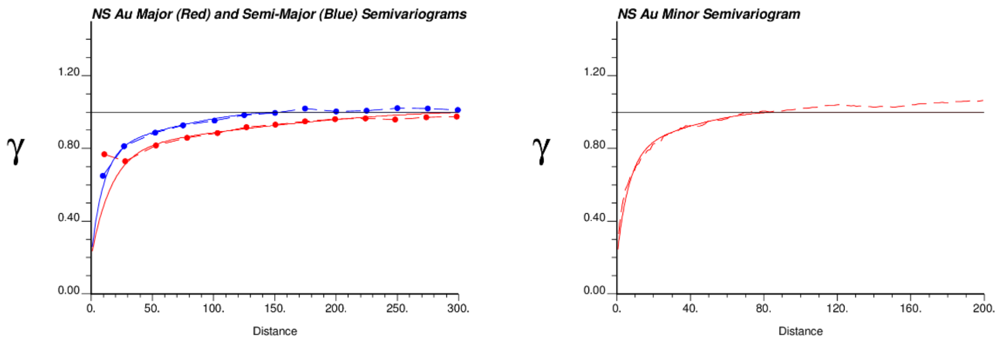


Figure 13-2: Normal Scores Variogram of Gold Grades within HG Subset Data

13.4.2.3 Model Parameters

The primary objective of this workflow was to generate contiguous areas that may be plausible grade domains for resource estimation. For this purpose, it was important to generate smooth estimation models; early experience with this technique has shown that this generally involves a combination of fine resolution grids and a large maximum number of samples being used in the estimation. In this study, SRK employed the following model settings and parameters:

- Block definition using a 5 x 5 x 5 m grid with model extents encompassing the region surrounding the high grade zones plus an additional 100 m on all sides;
- Search orientation of 180/-20/50 degrees for angles 1, 2, and 3 in GSLib orientation (corresponding to -180/-20/112 in Gems ADA convention);
- Search radii of 350 x 200 x 100 m aligned with the principle axes;
- Minimum number of composites = 2;
- Maximum number of composites = 128; and
- Ellipsoidal search.

13.4.2.4 Selection of Grade and Iso-Probability Shells

SRK calculated the probability to exceed five grade thresholds: 0.50, 0.75, 0.90, 1.0, and 1.10 g/t Au. These probability models (corresponding to each of the grade thresholds) were then imported into GOCAD® and iso-probability contours were generated.

A comparison of the iso-probability shell for a 20% probability to exceed 0.75 g/t Au showed reasonably good continuity of high gold grades for elevations higher than approximately 400 m. Continuity at lower elevations, however, could not be confirmed via this approach. This also corresponds to the region informed by boreholes that are spaced 200 metres apart. It appears that the 4800 high grade series of wireframes were generated by considering a structural geology interpretation that is not captured using this grade-driven MG kriging approach.

SRK provided to GSR several shells in DXF format corresponding to 0.75 g/t Au for more in-depth review. These shells were lightly cleaned by removing small triangulated solids with fewer than 50 facets.

The high grade zones considered in this resource model are a new approach to the resource modelling for Wassa. SRK independently confirmed continuity of high gold grades for elevations of 400 m and higher in regions identified by GSR. However, SRK was unable to confirm continuity of high grade for lower elevations, where informing boreholes are spaced 200 m apart. Despite this, the results of the high grade continuity assessment of this study is overall encouraging, and suggests that with additional drilling at depth between 19,100 and 19,500 Northing, the reasonableness of the HG wireframe extension at depth may be confirmed.

13.5 Statistical Analysis

SRK evaluated the four available databases for Wassa, and compared their summary statistics on a by-zone and by-series basis. Results from this comparison showed that the average grade from the grade control databases varied significantly from that of the borehole databases. SRK understands that GSR based the geology wireframes on the borehole databases, and the grade control data were subsequently tagged to the modelled domains. As such, this is likely a source for differences in summary statistics. Coupled with the inherent sample errors associated to the grade control data, the borehole database is deemed to be more reliable. Consequently, SRK chose to use data from only the borehole databases to condition the mineral resource model.

SRK analysed the length distributions from the 2002, AW, and the combined borehole databases. In the 2002 borehole database, approximately 90% of the assays are sampled at 1 metre or less, while 97% are sampled at 2 m or less. In the AW database, approximately 30%, 38%, and 98% are sampled at 1 metre, 2 metre, and 3 metre or less, respectively. On the basis of these length distributions, SRK chose to calculate composites at 3 metre lengths within the solid wireframes. This is consistent with previous Wassa resource models.

To avoid potential estimation bias due to short intervals, composites smaller than 0.30 metres

(10% of composite length) were removed from the composites database. SRK then analysed the composites database to determine if capping was required on a series-by-series basis. SRK selected the capping value by comparing probability plots of gold composites for each series, and plotting the mean grade and the number of affected data by the chosen cap value (Figure 13-4). Decile analysis was used to confirm the reasonableness of the chosen cap value for each series. Table 13-2 and Figure 13-3 summarize the results of this analysis. Summary statistics for both uncapped and capped composites are shown in Table 13-3 with further details in Figure 13-4.

Table 13-2: Summary of Capping Analysis

Series	Cap Value by Approach (g/t Au)		Cap Value (g/t)
	Probability Plots	Decile Analysis	
HG	30	60	30
8850s	15	16	15
8870s	17	15	17
8880s	15	15	15
8890s	15	18	15

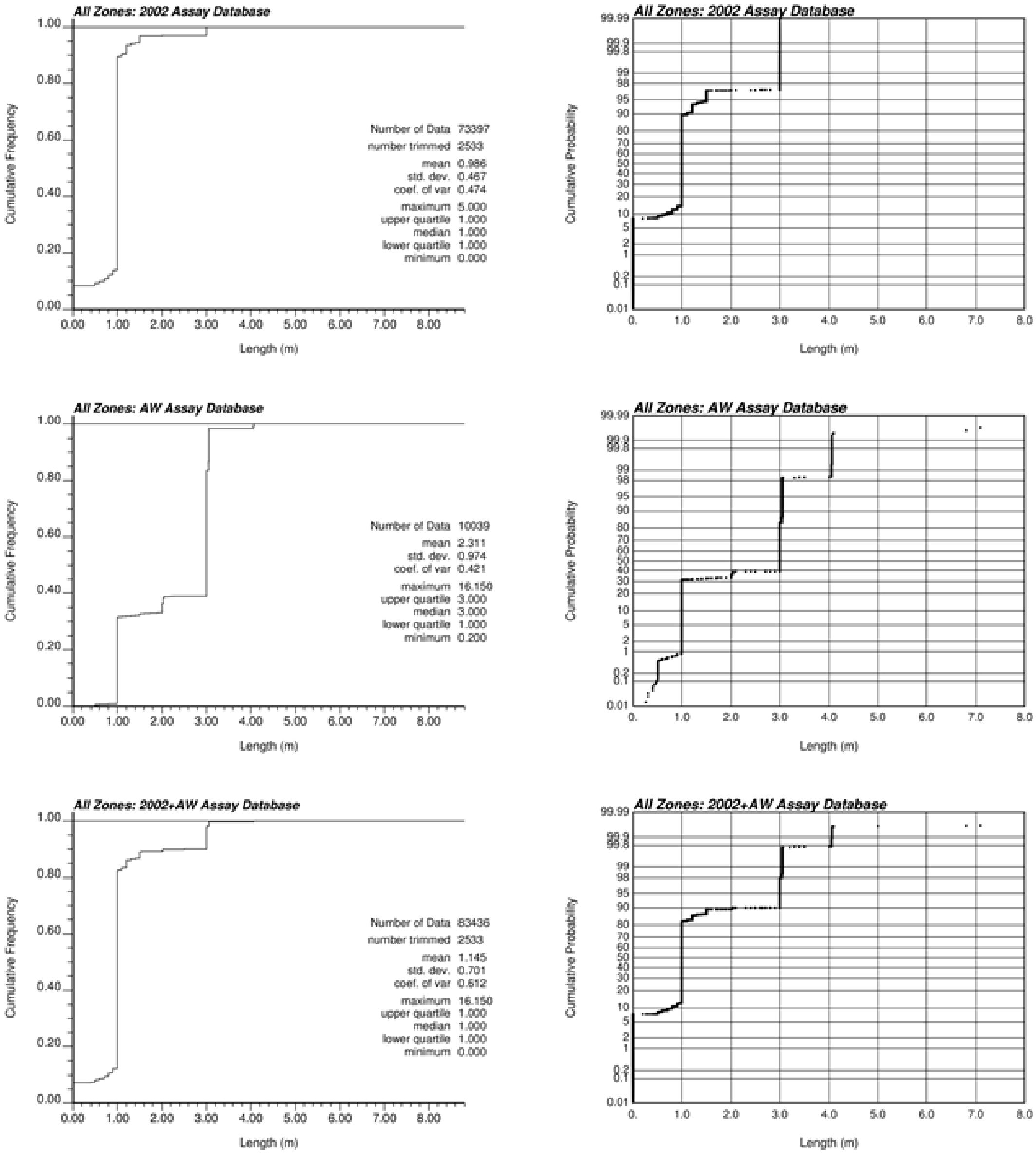


Figure 13-3: Assay Length Distribution for 2002 Database (top), AW Database (middle) and Combined Database (bottom)

Table 13-3: Summary Gold Statistics of Assays, Composites, and Capped Composites

Zone	Assays					Composites					Capped Composites	
	No. Data	Mean	StdDev	Min	Max	No. Data	Mean	StdDev	Min	Max	Mean	StdDev
All	**77,143**	**0.97**	**3.49**	**0**	**322.92**	**37,352**	**0.95**	**2.04**	**0**	**56.97**	**0.94**	**1.9**
HG series	**4,949**	**3.04**	**8.74**	**0.01**	**213**	**2,472**	**3**	**5.3**	**0.01**	**56.97**	**2.92**	**4.69**
4872	1,555	2.76	6.11	0.01	94.9	686	2.57	4.14	0.01	46.05	2.55	3.91
4881	1,731	4.02	12.75	0.01	213	995	3.66	6.76	0.01	56.97	3.48	5.74
4882	1,619	2.22	4.35	0.01	58.4	728	2.38	3.53	0.01	39.87	2.37	3.38
4883	44	3.53	6.95	0.02	43.5	63	4.29	5.37	0.03	27.94	4.29	5.37
8850 series	**8,332**	**1.21**	**4.91**	**0**	**322.92**	**4,525**	**1.07**	**1.99**	**0**	**22**	**1.06**	**1.86**
8850	1,590	1.01	2.63	0	42.57	912	0.99	1.87	0	19.88	0.98	1.77
8851	2,986	1.41	4.83	0	146.1	1,515	1.22	2.23	0	20.79	1.2	2.05
8852	1,423	1.32	7.69	0	322.92	786	1.11	1.96	0	22	1.1	1.87
8853	212	1.24	2.81	0.01	36.95	144	1.25	1.94	0.03	11.03	1.25	1.94
8854	1,002	1.57	5.77	0	89.85	556	1.22	2.28	0	22	1.2	2.14
8855	772	0.6	1.6	0	43.7	451	0.6	0.9	0	8.99	0.6	0.9
8857	347	0.62	0.98	0	6.99	161	0.61	0.78	0.01	6.27	0.61	0.78
8870 series	**20,154**	**0.78**	**2.27**	**0**	**93.5**	**9,509**	**0.75**	**1.42**	**0**	**25.17**	**0.75**	**1.4**
8870	3,161	0.92	2.51	0	93.5	1,456	0.91	1.67	0	18.07	0.91	1.67
8871	6,778	1.01	2.86	0	63.4	3,062	0.93	1.74	0	25.17	0.93	1.69
8872	5,155	0.63	1.53	0	63.8	2,618	0.59	0.91	0	17.72	0.59	0.91
8873	3,432	0.61	1.99	0	62.8	1,608	0.61	1.29	0	18.94	0.61	1.28
8874	633	0.44	1.9	0.01	37.3	356	0.42	1.09	0	12.03	0.42	1.09
8875	668	0.56	0.98	0.01	8.77	266	0.55	0.7	0.01	4.26	0.55	0.7
8876	327	0.62	1.91	0.01	27.97	143	0.62	1.21	0.01	10.4	0.62	1.21
8880 series	**27,088**	**0.74**	**2.45**	**0**	**166.26**	**12,543**	**0.7**	**1.21**	**0**	**22**	**0.7**	**1.18**
8881	5,602	0.78	3.4	0	166.26	2,779	0.7	1.32	0	22	0.7	1.28
8882	5,632	0.73	1.71	0	48.76	2,633	0.7	1.2	0	17.95	0.7	1.18
8883	3,805	0.82	2.61	0	125	1,741	0.79	1.33	0	17.6	0.78	1.31
8884	6,351	0.72	2.07	0	102	2,686	0.69	1.1	0	20.29	0.69	1.08
8885	4,910	0.7	2.28	0	94.2	2,264	0.66	1.07	0	16.54	0.66	1.06
8886	747	0.69	1.92	0	25.6	416	0.66	1.43	0	19.09	0.65	1.3
8887	41	0.36	0.44	0.01	1.75	24	0.36	0.38	0.01	1.1	0.36	0.38
8890 series	**16,620**	**0.91**	**2.68**	**0**	**211**	**8,303**	**0.87**	**1.5**	**0**	**23**	**0.87**	**1.47**
8890	3,068	0.8	2.18	0	72.9	1,413	0.75	1.22	0	13.73	0.75	1.22
8891	5,374	1.1	3.59	0	211	2,711	1.03	1.69	0	23	1.02	1.62
8892	6,464	0.82	2.13	0	55.15	2,984	0.79	1.41	0	16.2	0.79	1.41
8893	977	0.75	1.48	0	20	653	0.77	1.22	0	11.65	0.77	1.22
8894	348	0.73	1.49	0.01	15.7	175	0.7	1.04	0.01	6.62	0.7	1.04
8895	389	1.09	2.32	0	19.52	367	1.06	2.09	0	17.77	1.05	1.98

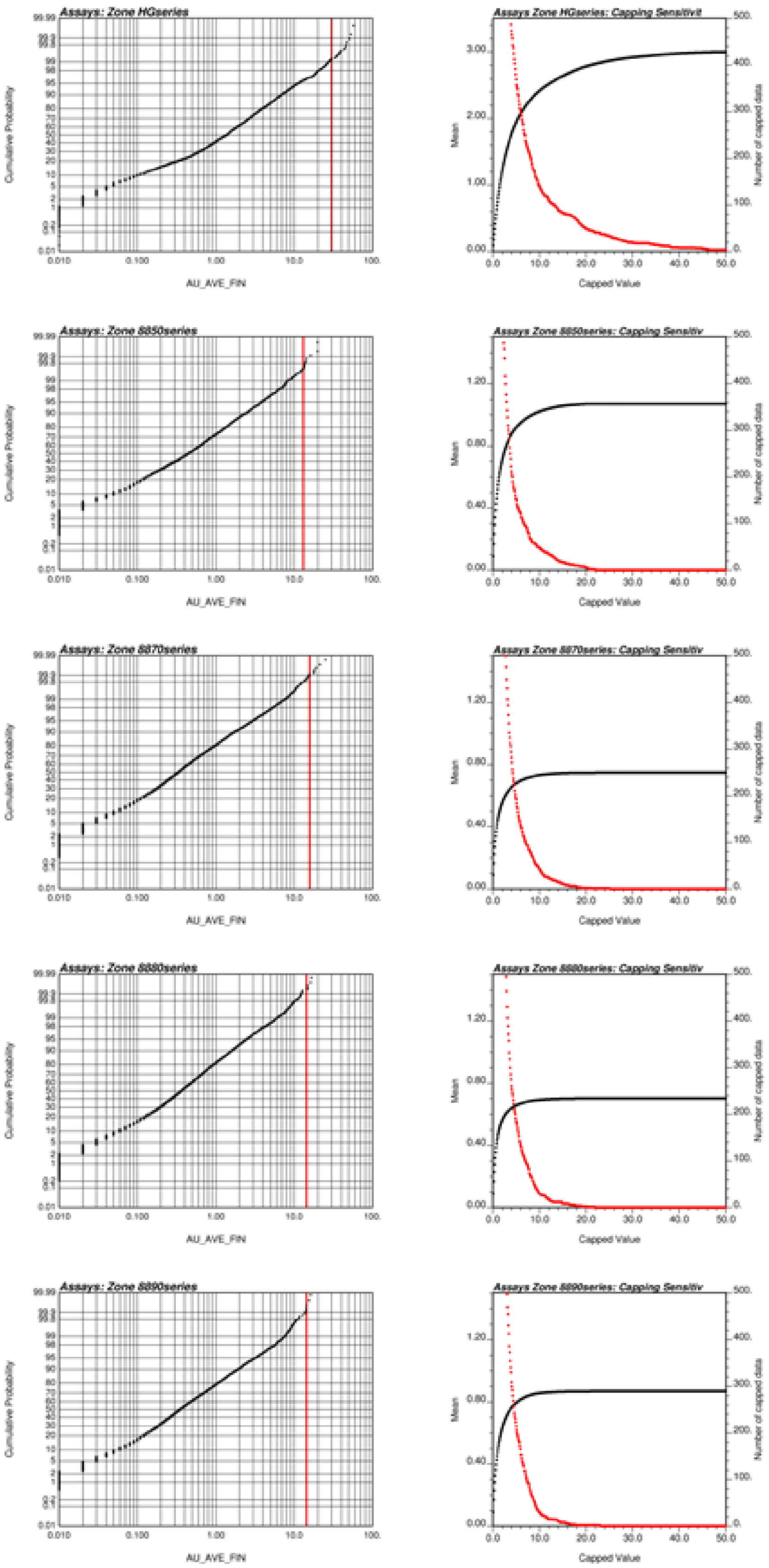


Figure 13-4: Probability and Capping Sensitivity Plots

13.6 Block Model and Grade Estimation

13.6.1 Block Model

A 3D block model including rock type, gold, percent, density and class was built in GEMS by GSR. The selection of the block size was driven by the borehole spacing and mainly by the geometry of the auriferous zones, but also based on mining parameters and in accordance to previous resource estimate conducted by Cube Consulting at the Wassa Mine. The block size was set at 10 x 10 x 3 m in the northing, easting and elevation directions, respectively along the mine grid. The block model origins can be seen in Table 13-4.

Table 13-4: Block Model Parameters

Coordinate	Origin	Block Size (m)	No. of Blocks
X	39,100	10.0	225
Y	17,700	10.0	360
Z	1,100	3.0	360

Rock code assignments from solid to block were made using auriferous wireframes along a horizontal needling level of three (along columns). Air blocks had to be 99.99% above the topographic surface to be assigned the rock code for air (500). Rock codes were modified accordingly for weathering profiles with block assigned 7000 rock codes for saprolite. A percent block model was used to evaluate tonnages. Tonnage for each respective block was obtained by weighting volumes corresponding to the interpreted auriferous zones and the respective mean specific gravity defined by weathering profile.

Rock code assignments from solid to block were made using auriferous wireframes along a horizontal needling level of three (along columns). Air blocks had to be 99.99% above the topographic surface to be assigned the rock code for air (500). Rock codes were modified accordingly for weathering profiles with block assigned 7000 rock codes for saprolite. A percent block model was used to evaluate tonnages. Tonnage for each respective block was obtained by weighting volumes corresponding to the interpreted auriferous zones and the respective mean specific gravity defined by weathering profile.

The block model bulk density data was coded based on weathering surface which was built to define saprolitic material from fresh material. The weathering surface defined the ‘top of fresh’ material, all blocks above the ‘top of fresh’ surface was designated as saprolite and material below the surface as ‘fresh’. The bulk density values assigned to the bloc model were based on series of measurements made over the various exploration phases going back to the initial Golden Star exploration program in 2002. The density values used for the tonnage estimate were provided by GSR, and are detailed below in Table 13-5.

Table 13-5: Bulk Density

Weathering Type	Assigned Bulk Density Value (t/m^3)
Oxidised	1.8
Fresh	2.7

13.6.2 Resource Estimation Methodology

SRK chose to use ordinary kriging with local varying angles and local variograms for the estimation of gold grades. The general steps required to implement the approach are:

1. Construct a locally varying angles models for dip and dip direction;

2. Calculate and model local variograms for each series, and interpolate these local variograms to construct a model of local variogram model parameters;
3. Estimate gold grades using ordinary kriging, calling upon the local models of dip, dip direction, and variogram models; and
4. Check estimated model using qualitative and quantitative methods.

SRK ran the four low grade and two high grade series independently with a block model for each series. The two high grade series were fully encompassed within the block model definition of the corresponding low grade zones. Therefore, the 4872 zone was modelled using the 8870 series block model definition, and the 4880 series of HG zones was modelled using the 8880 series block model definition. The main advantage of this setup was that it was computationally more efficient to estimate multiple series in parallel. Table 13-6 summarizes the block model definition for each series. In all cases, block sizes were 10 x 10 x 3 m.

Table 13-6: Block Model Definition by Series

Series	Number of Cells			Origin (Mid-pt)		
	X	Y	Z	X	Y	Z
8850	105	105	175	39125	19745	528.5
8870	80	310	260	39465	17765	321.5
8880	90	210	320	39715	18735	153.5
8890	45	275	215	39665	17805	459.5

13.6.3 Local Angle Models

SRK generated local angles derived from triangulated facets of the mineralized wireframes provided by GSR. This was achieved using CAE's Datamine Studio 3, and an initial angle data set for both dip and dip directions was obtained for each low grade series of wireframes.

For each series, SRK filtered the local angles within Datamine to remove any anomalous angles resulting from edge triangles from the mineralized solids (Figure 13-5 top). While this removed those angles that were easily identifiable on the edges of the wireframes, the possibility remained for more centrally located angles to be problematic as a result of the triangulation. For this reason, SRK applied a secondary smoothing of the angles using moving windows averaging (Figure 13-5 middle). Table 13-7 summarizes the parameters used to smooth the dip and dip directions within each series. The smoothed angles data set was then used to interpolate a block model of dip and dip directions, which was later called upon for local estimation (Figure 13-5 bottom). Table 13-8 shows the parameters used to estimate the angles. In all cases, the estimation of angles used inverse distance estimation with a power of one.

SRK notes that in both the smoothing and interpolation of angles, small search windows were used. This is primarily because the local angles calculated from wireframes yielded a dense data set as a data point was available for every triangulated facet. SRK ran sensitivities to assess the impact of search radii on the local angles, and quantified differences using the inner product to compare angles. The search ranges shown in Table 13-7 and Table 13-8 achieved a balance in mitigating any problematic angles while preserving local orientations.

Table 13-7: Parameters for Smoothing of Dip and Dip Direction Angles

Series	Search Angles			Search Ranges (m)		
	Ang1	Ang2	Ang3	hmax	hmin	hvert
8850	55	0	0	20	20	10
8870	0	0	0	20	20	10
8880	0	0	0	20	20	10
8890	0	0	0	20	20	10

Table 13-8: Interpolation Parameters of Dip and Dip Directions

Series	Search Angles			Search Ranges (m)		
	Ang1	Ang2	Ang3	hmax	hmin	hvert
8850	55	0	0	30	20	20
8870	0	0	0	30	20	20
8880	0	0	0	30	20	20
8890	0	0	0	30	20	20

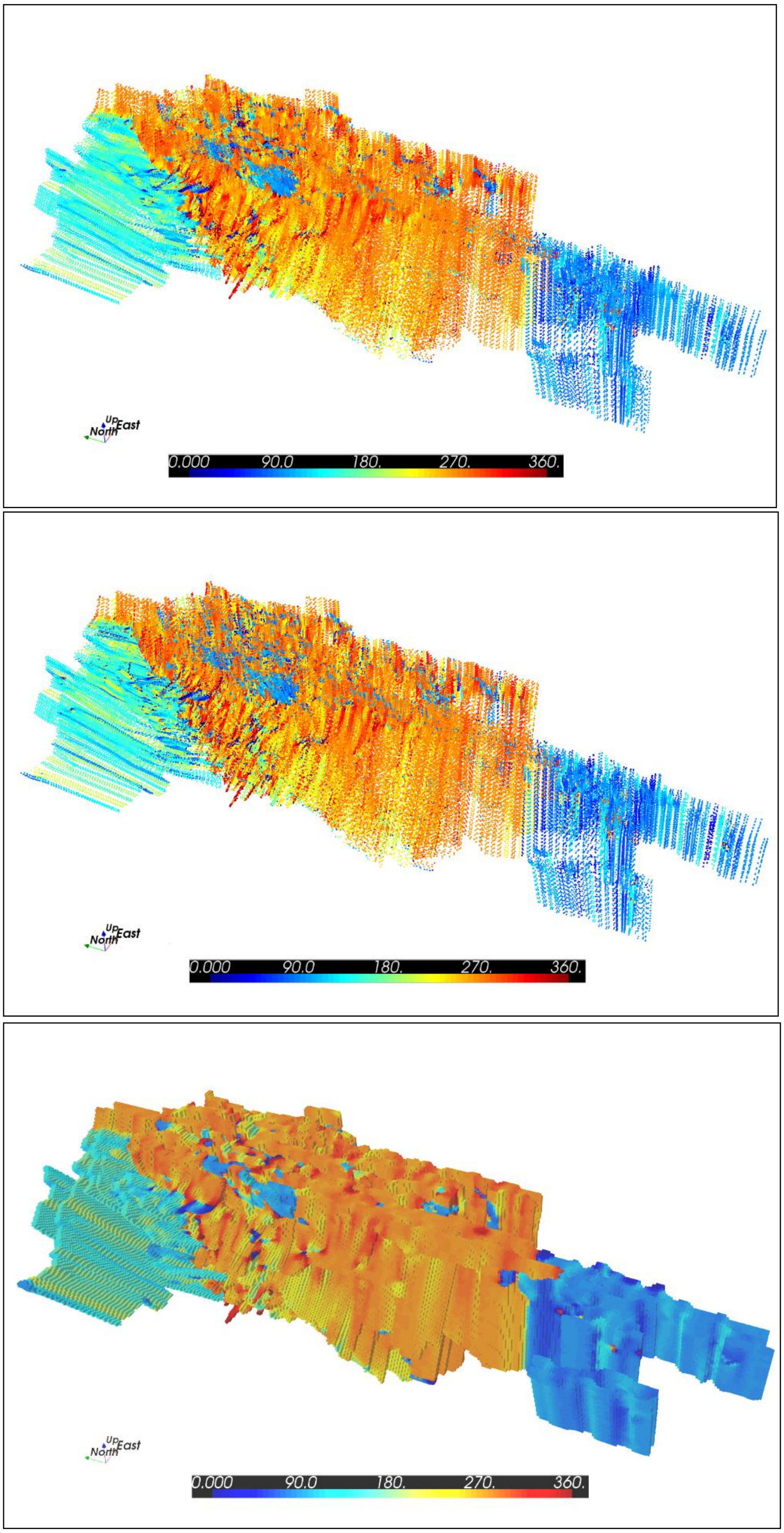


Figure 13-5: Filtered dip directions using CAE Datamine (top); smoothed dip directions using moving average (middle); interpolated dip directions into block model (bottom).

13.6.4 Local Variogram Models

The local estimation approach chosen for Wassa required the specification of local variogram models. In regions where data may be sparse, the "global" or zone variogram may be more reliable than a localized variogram model. For each series of low grade wireframes, SRK calculated both global and local variograms. SRK mainly considered composites from the borehole databases, but did assess the usefulness of the current and historic grade control data. Only the 8890 series of wireframes seemed to benefit from consideration of the grade control data, thus both local and global variograms for this series were based on the combined borehole and grade control databases. All other series relied only on composites from the borehole database.

Variograms corresponding to high grade zones were not modelled; SRK assumed that grade continuity within the high grade zones generally conformed to the spatial orientation of the surrounding low grade 8870 or 8880 series wireframes. The local and/or global variograms for these low grade zones were used for the corresponding high grade wireframes. For these two low grade zones, SRK calculated experimental variograms using only the low grade data and then using the combined low grade and the corresponding high grade data. The impact of including the high grade data were longer variogram ranges, so much so that SRK considered these ranges to be unreasonably large for a gold deposit. Consequently, SRK chose the variogram model associated to using only the low grade data. All variogram calculations and models were performed using GSLib.

Table 13-9 summarizes the global variogram orientations using GSLib convention. The corresponding Gems angle convention was also provided for Golden Star to confirm the major orientation. The variograms for the low grade zones were then calculated and modelled on a by-series basis. This is provided in Table 13-10. Figure 13-6 and Figure 13-7 shows the global variogram models.

Local variograms require the selection of anchor points ("AP"). The surrounding data are then weighted based on their proximity to the AP. These weights are used explicitly in calculating the local variograms, and these variograms are then modelled.

For each series, SRK chose AP locations by using k-means clustering of the local angle database within each zone, and then refined these locations to reflect the structural complexity of the series of wireframes. The specific AP locations and their local orientations for variogram calculation and modelling purposes are summarized in Table 13-11. The modelled local variograms are tabulated in Table 13-12, and illustrated in Figure 13-8 to Figure 13-16.

Table 13-9: Global Variogram Orientations in GSLib and Gems Angle Convention

Series	GSLib			Gems		
	ANG1	ANG2	ANG3	Azm	Dip	Azm
8850	155	-45	0	155	-45	65
8870	270	-65	0	-90	-65	180
8880	260	-55	0	-100	-55	170
8890	270	-50	0	-90	-50	180

Table 13-10: Global Variogram Models by Series

Series	GSLib			Nugget Effect	Structure 1 (Exp)				Structure 2 (Sph)			
	ANG1	ANG2	ANG3		CC	Ahmax	Ahmin	Ahvert	CC	Ahmax	Ahmin	Ahvert
8850	155	-45	0	0.3	0.5	20	20	10	0.2	90	90	10
8870	270	-65	0	0.2	0.57	45	20	10	0.23	50	150	12
8880	260	-55	0	0.2	0.5	15	25	4	0.3	150	125	12
8890	270	-50	0	0.3	0.55	13	25	7	0.15	40	90	7

Table 13-11: Local Variogram Orientations and Anchor Point Locations

Series	AP	GSLib			Gems			Coordinates		
		ANG1	ANG2	ANG3	Azm	Dip	Azm	X	Y	Z
8850	1	146	-44	0	146	-44	56	39,550	20,130	915
	2	150	-55	0	150	-55	60	39,745	20,400	945
	3	150	-40	0	150	-40	60	39,840	20,175	665
	4	165	-35	0	165	-35	75	39,968	20,580	1005
	5	150	-33	0	150	-33	60	39,758	20,204	975
8870	1	80	-75	0	80	-75	-10	39,800	18,225	983
	2	270	-80	0	-90	-80	180	40,040	18,869	960
	3	245	-60	0	-115	-60	155	39,950	19,510	675
	4	270	-60	0	-90	-60	180	40,075	19,425	908
	5	255	-40	0	-105	-40	165	39,956	19,970	787
	6	90	-60	0	90	-60	0	40,108	20,058	966
	7	250	-45	0	-110	-45	160	40,130	20,500	970
8880	1	270	-75	0	-90	-75	180	40,255	19,245	800
	2	260	-50	0	-100	-50	170	40,015	19,452	464
	3	270	-65	0	-90	-65	180	40,305	20,048	982
	4	270	-50	0	-90	-50	180	40,031	19,965	641
	5	275	-70	0	-85	-70	-175	40,102	20,640	919
8890	1	80	-65	0	80	-65	-10	39,980	18,230	945
	2	274	-77	0	-86	-77	-176	40,000	18,752	960
	3	268	-36	0	-92	-36	178	39,925	19,570	820
	4	277	-67	0	-83	-67	-173	39,950	20,208	950
	5	313	-44	0	-47	-44	-137	39,817	20,010	995
	6	249	-62	0	-111	-62	159	39,986	19,590	970

Variogram parameters (Table 13-12) were then estimated to the block model grid, to be read into the grade estimation. In general, the local variograms should be smoothly transitioning within the series. Abrupt changes in grade continuity, within a zone and between AP locations, are not expected. Highly localized changes were addressed by the selection of AP locations. To ensure smoothness of the local variograms, SRK used global kriging with the interpolation parameters shown in Table 13-13.

Table 13-12: Local Variogram Models

Series	AP	Nugget Effect	Structure 1 (Exp)				Structure 2 (Sph)			
			CC	Ahmax	Ahmin	Ahvert	CC	Ahmax	Ahmin	Ahvert
8850	1	0.3	0.55	20	20	10	0.15	90	90	10
	2	0.3	0.5	15	15	15	0.2	90	90	16
	3	0.3	0.5	20	20	7	0.2	90	90	7
	4	0.3	0.5	18	18	25	0.2	90	90	25
	5	0.3	0.5	12	12	12	0.2	90	90	12
8870	1	0.3	0.34	50	15	10	0.36	50	150	12
	2	0.2	0.63	50	25	6	0.17	50	150	12
	3	0.2	0.73	15	40	12	0.07	50	150	12
	4	0.25	0.55	50	15	10	0.2	50	150	12
	5	0.2	0.8	30	45	6	0	50	150	12
	6	0.25	0.56	10	45	4	0.19	50	150	12
	7	0.2	0.54	20	25	6	0.26	50	150	12
8880	1	0.2	0.52	15	35	12	0.28	150	125	15
	2	0.2	0.66	40	90	3	0.14	150	125	10
	3	0.3	0.3	15	20	8	0.4	150	125	12
	4	0.2	0.74	50	50	3	0.06	150	125	12
	5	0.3	0.34	30	20	4	0.36	150	125	20
8890	1	0.2	0.5	13	25	15	0.3	60	150	25
	2	0.3	0.45	20	30	15	0.25	80	150	25
	3	0.3	0.55	20	30	10	0.15	40	100	10
	4	0.3	0.55	13	25	7	0.15	40	90	7
	5	0.3	0.55	13	13	18	0.15	40	40	18
	6	0.3	0.5	13	50	18	0.2	40	90	18

Table 13-13: Global Kriging Parameters Used for Estimation of Local Variogram Model

Series	Search Angles			Search Ranges (m)		
	Ang1	Ang2	Ang3	hmax	hmin	hvert
8850	55	0	0	500	250	250
8870	0	0	0	500	250	250
8880	0	0	0	500	250	250
8890	0	0	0	500	250	250

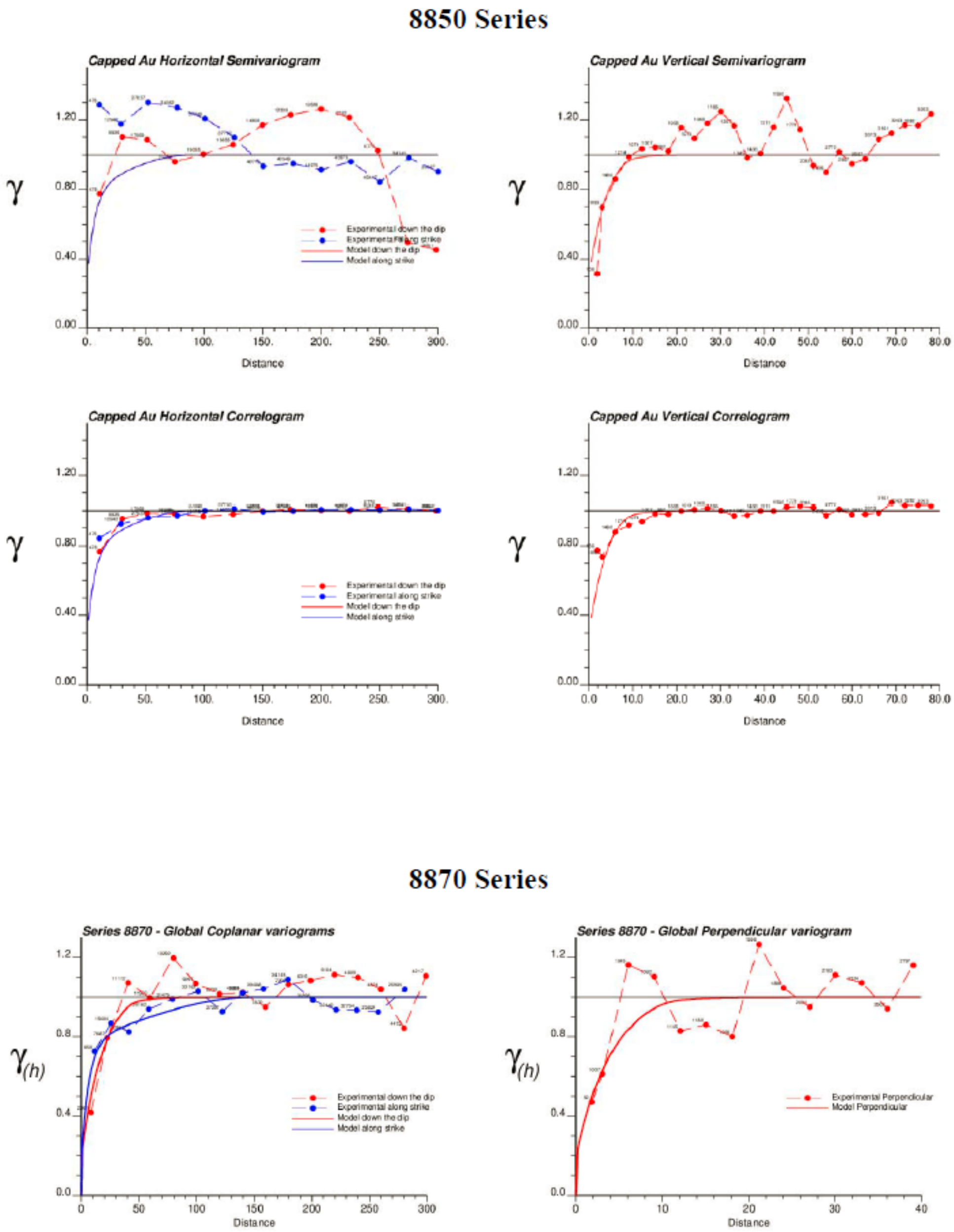


Figure 13-6: Global Variograms, 8850 and 8870 series.

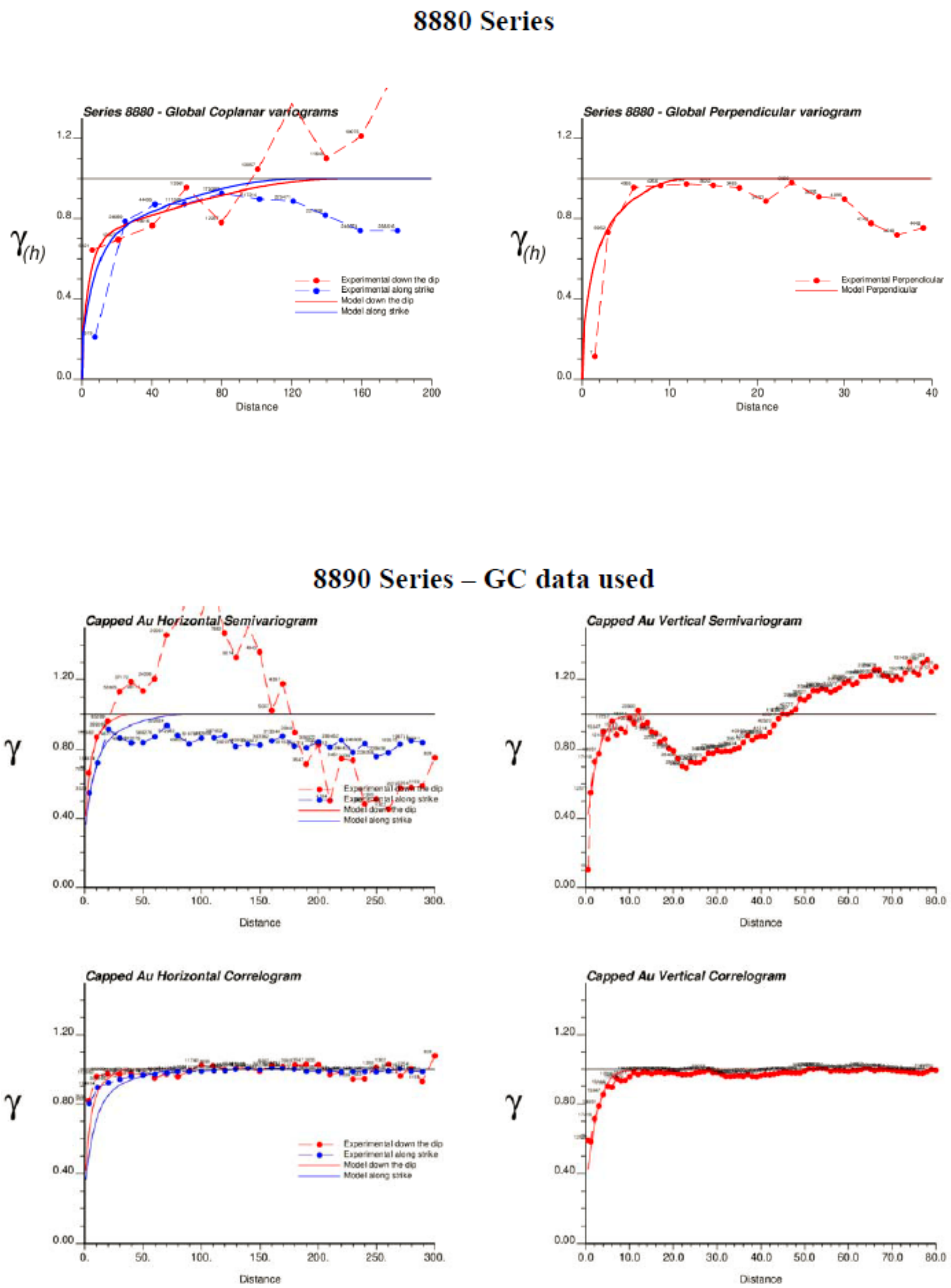


Figure 13-7: Global Variograms, 8880 and 8890 series.

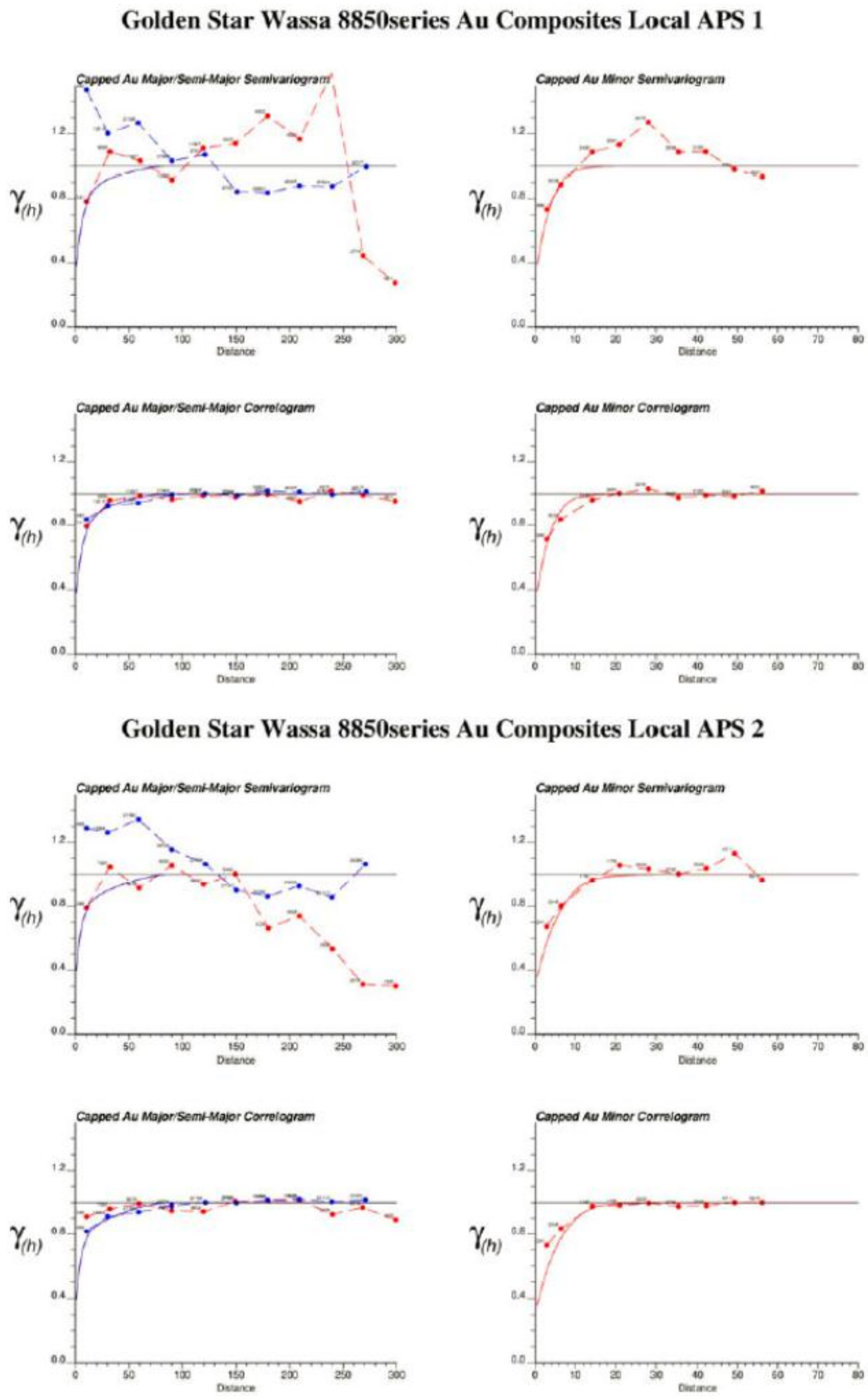


Figure 13-8: Local Variograms, 8850 series, APS 1 and 2.

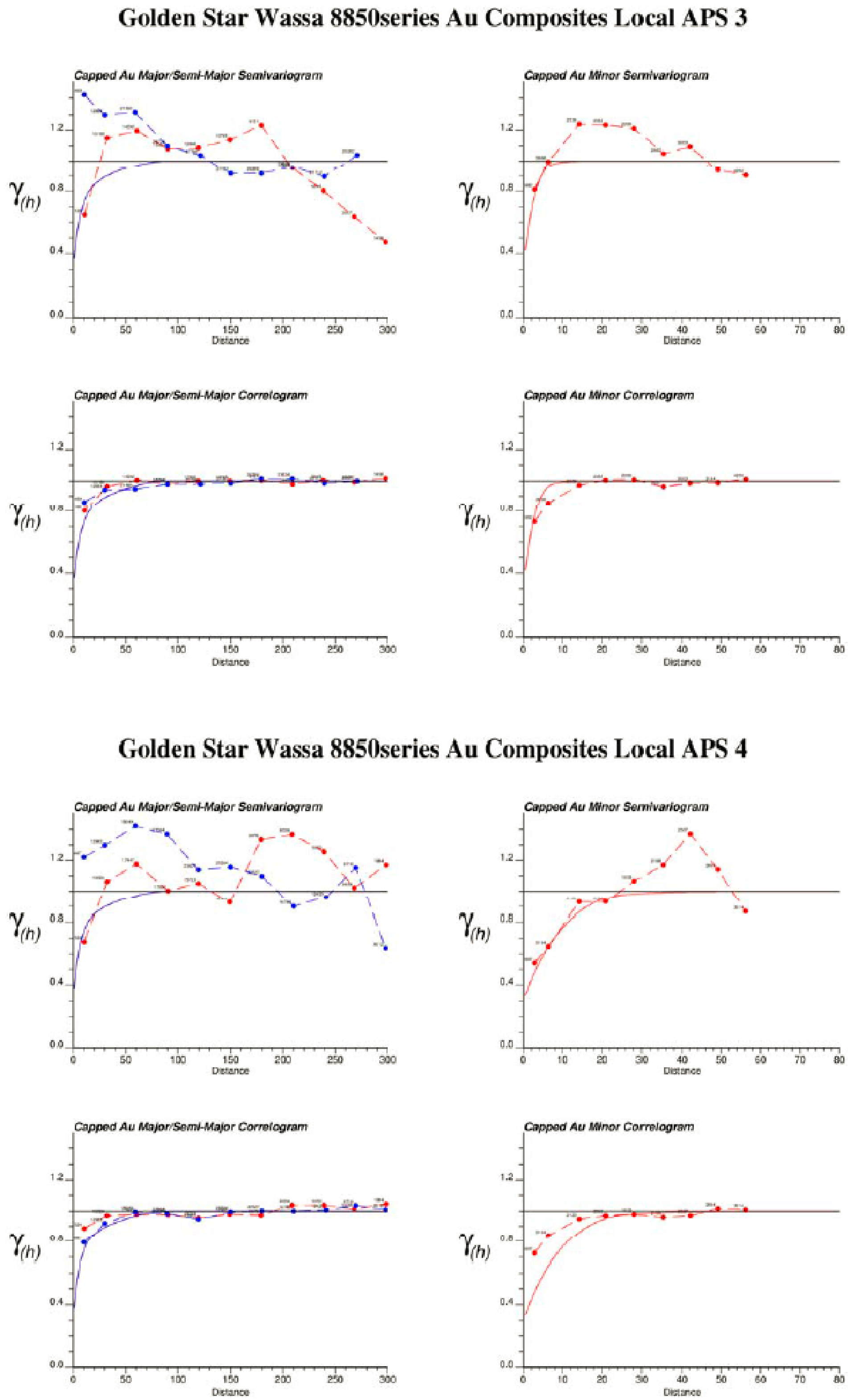


Figure 13-9: Local Variograms, 8850 series, APS 3 and 4.

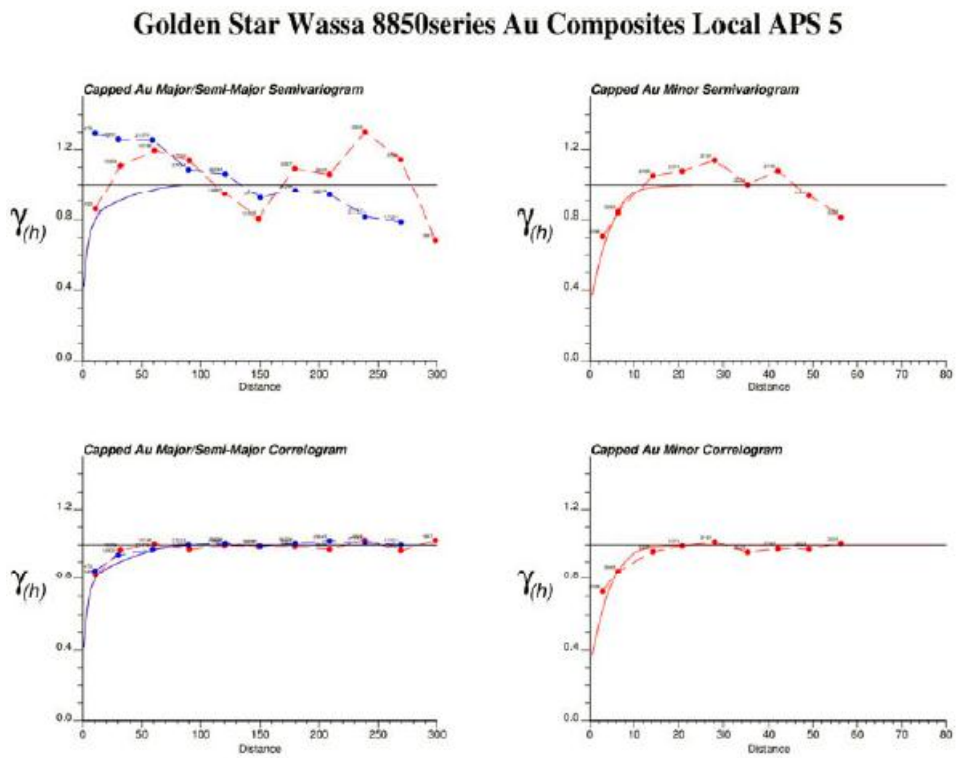


Figure 13-10: Local Variograms, 8850 series, APS 5.

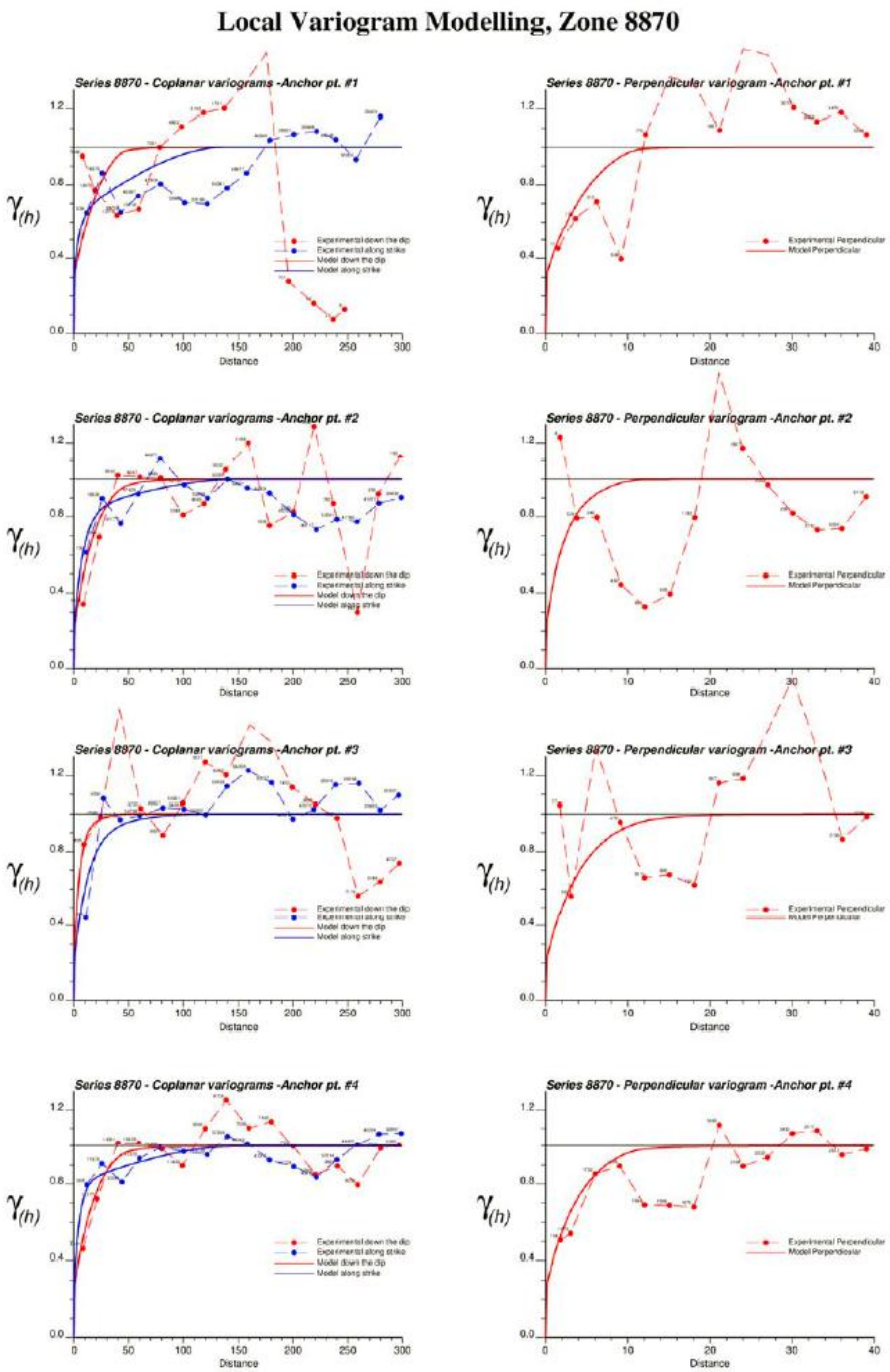


Figure 13-11: Local Variograms, 8870 series, APS 1 to 4.

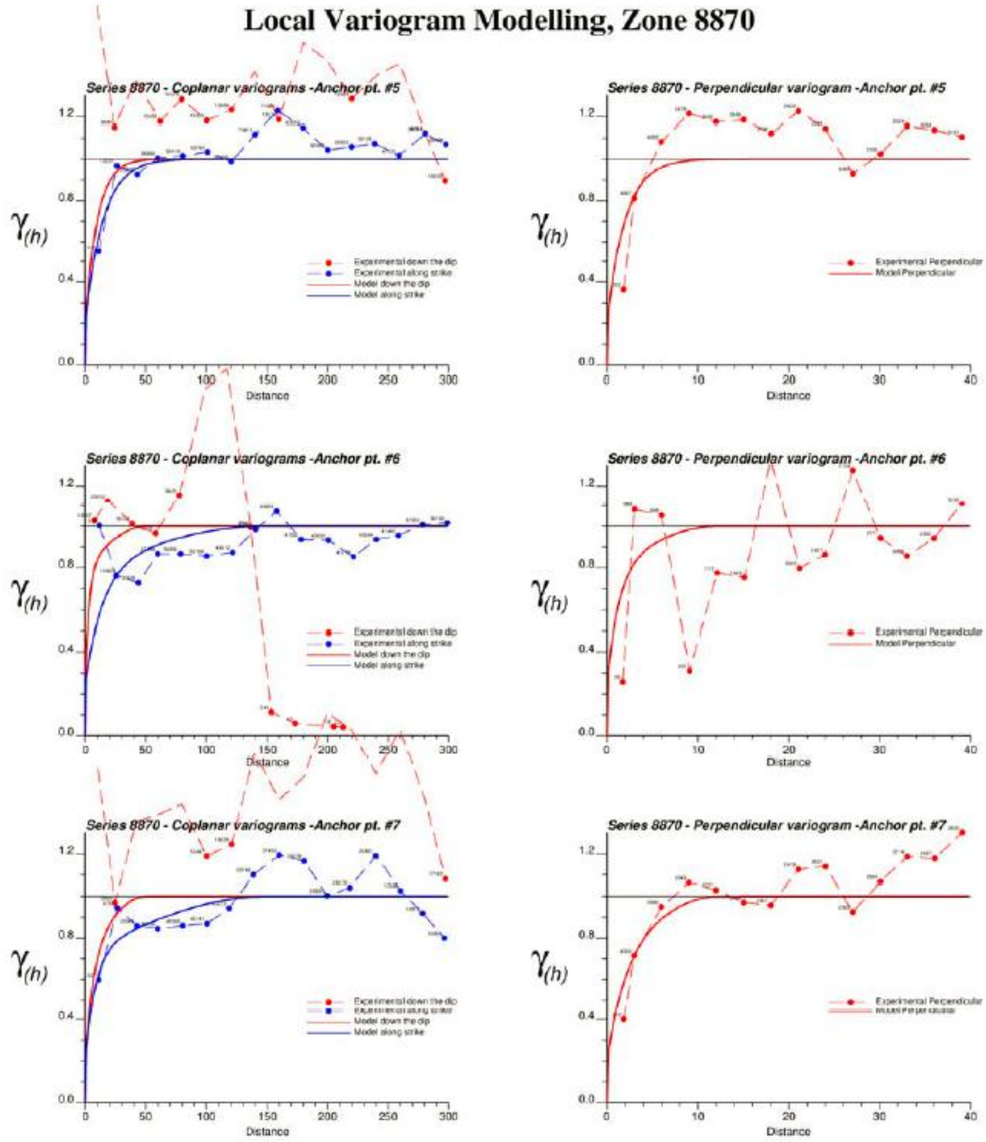


Figure 13-12: Local Variograms, 8870 series, APS 5 to 7.

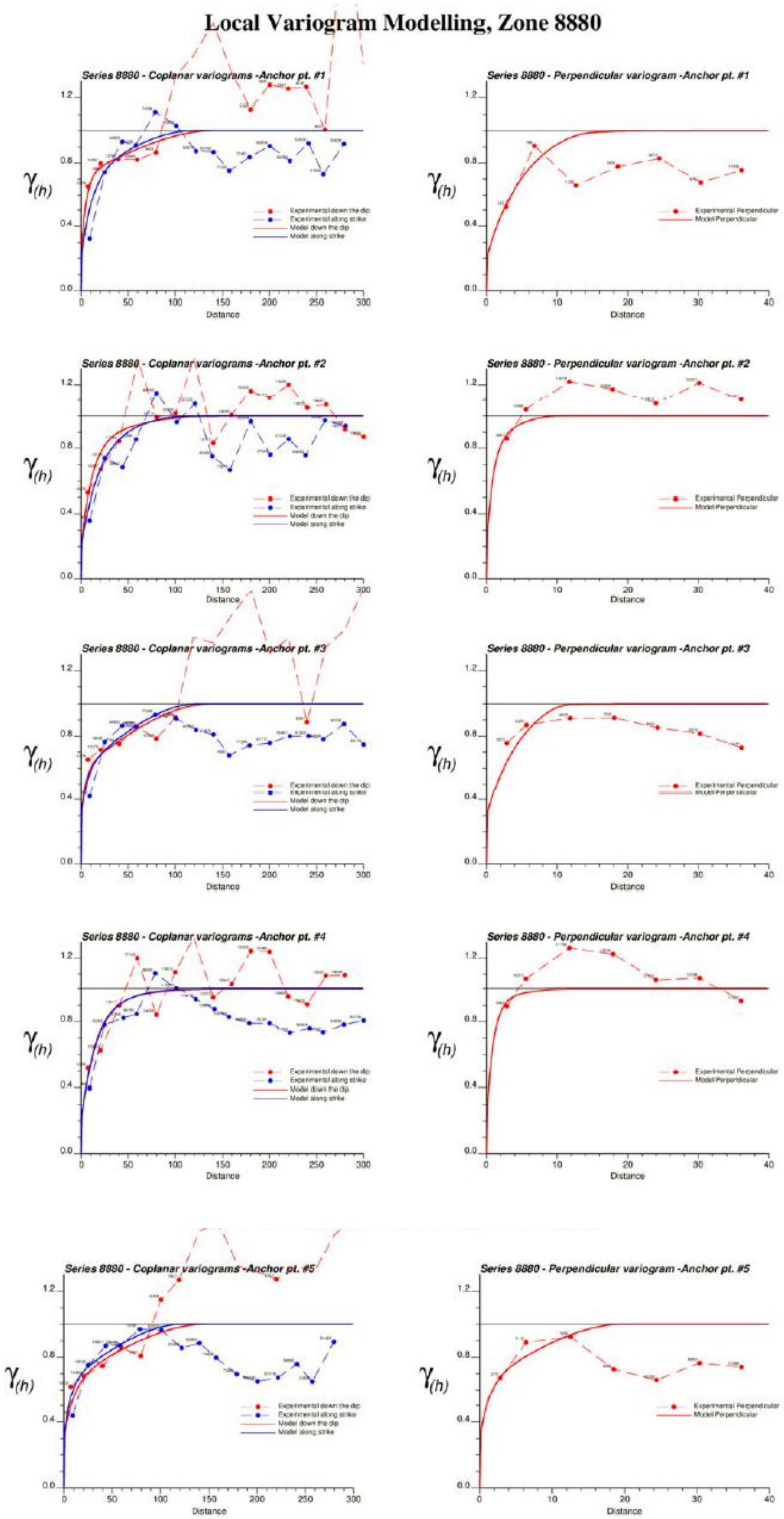


Figure 13-13: Local Variograms, 8880 series, APS 1 to 5.

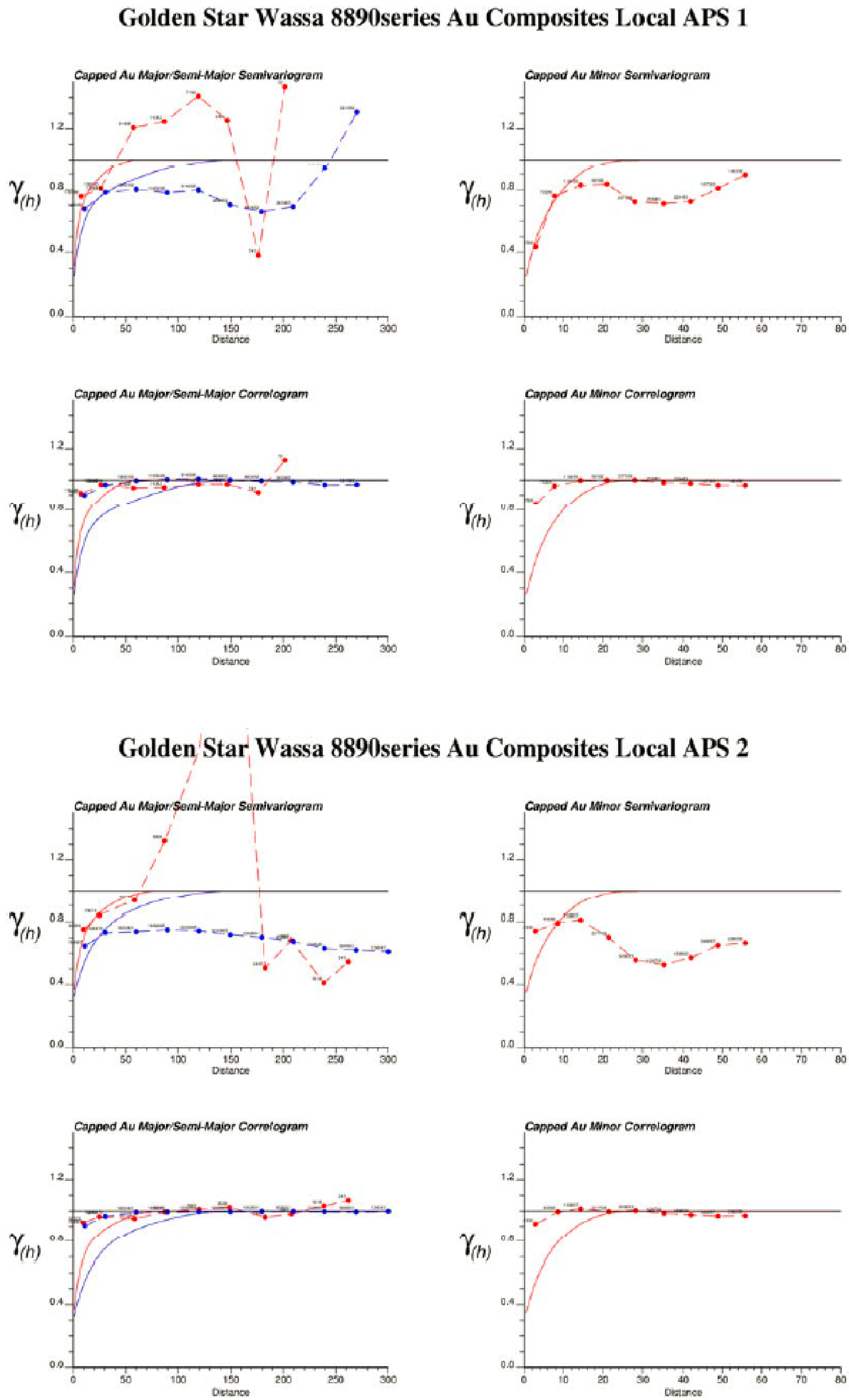
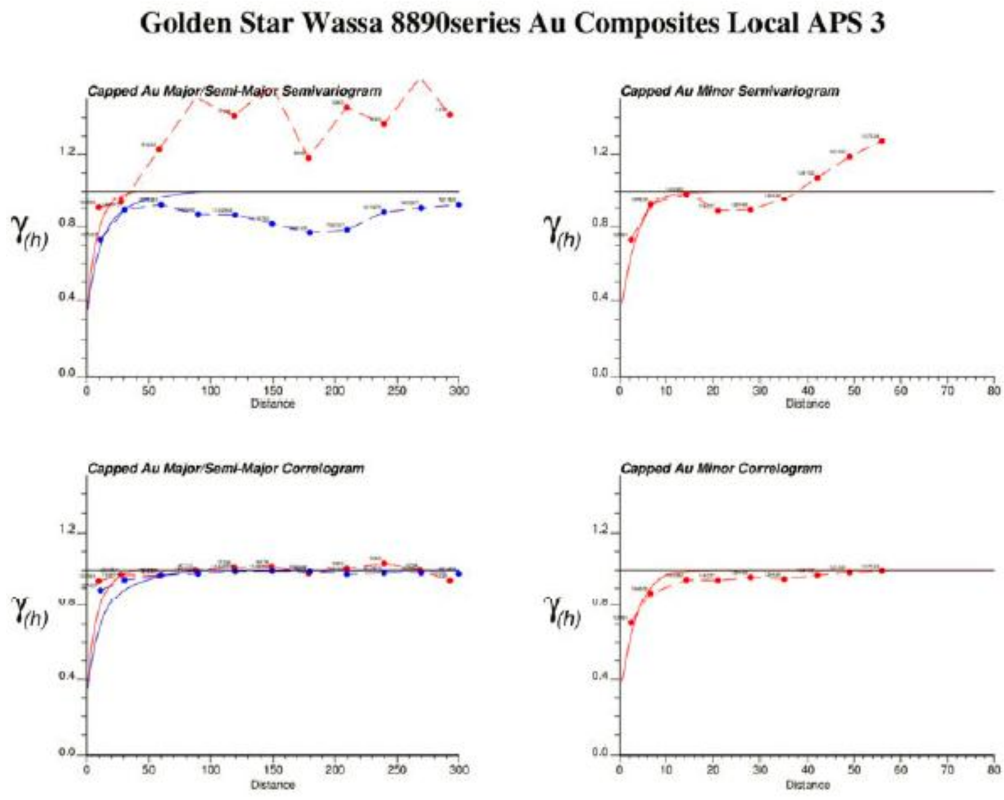


Figure 13-14: Local Variograms, 8890 series, APS 1 and 2.



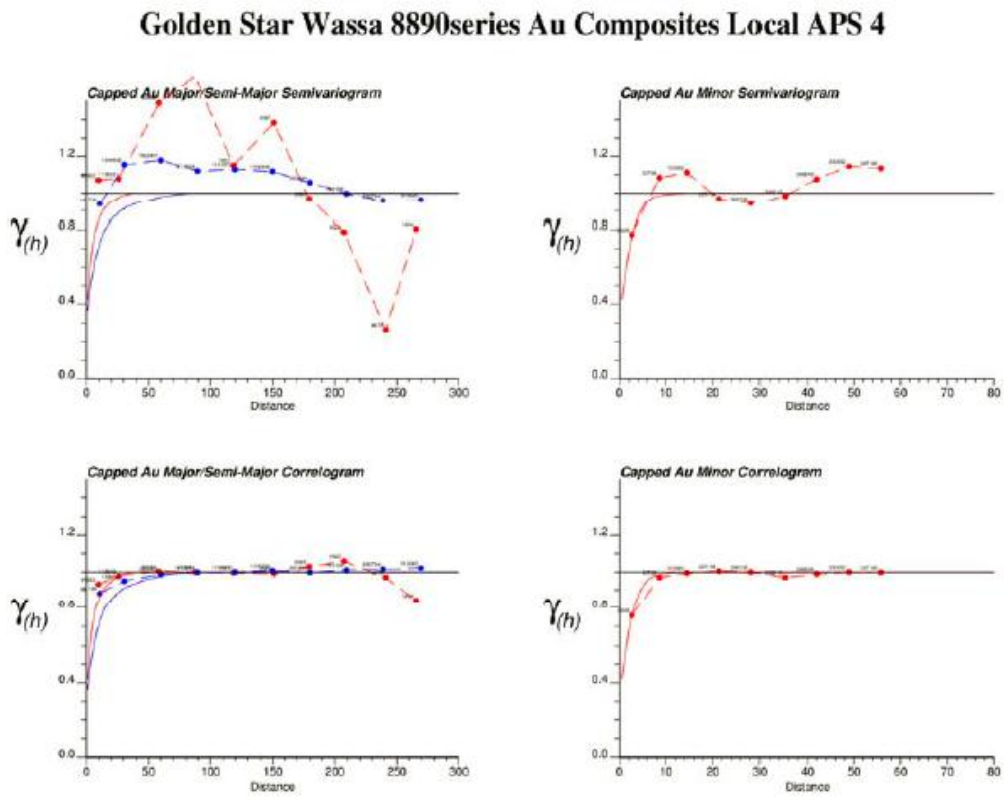


Figure 13-15: Local Variograms, 8890 series, APS 3 and 4.

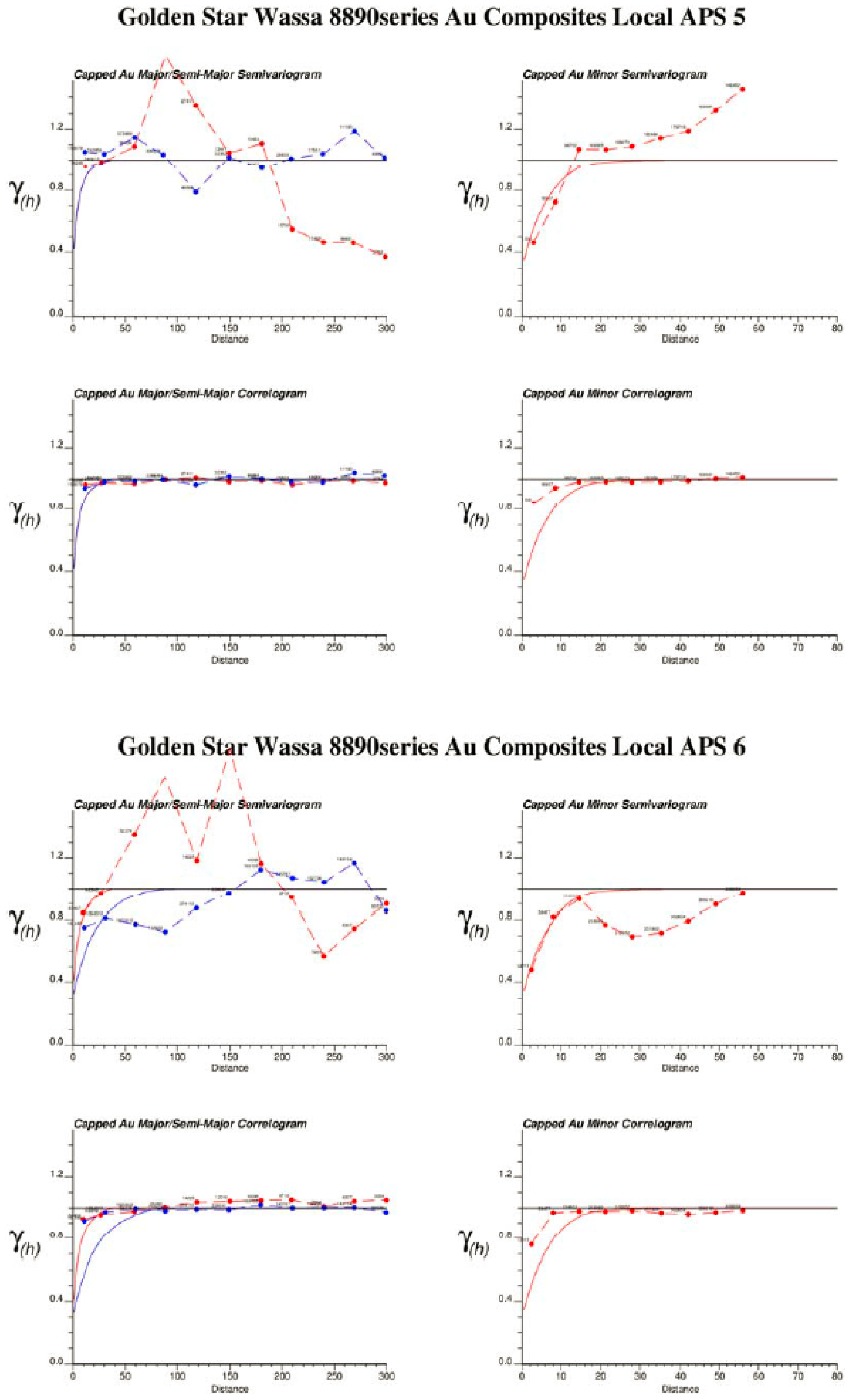


Figure 13-16: Local Variograms, 8890 series, APS 5 and 6.

13.6.5 Grade Estimation

Prior to grade estimation, SRK evaluated contact plots between series (Figure 13-17) to determine whether hard or soft boundary between wireframe series would be reasonable. A comparison of the mean grade across boundaries showed that a soft boundary approach may be reasonable between the 8870 and 8880 series, 8870 and 8890 series, and 8880 to 8890 series. This was not the case for the 8850 series; however, SRK notes that relative to the other series, there is much less data at the boundary with the 8850 series. Given the location of the 8850 series relative to the other wireframes, SRK chose to proceed with a soft boundary treatment to ensure a grade transition around the major NE (F4) fold plane.

SRK also evaluated the contact plot of the HG zones and the corresponding low grade series (see Figure 13-18). The abrupt change of the mean grade between the high grade zones to the surrounding low grade zones supports the use of a hard boundary. SRK ran estimation sensitivities to determine the impact of a hard and soft boundary within the high grade zones. Visual and quantitative comparisons indicated that any differences are immaterial; SRK chose to proceed with a soft boundary between the two series of high grade zones.

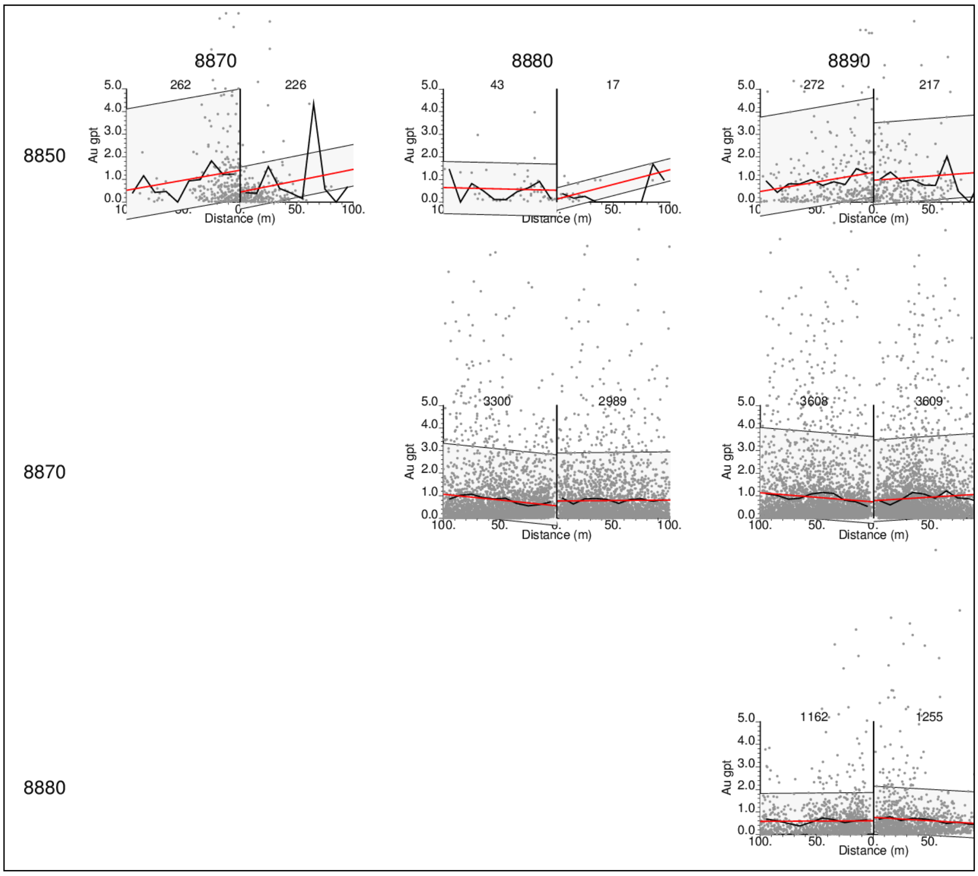


Figure 13-17: Contact plot between series.

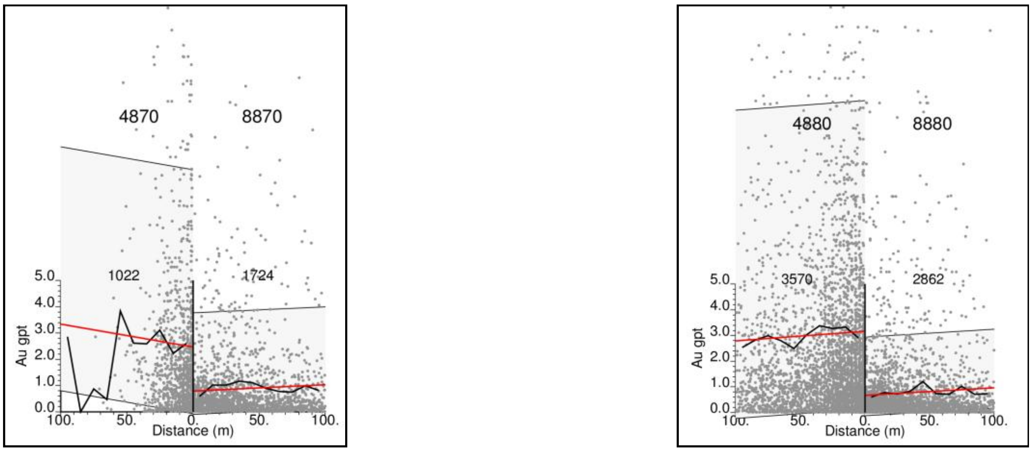


Figure 13-18: Contact Plots of high grade and low grade within series: 8870/4870 (left) and 8880/4880 (right)

Using these boundary treatments, and the models of local angles and local variograms, SRK performed the grade estimation using a local ordinary kriging methodology. The parameters for each of the series are summarized in Table 13-14. First and second estimation runs considered search neighbourhood sizes nominally based on the variogram ranges of the second structure. The main difference between these two passes was that the first pass required a minimum of two boreholes for an estimate to be obtained. The third estimation run considered search ellipses sized at least twice the variogram ranges. As the estimation considered a stationary search ellipsoid, these ranges were selected to ensure that local estimation yielded estimates that conformed to the local anisotropy.

Table 13-15 shows statistics about the percentage of blocks estimated on a by-pass basis per series. In all four major low grade series, over 50% of the mineralized blocks were estimated by the first pass, and approximately 80% were estimated in the first two passes. All high grade blocks were estimated within these three passes.

Table 13-14: Estimation Parameters

Series	Pass	Composites		Maximum composites per hole	Search Ellipse			GSLIB		
		Min.	Max		Svx* (metre)	Svy* (metre)	Svz* (metre)	A1	A2	A3
8850	1	4	12	3	50	50	50	155	-45	0
	2	3	16		90	90	50	155	-45	0
	3	1	16		200	200	100	155	-45	0
8870	1	4	12	3	50	50	50	270	-65	0
	2	3	16		100	150	50	270	-65	0
	3	1	16		200	200	100	270	-65	0
8880	1	4	12	3	50	50	50	260	-55	0
	2	3	16		150	125	50	260	-55	0
	3	1	16		200	200	100	260	-55	0
8890	1	4	12	3	50	50	50	270	-50	0
	2	3	16		80	90	50	270	-50	0
	3	1	16		200	200	100	270	-50	0

Table 13-15: Summary of Block Estimated by Estimation Pass

Series	Pass			Total Estimated Blocks	Total Blocks in Series	Total % Estimated	% Estimated by Pass		
	1	2	3				1	2	3
8850	71,683	22,251	14,280	108,214	108,214	100%	66%	21%	13%
8870+HG	171,213	56,250	27,028	254,491	276,093	92%	62%	20%	10%
8880+HG	253,339	153,044	49,250	455,633	473,804	96%	53%	32%	10%
8890	104,477	21,726	22,290	148,493	166,371	89%	63%	13%	13%
HG Series									
4870	7,783	204		7,987	7,987	100%	97%	3%	0%
4880	23,546	17,759	726	42,031	42,031	100%	56%	42%	2%

13.6.6 Model Validation

SRK checked the resultant block model by considering visual comparisons of block grades and nearby composites via a sectional approach. The results of this check were reviewed with GSR during a meeting in the SRK (Canada) office in Toronto on 13 August, 2014.

In addition to visual inspections, SRK also compared the block model distribution with the declustered, change-of-support corrected distribution of the informing composites. Declustering mitigates the influence of preferential sampling of borehole data; this often results in a distribution of composites whose mean statistic is often comparable to that of the estimated model. Further, a change-of-support correction is applied to account for the volume difference between the composite scale and the block volume scale. SRK applied a discrete Gaussian model (DGM) change-of-support as this model is the most robust and practical for volume correction. Figure 13-19 shows the result of this comparison, which shows that the block model reproduces well the analytically derived mean and standard deviation that may be reasonably expected. SRK notes that the 90th quartile for both distributions is approximately 2.0 g/t Au; thus deviations at the high end of the distribution (i.e., higher than 2.0 g/t Au) appear to be significant, these represent less than 10% of the grades.

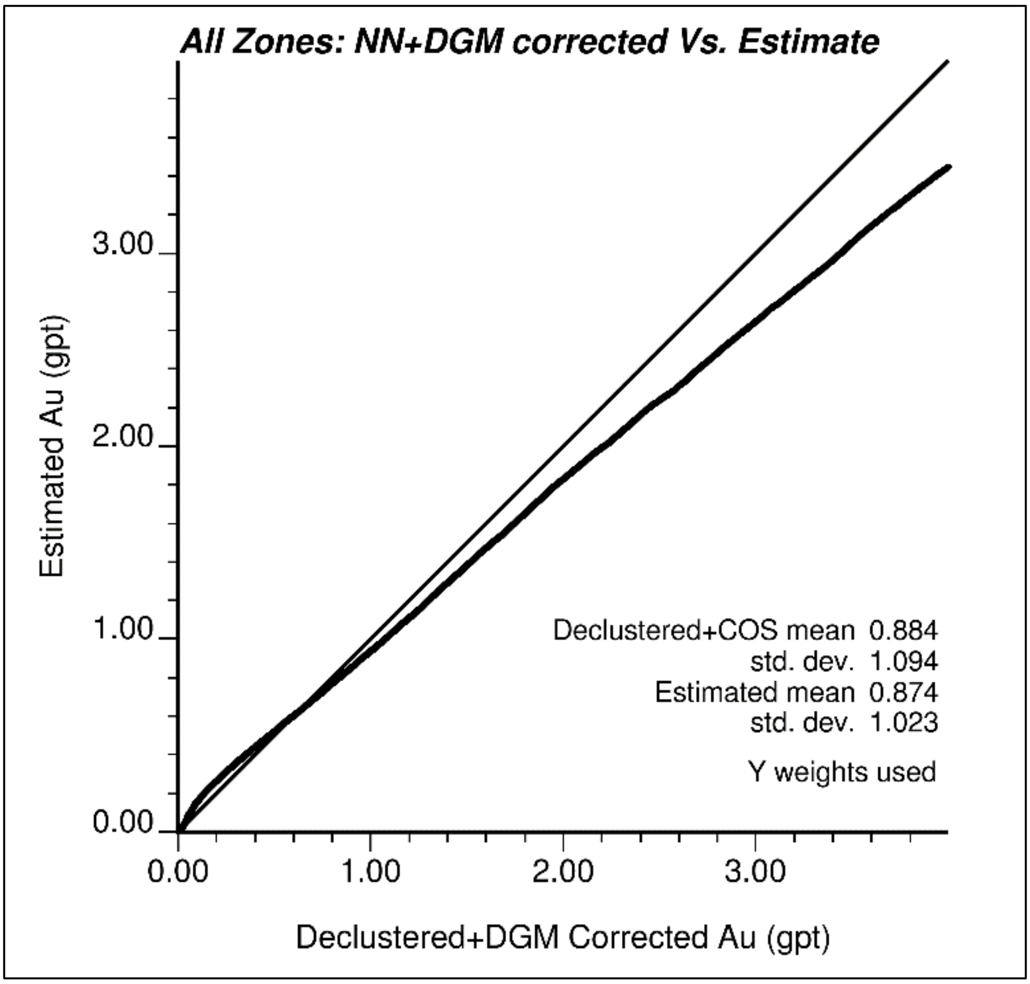


Figure 13-19: Quantile-Quantile comparison of Block Model grades to declustered change-of-support corrected gold distributions.

The model was delivered on 13 August, 2014 for further review by GSR. SRK understands that GSR compared this model with the previous resource model within various resource and reserve pit shells (which were based on the previous model) and relevant open pit and underground cut-off grades. The results of GSR review of the model was shared with SRK, and suggests that this resource estimate compares reasonably well against previous models while reflecting the additional drilling and revised geology model.

To facilitate GSR task of resource classification, SRK provided the following fields as part of the transferred block model:

- Number of boreholes used to estimate the block;
- Number of composites used to estimate the block;
- Average distance of the informing composites;
- Estimation variance;
- Local block covariance; and
- Pass number.

GSR requested SRK to provide some input into classification considerations. After review of the borehole data and their relative abundance to the various block model fields recorded (see above), SRK suggests that a combination of metrics be considered for classification. For instance, Indicated resources may consider those blocks estimated in Pass 1 with a minimum of three boreholes. Given the relatively short ranges used in this initial pass, this is nominally equivalent to 30 x 30 m drill spacing. SRK cautions that this criteria and/or values should not be used directly for classification. In conjunction with assessing confidence in the geology model, the quality control of the resource database and the reasonable prospect for economic extraction, Golden Star should conduct a thorough review of these quantitative factors for the resource estimation prior to determining the classification criteria.

13.7 Mineral Resource Classification

GSR has classified and reported the Wassa gold Mineral Resource in accordance with the NI 43-101 guidelines and the CIM standards on Mineral Resources.

A range of criteria were considered by GSR when addressing the suitability of the classification boundaries to the mineral resource estimate in accordance with SRK's recommendations which included:

- Geological continuity and structural control;
- Drill spacing and mining information;
- Modelling technique; and
- Estimation properties including search strategy, number of composites, average distance of composites from blocks and kriging quality parameters such as interpolated pass.

Indicated Mineral resources were defined where geological confidence for volume and grade definition was high, defined by:

- Good support from drilling, where drilling was averaging a nominal 20-30 m or less along strike/down dip spacing; and
- Areas where estimation quality was high and a majority of blocks estimated during the first interpolation pass.

Indicated blocks were mainly interpreted during the first interpolation pass. GSR estimates that drilling centres spaced within a nominal 25 m north/south were sufficient to classify the global resources as Indicated, given the current standards of drilling, sampling, assaying and geological understanding at Wassa. This classification was one where the level of geological knowledge and data are sufficient to assume the continuity of shape and grade characteristics to a reasonable level of confidence. Confidence in this estimate was sufficient to allow reasonable quantification of global metal content. It is therefore reasonable to expect that further resource definition drilling within the Indicated areas may result in significant material departures both positive and negative from the current estimate.

Inferred resources were defined where geological confidence for volume and grade definition was low, defined by:

- Areas where drilling was averaging a nominal 40 to 200 m along strike/down dip spacing; and
- Areas where the estimation quality was moderate to low with a majority of blocks estimated during the second and third interpolation pass.

The Inferred Mineral Resource is considered to be of significantly greater uncertainty than the Indicated Mineral Resource. It is therefore reasonable to expect that further resource definition drilling within the Inferred areas may result in significant material departures both positive and negative from the current estimate.

All blocks not meeting the Indicated and Inferred classification criteria above, including unestimated blocks remained unclassified.

Conversion of Inferred Mineral Resources to Indicated Mineral Resources will be achieved through underground drilling. The deeper Inferred Mineral Resources will be converted to

Indicated Mineral Resources from drilling conducted underground once decline access has been established. Underground development will be conducted well ahead of mining enabling additional conversion drilling to be done for further delineation of resources and reserves. Underground mining operations typically convert resources through subsurface drilling only once access has been provided and there is a need to increase resources and reserves to extend the mine life. Inferred resources are sufficient to justify underground exploratory development from which further drilling is to be conducted for resource conversion and detailed engineering studies for reserve consideration.

13.8 Mineral Resource Statement

The following section presents the combined Mineral Resource statement for the Wassa Main deposits. Mineral Resources are reported inclusive of the material which makes up the Mineral Reserve however due to the preliminary nature of this technical report there are no Mineral reserves to report. Further detail on the status of Mineral Reserves for Wassa is provided in Section 14 of this report. The Mineral Resource Statement is presented in accordance with the guidelines of the Canadian Securities Administrators' National Instrument 43-101.

SRK (Canada) was commissioned to construct a mineral resource model with estimated gold grades for the Wassa Main deposit, the methodology of the estimate was described in a memo dated 12 September 2014 and authored by Dr Oy Leuangthong and Dr David Machuca. The mineral resource classification and statement was conducted by GSR under the supervision of Mitch Wasel, a Qualified Person pursuant to National Instrument 43-101.

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 determine the quantities of material offering "reasonable prospects for economic extraction" by open pit mining, GSR used a pit optimizer and reasonable mining assumptions to evaluate the proportions of the block model (Indicated and Inferred blocks) that could be "reasonably expected" to be mined from an open pit.

The optimization parameters are based on actuals from the operations, apart from gold price, which is varied to reflect the variations in the Reserve gold price. The reader is cautioned that the results from the pit optimization are used solely for the purpose of testing the "reasonable prospects for economic extraction" by an open pit and do not represent an attempt to estimate Mineral Reserves. The results are used as a guide to assist in the preparation of a Mineral Resource statement and to select an appropriate resource reporting cut-off grade.

GSR considers that the blocks located within the conceptual pit envelopes show "reasonable prospects for economic extraction" and can be reported as a Mineral Resource.

In order to determine the quantities of material offering "reasonable prospects for economic extraction" by underground mining method, GSR used reasonable mining assumptions to evaluate the proportions of the block model (Indicated and Inferred blocks) that could be "reasonably expected" to be mined from underground.

Table 13 16 is the combined Mineral Resource statement for the Wassa Main project. In declaring the Mineral Resources for the Wassa Main deposit, the following are noted:

- The identified Mineral Resources in the block model are classified according to the CIM definitions for Measured, Indicated and Inferred categories and are constrained within a Whittle pit shell using a gold price of US$ 1,400/oz and below the end of month July topographic surface. The Mineral Resources are reported in-situ without modifying factors applied.

- This Mineral Resource estimate is based on appropriate cut-off grades for the oxide and fresh material, and reported within a conceptual Whittle shell. Pit optimisation using industry standard software has been undertaken on the Mineral Resource models using appropriate slope angles, modifying factors (mining recovery and dilution), process recovery factors, costs and a long term gold price of US$ 1,400/oz.
- The underground portion of the Mineral Resource estimate is based on a cut-off grade of 2.2 g/t and a gold price of $1,400/oz and is reported outside the $1,400/oz open pit resource Whittle shell.
- The Mineral Resource models have been depleted using appropriate topographic surveys, to reflect mining until the 31 July 2014.
- Geological modelling of the mineralization was undertaken by GSR, using a cut-off grade of approximately 0.2 g/t Au for the low grade envelopes and 1.0 g/t Au for the high grade wireframes, assays less than 1 g/t Au were often included within the mineralized wireframes in order to model continuity both down dip and along strike. The Mineral Resource estimates are derived from a combination of diamond and reverse circulation drilling techniques, supported by an industry best practice QAQC programme. Drilling is typically carried out on sections spaced at 25 m.
- The Mineral Resources were estimated using a block model with block sizes which typically reflect half the drillhole spacing within the Wassa Main deposit. The composite grades were capped for domains deemed necessary after statistical analysis. Local Ordinary Kriging was used to estimate the block grades.
- Block model tonnage and grade estimates were classified according to the CIM Definition Standards for Mineral Resources and Mineral Reserves (December 2005). The basis of the Mineral Resource classification included confidence in the geological continuity of the mineralized structures, the quality and quantity of the exploration data supporting the estimates, and the geostatistical confidence in the tonnage and grade estimates. Three-dimensional solids were modelled reflecting areas with the highest confidence, which were classified as Indicated Mineral Resources. All figures are rounded to reflect the relative accuracy of the estimate.
- Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

The Wassa Main Resource Statement, as of 31 July 2014, is given below in Table 13-16 using open pit cut-off grades of 0.61 g/t Au for oxide and 0.67 g/t Au for fresh, and underground cut-off of 2.27 g/t Au.

Table 13-16: Mineral Resource Statement as of 31 July 2014.

Source	MEASURED			INDICATED			INFERRED		
	Quantity	Grade	Metal	Quantity	Grade	Metal	Quantity	Grade	Metal
	kt	g/t Au	koz Au	kt	g/t Au	koz Au	kt	g/t Au	koz Au
Wassa Open Pit	-	-	-	25,582	1.41	1,160	237	1.56	12
Wassa Underground	-	-	-	10,116	4.27	1,389	8,841	3.95	1,122
Total	**-**	**-**	**-**	**35,698**	**2.22**	**2,549**	**9,078**	**3.89**	**1,134**

14 MINERAL RESERVE ESTIMATES (ITEM 15)

No Mineral Reserves have been estimated for the purposes of the PEA for the Wassa combined open pit and underground project in Ghana.

Recent Mineral Reserve estimates have been prepared by:

- SRK Consulting (UK) Limited – NI 43-101 Technical Report with effective December 31, 2012, filed 21 March 2013; and
- GSR – Year-end reserve estimate effective December 31, 2013.

Both of the above recent reserve estimates only relate to open pit operations while this PEA considers a combined open pit and underground operation.

15 MINING METHODS (ITEM 16)

15.1 Introduction

At the time of writing, mineral resources are recovered from the Wassa Main open pit mine and no mining utilizing underground methods is currently taking place. The Wassa PEA considers continued mining of the Wassa open pit in combination with underground mining of potentially economic mineralization.

The study supporting this preliminary economic assessment has not been carried out to a level of detail usually associated with a pre-feasibility study or feasibility study. Consequently, no Mineral Reserve estimate has been prepared and any Mineral Resources and conceptual production stated herein should not be considered to be representative of Mineral Reserves as defined by the CIM Standards on Mineral Resources and Reserves.

The resources available for mining that form the projected LoM plan are contained within a portion of the resource block model judged to be potentially mineable by underground methods. These resources include a significant quantity of Inferred material. For conceptual production, they have been modified to include estimates of mining dilution and mining recovery.

The proposed approach to mining in this PEA considers continuation of open pit mining at an initial rate of 3 Mtpa in 2015 which is depleted over a period of 5 years at a decreasing rate of production. Underground production would commence in 2015, initially as a supplemental feed, and ramp up to a production rate of 1 Mtpa for an extended period.

15.2 Open Pit Mining

15.2.1 Recent Production

The recent historical mine production at Wassa is provided in Table 15-1. The mine achieved an average ROM production rate of 2.4 Mtpa and total material movement of 18.5 Mtpa from Wassa and the other contributing satellite pits (Benso and Hwini Butre). The open pit mining plan for the purpose of this PEA is to source mineral resources only from the Wassa Main deposit.

Table 15-1: Recent open pit mine production for the Wassa Assets

Deposit	Type	Units	2009	2010	2011	2012	2013	H1 2014
Wassa Main	Total ROM	Mt	0.9	0.8	1	1.8	1.3	0.5
		g/t	1.42	0.97	1.66	0.96	1.23	1.32
	Waste	Mt	4.2	7.3	6.3	7.9	7.2	3.03
	TMM	Mt	5.1	8.1	7.4	9.7	8.5	3.5
	Strip Ratio	$t_{waste}:t_{min.}$	4.8	9	6.1	4.4	5.6	6.1
Benso	Total ROM	Mt	0.9	0.7	0.9	0.1	-	-
		g/t	4.09	2.69	1.96	1.79	-	-
	Waste	Mt	8.4	6.4	3.7	0.2	-	-
	TMM	Mt	9.4	7.1	4.6	0.3	-	-
	Strip Ratio	$t_{waste}:t_{min.}$	9.2	8.8	4.1	2.2	-	-
Hwini Butre	Total ROM	Mt	0.4	1	0.6	0.7	0.8	0.2
		g/t	4.46	3.63	3.22	4.7	5.07	3.34
	Waste	Mt	4.1	5.5	5.3	7.8	6.0	1.4
	TMM	Mt	4.5	6.5	5.9	8.5	6.8	1.6
	Strip Ratio	$t_{waste}:t_{min.}$	9.4	5.3	8.9	11.5	7.9	9.1
TOTAL	**Total ROM**	**Mt**	**2.2**	**2.6**	**2.5**	**2.6**	**2.1**	**0.65**
		g/t	**3.12**	**2.52**	**2.13**	**1.97**	**2.66**	**1.80**
	Waste	**Mt**	**16.7**	**19.2**	**15.4**	**15.9**	**13.3**	**4.4**
	TMM	**Mt**	**18.9**	**21.7**	**17.9**	**18.5**	**15.3**	**5.1**
	Strip Ratio	$\mathbf{t_{waste}:t_{min.}}$	**7.5**	**7.5**	**6**	**6.2**	**6.5**	**6.8**

15.2.2 Geotechnical

Pit Stability

The Wassa pit slopes are formed in a strong, competent but well jointed rock mass and are generally stable and able to stand at relatively steep inter-ramp angles. Small scale instabilities have occurred but these are generally restricted to the upper, weathered sections of the slope in areas of poor surface drainage. The main geotechnical problem at Wassa has been lack of berm retention due to the well jointed nature of the rock mass and the relatively narrow berms required to achieve steep inter-ramp angles. Whilst not impacting the overall stability of the pit slopes the lack of berms has given rise to an elevated rockfall risk at the base of the pit. GSR has trialled limit and pre-split blasting with variable success and SRK has suggested that the mine utilise double benching to increase berm width and improve berm retention.

Pit Slope Design

Because of the expansion of the resource at Wassa Main pit and the merging of the individual pits the current design pit is significantly deeper than the individual pits. The current pit design utilises slope angles developed for shallower pits. SRK considers that this design will require a detailed geotechnical assessment to support and confirm that the slope angles used are appropriate for the deeper pit. SRK understands that GSR staff are geotechnically logging the core from all the new diamond drill holes that have been used to expand the Wassa Main resource. This data will provide a robust database that can be used to confirm slope design parameters for the deeper pit.

Geotechnical Management

GSR retains in-house geotechnical capability, though SRK has provided geotechnical operational review and design services to GSR since 2003. The last full geotechnical review of the Wassa pits was carried out by SRK in June 2011. In September 2012 SRK visited site to provide training in geotechnical logging to the site geologists and geotechnical engineers as well as providing instruction to the geotechnical engineers in kinematic analysis.

Geotechnical Risks

Given the hard rock environment within which the Wassa pits are excavated the risk of large scale slope instability is considered to be low. As highlighted above the main operational risk relates to the difficulty of excavating catch berms of sufficient width to retain rockfall. This risk can be mitigated by a combination of improved limit blasting and double benching to increase berm width.

With regard to the expanded Wassa Main Pit, SRK considers that there may be a risk to the Reserve utilising the current slope angles. This risk can be mitigated by undertaking a detailed geotechnical assessment of the expanded pit slope designs, using the geotechnical information collected from the recent Resource drilling campaign. This will allow improved confidence in the slope angles used or modification to the angles depending on the outcome of the assessment.

15.2.3 Mining Methods and Equipment

A conventional approach to open pit mining is currently being used at the Wassa mine employing excavators and trucks which are considered typical for this type and style of gold mineralization. The same approach to equipment and open pit mining are used for the purposes of the Wassa PEA. Drilling and blasting is conducted over bench heights of 5 or 6 m and explosives are delivered to the hole by the manufacturer. Oxide or weathered material is generally only required to be lightly blasted or in some areas can be excavated as ‘free dig’. Hydraulic excavators are used to achieve good selectivity, and in conjunction with good blasting practice, mine to a 2.5 or 3.0 m flitch height. Broken rock is loaded to 95 t capacity off-highway haul trucks to a central stockpile or to the waste dump. A summary of the equipment in use at Wassa is tabulated in Table 15-2.

Table 15-2: Wassa Mining Equipment as at end-2013

Description	Total
Excavators	4
Haul Trucks	20
Blasthole Drill Rigs	5
Dozers	6
Graders	2
Water Truck	2
Total	**39**

The main production fleet in terms of excavators, haul trucks and rotary drill rigs is principally sourced from the Liebherr, Caterpillar and Atlas Copco manufacturers respectively. In addition to the main equipment units there are graders and water trucks as well as various support and service equipment in use. Operations are conducted on a 2 x 11 hr shifts per day basis on a

5-day roster.

Grade control drilling is undertaken and the results used to delineate the mineralized zones for excavation as well as low grade material and waste.

15.2.4 Modifying Factors and Cut-off Grade

The CoG for the Wassa open pit oxide and fresh material based on various estimates and assumptions, including:

- Gold price of US$ 1,300 per ounce;
- A Government Gross Revenue royalty of 5%;
- A process plant recovery for oxide and fresh material of 92.6%. Processing costs are based on US$16.57/t for oxide and US$17.50/t for fresh material treated;
- The base mining cost (including grade control) is estimated to ranging from US$2.07 to 2.52 per tonne;
- Haulage costs are estimated at between US$0.27 to US$0.48 per tonne;
- Rehabilitation costs of US$0.12 per tonne processed; and
- G&A cost of US$5.54 per tonne processed.

The technical and economic assumptions used in the derivation of the CoG are comparable to that historically achieved at Wassa and are summarized below in Table 15-3.

Table 15-3: Cut-off grades for oxide and fresh material for the Wassa Deposits

Deposit	Cut-off grade (g/t Au)	
	Oxide	Fresh
Wassa Main	0.69	0.72

The CoG includes a mining recovery of 95% and dilution of 10% which is considered appropriate for the mining methods and mineralization characteristics and is supported by mining experience.

15.2.5 Pit Optimization

A pit optimisation exercise was undertaken by GSR at the Wassa Main using Whittle Four-X software (“Whittle”) and imported resource models with an estimate of the topographic surface as at End of Month (“EOM”) July 2014. The imported block model is sized at 10 x 10 x 6 m blocks.

The Whittle pit optimisation utilised various technical and economic assumptions obtained from a combination of operating history, experience and company objectives with regards to gold price and pit shell selection. The Company’s Base Case gold price for 2014 mine planning is US$ 1,300 per ounce. A summary of the principal optimisation parameters is given in Table 15-4.

Table 15-4: Wassa Pit Optimisation Input Parameters

Parameter	Unit	Oxide	Fresh
Revenue			
Au price	US$/oz	1,300	
State royalty	%	5	
Selling costs (+5% royalty)	US$/oz	1	
Mining Parameters and Costs - Wassa			
Mining recovery	%	95	95
Dilution	%	10	10
Overall slope angle	deg.	45	52
Reference Elevation	m	154	154
Base Mining cost	US$/t	2.84	2.99
Incremental Pit Depth Cost	US$/t/m	0.003	0.003
Processing Parameters and Costs			
Haul to Plant	US$/t	0.27	0.48
Process plant recovery	%	92.6	92.6
Process Cost	US$/t	16.57	17.50
Other Costs			
G&A Cost	US$/t	5.54	5.54

GSR undertook a number of Whittle optimisations and used the $1,300/oz shell as a guide to the final mine design. However, only a portion of the $1,300/oz shell material is planned to be mined in this design. The majority of the pit is designed to an $1,100/oz pit shell in order to improve cash flow and reduce up front stripping requirements.

Figure 15-1 shows the pit optimisation results in terms of material movement, gold grade, strip ratio and undiscounted cash flow (“UDC Flow”) for a range of gold prices.

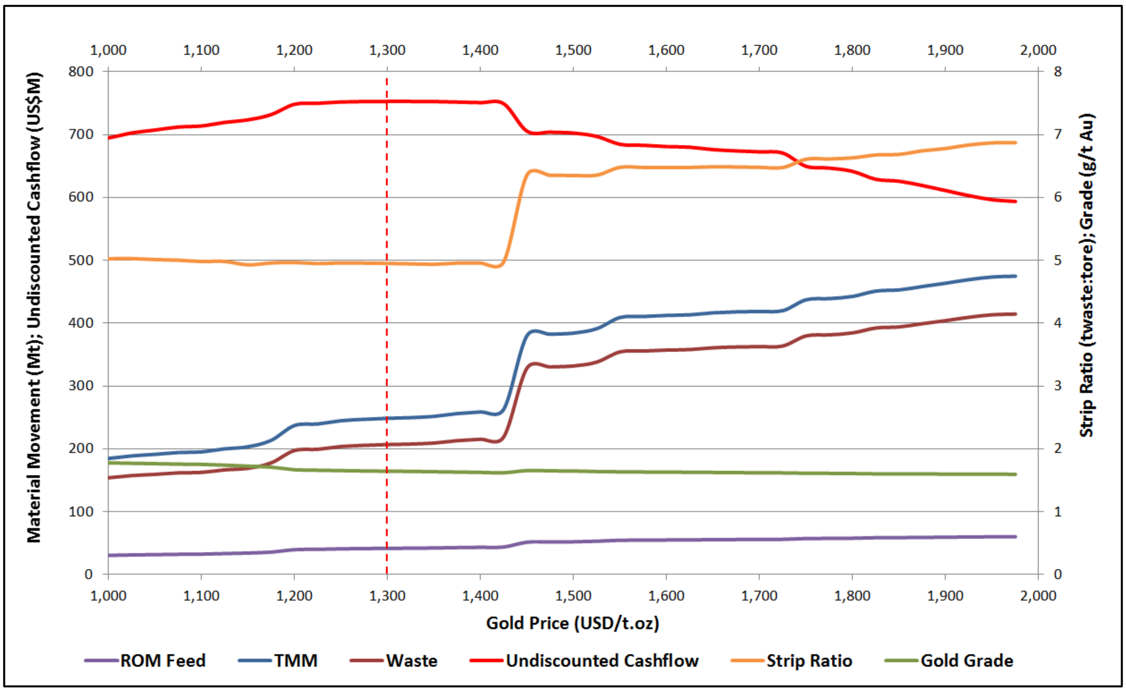


Figure 15-1: Wassa Pit Optimization Results – Material Movement, UDC Flow, Gold Grade and Strip Ratio

15.2.6 Open Pit Design

The topography as of EOM July 2014 was the starting point for the open pit design and is shown in Figure 15-2.

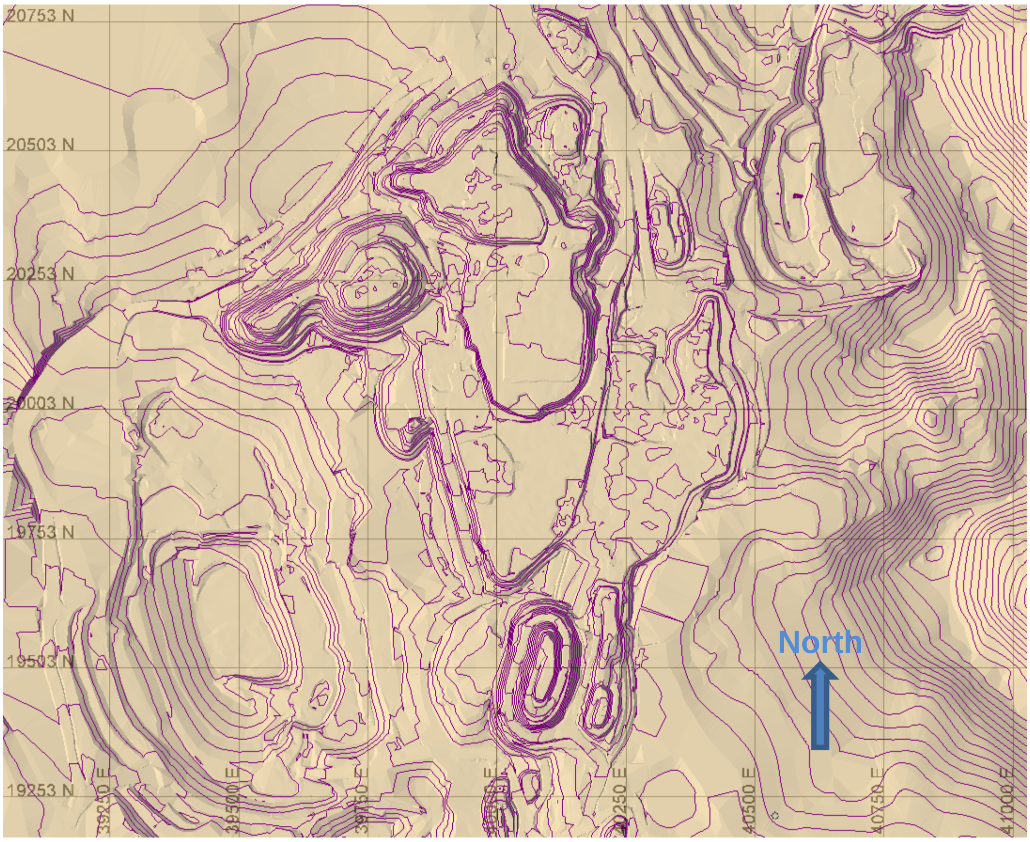


Figure 15-2: Wassa Main Topography as at July, 2014 (GSWL)

The final pit design slopes and benches are based on the following geotechnical parameters provided in Table 15-5.

Table 15-5: Wassa Open Pit Design Geotechnical Parameters

Parameter	Unit	Wassa Main
Oxide Slopes		
Berm Width	m	5
Bench Height	m	12
Batter Angle	deg.	65
Overall Slope Angle	deg.	45-50
Inter-ramp Slope Angle	deg.	45-50
Fresh Slopes		
Berm Width	m	5
Bench Height	m	12
Batter Angle	deg.	75
Overall Slope Angle	deg.	34-51
Inter-ramp Slope Angle	deg.	50-55

For Wassa Main, the slope parameters quoted above are based on the parameters for the individual pits developed by SRK for the feasibility study, and modified based on experience during mining. The pit slope parameters are height dependent.

The pit ramps are designed at a width of 20 m for two way haulage, reducing to 10 m for one-way traffic in the last 20 m vertical and installed at a gradient of 10%. A minimum mining width of 30 m is utilised assuming the space required for a CAT 777 haul truck to perform a 3-point turn.

The results of the pit design have been utilised in conjunction with the latest block model, topography and face positions to determine the contained mineable resource using the Measured and Indicated Mineral Resource categories only. Figure 15-3 provides a plan view of the final pit design with section lines which show the location of the following figures.

Figure 15-4 to Figure 15-6 provide typical section view of the Wassa Main pits with the Mineral Resources coloured green (Indicated) and blue (Inferred). The section views also provide the outlines for the following:

- Topography as of EOM July 2014; and
- Final Pit Design based on Whittle Pit Shell(s) using only Measured and Indicated Mineral Resources.

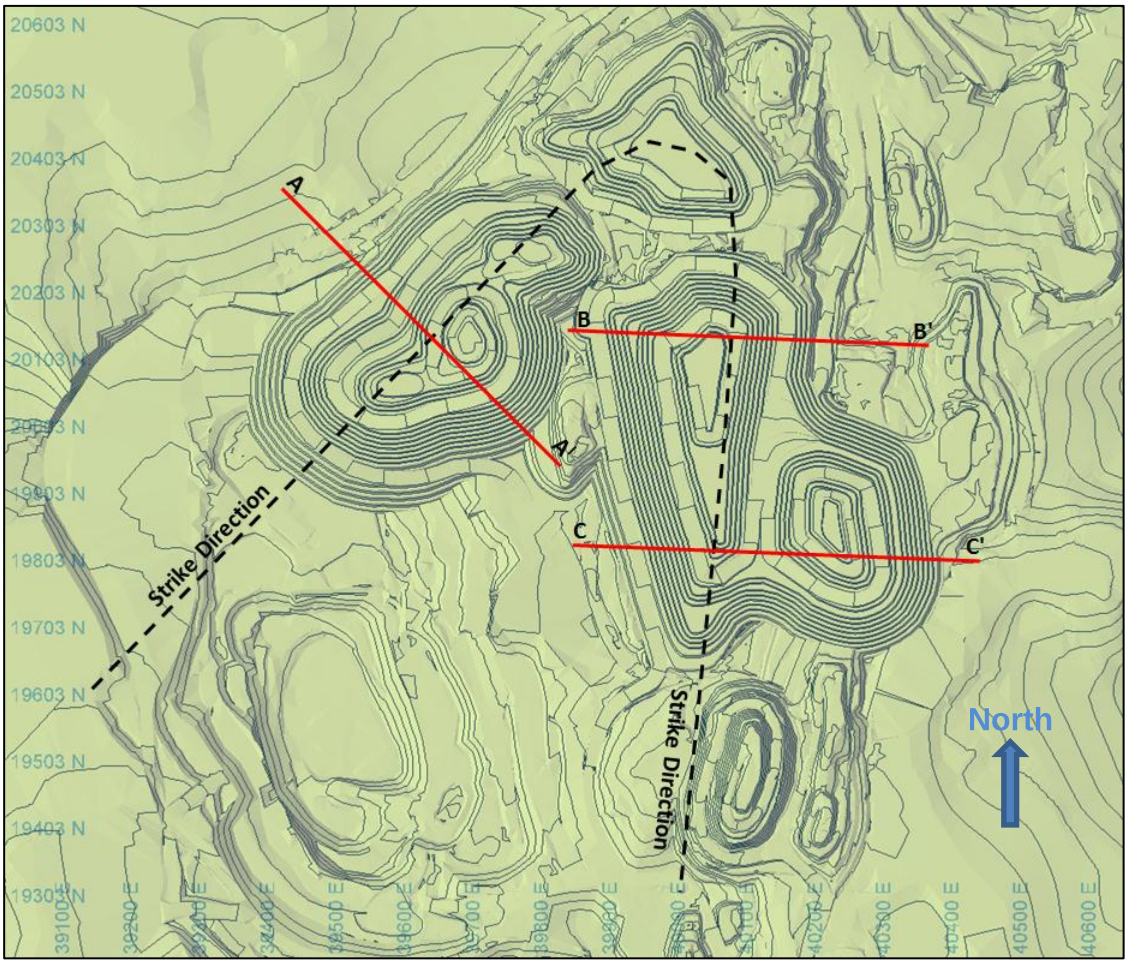


Figure 15-3: Wassa Main Final Pit Design (GSWL)

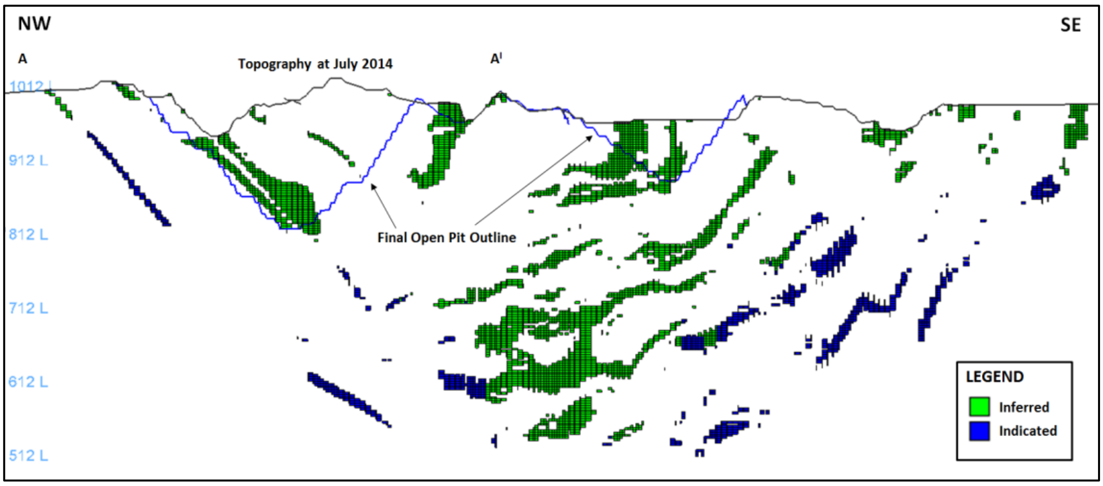


Figure 15-4: Section A – A' View of Wassa Main and mineral resources, looking North East

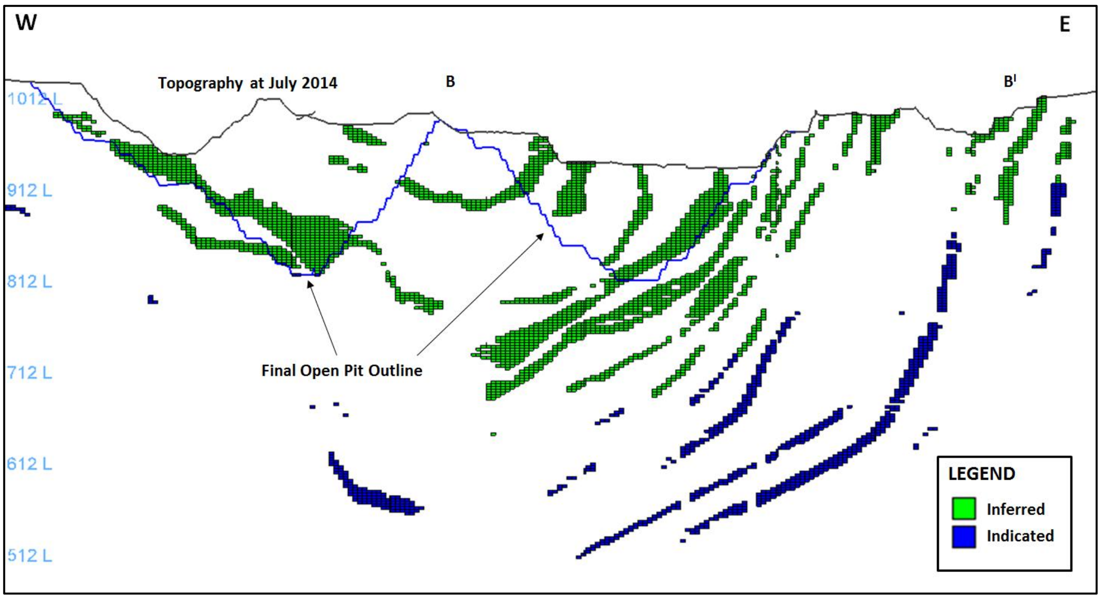


Figure 15-5: Section B – B^l View of Wassa Main and mineral resources, looking North

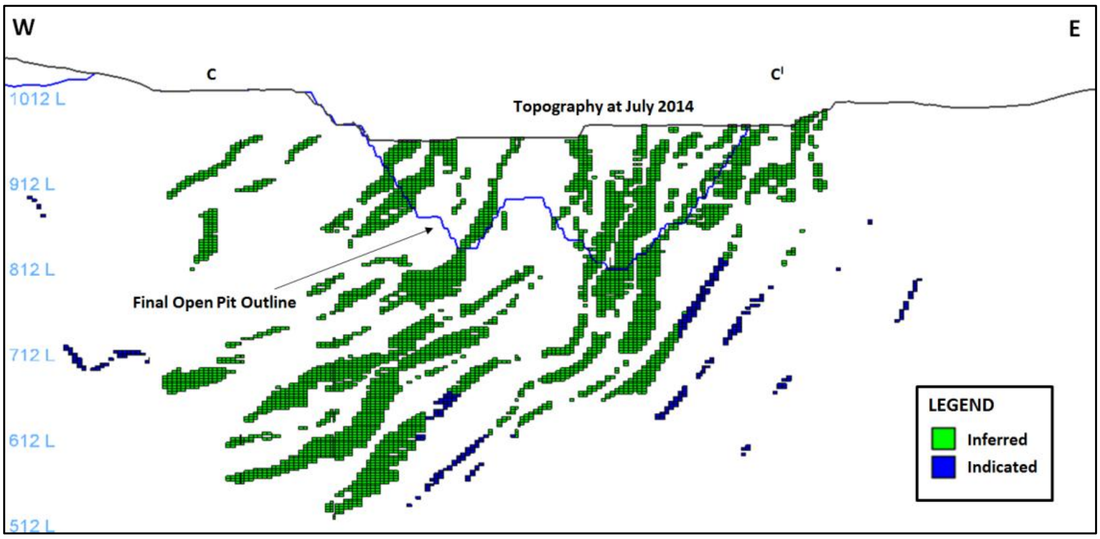


Figure 15-6: Section C – C[l] View of Wassa Main and mineral resources, looking North

15.2.7 Waste Storage Facilities

Waste dumps are located as close to the final pit limit as possible and include design parameters comprising a lift height of 10 m, berm width of 17 m and a batter angle of 35°. The final waste dump is re-profiled to an overall slope angle of 20°.

The waste storage designs for each open pit are incorporated into the mining schedule and have sufficient capacity for a number of years of operation. Further design work is required for waste storage facilities to cover the full mining schedule and opportunities to backfill completed mining areas need to be identified.

15.2.8 Dewatering

Current Dewatering

The pits are dewatered using in-pit pumps placed in sumps located in the lowest point of the pit. Wet season arrangements include drainage diversion channels to manage surface water runoff flows.

This water is managed by pumping it to sedimentation structures and then releasing the water to the receiving environment to comply with relevant permits.

To improve the overall management of surface run-off from the mining areas, five sedimentation structures were constructed in 2012. These are designed primarily to remove suspended solids, which may be elevated during storm events, from the run-off water.

Hydrogeological Setting

Below the weathered zone, where moderate hydraulic conductivities are expected, the hydraulic conductivity in the fresh bedrock is likely to be low with fracture flow being the dominant mechanism for any significant groundwater movement. Jointing in the rock mass may lead to some fracture flow but significant inflows would only be anticipated along larger, interconnected fractures and fracture zones.

Open Pit Inflows

Groundwater inflows will likely comprise a seasonally relatively constant but low contribution to overall pit inflows. Storm water runoff will comprise an additional contribution to pit inflow during discrete periods, typically in the wet season. Storm magnitudes for specific return periods have not been determined at this level of study.

Dewatering Risks

Unexpectedly high groundwater inflows have the potential to occur, particular associated with significant fractures/fracture zones. This risk can be mitigated through targeted hydrogeological testing as part of an integrated structural-geotechnical-hydrogeological investigation programme.

Stormwater runoff can be considerable in the event of a high return period storm event. This risk can be mitigated by both effective stormwater management infrastructure and by calculating storm water volumes entering the pit based on storm events of various return periods. Dewatering infrastructure can be designed accordingly.

15.3 Underground Mining

15.3.1 Introduction

The Wassa Main deposit is currently mined by open pit using conventional truck and shovel methods. A conceptual study was commissioned by GSR in 2013 to assess the underground mining potential below the Wassa Main pit. A high-level study performed by SRK Canada established that the concept showed economic potential and should be taken through to a more detailed PEA. SRK UK was subsequently tasked with the following scope for the PEA:

- A mining methods study, in conjunction with the geotechnical data analysis, in order to determine an optimum production method;
- A mine access study, to determine the best means of access for an underground extension to the mine, balanced against the spatial constraints imposed by the open pit;
- A mining backfill study, to examine backfill options in relation to the mine access and production method;
- An underground stope optimisation, using the Deswik Stope Optimiser software;
- Mining equipment trade-off study, taking into account the mine access, stope sizes production rates;
- A preliminary design and schedule for an agreed upon mining option;
- A preliminary ventilation study; and
- Capital and operating cost estimate, including a cut-off grade assessment, based on contract mining rates.

15.3.2 Geotechnical Considerations

SRK carried out a preliminary geotechnical assessment to provide input to the development of an appropriate mining method and appropriate stope dimensions and support requirements. This work included:

1. Global rock mass and structural characterisation of the Wassa rock mass.
2. A mining method selection assessment.
3. Estimation of stable stope dimensions.
4. Estimation crown pillar thickness between the open pit and underground workings.
5. Estimation of development support requirements.

The Wassa geology team had, under previous instruction from SRK, carried out basic geotechnical and structural logging of all of the cored boreholes used to define the Wassa underground resource. A database of geotechnical information, comprising geotechnical logging of 90,000m of borehole core, was available for this study.

Using this information SRK carried out global geotechnical characterisation of the mineralization, footwall and hangingwall rock masses at Wassa. The outcome of this work indicated that there was little difference in the engineering characteristics of these rock masses with all being of generally good to very good rock mass quality. The rock mass is characterised by widely spaced jointing, high RQD (>90%) and high intact rock strength (>150MPa). The Q classification values range from 5 to 32, with an average value of 22. Some minor weaker units were recorded in the mineralization and hangingwall which could relate to fault zones.

Mining method selection, using the Nicholas (1981) method confirmed that sub-level stoping or a variant thereof was the most suitable mining method for the prevailing geometrical and geotechnical conditions at Wassa.

An estimate of stable stope span was made by use of the Modified Stability Graph Method through the use of the Modified Stability Number, N', as described in Hutchinson and Diedrichs (1996). From that analysis SRK prepared two design charts for estimating stable stope plan dimensions and stable stope wall height to guide the PEA mine design which are presented as Figure 15-7 and Figure 15-8.

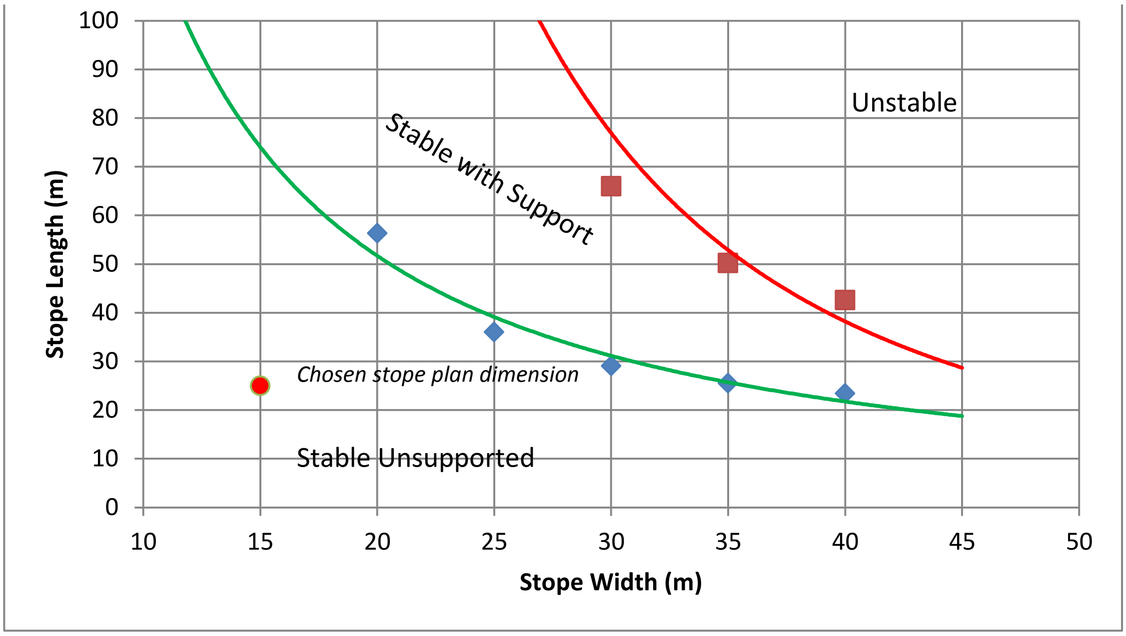


Figure 15-7: Stope Plan Dimension Design Chart

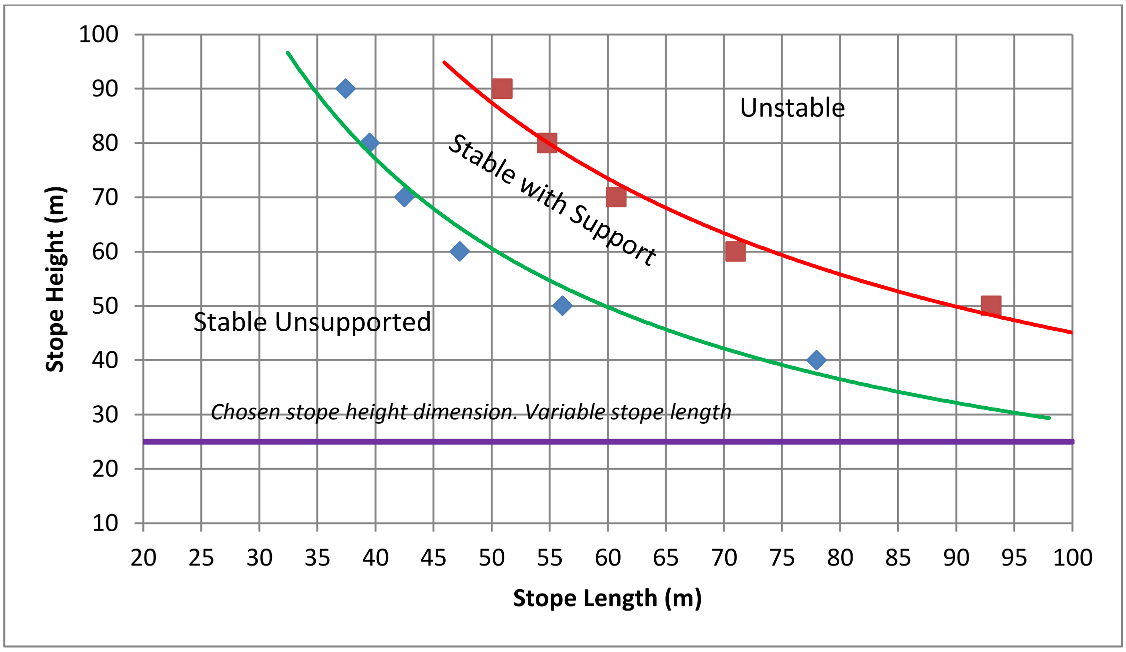


Figure 15-8: Stope Wall Dimension Design Chart

From this information, along with the continuity of the mineralized zones, SRK has defined a basic stope dimension of 15m wide by 25m high. Stope length will be dependent on mineralized width. At these dimensions stope should remain stable without support. Localised support, in the form of 6m long cable bolts, will be required to support specific blocks isolated by structures.

The nominal development span will be 5m. Several categories of development will need to be developed to extract the Wassa underground stopes:

- Temporary drilling and extraction drives and cross-cuts;
- Semi-permanent footwall haulages and ventilation development; and

- Permanent access development from surface.

Barton’s Q support classification system has been used to estimate support requirement for each category of development. The results of this assessment are presented in Table 15-6 for a Q range of 5 to 32 (assuming Jw and SRF ratings are both unity) for the mineralization and footwall.

Table 15-6: Development Support Assessment

Development Type	Span (m)	Excavation Support Ratio (ESR)	Equivalent Dimension (De = Span/ESR)	Support Requirements
Temporary	5	3	1.67	None. Spot Bolting only
Semi-Permanent	5	1.6	3.1	None. Spot bolting only
Permanent	5	1.2	4.1	2.4m long roof bolts at 2.0m spacing with mesh in low Q zones only.
Permanent (Intersections)	7	1.2	5.8	2.4m long roof bolts at 2.0m spacing with mesh in low Q zones only.

Due to the generally high rock competency there should be no general requirement for systematic rock support apart from in permanent development that intersects rock with a Q value of 10 or lower. Because of the jointed nature of the rock mass there will be a requirement for spot bolting where there is a block fall potential. Spot bolting will require designing on a site specific basis but will most likely comprise 2.4 m long rock bolts on a 1.5 to 2.5 m spacing.

15.3.3 Modifying Factors and Cut-off Grade

The mining modifying factors were selected on the following basis:

- Mining recovery of 90% was selected given the use of an cemented fill to provide support and eliminate the requirement for support pillars in mineralized material;
- An external dilution of 5% with no gold grade was applied to take into account blasting overbreak. It is assumed that given the 25m sub-level height, stope overbreak can be well-controlled. SRK notes that a key source of dilution will be from backfill when mining the secondary stopes adjacent to filled areas but this is accounted for in the 5% extermal dilution factor;
- A dilution skin of 1.0 m from the HW and 0.5 m from the FW was also included in the stope optimisation to take into account dilution from these contacts; and
- Internal dilution, material below cut-off which is included in the production stope design, is accounted for in the stope optimisation/design process.

15.3.4 Cut-off Grade for Stope Design

A summary of the Wassa cut-off Grade estimates are provided in Table 15-7. SRK has estimated the mining cost, whereas the processing and General and Administrative (“G&A”) costs have been provided by the client based on historical performance. SRK notes that for initial planning purposes a mining cost of US$55/t was assumed. This was later revised downwards to US$40/t after more technical work was completed as part of the PEA. The significant change in the unit mining costs was due to the fact that capital development was excluded from the revised unit cost.

Table 15-7: Wassa Underground Cut-off Grade

Parameter	Unit	COG
Mining Cost	US$/t RoM	55.00
Processing Cost	US$/t RoM	17.50
G & A Cost	US$/t RoM	5.54
Contingency (on operating cost)	US$/t RoM	11.71
Royalty		5%
Gold Price	US$/oz	1,300
Plant Recovery		93%
Cut-off Grade	g/t Au	2.9

15.3.5 Stope Optimization

Mineable limits

Stope dimensions were determined based on geotechnical recommendations and the mineralized geometry. Stope optimisations were run in Deswik's Stope Optimser using a 3g/t Au cut-off grade. Different combinations of widths, sub-level heights were tested in order to determine which dimensions would best balance maximum extraction of material above cut-off grade and average head grade. The 15 m wide by 25 m high stopes were selected as the basis for the mine plan as it was felt that higher productivities and lower mining costs would be achieved on a larger stope, without compromising on selectivity. Table 15-8 below presents the results of the stope optimisation.

Table 15-8: Summary of Mineable Stope Optimiser results

Stope Dimension	Tonnes (Mt)	Metal (Moz)	Grade (g/t Au)
15m wide by 25m high	8.6	1.3	4.7

Once the preferred mineable shape scenario was identified, individual mining shapes were eliminated if they were located within the designed pits or if they were distant from the main mining areas.

Figure 15-9 shows the spatial distribution of the mineable stopes prior to the stope selection process.

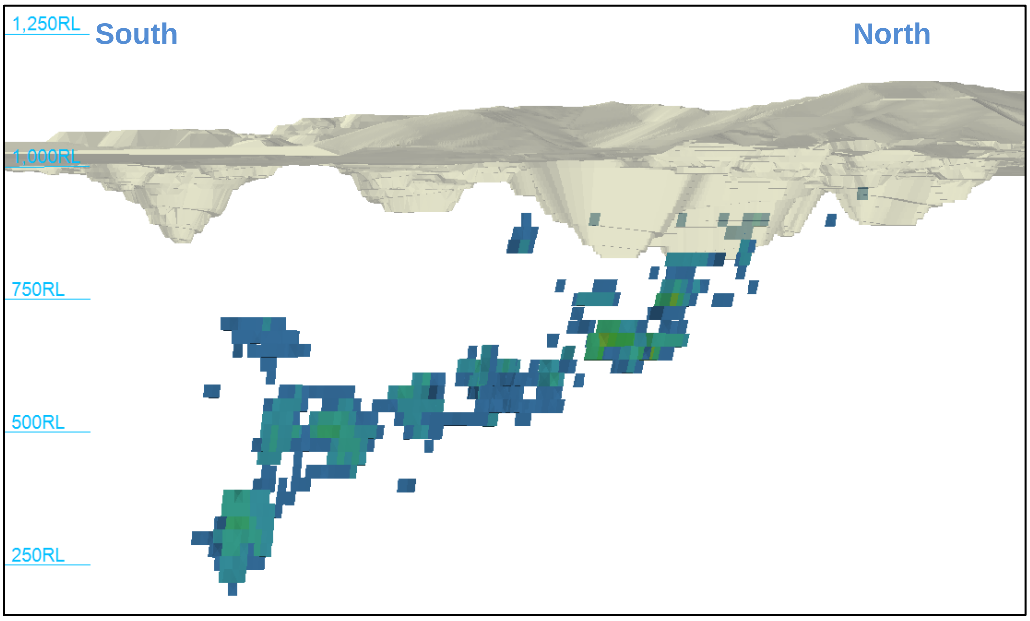


Figure 15-9: Isometric view looking west of 15 m wide by 25 m high stopes from the Deswik Stope Optimiser

Mineralized width variability

The in-situ tonnages reported from the stope shape interrogation include internal dilution, defined as mineralization below cut-off grade that must be mined in order to extract higher value material. In order to achieve the stope sizes that permit the economies of scale expected from a high-production, bulk mining method, a certain amount of internal waste must be mined. In the case of the Wassa Main deposit, this mineralization below cut-off is grade-bearing and its metal contribution is included in the stope optimisation metal inventory. This is estimated to be 23% of the in-situ tonnage.

Stope design process

In order to generate a production schedule, SRK used the stope wireframes generated from the mine optimisation process, with the economic selection criteria applied. Though no detailed design was produced for the stopes, a conceptual-level layout based on a 25 m sub-level spacing was completed for one of the main production zones in order to determine an appropriate factor for stope development quantities to incorporate into the production schedule. The ratio of stope tonnes to development in the main stopes is 282 t/m. The stopes grouped into zones based on the decline that provides access to them; this facilitated ventilation and footwall access designs. The stopes selected for inclusion in the production schedule are shown in Figure 15-10 and Figure 15-11.

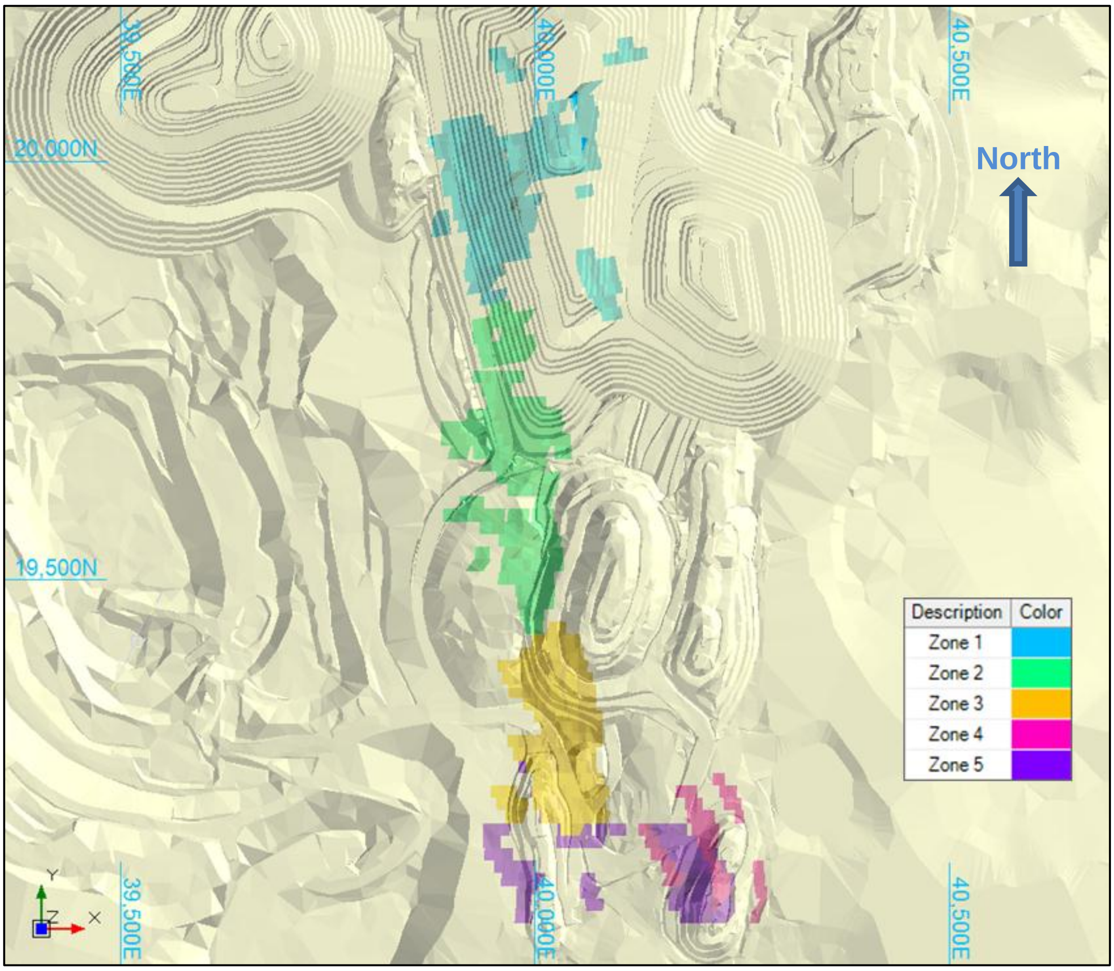


Figure 15-10: Plan view showing stopes included in Wassa Underground mine plan

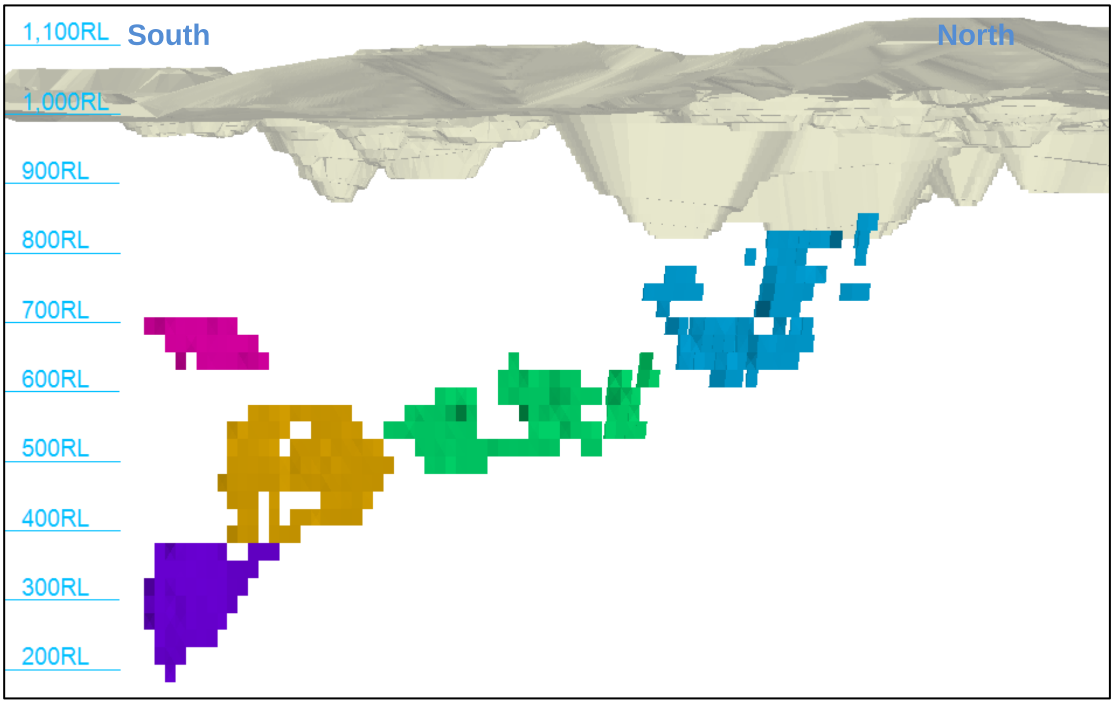


Figure 15-11: View looking west of stopes included in Wassa Underground mine plan

Results

The results of the stope optimisation and selection process, providing the basis for the production schedule, are presented in Table 15-9.

Table 15-9: Stope inventory for production scheduling

Classification	In-Situ Rock (Tonnes)	In Situ Metal (oz)	Grade (g/t Au)
Indicated (In-Situ)	4,150,000	704,000	5.28
Inferred (In-Situ)	3,980,000	593,000	4.63
Planned Waste	530,000	-	-
Stope Total	**8,660,000**	**1,297,000**	**4.66**

15.3.6 Underground Design

Ramp Access

The focus of the access infrastructure option study was on the placement of a decline to access the mineralization. A shaft option was discarded owing to the relatively shallow nature of the anticipated mining, expense of implementation and mobilisation time. A decline access would leverage elevation gains from the existing pits and be established with a relatively simple portal design in a pit highwall. In this case a boxcut would not be necessary. The use of a decline would also facilitate in-fill resource drilling and permit test mining and early production to begin while the primary development is still being undertaken.

Three potential locations were identified as a result of the site visit undertaken in February 2014. It was identified that the ramp could begin close to the bottom of the MSN pit which will be completed in December 2014. This provides a gain of 106 m in vertical depth and enabled a better alignment of the decline with the strike of the mineralization.

Figure 15-12 and Figure below illustrates the final decline design for the PEA. The ramp commences in the pit then heads south east towards the proposed mining areas. This configuration reduces the metres required to access the stopes and the location means that the surface haul to the process plant is minimised.

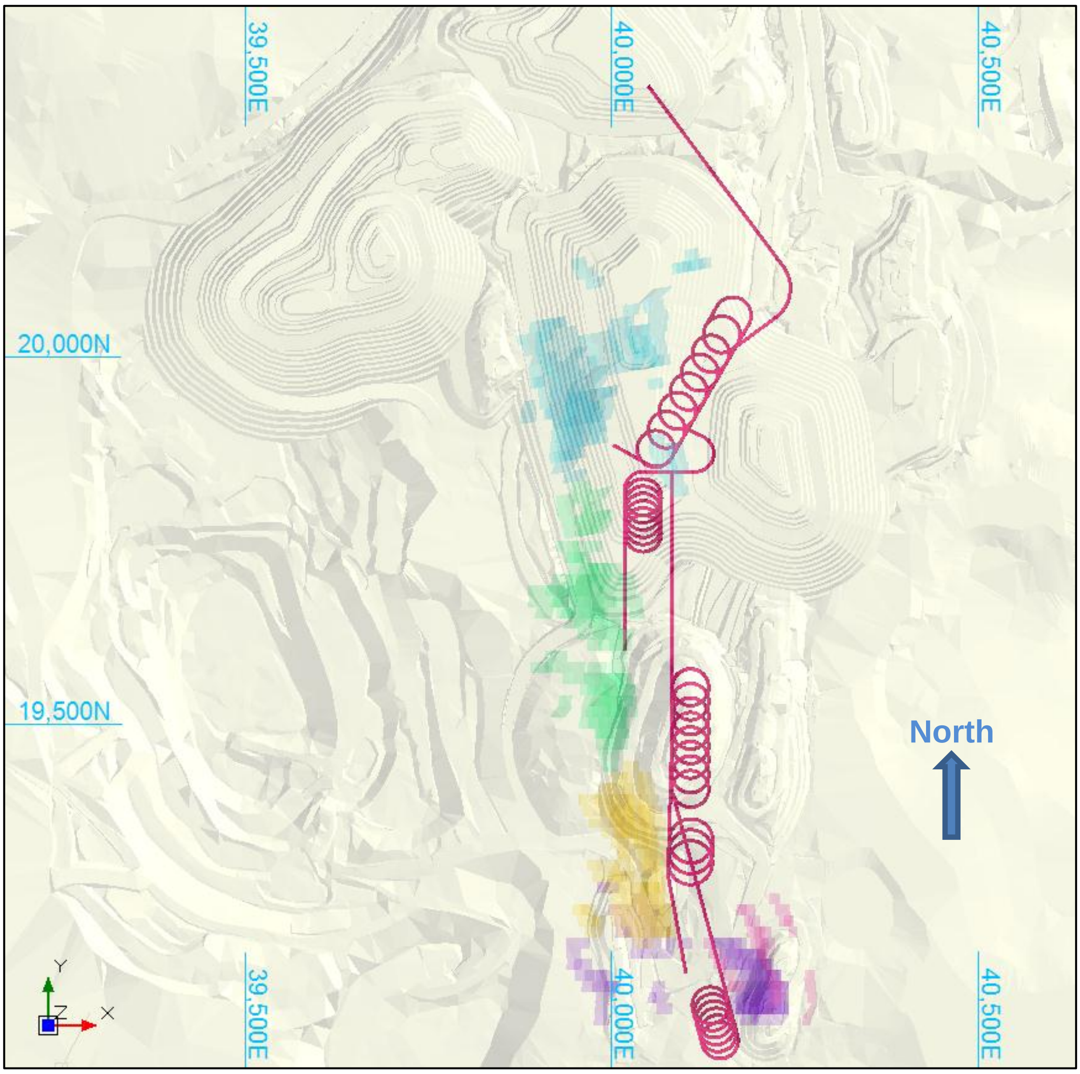


Figure 15-12: Wassa Decline Access Design

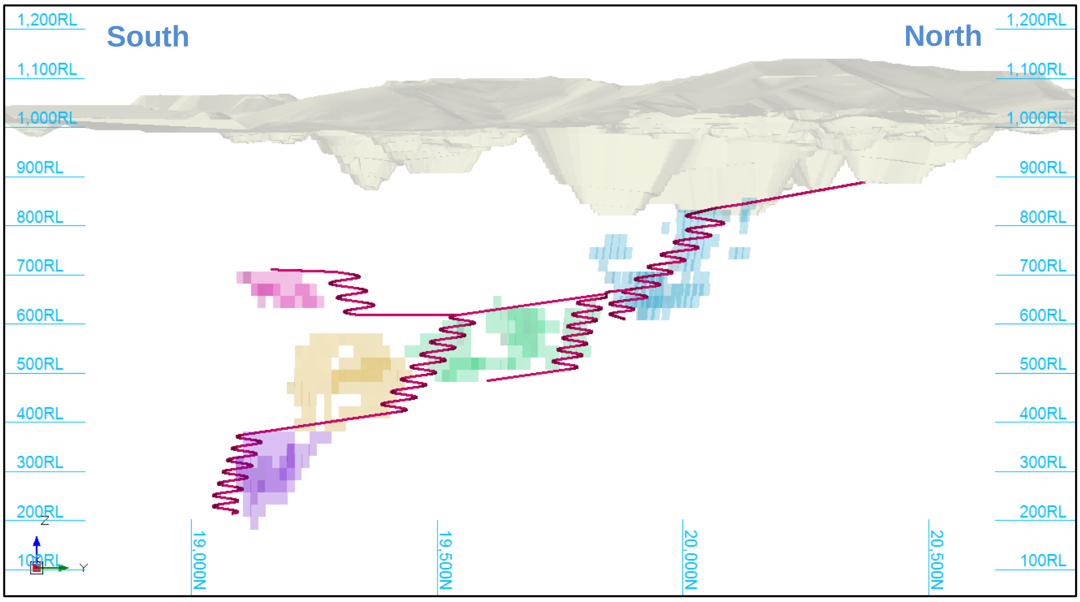


Figure 15-13: Long Section looking West of Wassa Decline Access Design

Stope Access

The concepts guiding the mine design are:

- Development of the main access ramp was aligned with the strike and dip of the target mineralization;
- Main stope accesses were aligned along the strike of the mineralization;
- Ventilation design that would permit early production and minimise additional development;
- Development of main ramps at a 15% (1:7) grade in order to balance equipment wear against vertical depth gains;
- Selecting ventilation raise locations with minimal interference with surface operations; and
- Maintaining a safe stand-off distance from the surface pits.

Ore accesses

Main production area sub-levels were designed on 25 m intervals, accessed from the main decline (Figure 15-14 and Figure 15-15). The footwall drifts are aligned with the strike of the proposed stopes and terminate at the strike limit of the mineralized body. The footwall drifts connect the stope accesses together and allow ventilation to be reticulated to the stopes. Trucks will be loaded in these footwall drifts close to the stopes.

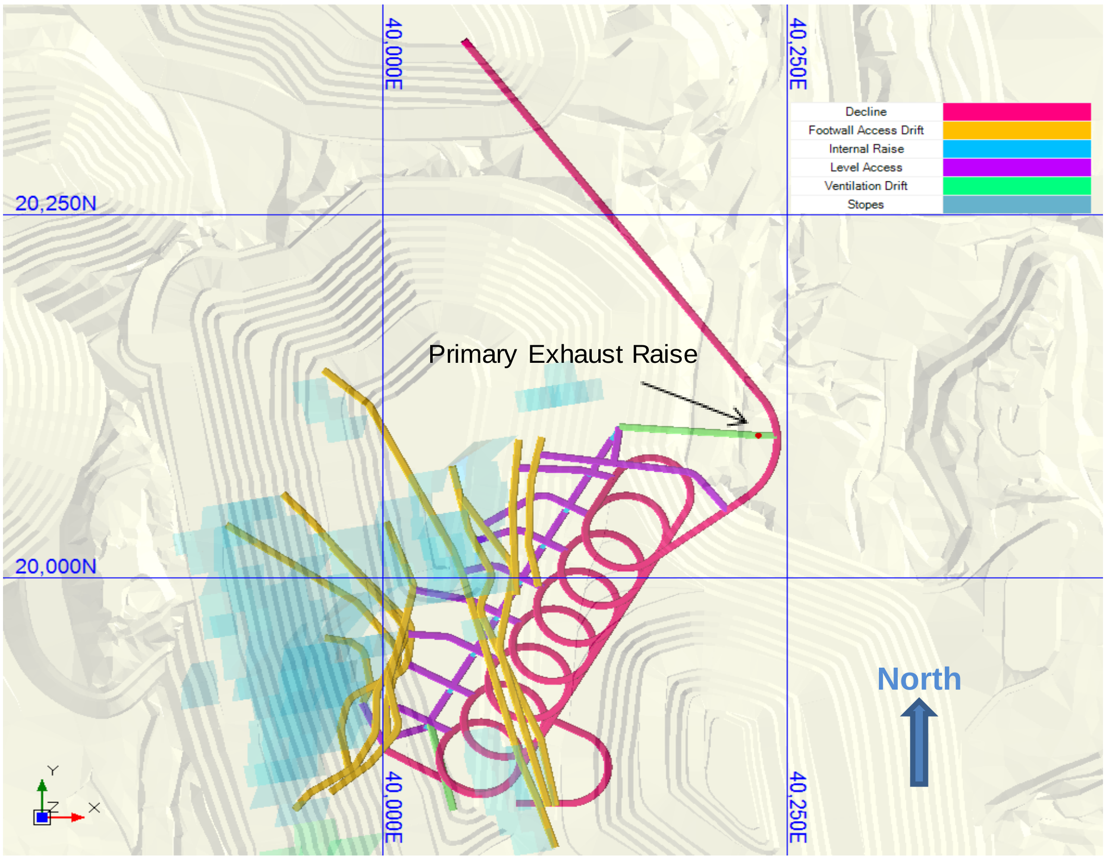


Figure 15-14: Plan View of Zone 1 Access Design

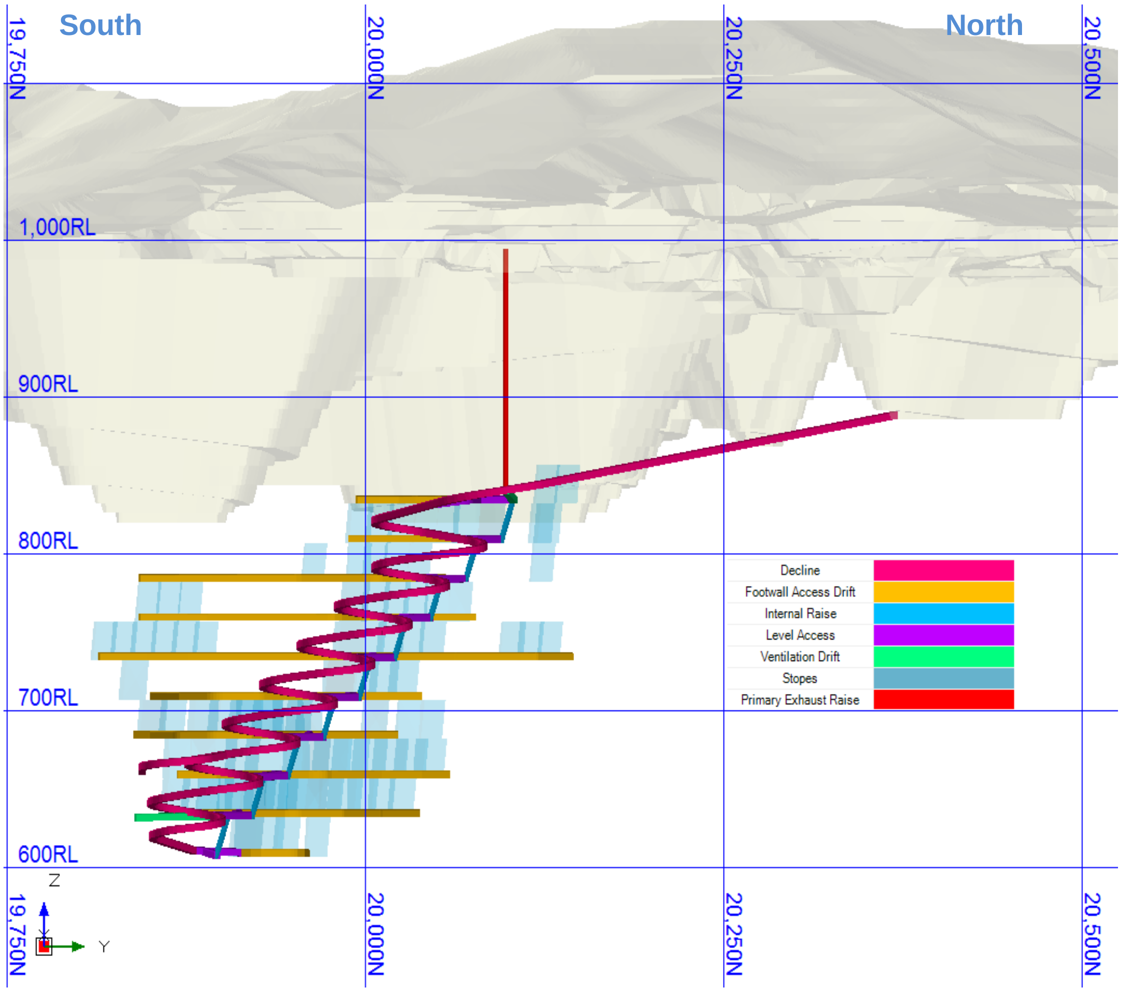


Figure 15-15: View looking East of Stopes and Development for Zone 1

A summary of development metres for the whole underground design is provided in Table 15-10 below.

Table 15-10: Summary of Development Requirements

Description	Planned development (m)
Horizontal Development	**50,850**
Decline Development	7,550
Level Development	12,600
Stope Development (factored)	30,700
Vertical Development (m)	**2,390**
3.1 m Raisebore	660
4 m Raisebore	1,100
3 x 3 m Internal Raise	630

Development Strategy

SRK proposes the following sequence of events for mine development:

1. Once decline construction reaches the 832RL level, a ventilation connection drift will be driven in order to raisebore the proposed exhaust raise from surface.

2. Decline development will continue while the surface raise is being constructed. The 832 and 807 level accesses and footwall access drifts will provide additional working headings.
3. An internal raise will be constructed between the 832 and 807 levels.
4. Once the internal and surface raises are completed, mining of the 807-level stopes can begin, while development of the decline and other level accesses continues.

Portal Access

Some construction will be required to establish and maintain an adequate portal access to the underground mine. This will include:

- Application of bolts, mesh and shotcrete to support the portal excavation;
- Adjustments to surface roads or traffic flows to accommodate underground haul trucks; and
- Construction of a small building to house the tag board and communication equipment.

15.3.7 Backfill

Introduction

An engineered backfill will be required to maintain support for mining the secondary stopes and increasing the overall level of extraction. Various types of backfill were considered for the PEA.

Paste Backfill

Paste fill is a high solid density mixture with several advantages as backfill:

- Reduced or eliminated need for fill drainage, compared to a slurry or hydraulic fill;
- Reduced binder consumption for equivalent or better slurry fill strengths; and
- High tailings usage, reducing surface tailings storage requirements. In most cases all size fractions of tails can be used.
- The disadvantages of using a paste backfill are:
- Paste fill plant facilities require a high level of technical precision to ensure safe, high quality backfill that meets design specifications;
- High plant capital costs and long construction lead times can be disadvantageous for shorter mine lives;
- Operating costs are relatively high mainly due to the need to substantially dewater the tailings for use as a paste; and
- Backfill delivery requires a well-maintained and operated pipeline infrastructure.

Cemented Rock Fill

Cemented, or consolidated, rock fill (“CRF”) is comprised of sized or un-sized aggregate mixed with binding agents and delivered to an open stope. It has the following advantages:

- Can be engineered to high strengths, enabling undercut mining and future exposure of fill walls from secondary stope mining;
- Can make use of waste material from underground or surface mining activities;
- Can be delivered by a loader or a truck; and
- Infrastructure requirements can be as simple as spraying a cement slurry into the back of a truck that contains waste or emptying cement bags into a cement truck for delivery to a stope.
- The disadvantages of using cemented rock fill are:

- Quality depends on proper fill placement and the method used to mix the fill;
- Segregation control may be difficult; and
- Additional material purchase costs may be incurred if nearby waste material quality or quantity is unsuitable for usage.

PEA Backfill Selection

For the Wassa Underground PEA, cemented rock fill was the preferred option for the following reasons:

- Development waste can be utilised underground saving the cost of transport and storage on surface ;
- Empty trucks returning underground from tipping at the surface RoM stockpile can be loaded with waste and measured dose of cement slurry. The trucks would tip directly into the stopes eliminating the need for back fill raises. In this way, the return journey of each truck can be utilised to benefit project economics. In order to minimise haulage cycle times in this scenario, a cement slurry preparation facility, such as a batch plant, will be located on surface as close to the RoM pad or portal as possible; and
- Potential use of nearby surface mining waste materials, reducing surface waste dump sizes and rehabilitation costs.

The use of paste backfill will be considered in detail during the next phase of the study.

15.3.8 Materials Handling

Transport of development waste will be by diesel powered trucks loaded using LHDs. This material would be tipped into stopes. RoM material would be trucked directly to surface by the same fleet.

Due to a shortfall of waste rock from development some backfill would need to be waste rock from surface dumps. This would be loaded directly into trucks that have deposited their loads of RoM material at the surface stockpile and are returning underground. Making use of the return journey in this manner reduces the number of trucks required.

Explosives will be transported underground by the mobile charging units which will be filled at the existing magazines that service the current open pit operation.

Other materials required for underground activities will be carried underground by the tool carriers and light vehicles planned as part of the equipment fleet.

15.3.9 Equipment

SRK has assumed that mining will be under taken by trackless methods using diesel powered units. The mine equipment fleet is presented in Table 15-11 and consists of:

- Twin boom drill jumbos to undertake development activities;
- Diesel loaders of 17 t capacity (LHDs) for loading RoM and waste into trucks;
- 40 t capacity diesel truck fleet for the transport of RoM material to surface and the movement of waste around the mine;
- An electro hydraulic longhole drill for stope drilling. This machine would drill 4” diameter holes for production;
- A multipurpose hydraulic drill (Cubex type) that can be used for production drilling, reaming slots in stopes and ventilation raises using a special attachment;
- A ‘Integrated Tool carrier’ type of service vehicle for underground installation work;
- A charge-up truck for delivering ANFO for production and development blasting;
- A grader to maintain the roads underground;
- Modified tractors for personnel transport; and

- Shotcrete machine. A fully mobile shotcrete delivery unit for rapid installation of ground support when required.

Table 15-11: Initial Mining Equipment Fleet

Equipment	*No.*	kW	Availability	Utilisation
LHDs (17t)	4	256	85%	75%
Jumbos (2 boom)	3	110	85%	15%
Trucks (40t)	3	375	85%	75%
Longhole rig / raisebore	1	55	85%	5%
CAT 930Ktool carrier	1	105	85%	75%
Normet Charmec explosive truck	2	96	85%	75%
Cat 12M grader	1	118	85%	60%
Kubota MX4700 tractor	8	35	85%	60%
Normet Variomec service vehicle	2	120	85%	75%

SRK notes that the Variomatic is a multipurpose vehicle that can utilise a number of cassettes to deliver fuel, transport people and apply shotcrete.

15.3.10 Mine Ventilation

SRK has carried out an analysis of ventilation requirements at Wassa using VENTSIM VISUAL software. The aim of this analysis is to determine fan, duct and airway sizes for the LoM plan. The principal objectives of the ventilation design are:

- Remove the diesel exhaust fumes from mechanised mobile equipment;
- Remove blasting fumes from the workings and provide for a reasonable re-entry period; and
- To maintain working conditions in the mine in accordance with mine regulations.

Ghanaian Mining regulations stipulate the following:

- Maximum 6 m/sec in travelling roadways;
- For diesel engine equipment, not less than 0.06 m^3/kw/sec;
- Minimum velocity of 0.2 m/sec in headings and 0.1m/s in large openings;
- 32.5 degrees Celsius wet bulb is the maximum men are allowed to work in; and
- CO must be continuously monitored in return airways and information transmitted to surface.

Ventilation Quantities

Calculated airflow requirements take into account the fleet scheduled as per

Table 15-12 and Table 1-13. There are a number of methods of determining the ventilation requirements and considering that the removal of diesel fumes is generally the primary concern in trackless mining operations the calculation is based on a diesel dilution rate of 0.06 m^3/s/kW of diesel engine flywheel power using the entire fleet with their modelled availability and utilisation.

The mobile fleet was modelled at an early stage for Phase 1 modelling and later deeper mine-life stage for Phases 2 and 3.

Table 15-12: Equipment and Ventilation Requirements Phase 1

Equipment	*No.*	kW	Availability	Utilisation	m^3/sec per unit	Total m^3/sec
LHDs (17t)	4	256	85%	75%	15.4	39.2
Jumbos (2 boom)	3	110	85%	15%	6.6	2.5
Trucks (40t)	3	375	85%	75%	22.5	57.4
Longhole rig / raisebore	1	55	85%	5%	3.3	0.1
CAT 930K	1	105	85%	75%	6.3	4.0
Normet Charmec	2	96	85%	75%	5.8	7.3
Cat 12M	1	118	85%	60%	7.1	3.6
Kubota MX4700	8	35	85%	60%	2.1	8.6
Normet Variomec	2	120	85%	75%	7.2	9.2
Leakage					15.0%	19.8
Contingency					10.0%	13.2
				Total ventilation quantity required		**164.9**

SRK notes that in order to support the proposed fleet a primary circuit of around 165 m^3/s is required in Phase 1 and 263m^3/sec for Phases 2 and 3.

Table 15-13: Equipment and Ventilation Requirements Phases 2 and 3

Equipment	*No.*	kW	Availability	Utilisation	m^3/sec per unit	Total m^3/sec
LHDs (17t)	4	256	85%	75%	15.4	39.2
Jumbos (2 boom)	3	110	85%	15%	6.6	2.5
Trucks (40t)	9	375	85%	75%	22.5	129.1
Longhole rig / raisebore	1	55	85%	5%	3.3	0.3
CAT 930K	2	105	85%	75%	6.3	8.0
Normet Charmec	2	96	85%	75%	5.8	7.3
Cat 12M	1	118	85%	60%	7.1	3.6
Kubota MX4700	10	35	85%	60%	2.1	10.7
Normet Variomec	2	120	85%	75%	7.2	9.2
Leakage					15.0%	31.5
Contingency					10.0%	21.0
				Total ventilation quantity required		**262.4**

System Design

The mine shall be developed from a portal and decline commencing in an open pit base. Forced ventilation will be initially used until the raisebore position of Ventilation Raise (“VR”) 1 is reached (at 340 m after portal). After the establishment of VR1 it becomes the mines exhaust with the decline as sole intake airway. The decline and return are connected on each 25 m vertical level to create a primary circuit loop.

As the mine extends to the south two more primary raises are designed through to surface, VR2 and VR3. As each of these become operational they are used as the primary return with previous raises becoming intake airways.

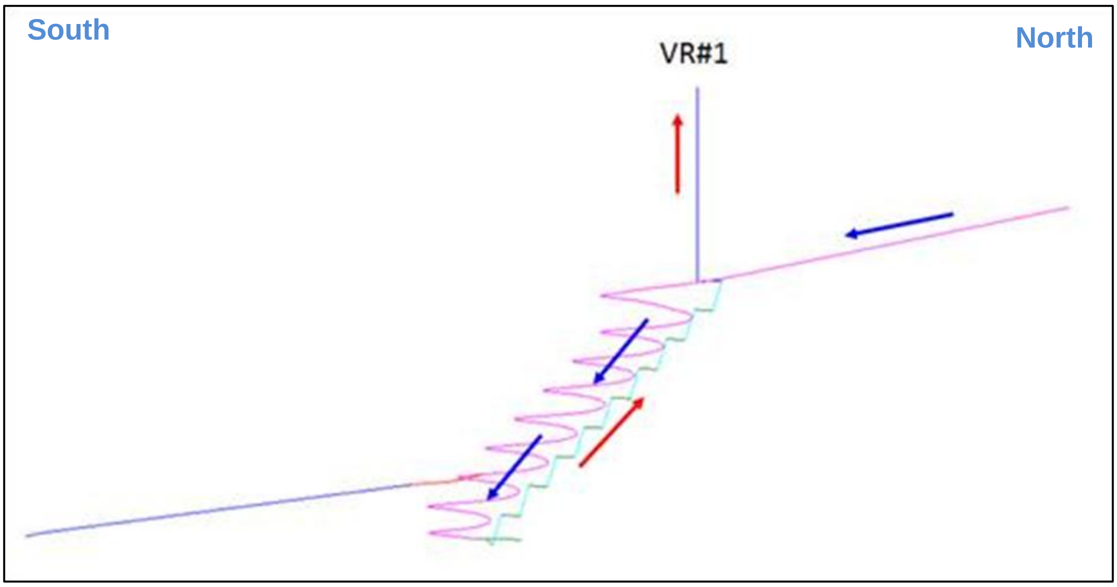


Figure 15-16: Phase 1 primary ventilation circuit looking west (not to scale)

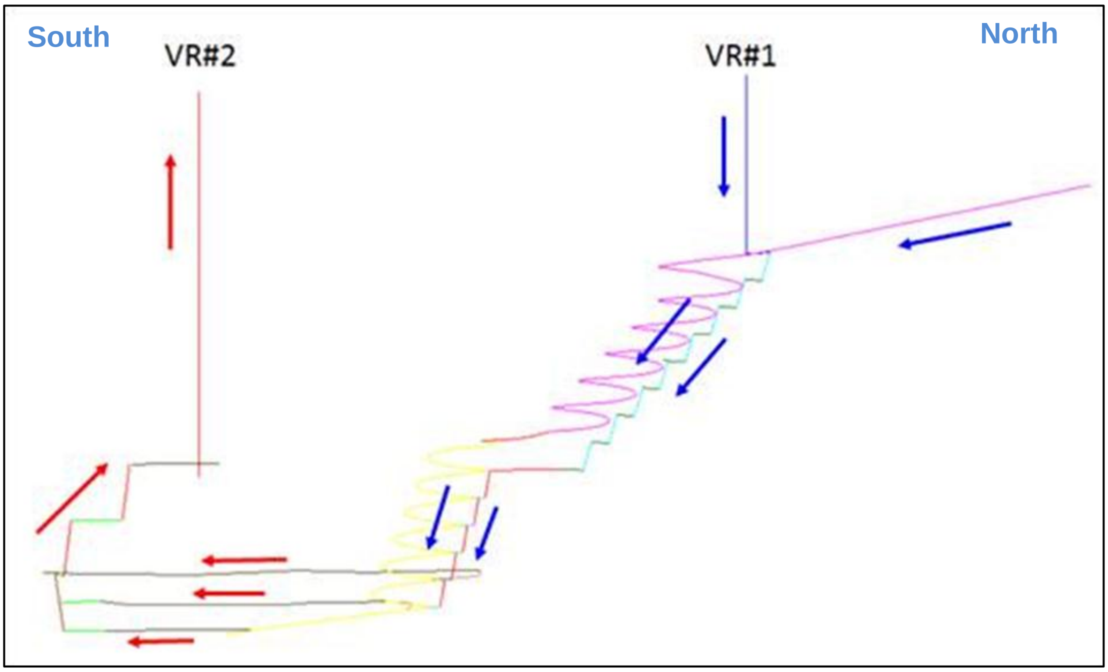


Figure 15-17: Phase 2 primary ventilation circuit looking west (not to scale)

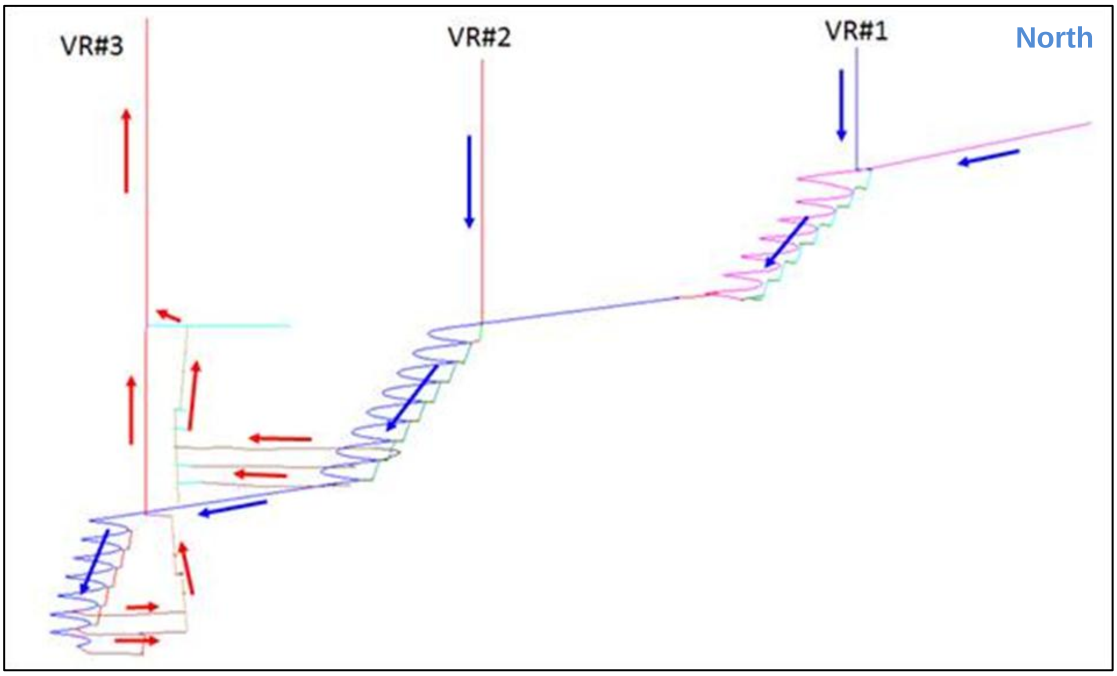


Figure 15-18: Phase 3 primary ventilation circuit looking west (not to scale)

Airflow Simulation Inputs and Results

Inputs

The following key system inputs were used in the simulation:

- All major development headings for truck access were set at 5.8mH x 5.5mW cross-section size;
- The primary raises to surface are 4m diameter raisebores;
- The drop raises between each 25m sublevel or decline loop are 4m x 4m;
- Primary raises are assumed raisebore (friction factor, 0.0047 kg/m^3);
- All other raises and headings assumed to be average blasted (0.0115 kg/m^3); and
- Shock losses are automatically modelled at raise entrances and exits.

Phase 1

Fixing the exhaust quantity at 165 m^3/sec gives a collar total pressure of 1,574 Pa, giving a fan shaft power of 346 kW, allowing for electrical efficiency and for power factor a 400 kW motor is likely required for installation.

The decline velocity is 5.5 m/sec and the final upcast velocity is 13.1 m/sec. 13.1 m/sec is just above the theoretical water suspension limit. A variable speed fan is recommended to handle the wide range of duties over the mines life and in Phase 1 the speed could be slightly increased if water suspension in the upcast becomes evident.

During this phase the return raise accesses are located in the access connection between the decline and the footwall drives, so there does exist the ability to exhaust directly off each level. However it is likely that only the bottom most connection with the exhaust is open. In this case used air off each upper sub-level rejoins the decline flow and is re-used. Dust, backfilling fumes and blasting cases in this case will need to be monitored.

Phase 2

The exhaust quantity from VR2 was fixed at 263 m^3/sec with the decline and VR1 and associated drop raises acting as parallel intakes. The intake system will have low pressure loss due to the parallel arrangement. The major pressure loss will be on the exhaust system. A series of simulations with changed return airway sizes were modelled.

Table 15-13: Phase 2 ventilation simulation scenarios

Scenario	Fan Total Pressure	Electrical input	Annual power cost	System characteristics
1	1,695 Pa	667 kW	US$1.05M	All raisebores 4m diameter. All drop raises 4m x 4m
2	1,283 Pa	477 kW	US$0.75M	VR2 at 5m diameter
3	1,038 Pa	393 kW	US$0.62M	VR2 at 5m diameter and exhaust drop raises at 5m x 5m

Scenario 3 is chosen as the base case.

Phase 3

The exhaust quantity from VR3 was fixed at 263 m^3/sec. The raise system for Phase 2 scenario 3 was utilised. The drop raises after VR2 were reduced to 4m x 4m and raisebores (upper and lower) for VR3 position were modelled at both 4m and 5m diameters. Some regulation was introduced to:

i) ensure a minimum 100 m^3/sec in the primary decline generally; and

ii) have a maximum 185 m^3/sec in the decline across to the southern most spiral (as there is no parallel intake in this portion

Table 15-14: Phase 3 ventilation simulation scenarios

Scenario	Fan Total Pressure	Electrical input	Annual power cost	System characteristics
1	1,993 Pa	772 kW	US$1.22M	both VR3 raises 4m diameter
2	1,494 Pa	551 kW	US$0.87M	Upper raise 5m, lower 4m
3	1,307 Pa	486 kW	US$0.77M	both VR3 raises 5m diameter

Scenario 3 is chosen as the base case. SRK notes that due to the level of detail in the simulation, no bulkhead leakage modelling has been performed. The effect of leakage is to reduce the overall system resisance at the expense of total flow to the lower levels.

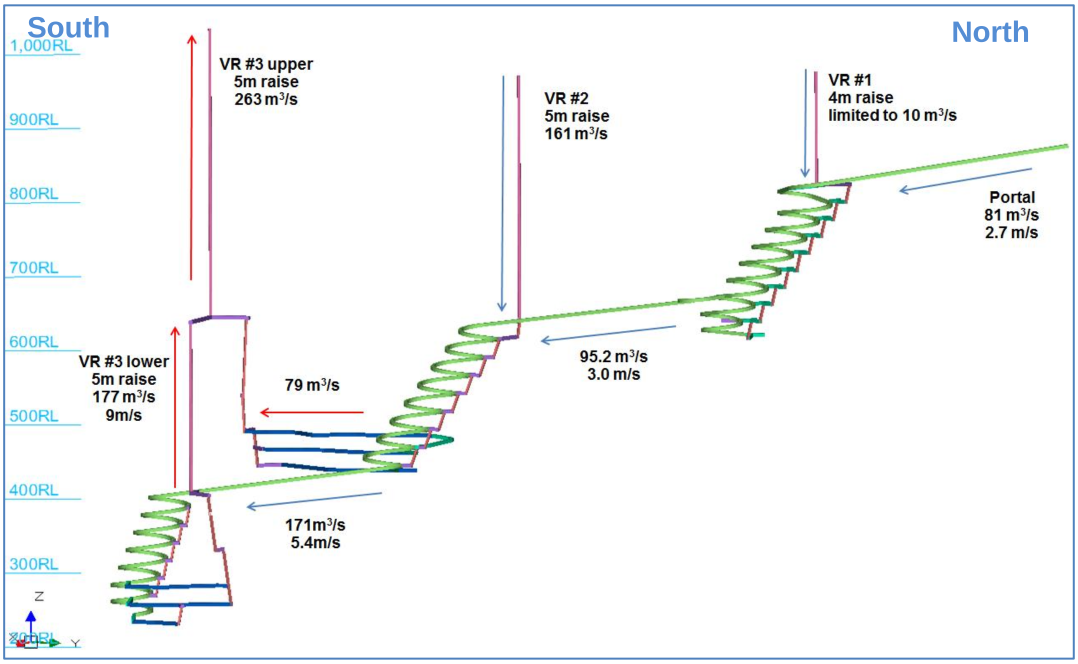


Figure 15-19: Phase 3 – Long Section from VENTSIM looking west (not to scale)

Primary Fan Recommendation

SRK envisages that just one fan purchase will be made and this fan will be progressively moved south. An axial flow fan is recommended for lower capital purchase and installation costs. SRK would recommend putting those capital savings into ensuring the fan is purchased with a variable speed drive. There are large variations in the fan duty over the life of the operation and hence there is a significant saving in electrical power input that could be realised.

SRK recommends the purchase of a 550 to 600 kW fan together with 'lobster back' ducting arrangement and diffuser to minimise exhaust shock losses.

Secondary Ventilation

Decline and Footwall Haulages

For driving the blind decline headings SRK has modelled the use of twin stage contra-rotating high pressure 150 kW (2 x 75 kW-1254 mm diameter) fans for development with 1400 mm flexible PVC ducting. At 500m these will deliver 30 to 35 m^3/sec depending on duct condition from an initial fan output of 45 m^3/sec.

In order to provide fresh air down the decline from the portal during initial development (to allow more than one LHD and one truck), twin fans with independent 1400 mm double ducting systems have been applied. Each duct provides 20 to 28 m^3/sec of fresh air at 1000 m duct length depending on the condition of the ducting.

Stope Access Drifts

For secondary ventilation purposes the mineralization has been divided into five geographical regions as per Figure 15-19.

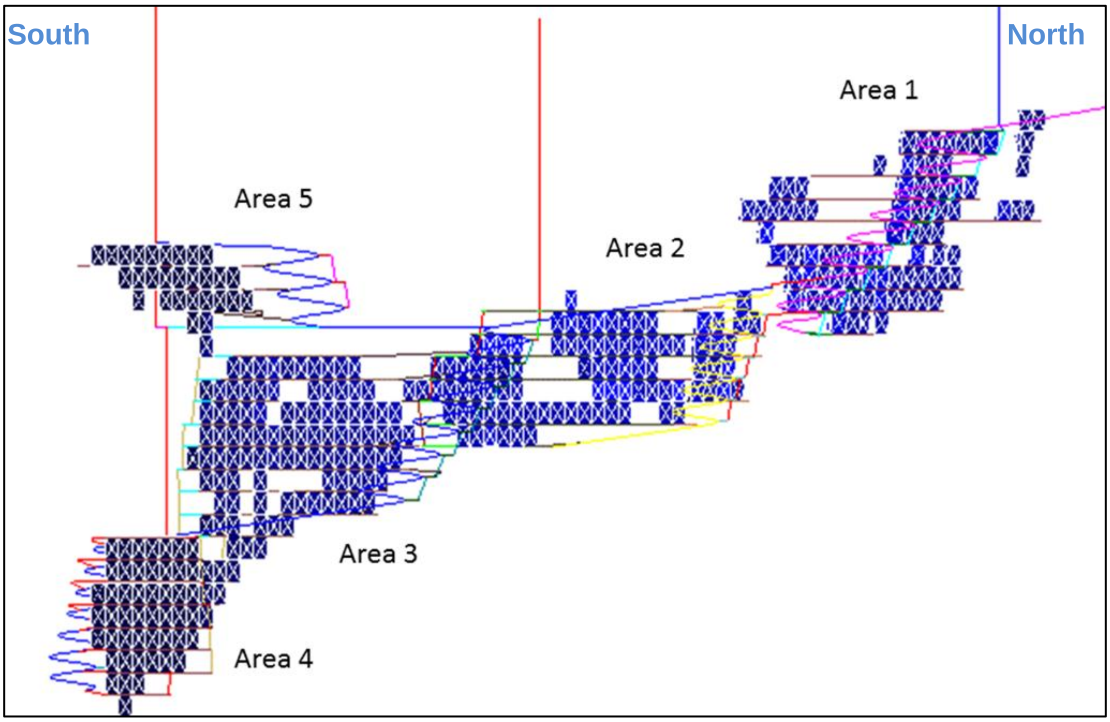


Figure 15-20: Secondary ventilation districts, isometric long view looking west not to scale

Areas 2 to 4 generally use footwall development with a primary airflow requiring short secondary forcing arrangements. Areas 1 and 5 are dead ends for ventilation purposes.

For Areas 2 to 4, once footwall drive primary flow is established, secondary ventilation of maximum 100 m will be required to ventilate the stope drives and stopes. The largest engine in this area is the LHD during mucking. This requires 15.4 m^3/sec.

The stope drifts will be ventilated by 30 kW (1070 mm diameter) fans situated in the footwall drive prior to the exhaust raises that will deliver around 15 to 18 m^3/sec over up to 200 m using 1000 mm diameter flexible ducting.

It will be necessary to ventilate more than one stope drive on each level. Air returning from the stope drives to the footwall drives will need to be re-used into other headings. For this reason it will likely be required to have a number of 30 kW fans placed down the footwall drive re-using the same air.

A schematic of this is shown in Figure 15-20. Note that Clemcorp fans were used as the basis for this investigation. In the diagram fresh air is the blue arrows and exhaust air in shown as red arrows. The yellow represents a flexible ventilation duct.

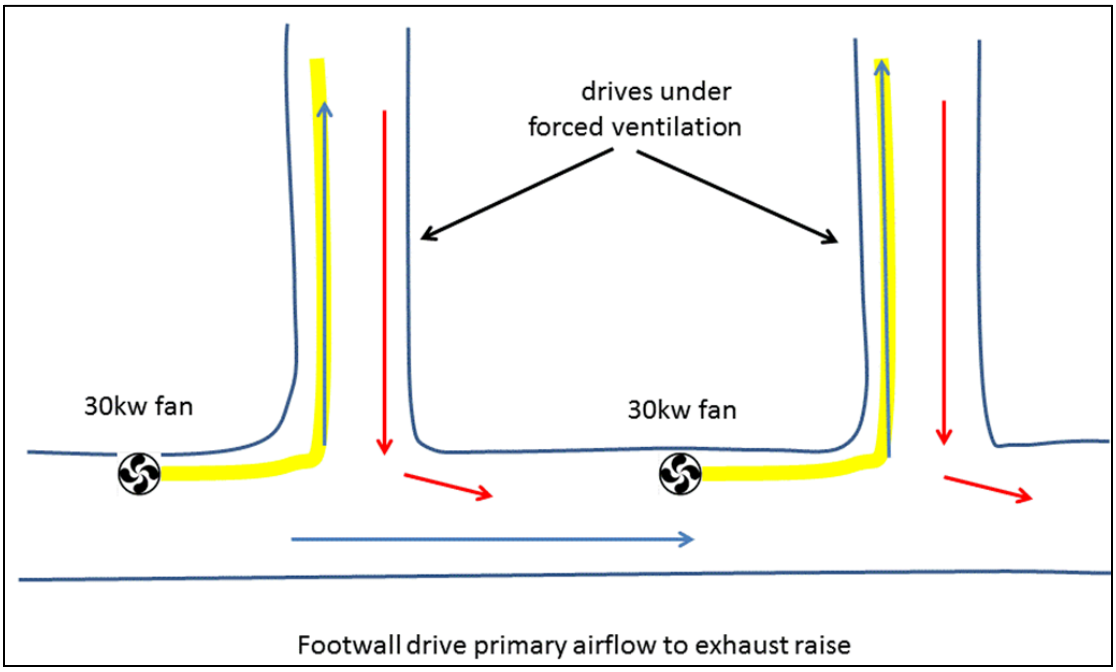


Figure 15-21: Plan View of a Typical Level, Showing Re-Use of Air for Areas 2-4

Area 1 will require force ventilation from the decline (Figure 15-21). The number of stope drives and stopes in this area on each sub-level will up in the order of 12. Considering the primary secondary nature of the mining method, then six may be in the production cycle. Two fans can be mounted in the decline primary airflow and could deliver 30 to 35 m^3/sec each. If each ventilation line was dedicated to the north and south, these lines could then be further branched off into the three active headings on each end.

LHD operations requires almost 16 m^3/sec, hence it is likely only one other activity could be performed off that vent line to ensure adequate conditions for mucking.

Neither the north nor south ends under this scenario have enough air for truck loading operations. So loading activities will ideally need to occur around the x-cut or decline area.

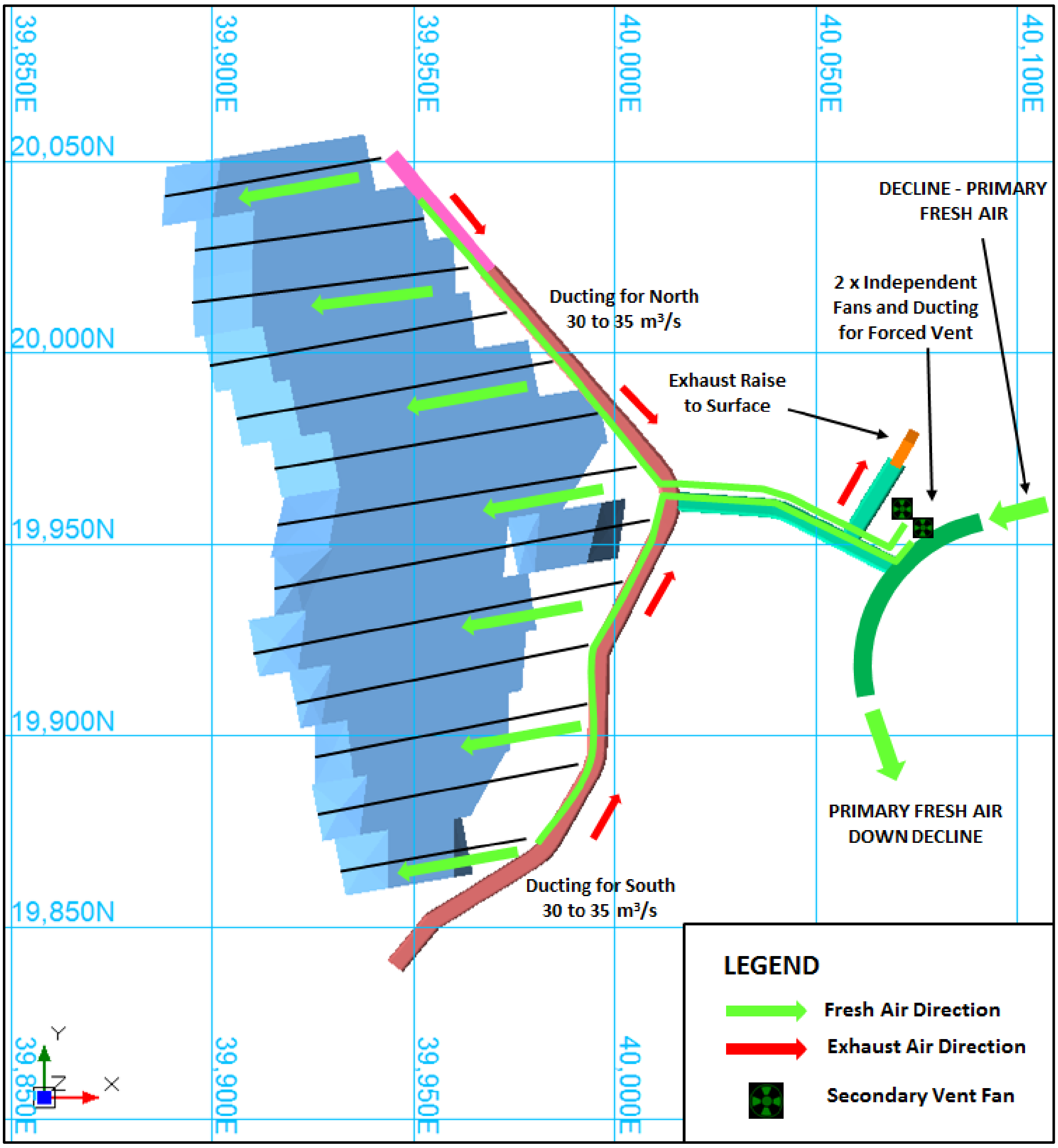


Figure 15-22: Plan view 861mRL level in Area 1 – secondary ventilation

15.3.11 Electrical Distribution

The underground electrical system would be fed from the existing surface substations at Wassa. The expected load for the UG Project would be around 4MW. A total capital cost of US$1.5M has been allocated for the HV backbone for the Wassa underground.

15.3.12 Communication

SRK has assumed that the capital costs would allow for the installation of an underground WIFI backbone which would allow a number of activities to be coordinated including vehicle and worker tracking and voice and data communication.

15.3.13 Manpower

SRK has assumed for that the Wassa underground operation that both initial access development and production would be carried out by a contractor. The total estimated workforce would be between 200 to 350 persons depending on the stage of the project.

SRK has assumed that a rotating shift system of three shifts per day of eight hour duration would be utilised. An hour per shift would be allowed for blasting to be carried out. After travel and shift change time is allowed, 6 hours per shift would be considered as the effective working time.

The mine will be developed and mined using a contractor because GSR does not currently have the necessary underground mining expertise. The contractor will supply mobile equipment, supervision, training and established systems, processes and procedures. The contractor will be expected to hire and train a significant portion of their workforce from local communities.

Ghana and West Africa has a number of established, local and international contracting companies with the experience necessary to carry out the scope of work.

15.3.14 Mine Services

Water Supply and Pumping

SRK has allocated a total of US$0.66M for primary and secondary pumps capable of dewatering the maximum expected mine inflow of 35 l/s. The primary pump would be a mono type of pump selected for its versatility and the secondary pumps would be electrically powered submersible pumps situated in the development ends. It has been assumed that simple sumps would be established throughout the mine to collect and reticulate groundwater to be recycled underground for service use. Any excess would be pumped to surface.

SRK notes that a detailed water balance for the underground mine needs to be undertaken in the next stage of study. There may be an excess of water which may require treatment before discharge.

Compressed Air

Due to the mechanised nature of the proposed operation it is not planned to use much equipment at Wassa UG that relies on compressed air. It is expected that compressed air would only be needed for minor service work. To this end, a 4” poly pipe would be reticulated underground down the main decline. A single 250 kW compressor has been assumed to be adequate for this duty.

Emergency Egress

SRK has not examined the issue of emergency egresses in detail for this PEA but notes the following:

- Provision will need to be made in future studies to comply with regulatory requirements in Ghana and it is not clear what the current planning guidelines are;
- There are options to provide emergency egresses using the primary vent raises where ladder ways or Alimak lifts could be installed; and
- Provision of well-equipped portable refuge stations.

Magazines

Detonators, explosives and blasting agents will be stored at the existing Wassa magazine on surface delivered from existing suppliers in Ghana. Charging units and utility vehicles will transport the explosives underground for immediate use or for storage in approved underground magazines that will be constructed.

Underground Workshop

The contractor and owner crews will construct satellite workshops in the underground workings for small repairs and servicing but all major mechanical work will be undertaken in surface workshops.

15.3.15 Blasting

Development blasting has been planned to be carried out primarily using Ammonium Nitrate Fuel Oil ("ANFO"), Nonel detonators and emulsion primers. Wet holes would be charged with emulsion cartridges. The WIFI system installed in the mine for communication will also be used for remote blasting of all stopes and development ends at the end of each shift.

Stope blasting would be carried out using ANFO as the primary explosive double primed with pentolite boosters and initiated with Nonel stope detonators.

15.4 Surface Infrastructure

15.4.1 Site Layout

The current site layout for the Wassa mine is provided below in Figure 15-22 and shows the existing location for the following mining areas and major infrastructure:

- Main roads, towns and power lines;
- Open pits and waste storage areas;
- Processing facilities;
- Tailings storage facilities; and
- Site accommodation.

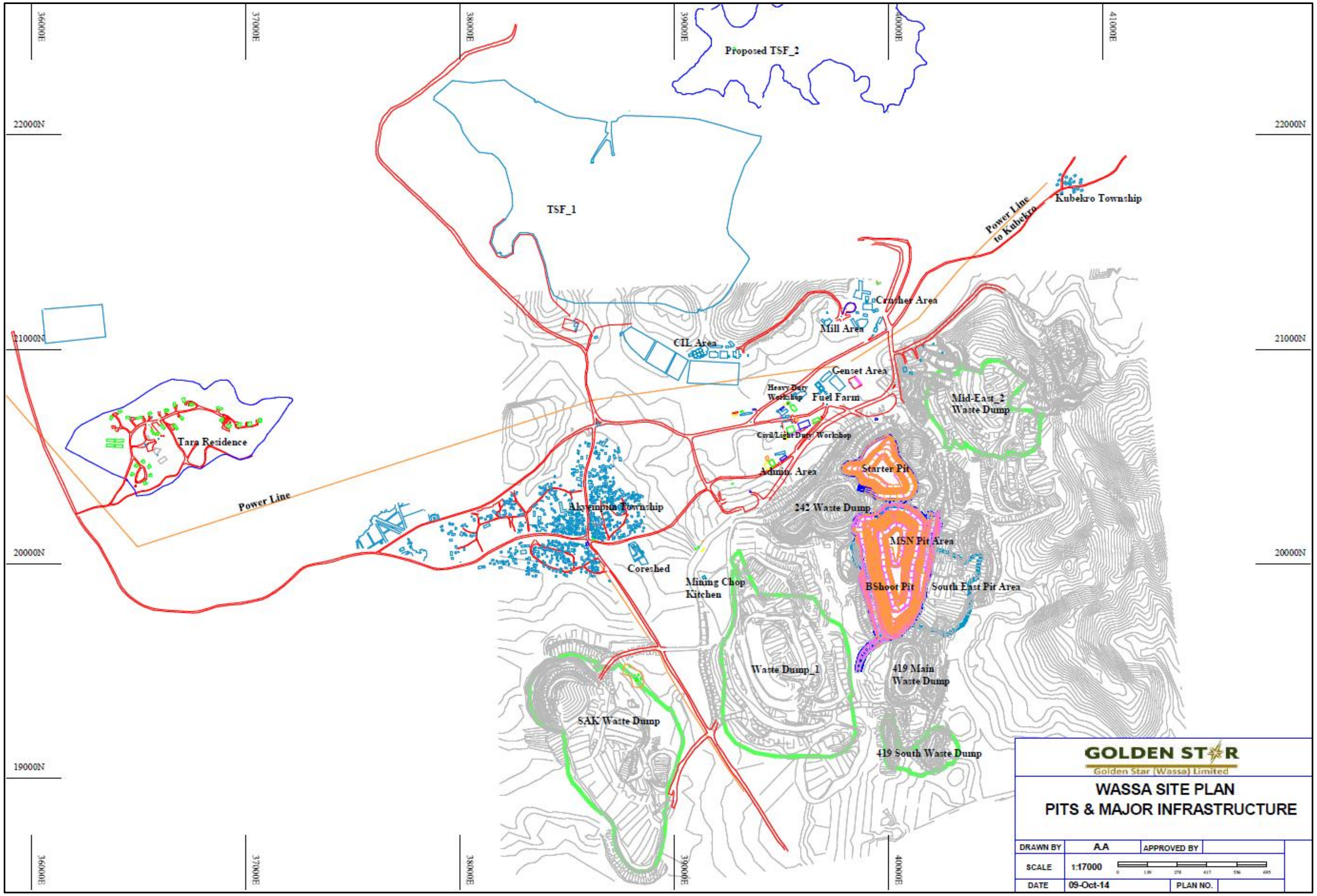


Figure 15-23: Existing Site Layout – Wassa Mine

15.4.2 Mine Office

It has been assumed that the contractor would provide their own facilities including offices, change house and stores. The owner would construct simple office building to provide working places for the supervision team and change house would be constructed for the owner crews working underground. The existing Wassa facilities would be used to provide administration support, warehousing, purchasing and catering.

15.4.3 Workshop Facilities

The contractor would construct its own workshops on surface. The owner would use the existing facilities that service the open pit equipment fleet.

15.4.4 Explosive Magazines

The existing surface magazine would be used to accommodate detonators, explosives and blasting agents. The facilities would act as a storage buffer to supply the small underground magazines on a daily basis.

15.4.5 Backfill Plant

SRK proposes the use of a cemented rock fill in the primary stopes in order to maximise extraction of the secondary stopes. It is envisaged that the fill would be delivered to the stopes by the haul trucks in a round-haulage scenario, eliminating the need to put in place backfill raises. In order to minimise haulage cycle times in this scenario, a cement slurry preparation facility, such as a batch plant, will need to be located on surface as close to the RoM pad or portal as possible. Trucks returning underground will be filled with waste and have a measured dose of cement slurry applied to the load. This will then be tipped directly into stopes. SRK has assumed that the cement dosage for planning purposes in this PEA will be 10% of the fill mass.

15.4.6 Water Supply

The UG mine at Wassa would receive initially receive service water from a header tank at the portal fed from the Wassa pit dewatering system. Water would be reticulated underground by gravity via a 150mm diameter poly pipe carried down with the decline. Smaller diameter poly pipes would reticulate the water into the levels and sublevels. Subsequently water would be fed into the underground system from the various sumps positioned to collect groundwater and water from the drills and washing down.

15.5 Mineral Inventory & Schedule

15.5.1 Introduction

A mine schedule was generated using the Deswik mine planning package, which integrates design, sequencing and scheduling functions into a single suite of software. Current open pit designs were provided, along with open pit productivities in order to inform a combined open pit and underground schedule. In the schedule, the underground production would come on line as soon as possible to feed high grade material to the mill, with the mill feed being supplemented with open pit material in order to achieve a feed rate of 2.6 Mtpa in the first 5 years.

15.5.2 Open Pit

Scheduling Methodology

The open pit designs were split into bench blocks and linked in a logical sequence for each pit, based on elevation. A ramp proximity priority was applied as a scheduling parameter in order to ensure that material closest to the ramp was mined first.

The open pit schedule commences in 2015 at a RoM feed of 3 Mtpa and finishes at the end of 2020. The average RoM feed over this period is 1.7 Mtpa with an average grade of 1.44 g/t Au and a stripping ratio of 5.0 (t_{waste}:t_{ore}). The annual depletion from a long section and plan view are shown in Figure 15-23 and Figure 15-24 respectively. Figure 15-25 shows the annual production profile for the Wassa open pit.

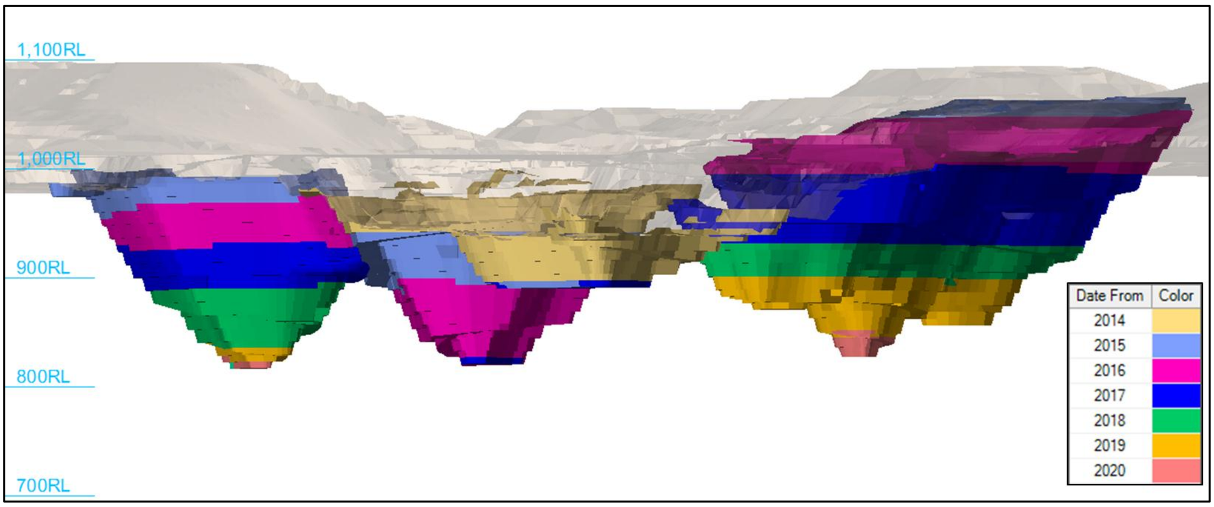


Figure 15-24: Wassa Open Pit Schedule (Long Section View)

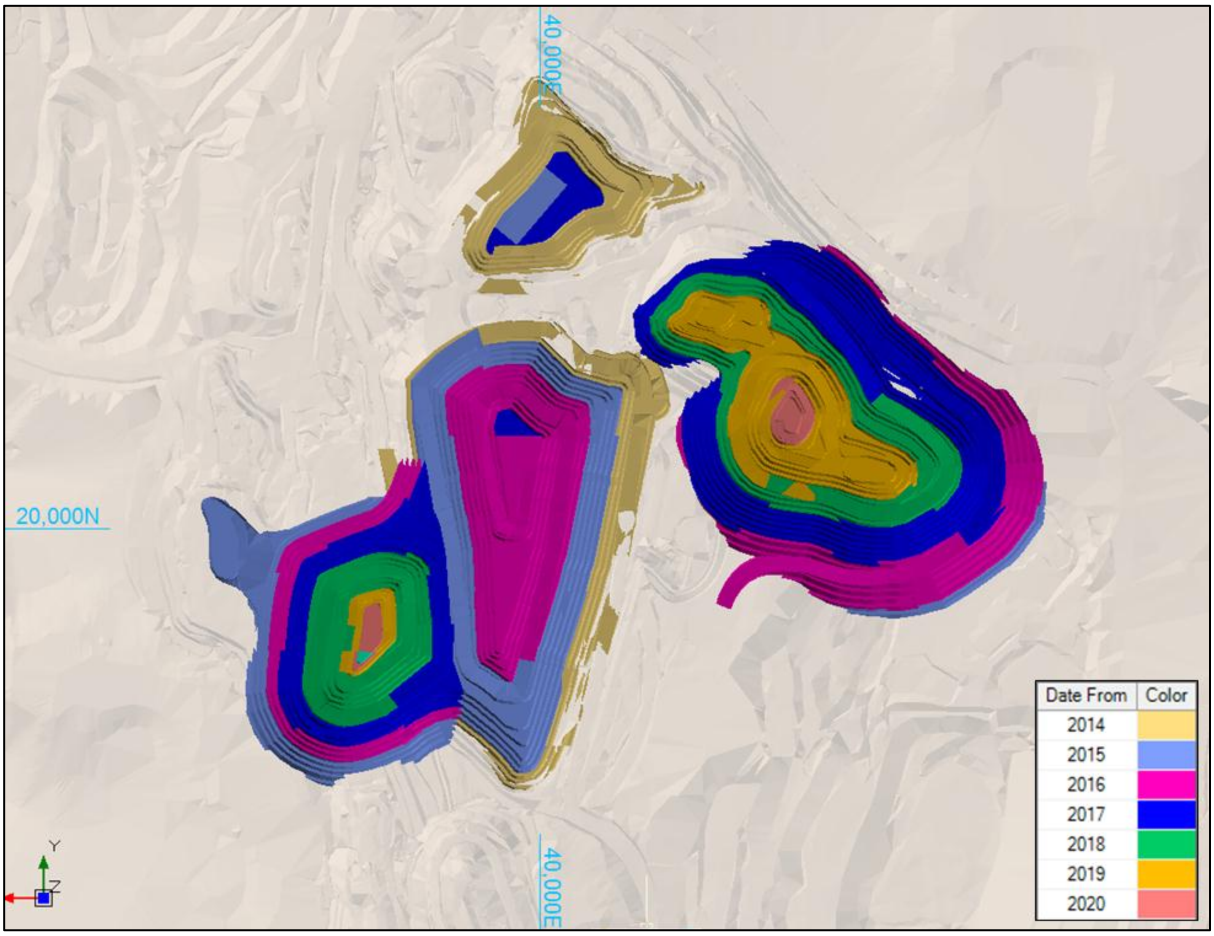


Figure 15-25: Wassa Open Pit Schedule (Plan View)

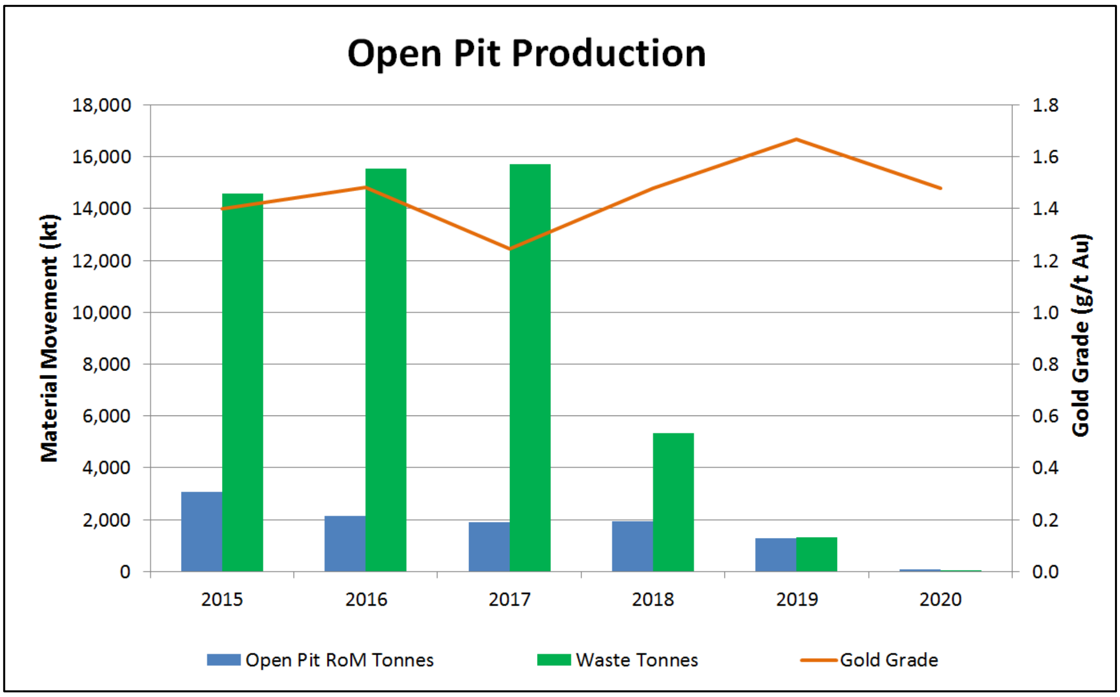


Figure 15-26: Open Pit Production profile for Wassa

The waste movement over the schedule totals 52.5 Mt over the 6 year period and the approach to scheduling takes into account the current performance of the open pit operations to ensure that the production rates are achievable. The waste storage designs for each open pit are incorporated into the mining schedule and have sufficient capacity for a number of years of operation. Further design work is required for waste storage facilities to cover the full mining schedule and opportunities to backfill completed mining areas need to be identified.

15.5.3 Underground

Scheduling Methodology

The underground production stopes and development were sorted into underground zones in order to facilitate sequencing and scheduling. Figure 15-26 shows the designation of these areas. The design entities were then linked together in a sequence, with the following guiding principles:

- Ventilation drifts and raises are to be developed as soon as possible;
- Production from the 807 level (the highest sub-level) can begin as soon as sub-levels and the primary exhaust raise to surface was in place, while main decline development continues, in order to avoid leaving a crown pillar;
- Stope production will be conducted in a bottom-up sequence;
- Sub-level access development to be delayed until required for production;
- Stope development tasks begin before production,;
- Mining on a level progresses in a primary-secondary sequence in a transverse stoping arrangement; and
- Development of a given underground zone can only begin once mining in the previous area is well-advanced, in order to avoid opening up excessive amounts of production areas.

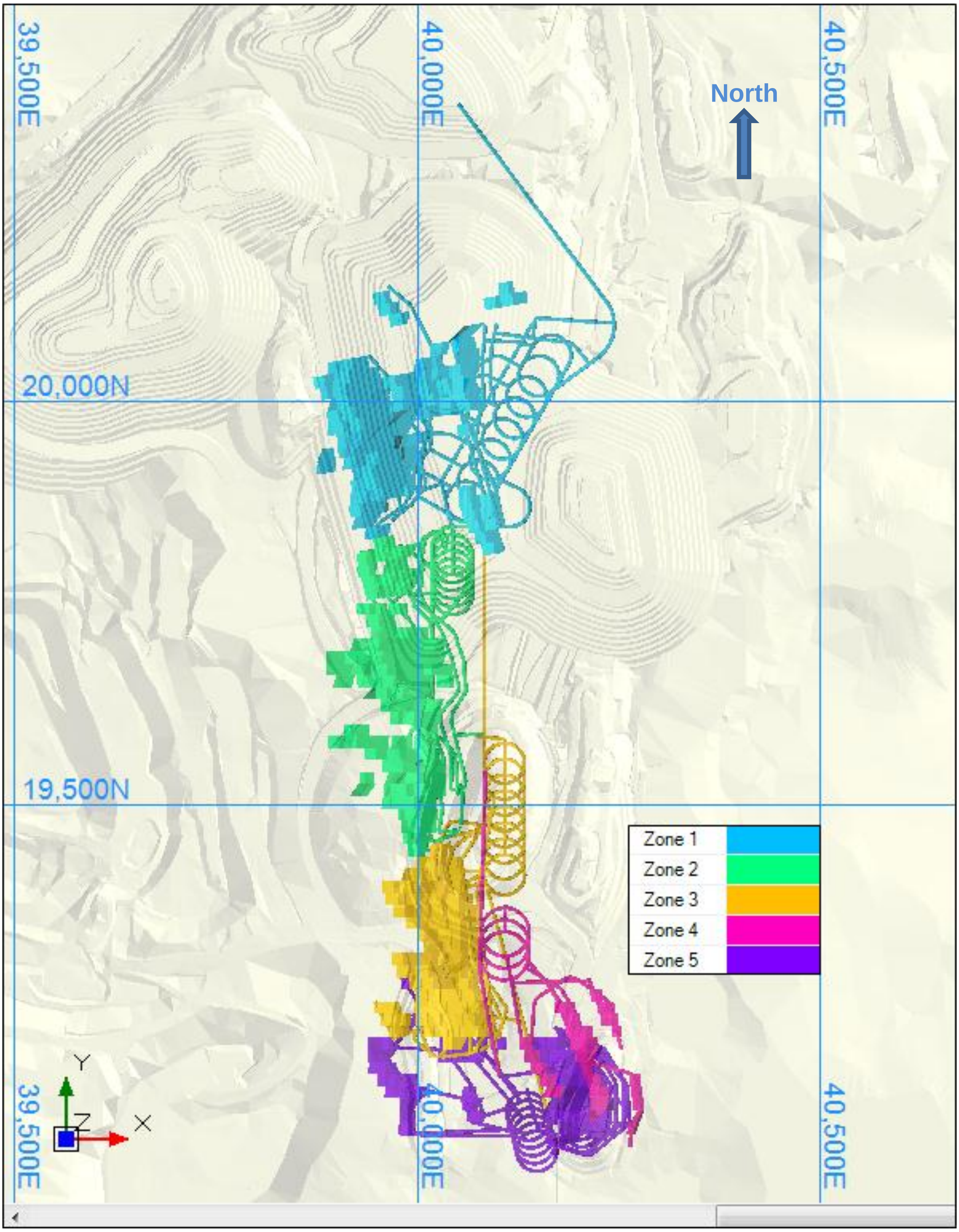


Figure 15-27: Underground mining zones plan view

Scheduling

The scheduling parameters used to develop the mine plan for the Wassa underground mine are outlined in Table 15-15. The stope production rate has been estimated based on the cycle times presented in Table 15-16.

Table 15-15: Mine Scheduling Parameters

Description	Unit	Planned Rate
Decline Development	m/month	110
Accesses	m/month	110
Stope Development	m/month	110
Vertical Development	m/day	5
Stope Production	t/day/stope	650

Table 15-16: Estimated Production Stope Cycle Time

Activity		
Stope Dimensions	15m wide x 15m high x 25m long	
	25,300 t/stope	
Drilling Time	6.6	days
Charging Time	2.0	days
Mucking Time	17.0	days
Backfill	15.0	days
Average Individual Stope Productivity	623	tonnes per day

Development and Mining Sequence

The development of the main decline is scheduled to begin in January 2015; the first phase, allowing access to the stopes in zone 1, will be completed in December 2017. The final phase of the decline will be complete in 2021. The establishment of the main exhaust raise, the ventilation drift connections and the internal spiral raises by Q3 2015 will allow production to begin in Q4 2015 while development of the lower part of the mine continues. Once the main intake raise is established, full production from the main stopes can begin. Stopes will be mined bottom up in a primary/secondary sequence, relying on a cemented backfill to provide support for mining secondary stopes. The proposed production cycle is:

1. Establish an extraction drift at the base of the stope and a drill drift at the sub-level interval height above the extraction drift;
2. Drill, blast and muck the stope; and
3. Fill the stope to the sill of the drill drive, which then becomes the extraction drift for the stope above.

Schedule Results

The Wassa underground stopes can support an eleven year mine life, with a three year ramp-up to a nominal production rate of 750 ktpa, and then a further expansion to 1 Mtpa seven years after the start of underground mining. The development and production schedules are coloured by year of completion in Figure 15-27 and Figure 15-28 respectively. The physical production schedule for the Wassa underground mine is presented in Table 15-17.

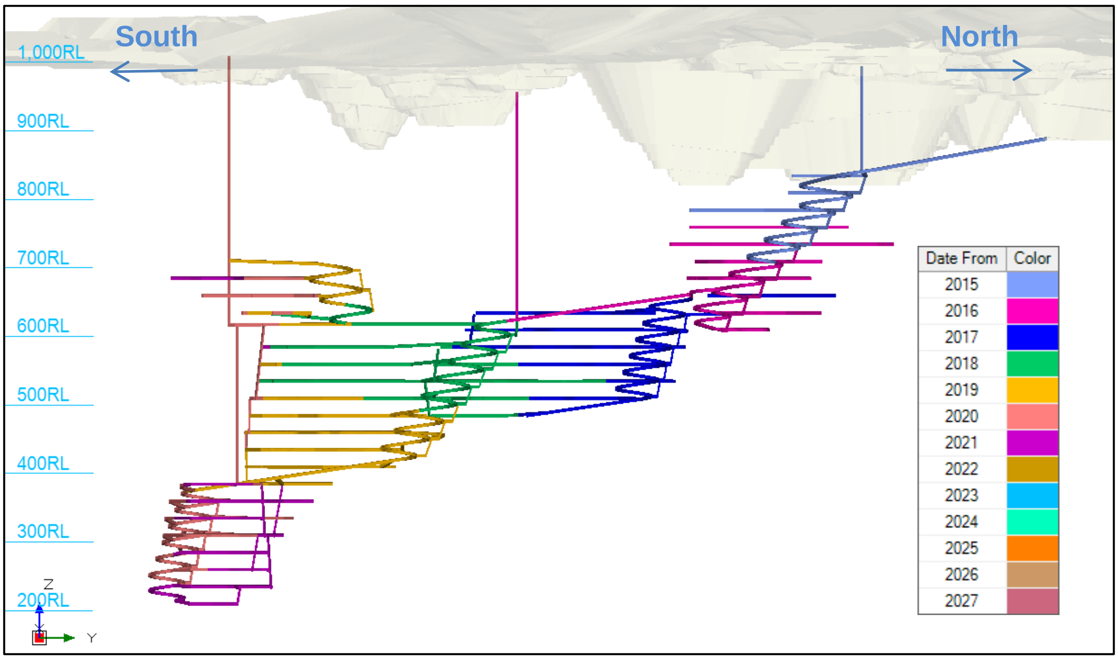


Figure 15-28: Development design coloured by period – Long Section looking West

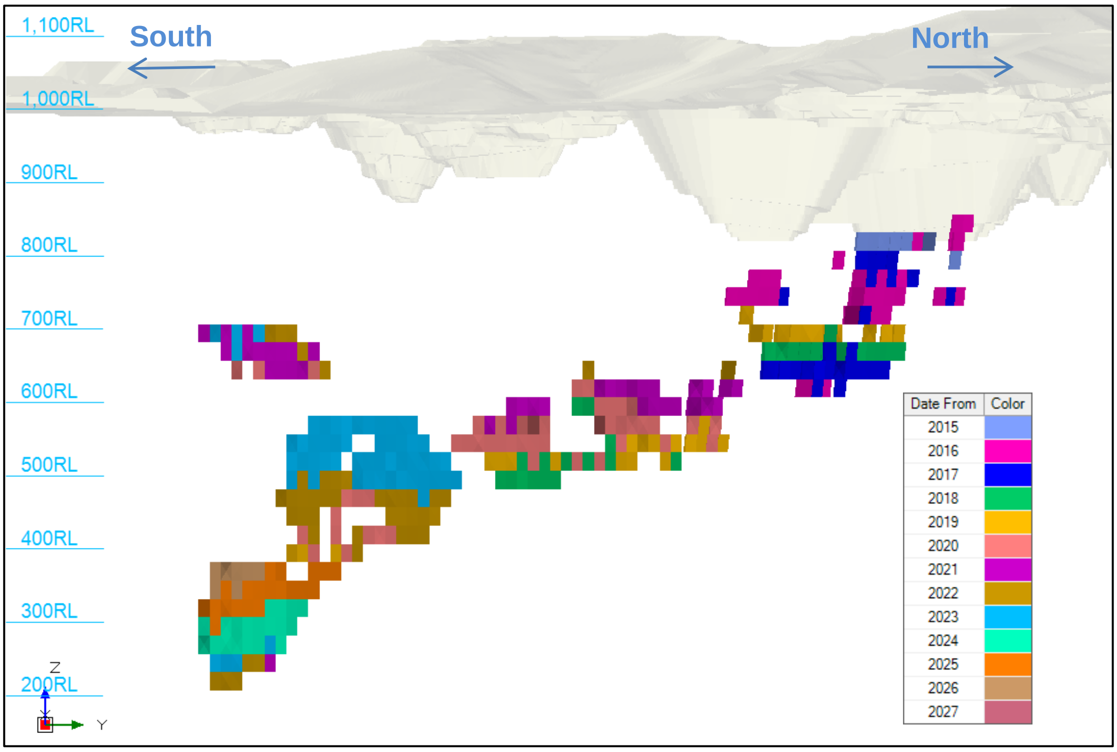


Figure 15-29: Production stopes coloured by period – Long Section looking West

Table 15-17: Wassa combined open pit and underground mine development and production schedule

		LOM	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026
Underground Physicals															
Mine Development (waste)															
Horizontal Development	m	22,211		2,556	2,860	3,626	3,515	3,835	3,325	2,496					
Vertical Developmnet	m	1,103.3		151	333				619						
In-stope Development	m	30,664		600	2,335	2,626	3,440	3,102	3,524	3,675	3,505	3,204	2,250	1,648	755
RoM Material	kt	**8,299**		**102**	**506**	**750**	**753**	**816**	**986**	**1,018**	**1,020**	**1,020**	**610**	**506**	**212**
RoM From Development	kt	2,070		41	158	177	232	209	238	248	237	216	152	111	51
RoM from Stopes	kt	6,229		61	348	573	521	607	748	770	783	804	458	395	161
RoM Metal	koz	**1,198**		**11.9**	**73.5**	**136.5**	**126.5**	**111.2**	**120.7**	**139.3**	**139**	**141.7**	**85.6**	**77.5**	**35**
RoM From Development	koz	310.0		4.94	26.29	32.31	37.58	28.42	30.74	34.88	34.67	30.91	22.93	17.6	8.73
RoM from Stopes	koz	888.		6.99	47.23	104.21	88.89	82.77	90.01	104.41	104.36	110.77	62.7	59.94	26.23
RoM Metal	g/t	**4.49**		**3.64**	**4.52**	**5.66**	**5.22**	**4.24**	**3.81**	**4.26**	**4.24**	**4.32**	**4.37**	**4.77**	**5.13**
RoM From Development	g/t	4.66		3.75	5.17	5.68	5.04	4.23	4.02	4.37	4.55	4.45	4.69	4.93	5.32
RoM from Stopes	g/t	4.44		3.56	4.22	5.66	5.31	4.24	3.74	4.22	4.15	4.29	4.26	4.72	5.07
Open Pit Physicals															
RoM Material Mined	kt	11,977	**1,539**	**3,080**	**2,155**	**1,896**	**1,937**	**1,287**	**82**						
RoM Grade Mined	g/t	1.40	1.18	1.40	1.48	1.25	1.48	1.67	1.48						
Waste Material Mined	kt	58,373	**5,858**	**14,567**	**15,541**	**15,718**	**5,338**	**1,305**	**46**						
Plant Feed Plan															
Plant Feed Tonnes	kt	20,277	**1,075**	**2,616**	**2,656**	**2,645**	**2,690**	**2,620**	**1,585**	**1,020**	**1,020**	**1,020**	**612**	**507**	**212**
Plant Feed Grade	g/t	2.67	1.35	1.51	2.08	2.51	2.53	2.38	2.69	4.25	4.24	4.32	4.36	4.77	5.13

The production profile in Figure 15-29 shows a peak in gold grade within three years of the start of mining (2017), with a trough in 2020 and steadily increasing gold grades thereafter. No schedule optimisation has been performed to prioritise high grade stopes over lesser ones.

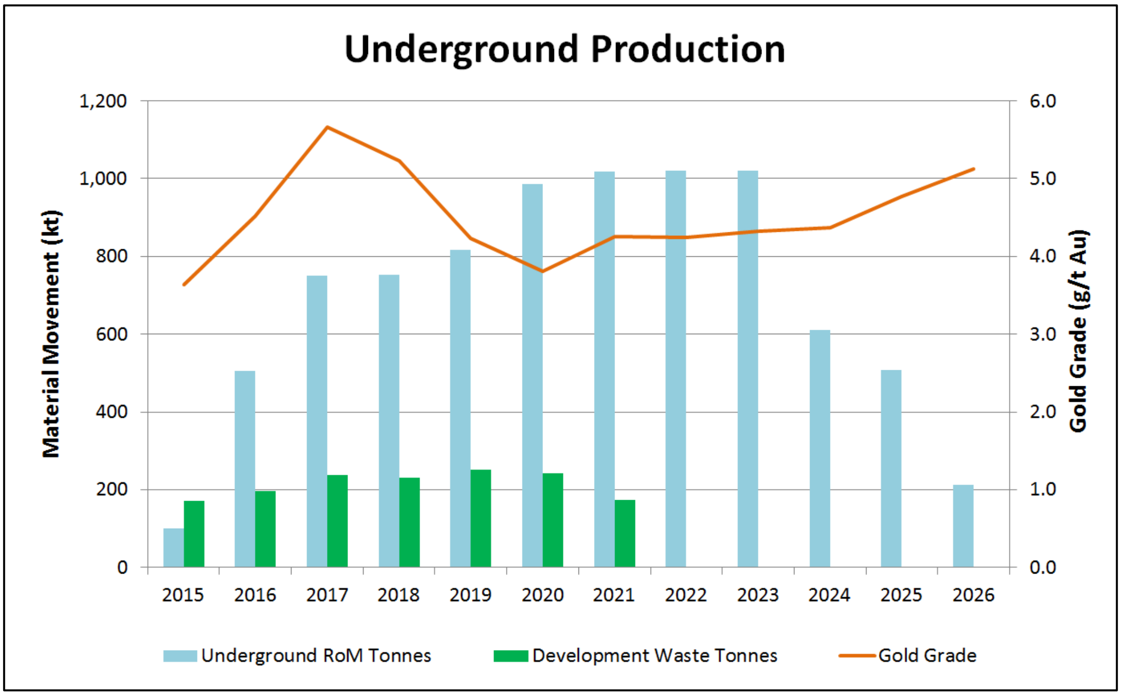


Figure 15-30: Underground Production profile for Wassa

SRK notes the following with regard to the Wassa underground production sequence and schedule:

- A steady production rate is sustainable post-ramp up until the end of the planned mine life;
- The production rate for the mining of Zone 1 is planned to be nominally 750ktpa, with a ramp-up to 1Mtpa for subsequent zones;
- Peak production is achieved in 2021 to 2023 at an average of 2,775 tpd;
- Peak gold production is in 2023; and
- Opportunity exists in future stages of more detailed study to optimise the grade profile and bring gold ounces forward by looking at more detailed sequencing for the primary production stopes.

15.5.4 Combined Open Pit and Underground Schedule

The combined Wassa open pit and underground production schedule for RoM Feed and gold grade is provided in Figure 15-30 which shows a reduction in production tonnage over time and an increasing gold grade with depth as the underground becomes the primary source of RoM Feed.

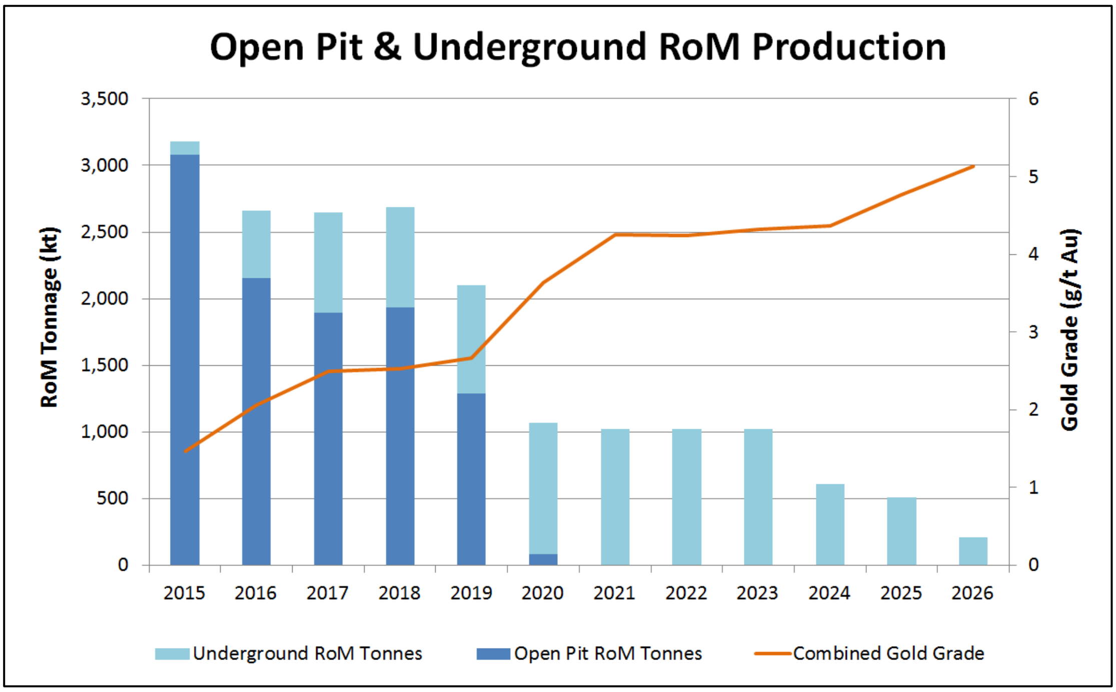


Figure 15-31: Combined Open Pit & Underground Production profile for Wassa

16 RECOVERY METHODS (ITEM 17)

16.1 Flow Sheet Description

Gold recovery is achieved at Wassa through the use of conventional CIL technology, although the plant itself contains a few atypical features due to its history and development. The operation has its roots in a heap leach operation that ran from 1998 to 2001: this operation had a design capacity of 3 Mtpa and resulted in the establishment of two heap leach pads.

On obtaining ownership of the project, GSR commissioned a FS for a CIL operation, the process engineering component of which was conducted by MDM of South Africa. The FS was completed in 2003, and the CIL plant commenced operations in late 2005. The crushing circuit from the previous heap leach operation was retained and the new ball mills were installed in the same general location. The CIL circuit was constructed near to the previous heap leach operation's adsorption, desorption and recovery ("ADR") circuit, a distance of approximately 900 m from the comminution circuit. Cyanide is added to the slurry from the comminution circuit, which is then pumped the 900 m to the CIL circuit; this pipeline effectively operates as a "pipe reactor", with typically 80% of the leaching taking place within the pipe before the slurry reaches the CIL tanks. The main features of the plant are:

- Four stages of crushing to a nominal product size of 80% -10 mm. This circuit had an original design capacity of 3 Mtpa to a product size of 80% -13 mm.
- Grinding in two parallel ball mills, each 5.0 m diameter x 6.9 m long with 3 MW motors, to a nominal product size of 80% -75 μm. The design product size for the grinding circuit was 80% -106 μm; the lower target product size now used is in response to the need to grind the fresh material finer in order to maximise the gold recovery.
- Gravity gold recovery with a 48" Knelson Concentrator in closed circuit with each mill. The concentrate from the Knelson Concentrators are combined and upgraded using a Gemini Table. The concentrate from this stage is smelted separately to the CIL circuit gold for accounting purposes.
- Cyanide and oxygen are added at the CIL feed hopper, which is at the feed end of the "pipe reactor".
- The CIL circuit consists of six tanks, with a total nominal residence time of 18 to 21 hours, depending on throughput & CIL feed slurry density. Oxygen is added to the first tank, hydrogen peroxide to the second tank, and air to the remaining tanks. No further cyanide is currently added in the CIL circuit.
- The carbon treatment circuit (acid wash, elution & regeneration) is based on an 11.5 t carbon batch size.
- There are two Pressure Swing Adsorption ("PSA") oxygen plants.
- The CIL plant has a nominal design capacity of 3.5 Mtpa that was historically achieved using a feed blend of 45% Fresh and 25% Oxide and 30% reclaimed spent heap leach material. With the transition to largely or wholly fresh material feed, and the need to grind this material finer in order to maximise the gold recovery, the plant currently operates at a throughput rate of 2.7 Mtpa and has been operating on 100% Fresh material since July 2014.

Metallurgical accounting is based on ball mill feed weightometers, samples of the CIL feed and tailings, and the estimated or actual Gravity Recovered Gold ("GRG"). Estimated GRG is used to determine the back-calculated head grade daily, whilst the reconciled head grade is determined weekly based on the actual GRG poured and the change in estimated gold inventory.

Gold values are determined using a cyanide leach assay procedure. Fire assays are conducted on high grade samples and as a periodic check on the cyanide leach assay tails. Fire assays are also used to provide an estimate of silicate-hosted gold in the CIL tails, the value of which is added to the cyanide leach assay value to give the reported CIL tails grade.

The operation achieved compliance with the International Cyanide Management Code in early 2010 and is currently Code certified.

16.2 Historical Production

Production statistics for the CIL operation for the six years since 2007 are shown in Table 16-1.

Table 16-1: Historical Production Statistics – CIL

Item		Unit	2009	2010	2011	2012	2013	2014 H1*
ROM		kt	2,506	2,434	2,391	2,499	2,549	1,458
		g/t Au	2.78	2.36	2.04	2.09	2.40	1.59
Heap Leach		kt	147	214	188	7.7	146	96.4
		g/t Au	0.73	0.59	0.39	0.24	0.3	0.3
Total Feed		kt	2,653	2,648	2,579	2,507	2,695	1,555
		g/t Au	2.67	2.22	1.92	2.09	2.29	1.51
Recovery	**Gravity**	%	22.7	26.5	27.0	49.6	54.4	38.5
	CIL	%	72.4	68.3	67.3	45.0	45.6	54.2
	Total	%	95.1	94.8	94.3	94.6	94.5	92.7
Au Produced		oz	223,848	183,931	160,616	160,917	183,788	71,238
Operating Cost		US$/t processed	8.54	12.49	15.63	17.02	17.38	16.59
		US$/oz Au	101	179	253	268	255	362

* Jan-Jul 2014 incl.

The historically-reported gold recoveries correspond very well with those predicted from the original metallurgical testwork. The gravity circuit contributes significantly to the total gold production; this is also consistent with the results of the original laboratory testwork.

The major contributing elements to the Wassa CIL operating cost are electrical power; grinding media, cyanide and labour, which between them make up 50 to 60% of the total. The principal reagent consumptions are approximately 0.5 kg/t for NaCN and approximately 0.6 kg/t for lime which are similar to the figures reported from the laboratory testwork, which are typical for a CIL operation.

16.3 Plant Improvements

GSR is currently in the process of modifying the plant circuit to include new cyclone banks and a thickener. The current cyclone banks are incorrectly sized for 100% Fresh material feed, resulting in an excess of fine material in the cyclone underflow returning to the ball mills and an excess of coarse material in the overflow feeding the CIL tanks. This results in a reduced ball mill throughput and inconsistent density, poor leach kinetics and high power draw in the CIL. The following plant modification work is being undertaken:

- Install new cyclone banks and thickener;
- Recalibrate current cyclone banks and feed overflow to new cyclone bank;
- New cyclone bank overflow to thickener; and
- Underflow from new cyclone banks and thickener to the pipe reactor feeding the CIL

tanks.

This new process is expected to reduce the recycle load to the ball mills and it is estimated that the improved efficiency will result in a 12% improvement in mill throughput of fresh material from 2.7 to 3.0 Mtpa. The new process is also expected to result in a consistent CIL tank density of 1.42 t/m^3, which should improve the leach kinetics and reduce the CIL agitator power draw. Figure 16-1 shows a simplified schematic of the current flow sheet in black with the upgrades shown in red.

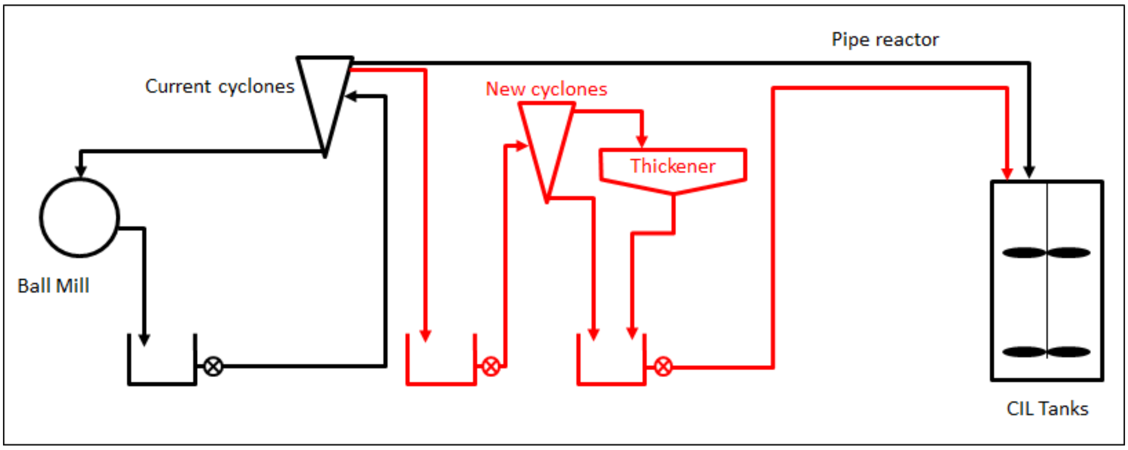


Figure 16-1: Schematic view of classification circuit improvements

The procurement process is underway and it is expected that the new process will be commissioned by June 2015 at a capital cost of US$3.5M.

16.4 Impact of High Grade Underground Feed

The proposed Wassa underground operation will provide of the order of 2,500 tpd of high grade (4 to 6 g/t Au) feed to the mill. Table 16-2 shows the future production profile for 2015 to 2020 based on the underground and open pit mine plans developed in this PEA (see Table 15-17). The Au production figure shown in Table 16-2 assumes a 92% Au recovery.

Table 16-2: Future Production Statistics – CIL

Item	Unit	2015	2016	2017	2018	2019	2020
OP Feed	kt	2,514	2,150	1,895	1,937	1,804	599
	g/t Au	1.42	1.51	1.26	1.48	1.54	0.85
UG Feed	kt	102	506	750	753	816	986
	g/t Au	3.64	4.52	5.66	5.22	4.24	4.02
Total Feed	kt	2,616	2,656	2,645	2,690	2,620	1,585
	g/t Au	1.51	2.08	2.51	2.53	2.38	2.69
Au Produced	koz	113	158	190	195	179	122

While the comminution circuit improvements described in Section 16.3 are expected to provide scope to increase the plant capacity to 3.0 Mtpa with 100% fresh RoM feed, the projected plant feedrates shown in Table 16-2 are lower (not exceeding 2.7 Mtpa) in order to take into consideration the expected increase in mineralized material hardness with increasing depth, as described in Section 12.1.

The forecast gold production figures are lower than the historical production achieved to date (see Table 16-1) and so the gold recovery section of the plant (i.e. leaching, adsorption, elution and electrowinning) will be capable of achieving the forecast levels of gold production.

The current open pit LoM extends to 2020 only, and so from that point on, the total LoM shows a reduced feed rate from the underground only. However, with the higher grade of the underground feed material, the overall gold production rate remains at a similar level to that shown for 2020 until the underground operation starts to ramp down in 2024.

17 PROJECT INFRASTRUCTURE (ITEM 18)

17.1 Overview

The development of the underground mine will be carried out in conjunction with the existing open pit mining operation. It is stated that as result of the introduction of an underground mine, the production rate of the tailings will not change. There are two tailings facilities that will accommodate the anticipated tailings production, with the existing facility referred to as TSF1 and a new facility referred to as TSF2.

17.2 Existing Tailings Storage Facility (TSF1)

17.2.1 Introduction

The TSF1 is located northwest of the plant site at the head of a southerly draining valley, immediately adjacent to and then over some pre-existing leach pads. Ground levels in the valley range from 1000 m RL on the valley floor to more than 1060 m RL on the surrounding hills. The TSF1 is a cross valley impoundment created by the construction of a Main Embankment in the south with confining Saddle Embankments at the north of the facility. Natural ridges provide containment at the east and west of the storage area. Access to TSF1 is via an unpaved access road west of the plant site area. The catchment area of TSF1 is estimated to be about 140 ha, of which about 124 ha will be covered with tailings at the end of the facility design life.

17.2.2 Construction Sequence

The TSF1 facility has been raised in stages with the first stage being constructed in 2004. The first stage comprised a main cross-valley embankment and a small extension embankment located to the east of the main embankment, which were both initially constructed to 1018.5 m RL.

Construction of the Stage II was completed in November 2007 and was raised to 1028.5 m RL. This stage comprised a major raise on the main and extension embankments (downstream method on the main embankment and centreline method on the extension) and the construction of saddle embankments 2 to 5 around the periphery of the facility.

The Stage III works comprised upstream raises on all the embankments except Saddle Embankment 3 (which was raised by centreline method) to a crest level of 1031.0 m RL. This work was carried out in the period between June 2008 and February 2009.

The Stage IV works comprised the raise of all the embankments up to an elevation of 1034.0 m RL and was completed in November 2010.

The Stage V construction raise included raising of the facility by 3.0 m to 1037.0 m RL and was completed in November 2012.

The current Stage VI raise was permitted to 1039.0m RL but has been constructed to 1038.0m RL by the end of April 2014. Preparation of the site for the final 1m raise to 1039.0m RL elevation is currently being organised and is expected to be completed before the end of 2014.

Permitting is underway to include the heap leach area as part of the TSF1. The Environmental Scoping Report was submitted to the EPA in July 2014. A plan view of the TSF1 layout is shown in Figure 17-1.

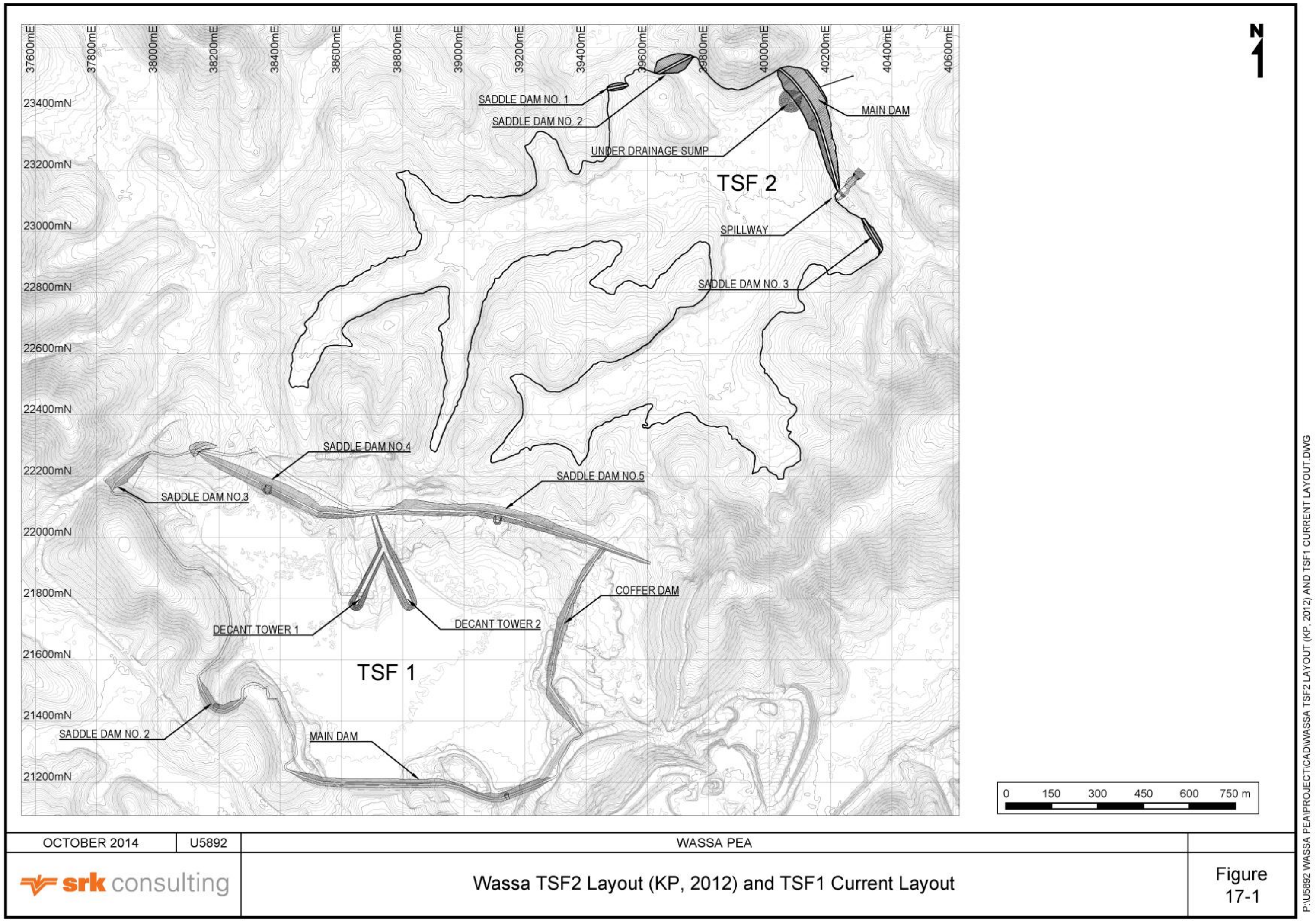


Figure 17-1: TSF1 and TSF2 Layout

17.2.3 Design Parameters

The TSF1 Stage 6 extension design was based on the following parameters:

- Annual throughput 2.7 Mtpa;
- Additional storage for 2.5 years (May/June 2013);
- Tailings solids (45% solids by weight, 55% water); and
- Average tailings dry density of 1.40t/m^3.

17.2.4 TSF1 Operation

Tailings are currently distributed via a 450 mm diameter high-density polyethylene ("HDPE") pipeline, which runs around the perimeter of the structure. Discharge occurs through a series of 30 mm diameter plastic spigots, which are attached to the main perimeter pipeline at approximately 10 m intervals. This distribution method controls the beach angle around the perimeter of the TSF and ensures that ponding is minimised for effective drainage. A supernatant pond currently exists in the north of the TSF area as beaches slope away from the main embankment. Deposition occurs around the entire periphery to control the position of the supernatant pond.

Until 2009, clarified supernatant was returned to the plant via two barge pumps at a rate of up to 800m^3/hr. During the Stage 3 construction program in 2009, two permanent decant towers, which comprise perforated concrete rings, were constructed. An emergency spillway has been constructed at saddle embankment 4 that would be available to discharge excess water arising from any extreme rainfall events.

17.2.5 TSF1 Monitoring

KP has been responsible for the monitoring of the TSF1 since October 2004. The facility has been monitored regularly in order to confirm that the facility is constructed / operated according to the original design. The monitoring reports have been issued on quarterly bases that include visual inspection of the embankments, tailings beach, pipeline, drainage structures and data collection and analysis of ground water monitoring wells, piezometer readings, drain flows.

Glocal Engineering has been responsible for completion of construction and operational monitoring of the facility and provides GSWL with regular reports.

17.3 New Tailings Storage Facility (TSF2)

17.3.1 Introduction

SRK has undertaken a review of the "Golden Star Wassa Limited - TSF2 detailed design report" prepared by Knight Piesold Ltd. ("KP") in 2013. The site was not visited for this review. After two LoM studies by KP in 2008 and 2010, a detailed design for the preferred storage option TSF2 was produced for tailings from the continued open pit mining operation. This was revised following the issuance of the environmental approval, which required the use of a liner.

The TSF2 design herein is based on KP design and their assumptions.

17.3.2 TSF2 Design

The objectives of the TSF2 design are to provide:

- Secure confinement for process water;
- Permanent and secure confinement for all solid tailings;
- Cost effective embankment construction using local borrow and/or mine waste;
- Effective drainage systems that ensures maximization of tailings settled densities, and

- Protection of the environment, community and other resources.

Design Philosophy

The design criteria employed for TSF2 include:

- Provision of a design with minimal environmental impact;
- Provision of a design that limits excessive and uncontrolled seepage beneath impounding structures and into groundwater;
- Provision of storage capacity for the current life of mine, approximately 12 years at 3.5 Mtpa;
- Actual capacity is forecasted to be approximately 41 Mt;
- Provision of a design that provides staged cell development for off-setting capital expenditure;
- Provision of a design that allows sub-aerial deposition of tailings slurry to optimise the density of deposited tailings;
- Provision of a design that allows for ease of supernatant water reclamation for processing plant operation and maintaining optimum water storage on the facility; and
- Provision of appropriate monitoring equipment for assessment of potential downstream environmental impacts.
- TSF2 to be designed in accordance with recommendations of the International Commission on Large Dams ("ICOLD"), Australian Commission on Large Dams ("ANCOLD") and to meet Ghana Draft Mining Regulations.

Design Criteria

The design criteria are provided in the Table 17-1 and KP proposed TSF2 layout in Figure 17-1.

Table 17-1: Summary of Tailings Dam Design Criteria

Description	Design Criteria
Milled Throughput	
Monthly	291,666 tonnes
Total	3,500,000 tonnes per annum
Storm Capacity	1:100 year 24 hour event 1:100 year 12 month wet of average rainfall sequence 1:100 year 24 hour event Provide a minimum freeboard, of 1.0 m (between the pond and crest level).
Spillway	
Intermediate Stages	1:100 year 24 hour event when supernatant pond is at maximum designed operation level
Closure	24 hour PMF event when supernatant pond is at the invert of the spillway. Closure spillway will be designed at time of TSF2 closure.
Slope Stability Factors of Safety	
Static (Operations)	1.7 (as per Ghana Requirements)
Static (Closure)	1.7 (as per Ghana Requirements)
Psuedo Static (Operations)	1.1
Psuedo Static (Closure)	1.1
Earthquake Loads	
Operating and Closure	Maximum Credible Earthquake (MCE) = 7.0, 0.1g maximum ground acceleration (Seismic Activity in Ghana: past, present and future, P.E. Amponsah) (Ref. 7)
Slurry Characteristic	
Solids Concentration (Range)	35% - 45 % solids by weight
Solids Concentration (Average) Particle	40%
Specific Gravity	SG = 2.81
In-storage Density	Initial In-storage Dry Density = 1.1 t/m^3. In-storage Dry Settled Density = 1.4 t/m^3. Achievable In-storage Density =1.50 t/m^3
	1.347 t/m^3
	60% < 90 microns
Slurry Density	60% < 90 µm
Tailings Particle Size Distribution	
Tailings Deposition	
Beach Slopes	1 vertical : 200 horizontal
Subaqueous Slopes	1 vertical : 50 horizontal

17.3.3 Geotechnical Investigations

KP carried out a geotechnical field investigation that comprised of in total of 35 test pits and 13 boreholes. These were conducted within the alignment of the proposed embankments, the TSF basin floor and in areas of the hill sides and high ground. The various test locations were designed and investigated to:

- Determine the properties of the foundation materials;
- Determine the permeability of the basin geology;
- Evaluate groundwater levels;
- Evaluate the geotechnical characteristics of the natural slopes within the TSF basin, and
- Identify and evaluate potential sources of construction materials.

Laboratory testing on a number of samples obtained from the geotechnical investigation were conducted. The tests carried out included natural moisture content, particle densities, bulk density tests, particle size distribution, Atterberg limits, Standard Proctor compaction tests, permeability tests and shear strength estimations.

KP report concludes that the testing of the local in-situ soils proved that the soils are considered suitable to be used as construction material. It was however recommended that during construction the testing of borrow materials needs to be carried out to ensure that acceptable parameters are being achieved.

17.3.4 Embankment Design

KP considered different construction methods for TSF2 embankments including upstream, downstream and centreline method. Due to the overall stability of the embankment and as per Ghana Minerals and Mining Requirements the downstream method has been adopted. It is anticipated that all further raises will be developed using the downstream approach, however, it is stated that this needs to be confirmed during the design of the future raises. SRK agrees with this approach to use downstream construction method for embankments.

The embankments, main embankment and associated saddle dams will be constructed in stages. The staging will be provided in the next level of the design.

The embankments will comprise of the following:

- All embankments will have upstream slopes of 1V:2H and the downstream slopes of 1V:3H, with 8 m wide crest.
- A 500 mm high safety berm will be constructed on the downstream and upstream crests.
- A rip rap layer of 1.0m thickness will be placed on the upstream slopes.
- The crest will be topped with a 150 mm thick layer of wearing course material.
- The wearing course will include a 2% gradient, sloping into the TSF2 Cells.
- A chimney and blanket drains.

17.3.5 Slope Stability

KP carried out slope stability analyses using the limit equilibrium program Slope/W. In the program the Morgenstern Price method of analysis was employed which considered circular slips for both force and moment equilibrium. Stability was modelled in both drained and un-drained conditions for static cases and in the un-drained case for seismic cases.

A number of stability analyses were carried out and the review of the results indicates that the slopes of the embankments will have high factor of safety and that they are greater than the required minimum acceptable Factor of Safety (“FOS”) adopted for the analyses. The minimum FOS are presented in the Table 17-1.

Based on the information evaluated, design parameters and analysis undertaken, it was concluded that the tailings embankments meet normally accepted stability requirements.

17.3.6 Drainage Design

The drainage system has been designed to follow the natural drainage path of the TSF cells.

There will be two systems of drainage; under drainage and groundwater drainage.

The under drainage system will comprise of finger drains, branch drains, and collector drains and were sized to accommodate the anticipated flows through the tailings bed for all phases of the facility.

The groundwater drainage systems will consist of a number of finger drains constructed using 100 mm fully slotted uPVC drain coil pipe embedded in clean gravel that is surrounded by geotextile.

17.3.7 Decant System

The decant system will consist of a barge; 2 pumps mounted on the floating barge and decant return pipe. A floating barge was chosen due to the ease of operation with respect to positioning the barge within an acceptable body of water.

The decant water return system has been designed to provide up to 95% of the processing plant monthly water requirements as well as the capability to transport approximately 0.5 Mm^3 to the detoxification plant for treatment and release to the environment. Additionally, at times the system will also be responsible for removing water to the detoxification plant after large volume storms.

17.3.8 Tailings Deposition Strategy

Deposition of the tailings material will occur from evenly spaced spigots with slotted dropper pipes. It was stated that the dropper pipes will reduce tailings flow and achieve maximum particle separation at the optimum flow velocity. Tailings will be deposited sequentially to control the location of the supernatant pond and to achieve the maximum dry tailings density possible.

17.3.9 Storm Water Analysis and Emergency Spillway

The storms water analyses were carried out for 1:100 year, 24 hour storm event as per Ghana Minerals and Mining Regulations requirements. In addition, the analysis included 1:1000 year, 24 hour (331 mm) storm event, as the dam has been classified as Significant in accordance with the Canadian Dam Association (CDA) Dam Safety Guidelines 2007 (19).

17.3.10 Liner Design

Following the review of the report (Environmental Impact Assessment of February 2013), the Environmental Protection Agency gave notice (EPA/EIA/383) to the Mine on the 5th of April 2013 that the construction of the facility was permitted, provided the entire basin was lined with an HDPE liner. It is stated that the HDPE liner design over the entire basin and upstream slope has been incorporated into the TSF2 design.

17.3.11 Post Construction

Post construction activities have been identified and these will include water quality, geotechnical and environmental monitoring.

Continual tailings management, inspection and monitoring are critical to ensure that the facility is operated in accordance with accepted international codes of best practice and Ghanaian regulatory requirements.

17.3.12 Closure and Rehabilitation

GSWL have documented a provisional closure plan within the Environmental Impact Statement (“EIS”) for the environmental and socio-economic permitting of TSF2. As documented within the EIS, GSWL will incorporate the TSF2 closure plan with the wider GSWL closure plan.

Closure of TSF2 will be in accordance with Section 5.16 of the Environmental Permit issued 5 April 2013 and will include details of proposals for the control and discharge of surface water and storm water, the long term monitoring of discharge and seepage, the long term monitoring and auditing of the facility, as well as the rehabilitation and proposals for reuse of land on the embankment slopes and tailings beaches.

17.3.13 Conclusions

SRK agrees with the principal recommendations made in the KP report and the report meets the requirements for this PEA level of study.

SRK's assessment of the geotechnical investigation and embankment stability and seepage analyses attests that they are relevant to the size and scope of the project and demonstrate satisfactory results.

Social, environmental, and hydrogeological criteria used in the site evaluation are reported to be described in previous studies but are not stated in the reviewed document. This is acceptable as long the studies are undertaken and referenced.

18 MARKET STUDIES AND CONTRACTS (ITEM 19)

GSWL have conducted appropriate market studies and appropriate contracts are in place. The assumptions made concerning commodity price projections, product valuations and product specification requirements which are used for planning purposes of the current LoM are considered appropriate and are discussed further in Section 21.

SRK notes that contract mining is planned for the Wassa underground mining activities and there exist in Ghana several experienced international mining contractors who will be considered as part of a structured bidding process. No contracts are currently in place.

19 ENVIRONMENTAL AND SOCIAL APPROVALS AND MANAGEMENT (ITEM 20)

19.1 Environmental Approvals

19.1.1 Relevant Legislation and Required Approvals

Required Approvals

The Mining Act (Act 703 of 2006) 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 19-1. Ghanaian environmental legislation is well developed, and is enforced by the Environmental Protection Agency ("EPA").

Table 19-1: Primary environmental approvals that have to be obtained for mining operations

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

The overarching Act that regulates the environmental regime of Ghana is the EPA Act

(Act 490 of 1994). The main legal framework used by the EPA for regulating and monitoring mineral operations is the Environmental Assessment Regulations, Legal Instrument 1652 of 1999 (LI 1652). The EPA grants environmental 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 environmental impact assessment, which also covers social aspects, which is documented in an Environmental Impact Statement ("EIS"). For a mine, an EIS must include a reclamation plan (Regulation 14 of LI 1652) and a provisional EMP. The EIS is subject to a public hearing and review by the EPA before a permit is granted. An EMP must be submitted within 18 months of commencement of operations and must be approved by the EPA.

Operations that pre date LI 1652 are also required to obtain Environmental Permits. Generally, the approval of existing operations is based on an EMP, rather than an EIS.

All mines in Ghana are required to have a reclamation plan (Regulation 14 of LI 1652). Mines are also 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). 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.

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 indicated in the EIS or EMP; and submission of annual environmental reports to the EPA.

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 an EIS, EMP, and Reclamation Plan. They also include guidelines on environmental impact assessment 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 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 of other approvals to the facility until notified otherwise by the EPA.

Six mining regulations were promulgated in 2012 to define and facilitate implementation of concepts in the Minerals and Mining Act, 2006 (Act 703). The following regulations have particular relevance to environmental and social management:

- Minerals and Mining (Health, Safety and Technical) Regulations 2012 (LI 2182) – these regulations define 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 (LI2173) – 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 & 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, first and then by the Minister responsible for Mines.

GSWL has a localization plan that has been approved by the Minerals Commission that covers expatriate staff and is in full compliance with the regulation requirements. GSR is listed on the Ghana stock exchange and continues to submit its annual financial reports as required by the law.

19.1.2 Status of Environmental Approvals

For the Wassa Project, it will be necessary to undertake an Environmental Impact Assessment ("EIA") and submit an EIS to the EPA to obtain an Environmental Permit for the operation. The EIS must contain a closure plan for the project. GSWL does not foresee that there will be obstacles to the permitting of the project, as it is in a disturbed area and the project will have limited environmental effects as the majority of the new development will be within the existing disturbed area.

A summary of environmental approvals held by GSWL is provided in Table 19-2. GSWL received its first Environmental Certificate for the period 21/09/2006 to 20/09/2009. The new certificate, which was issued in April 2011, expired in April 2014. The certificate renewal process has been initiated with the submission of a revised copy of the EMP to the Agency in October 2013. Comments were received from the EPA in 2014 and the revised and finalized EMP was submitted to the EPA for approval in June 2014 which is pending.

The Certificate confirms the company has been compliant with all its statutory reporting obligations throughout its operational period.

The Environmental Certificate and the EMP are for the overall Wassa operation; they cover the Wassa operation, the suspended Hwini Butre and Benso operations, and all associated infrastructure, including the Hwini Butre and Benso access road.

Commitments under the schedule to the environmental permits and in the EIS for the operations include:

- The Company must post a reclamation bond within one year of commencement of operations – GSWL posted the reclamation bond in November 2004. The bond which has been renewed is left with the signing of the Reclamation Security Agreement with the EPA.
- Compliance with mining legislation.

Table 19-2: Environmental Approvals obtained for the Wassa Mine

Approval	Permit No.	Agency	Date of Issue	Expiry Date	Comments
Environmental Certificate	EPA/EMP/093	EPA	15/04/2011	14/04/2014	EMP submitted for renewal
Environmental Permit to pursue operations	EPA / EIA/112	EPA	18/03/2004	N/A	
Environmental Permit of the Wassa Power Project	Form D (0010335)	EPA	07/05/2004	N/A	
South Akyempim Environmental Permit	EPA/EIA/190	EPA	02/06/2006	NA	
Hwini-Butre/Benso Project Environmental Permit	EPA/EIA/247	EPA	02/10/2007	NA	
Reclamation Bond		EPA	22/07/2011	01/05/2012	Applied for renewal
Water Use Permit	N/A	Water Resources Commission	09/08/2010	09/08/2013	Applied for renewal
Permission to divert Adehesu creek at South Akyempim	NA	Water Resources Commission	06/12/2006	N/A	
Water Use Permit Diversion of Ben and Subri Streams	N/A	Water Resources Commission	27/03/2008	N/A	
Water Use Permit (HBB Abstraction)	N/A	Water Resources Commission	01/07/2010	01/07/2013	Applied for renewal

The mining leases listed in Section 3 also contain conditions relevant to environmental management. The Wassa Mining Lease LVB 7618/94, Benso Mining Lease LVB26871/07, and Hwini Butre Mining Lease LVB1714/08 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.

19.2 Environmental and Social Setting

19.2.1 Administrative Setting and Nearest Settlements

The Wassa Mine is adjacent to the Akyempim community in the Wassa East District of the Western Region of Ghana and is 62 km north of the district capital, Daboase, and 40 km east of Bogoso. Cape Coast is approximately 90 km to the south. The Wassa project area is predominantly rural and there are no major urban settlements within 30 km of the operations.

The main access to the site is from the east, via the Cape Coast to Twifo-Praso road, then over the combined road-rail bridge on the Pra River. Additionally, there is access directly from the south through Mpohor and then via the access road from the suspended Hwini Butre operations. Akyempim is linked to Tarkwa and Ateiku by a network of feeder roads that also link it to Huni Valley, Abosso, and Bogoso.

The two largest villages within the Wassa mining lease boundary are Nsadweso and Akyempim. The villages of Akyempim, Akyempim New Site (formally Akosombo that was resettled by the company) and Kubekro are the closest communities to the Wassa operational site. The Togbekrom community was recently resettled to create space for construction of TSF2. Historic gold workings are also known to occur in the lease area but are on a relatively small scale.

19.2.2 Climate

The climate of the project site is described in Section 4.3. The average annual rainfall is 1,996 ± 293 mm as defined by the Ateiku meteorological station. Using data from the GS Wassa weather station, local rainfall is about 1,750 mm / year.

19.2.3 Topography, Elevation and Vegetation

The project area is characterized by gently rolling hills incised by an extensive drainage network. The area is relatively wet, with many low lying swampy areas. It is located in the wet evergreen ecological zone of Ghana. Extensive subsistence farming occurs throughout the area, with plantain, cassava, pineapple, maize, and cocoyam being the principal crops. Some small scale cultivation of commercial crops is also carried out, with cocoa, teak, coconut and oil palm being the most common.

Most vegetation within the area is highly modified and dominated by cocoa stands, weeds, secondary growth associated with abandoned farmland, and other agricultural crops. Areas of remnant vegetation within the general area are heavily degraded. Although a small number of globally conservation-significant species are present, the vegetation is only considered to be of local ecological significance. The floral community degradation is reflected in heavy under storey clearing, wide spread crop planting, and the abundance of *Trema orientalis, Musanga cecropoides, Chromolaena odorata* and *Panicum maximum.*

A number of fauna were identified in various field surveys; the area supports a population of bird species; however, the low number of large mammal species present reflects indiscriminate hunting and clearing of forest for agricultural purposes.

19.2.4 Hydrogeology

In 1995, Satellite Goldfields Limited commissioned Minerex Environmental Limited (“MEL”) to conduct a detailed and extensive hydrogeological assessment, using over 200 boreholes, of the Wassa concession as part of the specialist baseline studies for the then proposed Wassa Gold Project.

The study found that generally the groundwater gradient dips steeply off the plateau areas following, but not as steep as, the topography. Water levels fluctuate seasonally by only 1 to 2 m and in nine months of monthly sampling (wet and dry season), were slightly higher in November (gradient 0.048) than in February (gradient 0.043), with water levels falling in March, but rising in May through July (MEL 1996 c). Analysis of the Very Low Frequency (“VLF”) geophysics data for conductive zones (MEL 1996 b) indicated that the potential for significant water inflows was generally low, and in some valley areas, confined groundwater was discharging into swamps. The study reports (MEL 1996 a, b and c) present detailed groundwater piezometric contours, as well as a refined stratigraphic and hydrogeological model.

The study found that whilst the groundwater hydrochemistry could not be clearly sub-divided into groups, a correlation did appear to exist between the confined nature of groundwater in the boreholes and the hydrochemistry. Groundwater in the valley areas had a higher calcium bicarbonate signature than the groundwater from more elevated plateau areas, which had a neutral ionic signature and low ionic strength, indicating that the groundwater resident in the aquifer longer has become more saturated with respect to calcium carbonate.

From a hydrogeological perspective, distinct lithological units are apparent with an upper oxidized zone and a lower fresh rock zone (Table 19-3).

Table 19-3: Baseline study identified hydrogeological units (MEL 1996 c)

Unit no.	Hydrogeological unit	Weathered state	EC (µS/cm)
A	Phreatic aquifer	Oxidised	1 to 5
B	Confined aquifer in valleys and unconfined on plateaus	Unoxidised	0.01
C	Quartz veins	Unoxidised	1 to 5

The upper aquifer was found to be generally phreatic and the principal groundwater flow occurs where vein quartz occurs more abundantly.

The lower aquifer is within the unoxidised bedrock. It is unconfined in topographically elevated areas and semi-confined in the valleys where there is a vertical upward head gradient. The recharge for this aquifer is on the topographic ridges local to the area where a downward vertical head gradient exists. The groundwater has higher mineralization in the confined zones with the presence of H_2S and slightly higher iron and manganese concentrations than the groundwater in the saprolite (oxide) (MEL 1996 a, MEL 1996 c).

The confined aquifer may be very static with low throughput and not currently discharging to any zone in large quantities. The hydrochemistry may indicate a very long residence time and no direct discharge point, only small dispersed seepages through the aquitard zones (MEL 1996 c).

19.3 Existing Approach to Environmental and Social Management

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 (submitted for final review to the EPA in June 2014) and the associated Environmental Certificate provide legal framework for GSWL environmental management.

Community management at GSWL is carried out by the Department of Environment and Social Responsibility. GSWL has established a series of Community Mine Consultative Committees (“CMCCs”) within the local stakeholder communities. An Apex CMCC collects the recommendations and then makes them to the corporate and company entities (such as the Golden Star Development Foundation) on behalf of the three functional areas. This aims to ensure that full representation across the GSWL operations occurs without interference from GSWL.

The CMCCs 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.

19.4 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.

19.4.1 Legacy Issues

When GSR (through GSWL) took over the Wexford operation, the heap leach area was already disturbed (now the TSF) and most of the infrastructure was in place. The development of the CIL processing plant and the incorporation of the tailings into the former heap leach area allowed the use of mostly brownfields sites for a lot of the new infrastructure development associated with the then revised project. The establishment of the reclamation and closure provision in consultation with the EPA allowed the asset retirement obligations to be addressed. There are no other legacy issues associated with the GSWL site.

19.4.2 Community Expectations and Sensitivities

Employment

The main socioeconomic concern for most stakeholders is employment. The local community around the Wassa operation see working at the mine as one of the preferred occupations. The extension of the mine life with the development of the underground mine is expected to receive local support. Employment levels for the Wassa underground mine have yet to be developed; however, community expectations will be managed via the normal community consultative methods.

Although GSR is unable to employ all the people seeking work, there is a local hiring policy in place that provides affirmative action for the employment of local stakeholder communities. All vacant positions are advertised locally first and then nationally. Local people are used exclusively for unskilled positions, and as much as possible for all other positions within the operation. The project will draw most of its required workforce from within the Western Region. The Wassa operation has started a local training program along the lines of an apprenticeship where people from the local community are offered the opportunity to train in work areas where the mine may need workers in the future.

Access to land, noise and blasting effects

Other community concerns include access to land, and noise and blasting effects. However, these are minor for the project as the mine is underground and blasting will be on a small scale when compared to the current open pit operations. The land requirement for the project should be limited with very limited surface expressions for infrastructure beyond the areas already developed.

Resettlement

No resettlement is required for the development of the Wassa Project.

19.4.3 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. GSR has reported that galamsey in the area of the project has little potential to affect the operations. In general, the removal of the galamsey from the work areas is not a problem, when asked to leave, they move on to other areas. As the underground mine will start within the current open pit, unauthorized small scale mining is not expected to adversely affect the project.

19.4.4 Water Management

Based on preliminary estimations inflows into the underground mine could be in the range of between 9 and 35 l/s. This preliminary analysis predicts that the cone of depression that will surround the underground mine in response to dewatering may extend between 0.5 km and 1.6 km away from the mine. Further work will be required during the next stage of study to define potential impacts to community water supplies and water management requirements for water entering the mine.

The water management at the GSWL site has been such that discharges to the receiving environment from the tailings storage facility have not been required since 2010. The Wassa operation has an approved detoxification plant to treat elevated CN concentrations that is available for the treatment of water should a discharge be required. However, the water balance model for the current configuration of the site indicates that under normal conditions discharges should not be required. The current detoxification plant will be upgraded with the construction of the new tailings storage facility (TSF2).

Normal mining operations continue with the operational requirement for the installation of sumps and the removal of rainfall and groundwater that enters the mining areas. The management of this water is to pump the water to sedimentation structures and then release the water to the receiving environment. This is carried out in compliance with the permits. To improve the overall management of surface run-off from the mining areas, five sedimentation structures were constructed in 2012. These are primarily to remove suspended solids from the run-off water that may be elevated during storm events.

Geochemical test work undertaken for existing operations indicates that waste rock and tailings at the Wassa mine site do not have acid generating potential but waste rock from the Benso site has acid generation potential. Further geochemical testwork will be required for the Wassa Project to confirm that there is no potentially acid generating material and consequent need for special measures in the design of the mine waste disposal facilities.

19.4.5 Closure Planning and Cost Estimate

As explained in Section 19.1.2, an environmental permit will need to be obtained for the Wassa Project. The EIS submitted to the EPA to obtain the permit will have to contain a provisional closure plan and cost estimate.

The project surface expression is expected to be limited as most of the waste will be stored in the current Wassa waste dumps / within the Wassa open pit or underground. The additional infrastructure required for the project above the existing Wassa mine infrastructure is minimal.

The rehabilitation and closure of the support facilities (e.g. processing plant, tailings storage facility, transportation corridor) are covered under existing GSWL Asset Retirement Obligations (“ARO”). The closure plan for the existing operation is given in the EMP. Both LoM and ARO closure costs have been estimated for the mine and are updated on a regular basis. In 2013, ARO costs for the existing operations were estimated to be US$13.2 million.

For the Wassa Project, a closure cost estimate will be prepared in the next stage of project planning and will be based on the following principles:

- No allowance for scrap value.
- Progressive closure will be integrated with on-going operations.
- Costs will be 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 Wassa main pit.
- Community post-closure issues will not be included.

GSR expects the EPA to request either a reclamation bond for the project or a modification to the existing GSWL reclamation bond through the reclamation security agreement.

20 CAPITAL AND OPERATING COSTS (ITEM 21)

20.1.1 Introduction

Capital costs have been estimated in US$ real terms and are valid as at the effective date of this report.

20.2 Capital Costs

20.2.1 Open Pit and Processing Facilities

Estimates for the Open Pit and Processing Facilities have been produced by GSWL and are summarised in Table 20-1. These cost estimates are based on the expected requirements for the remaining LoM and current budgets in place at the Wassa operations.

Table 20-1: Wassa PEA Open Pit and Plant Project Capital

Capital Costs	LoM (US$M)
TSF2 tailings facility construction	23.46
Wassa mining area	2.10
Other mining areas	0.40
New mobile equipment	7.48
Replacement equipment - process	1.02
Community and environment	0.57
Other	0.23
ARO	13.19
Process Modifications & Upgrades	0.91
Total Open Pit and Plant Project Capital	**49.35**

20.2.2 Underground

SRK has derived a capital cost estimate for the Wassa Underground Project totalling US$109.5M for Project Capital and US$34.1M for Sustaining Capital as reflected in the table below. SRK notes that the capital costs for underground development works have been estimated using unit rates for contractors taken from similar projects with a capitalised mobilization cost of US$2M applied. The line items for capital costs in Table 20-2 have been derived from recent feasibility studies in Ghana and are based on recent quotes. A contingency of 25% has been added to the underground capital costs to reflect the level of accuracy of the planning work that has been undertaken.

Table 20-2: Wassa PEA Underground Capital Costs

Capital Costs	LoM (US$M)
Total Capital Development	**73.27**
Horizontal Development (Capital)	67.75
Surface Raisebore Development (Capital)	5.52
Total Other U/G Capital	**14.32**
Feasibility Study	1.20
Contractor Mobilisation	2.00
Workshop	0.50
Camp	1.00
Compressors	0.30
Ventilation fan - main	1.70
Ventilation fan - secondary	0.60
Refuges	0.33
Portal	0.35
Offices	0.40
Changehouse	0.25
Light vehicles	0.65
Pumps - primary	0.30
Pumps - secondary	0.36
HV backbone	1.50
Miscellaneous	1.00
Closure Cost	1.00
U/G Project Capital Contingency	**21.90**
U/G Project Capital	**109.49**
U/G Sustaining Capital	**34.11**
Total U/G Capital	**143.59**

20.3 Summary

Figure 20-1 provides an annual breakdown for the following capital cost categories over the mine life of the Wassa Project including:

- Open pit and plant capital;
- Underground project capital; and
- Underground sustaining capital.

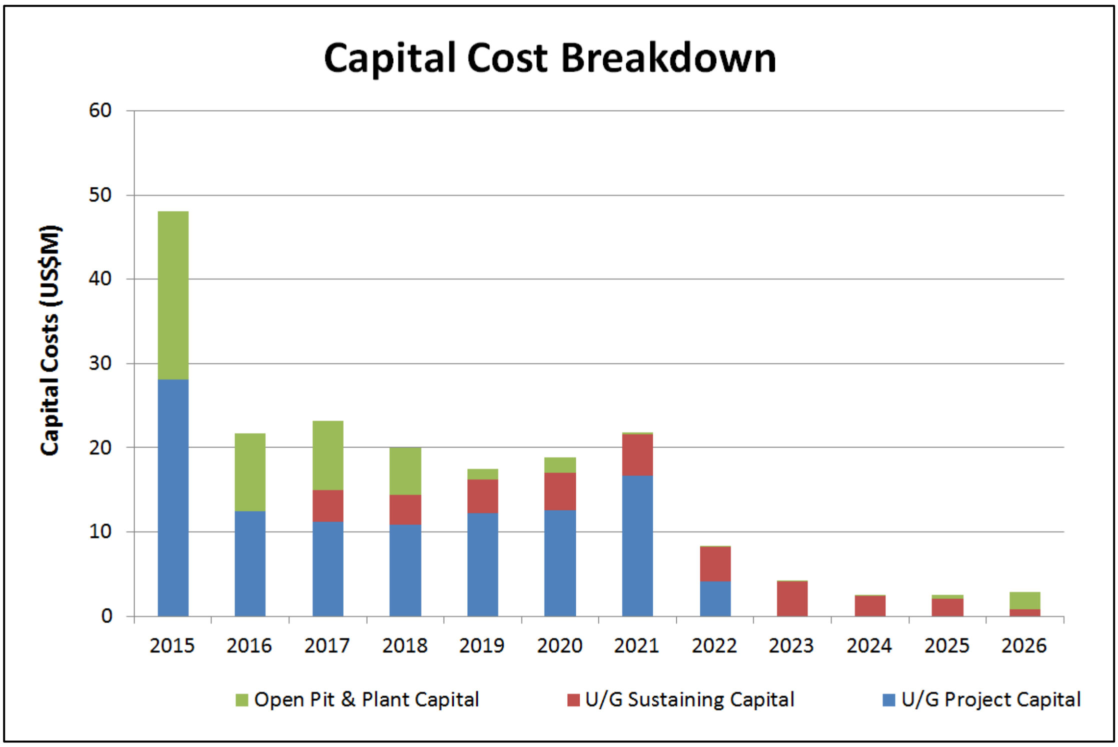


Figure 20-1: Annual Capital Expenditure

20.4 Operating Costs

The open pit operating costs for the Wassa Project have been estimated based on past historical performance and from the 2014 budget currently in place at the operation. All costs are estimated in US$ terms and are valid as at the effective date of this report.

SRK has estimated underground mining costs for the Wassa Project to be US$40/t based on a Contractor mining scenario and covering all direct mining costs excluding the capital cost of the access declines and primary vertical development.

Processing costs and G&A costs are assumed to be US$17.50/t and US$5.54/t, respectively. Refining charges are assumed to US$5/oz Au. A 15% contingency has been applied to the underground operating costs. The unit costs are summarised in the Table 20-3 and Table 20-5 below.

Table 20-3: Summary of Wassa Project Unit Operating Costs - Mining

Operating Cost - Unit Rates		
U/G Stoping Unit Cost	(US$/t U/G RoM)	40
U/G Development Unit Cost	(US$/m)	4,500
U/G Mining Operating Unit Cost RoM	(USD/t U/G RoM)	44.96
O/P Mining Operating Unit Cost TMM	(US$/t O/P TMM)	3.13
O/P Mining Operating Unit Cost RoM	(US$/t O/P RoM)	18.88

Note: TMM = Total Material Moved

Table 20-4: Summary of Wassa Project Unit Operating Costs per Total Tonne Milled

Operating Cost - Unit Rates		
U/G Mining Unit Cost	(US$/t milled)	19.44
O/P Mining Unit Cost	(US$/t milled)	10.26
Processing Unit Cost	(US$/t milled)	17.50
Site G&A Unit rate	(US$/t milled)	5.54
Refining Charge	(US$/t milled)	0.41
Royalty	(US$/t milled)	5.33
Contingency	(US$/t milled)	2.92
Total Operating Cost	**(US$/t milled)**	**61.39**

Table 20-5: Wassa Underground PEA Unit Operating Costs (per Ounce Gold Recovered)

Operating Costs	LoM (US$/oz)
Total Mining U/G	237.07
Total Mining O/P	125.17
Total Processing	213.45
Total G&A	67.57
Total Refining	5.00
Total Royalty	65.00
Contingency	35.56
Total Operating Cost	**748.82**

20.4.1 Summary

Figure 20-2 provides an annual breakdown of the various components of the operating costs over the mine life of the Wassa Project.

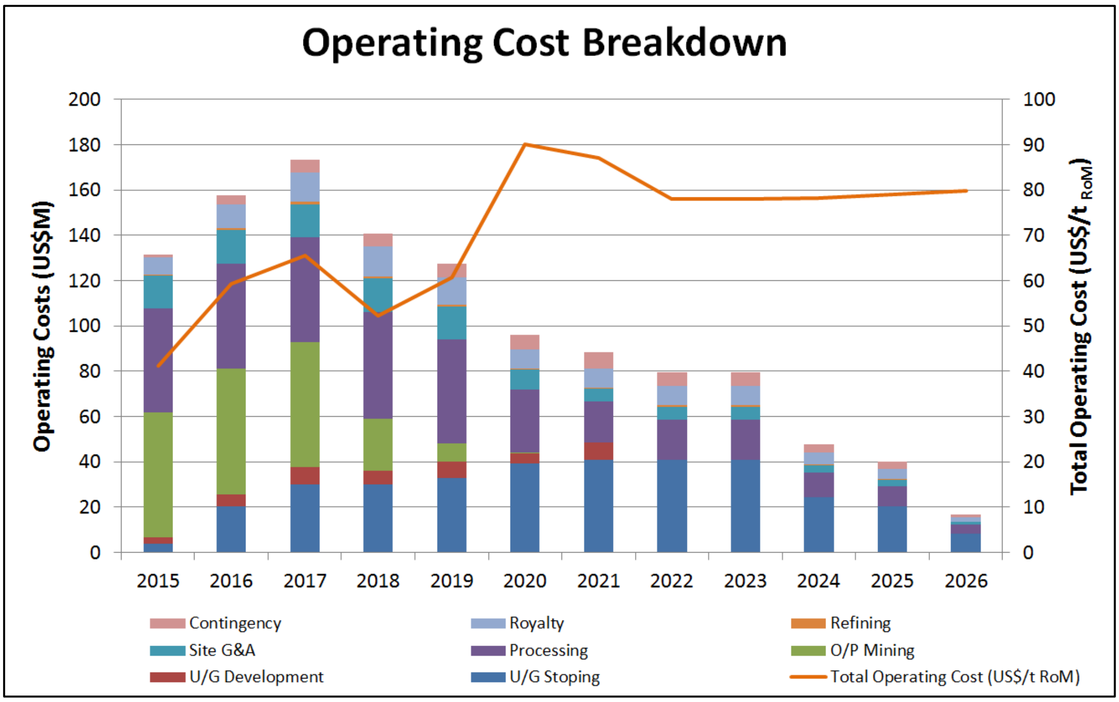


Figure 20-2: Annual Operating Costs

21 ECONOMIC ANALYSIS (ITEM 22)

21.1 Introduction

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

SRK has derived an independent Technical Economic Model (“TEM”) for the PEA using the key assumptions outlined in Section 21.2. The mine plan reflects production commencing in 2015 with combined open pit and underground production till 2020. In 2021, the underground is the only source of RoM feed at a production rate of 1 Mtpa till 2025. Total material mined amounts to 19.2 Mt at an average grade of 2.74 g/t Au producing some 1,574 koz gold. It should be noted that the underground grades are significantly higher than the open pit.

21.2 Key Assumptions

The following general assumptions have been applied to the TEM:

- All cost and revenues are in 2015 terms;
- The start date of the valuation is January 2015;
- A base case discount rate of 5% has been applied for Net Present Value calculations;
- Gold price of US$1,300/oz;
- Royalty of 5% of revenue applied;
- Corporate income tax rate is 35%;
- SRK has relied on the Company’s advice that implementation of a Windfall Profits Tax (“WPT”) has not yet been ratified by the Ghanaian government, therefore WPT has not been provided for in the TEM; and,
- Capital investment is depreciated on an annual fixed percentage basis as per the fiscal regime of Ghana. It has been assumed that all capital items have been fully depreciated and at the end of the mine life there is no terminal value to consider.

21.3 Cash Flow Model

Table 21-1: Wassa PEA TEM - Physicals

Year Base Case	Units	Total/Ave	Year 1 2015	Year 2 2016	Year 3 2017	Year 4 2018	Year 5 2019	Year 6 2020	Year 7 2021	Year 8 2022	Year 9 2023	Year 10 2024	Year 11 2025	Year 12 2026
Mine Production														
Operating Development - Contractor - Metres														
Total On Reef - RoM	(m)	30,664	600	2,335	2,626	3,440	3,102	3,524	3,675	3,505	3,204	2,250	1,648	755
Horizontal Development (Operating) - Waste	(m)	9,157	555	1,194	1,702	1,339	1,638	963	1,766					
Total Development	**(m)**	**39,821**	**1,155**	**3,529**	**4,329**	**4,778**	**4,740**	**4,487**	**5,441**	**3,505**	**3,204**	**2,250**	**1,648**	**755**
Operating Development - Contractor - Tonnage														
Total Development	(kt)	2,070	41	158	177	232	209	238	248	237	216	152	111	51
Operating Development - Contractor - Grade														
On Reef development meters	(g/t Au)	4.66	3.75	5.17	5.68	5.04	4.23	4.02	4.37	4.55	4.45	4.69	4.93	5.32
Capital Development - Contractor - Metres														
Horizontal Development	(m)	15,056	2,001	2,001	1,666	1,923	2,176	2,197	2,362	729				
Surface Raisebored Development	(m)	1,255	151	151	333				619					
Total Capital Development	**(m)**	**16,310**	**2,152**	**2,152**	**1,998**	**1,923**	**2,176**	**2,197**	**2,982**	**729**				
Underground Mining Production Waste Tonnage														
Waste Development	(kt)	1,499	171	196	238	231	250	242	172					
Underground Mining Production RoM Tonnage														
Development	(kt)	2,070	41	158	177	232	209	238	248	237	216	152	111	51
Stoping	(kt)	6,230	61	348	573	521	607	748	770	783	804	458	395	161
Total U/G RoM	(kt)	8,299	101	505	750	753	816	986	1,019	1,020	1,020	610	507	212
Development	(g/t Au)	4.66	3.75	5.17	5.68	5.04	4.23	4.02	4.37	4.55	4.45	4.69	4.93	5.32
Stoping	(g/t Au)	4.44	3.56	4.22	5.66	5.31	4.24	3.74	4.22	4.15	4.29	4.26	4.72	5.07
Total U/G RoM	(g/t Au)	4.49	3.64	4.52	5.66	5.22	4.24	3.81	4.26	4.24	4.32	4.37	4.77	5.13
Open Pit Tonnage														
Waste Tonnes	(kt)	52,515	14,567	15,541	15,718	5,338	1,305	46						
Tonnes O/P RoM	(kt)	10,437	3,080	2,155	1,896	1,937	1,287	82						
Total Tonnes Moved	(kt)	62,951	17,647	17,695	17,613	7,275	2,592	128						
Grade	(g/t Au)	1.44	1.40	1.48	1.25	1.48	1.67	1.48						
Stripping Ratio	(t_{waste}/t_{ore})	5.03	4.73	7.21	8.29	2.76	1.01	0.57						
Processing														
Tonnes Milled	(kt)	19,200	2,616	2,656	2,645	2,690	2,620	1,585	1,020	1,020	1,020	610	507	212
Grade	(g/t)	2.74	1.51	2.08	2.51	2.53	2.38	2.69	4.25	4.24	4.32	4.37	4.77	5.13
Contained Gold	(koz)	1,693	126.9	177.4	213.4	218.9	200.8	136.8	139.5	139.0	141.7	85.6	77.6	35.0
Recovery	(%)	93.0%	93.0%	93.0%	93.0%	93.0%	93.0%	93.0%	93.0%	93.0%	93.0%	93.0%	93.0%	93.0%
Recovered Gold	(koz)	1,574	118.0	165.0	198.4	203.5	186.8	127.2	129.7	129.3	131.8	79.7	72.2	32.5

Table 21-2: Wassa PEA TEM – Preliminary Economics

Year Base Case	Units	Total/Ave	Year 1 2015	Year 2 2016	Year 3 2017	Year 4 2018	Year 5 2019	Year 6 2020	Year 7 2021	Year 8 2022	Year 9 2023	Year 10 2024	Year 11 2025	Year 12 2026
Processing														
Tonnes Milled	(kt)	19,200	2,616	2,656	2,645	2,690	2,620	1,585	1,020	1,020	1,020	610	507	212
Grade	(g/t)	2.74	1.51	2.08	2.51	2.53	2.38	2.69	4.25	4.24	4.32	4.37	4.77	5.13
Contained Gold	(koz)	1,693	126.9	177.4	213.4	218.9	200.8	136.8	139.5	139.0	141.7	85.6	77.6	35.0
Recovery	(%)	93.0%	93.0%	93.0%	93.0%	93.0%	93.0%	93.0%	93.0%	93.0%	93.0%	93.0%	93.0%	93.0%
Recovered Gold	(koz)	1,574	118.0	165.0	198.4	203.5	186.8	127.2	129.7	129.3	131.8	79.7	72.2	32.5
Macro Economics														
Gold Price	(USD/oz)	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300
Revenue														
Gold Revenue	(US$M)	2,046.4	153.4	214.4	258.0	264.6	242.8	165.4	168.7	168.1	171.3	103.5	93.9	42.3
Total Revenue	**(US$M)**	**2,046**	**153.4**	**214.4**	**258.0**	**264.6**	**242.8**	**165.4**	**168.7**	**168.1**	**171.3**	**103.5**	**93.9**	**42.3**
Operating Cost - Unit Rates														
U/G Stoping Unit Cost	(US$/t U/G RoM)	40.00	40.00	40.00	40.00	40.00	40.00	40.00	40.00	40.00	40.00	40.00	40.00	40.00
U/G Development Unit Cost	(USD/m)	4,500	4,500	4,500	4,500	4,500	4,500	4,500	4,500					
U/G Mining Operating Unit Cost RoM	(USD/t U/G RoM)	44.96	64.66	50.64	50.21	48.00	49.03	44.39	47.80	40.00	40.00	40.00	40.00	40.00
O/P Mining Operating Unit Cost TTM	(USD/t O/P TTM)	3.13	3.13	3.13	3.13	3.13	3.13	3.13						
O/P Mining Operating Unit Cost RoM	(USD/t O/P RoM)	18.88	17.93	25.70	29.08	11.76	6.30	4.90						
U/G Mining Unit Cost	(US$/t milled)	19.44	2.50	9.63	14.24	13.43	15.28	27.63	47.74	40.00	40.00	40.00	40.00	40.00
O/P Mining Unit Cost	(US$/t milled)	10.26	21.12	20.85	20.84	8.46	3.10	0.25						
Processing Unit Cost	(US$/t milled)	17.50	17.50	17.50	17.50	17.50	17.50	17.50	17.50	17.50	17.50	17.50	17.50	17.50
Site G&A Unit rate	(US$/t milled)	5.54	5.54	5.54	5.54	5.54	5.54	5.54	5.54	5.54	5.54	5.54	5.54	5.54
Refining Charge	(US$/t milled)	0.41	0.23	0.31	0.38	0.38	0.36	0.40	0.64	0.63	0.65	0.65	0.71	0.77
Royalty	(US$/t milled)	5.33	2.93	4.04	4.88	4.92	4.63	5.22	8.27	8.24	8.40	8.49	9.26	9.97
Contingency	(US$/t milled)	2.92	0.38	1.44	2.14	2.02	2.29	4.14	7.16	6.00	6.00	6.00	6.00	6.00
Total Operating Cost	(US$/t milled)	61.39	50.20	59.31	65.51	52.25	48.69	60.69	86.84	77.91	78.08	78.18	79.02	79.78
Operating Cost Net														
U/G Stoping Operating Cost	(US$M)	332.0	4.1	20.2	30.0	30.1	32.7	39.5	40.7	40.8	40.8	24.4	20.3	8.5
U/G Development Operating Cost	(US$M)	41.2	2.5	5.4	7.7	6.0	7.4	4.3	7.9					
O/P Mining Operating Cost	(US$M)	197.0	55.2	55.4	55.1	22.8	8.1	0.4						
Processing Cost	(US$M)	336.0	45.8	46.5	46.3	47.1	45.9	27.7	17.9	17.9	17.9	10.7	8.9	3.7
Site G&A Rate	(US$M)	106.4	14.5	14.7	14.7	14.9	14.5	8.8	5.7	5.7	5.7	3.4	2.8	1.2
Refining Charge	(US$M)	7.9	0.6	0.8	1.0	1.0	0.9	0.6	0.6	0.6	0.7	0.4	0.4	0.2
Royalty	(US$M)	102.3	7.7	10.7	12.9	13.2	12.1	8.3	8.4	8.4	8.6	5.2	4.7	2.1
Contingency	(US$M)	56.0	1.0	3.8	5.7	5.4	6.0	6.6	7.3	6.1	6.1	3.7	3.0	1.3
Total Operating Cost	**(US$M)**	**1,178.8**	**131.3**	**157.5**	**173.3**	**140.6**	**127.6**	**96.2**	**88.6**	**79.5**	**79.6**	**47.7**	**40.0**	**16.9**
Capital Costs														
Total Capital Development	(US$M)	73.3	9.7	9.7	9.0	8.7	9.8	9.9	13.4	3.3				
Total Other U/G Capital	(US$M)	14.32	12.8	0.3	0.0	0.0	0.0	0.2						
U/G Project Capital Contingency	(US$M)	21.9	5.6	2.5	2.2	2.2	2.4	2.5	3.3	0.8				
U/G Project Capital	**(US$M)**	**109.49**	**28.1**	**12.5**	**11.2**	**10.8**	**12.2**	**12.6**	**16.7**	**4.1**				
U/G Sustaining Capital	**(US$M)**	**34.11**	**0.0**	**0.0**	**3.8**	**3.6**	**4.0**	**4.4**	**4.9**	**4.1**	**4.1**	**2.4**	**2.0**	**0.8**
Total U/G Capital	**(US$M)**	**143.59**	**28.1**	**12.5**	**15.0**	**14.4**	**16.2**	**17.0**	**21.6**	**8.2**	**4.1**	**2.4**	**2.0**	**0.8**
Total Open Pit and Plant Project Capital	(US$M)	49.35	20.0	9.2	8.3	5.5	1.3	1.9	0.3	0.2	0.1	0.1	0.5	2.0
Total Capital	**(US$M)**	**192.94**	**48.1**	**21.7**	**23.2**	**20.0**	**17.5**	**18.9**	**21.9**	**8.4**	**4.2**	**2.6**	**2.5**	**2.8**
Economics, Real: BASE DATE	**01 January 2015**													
Sales Revenue	(US$M)	2,046	153	214	258	265	243	165	169	168	171	104	94	42
Operating Costs	(US$M)	1,179	131	158	173	141	128	96	89	79	80	48	40	17
Operating Profit - EBITDA	(US$M)	868	22	57	85	124	115	69	80	89	92	56	54	25
Tax Liability	(US$M)	207	0	9	18	30	32	15	21	24	26	14	14	3
Capital Expenditure	(US$M)	193	48	22	23	20	18	19	22	8	4	3	3	3
Working Capital	(US$M)	0	2	2	2	0	-2	-4	2	0	0	-2	0	-1
Net Free Cash Flow	**(US$M)**	**468**	**-28**	**24**	**42**	**74**	**67**	**39**	**35**	**56**	**61**	**41**	**37**	**20**

21.4 Economic and Sensitivity Analysis Results

The results of the cash flow and unit cost analysis from the TEM are shown in Table 21-3.

Table 21-3: Summary of Cash Flows and Unit Costs

Description	Units	Value
Revenue	(US$M)	2,046
Operating Costs	(US$M)	(1,076)
Royalty	(US$M)	(102)
Operating Profit	**(US$M)**	**868**
Tax	(US$M)	(207)
Capital Expenditure	(US$M)	(193)
Cash Flow	**(US$M)**	**468**
RoM Feed produced From U/G	(kt)	8,299
RoM Feed produced From O/P	(kt)	10,437
RoM Feed From O/P Stockpile	(kt)	464
Total RoM Feed	(kt)	19,200
Waste Mined U/G	(kt)	1,499
Waste Mined O/P	(kt)	52,515
Contained Au	(koz)	1,693
Recovered Au	(koz)	1,574
Mining cost (U/G & O/P)	(US$/t $_{\text{Total feed}}$)	(29.70)
Processing cost	(US$/t $_{\text{Total feed}}$)	(17.50)
G&A cost	(US$/t $_{\text{Total feed}}$)	(5.54)
Refining cost	(US$/t $_{\text{Total feed}}$)	(0.41)
Contingency on U/G Mining Cost	(US$/t $_{\text{Total feed}}$)	(2.92)
Royalty	(US$/t $_{\text{Total feed}}$)	(5.33)
Total	**(US$/t $_{\text{Total feed}}$)**	**(61.39)**
Revenue	(US$/oz)	1,300.00
Operating Costs	(US$/oz)	(748.82)
Operating Profit	**(US$/oz)**	**551.18**

The following Table 21-4 shows the Net Present Value (“NPV”) post tax over a range of gold prices and discount rates for Wassa. The NPV for the Project is US$350M at a 5% discount rate and a gold price of US$1,300/oz.

Table 21-4: Net Present Value

Summary of NPV's

Discount Rate	0%	5%	8%	10%	12%	15%
NPV (US$M) @ US$1,100/oz Au Price	273	191	156	136	119	97
NPV (US$M) @ US$1,300/oz Au Price	468	350	298	269	244	212
NPV (US$M) @ US$1,500/oz Au Price	670	515	446	407	373	330

The breakeven gold price assuming a 5% discount rate is US$879/oz. This is the gold price when the NPV is zero using a 5 % discount rate.

Table 21-5: Breakeven Gold Price

Discount Rate	Break Even Price (US$/oz)
5%	879.2
10%	909.8

An NPV sensitivity chart for operating costs, capital expenditure and sales prices is presented in Figure 21-1. The Project is most sensitive to gold price/revenue with a 32% reduction resulting in breakeven. Sensitivity to operating costs is low with a greater than 56% increase required for breakeven. The operation is least sensitive to changes in capital expenditure with a substantial increase required to reach breakeven.

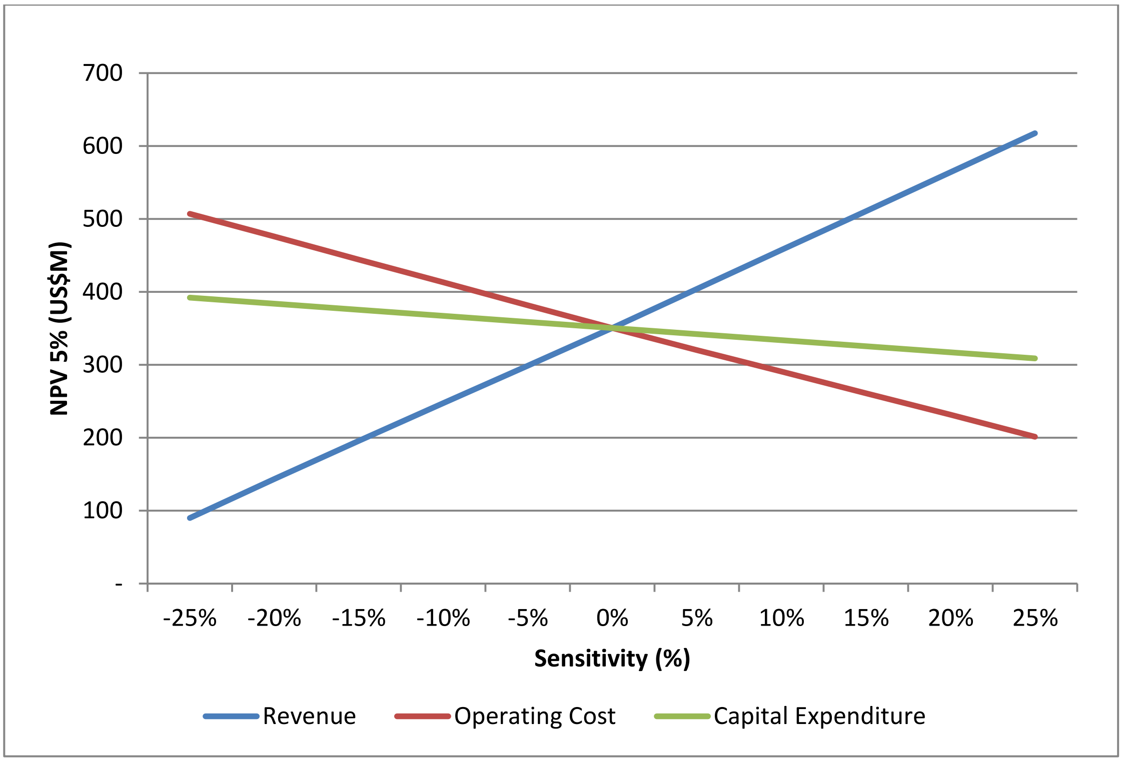


Figure 21-1: NPV Sensitivity Chart

With the on-going open pit operation contributing significantly to initial cash flows as the underground operations are established the project reflects a high IRR of 129%. The contribution from the open pit also results in a rapid payback being realized in year 3 of the Wassa Project.

22 ADJACENT PROPERTIES (ITEM 23)

There is no other relevant data available about adjacent properties.

23 OTHER RELEVANT DATA AND INFORMATION (ITEM 24)

There is no other relevant data available about the Wassa Project.

24 INTERPRETATION AND CONCLUSIONS (ITEM 25)

24.1 Conclusions

Based on the work carried out for this PEA, SRK concludes the results indicate that there is potential for a combined open pit and underground operation at Wassa. SRK notes that further detailed technical work and investigation is required to confirm the optimal approach and economic viability through improving the accuracy of mine planning and cost estimates.

In SRK's opinion an underground mine at Wassa is practically achievable and the PEA indicates that a sub-level stoping method will be suitable together with a consolidated fill to maximise the extraction ratio. A single decline would appear to be the most suitable means for accessing the identified production areas using a trackless haulage for materials handling to surface. The study also indicates that an initial production rate of 0.75 Mtpa is achievable from the underground which can be increased to around 1 Mtpa over a number of years.

GSWL has a long history of mining at Wassa and has the required understanding of how to implement the Wassa Project into their current operations.

24.2 Risks & Uncertainties

GSR has taken a focussed and technically disciplined approach to exploration for additional Mineral Resources and gathering the necessary data to reduce the risks and uncertainties with mine planning. Taking GSR's diligence into account and the work completed for the purposes of this PEA the risks appear to be fairly well understood and low for future development of the Wassa Project.

It should be noted that additional exploration drilling, metallurgical testwork and technical work is required to understand the full economic potential of the Wassa Project however this PEA will assist in focussing future efforts and investment.

The Project is most sensitive to sales price/revenue with a 32% reduction resulting in breakeven. Sensitivity to operating costs is low with a greater than 56% increase required for breakeven. The operation is least sensitive to changes in capital expenditure with a substantial increase required to reach breakeven.

25 RECOMMENDATIONS (ITEM 26)

25.1 Key Recommendations

Based on the work carried out for this PEA, SRK recommends that consideration is given to advancing the Wassa Project to a feasibility level of study, using this PEA as a basis for development of the mining approach and technical detail. Further detailed investigation is required to confirm the following aspects in sufficient detail before committing to construction:

- Geology and Mineral Resources
  - Focus on proving up sufficient Mineral resources to extend the life of the open pit feed and where possible oxide material from Wassa Main and the satellite deposits.
- Water Management
  - Advance the hydrogeological analysis for the Wassa Project and complete a suitably detailed water balance covering all aspects related to mining, processing and tailings.
- Open Pit Mining
  - Evaluate opportunities to extend the life of the Wassa open pit and other satellite deposits.
- Underground Mining
  - A feasibility level backfill study to assess opportunities to increase the extraction ratio and reduce external dilution from backfill.
  - Confirm the mining method approach based on the results of the backfill study and through more detailed geotechnical analysis determine the optimal stope dimensions.
  - Revision of the ventilation aspects to determine optimal design at different stages in the life of the project.
- Mineral Processing and Testwork
  - Additional metallurgical testwork on representative samples at depth for the proposed underground mine. This testwork should principally cover the expected increase in rock hardness with depth, and any potential variation in the recovery response with grind size. The other key parameter to be covered will be the settling and tailings characteristics properties of the future ores.
- Environmental and Social Aspects
  - Focus on keeping existing permits renewed and current with the Ghanaian regulatory authorities.
  - Ensure a clear understanding of the requirements to progress from the current open pit only operation to a combined open pit and underground mine.
- Economic
  - Identify long lead time items to avoid delays in the key development activities identified to progress the Wassa Project. This also extends to communicating with underground development contract groups to improve the accuracy of cost estimates through quotes.

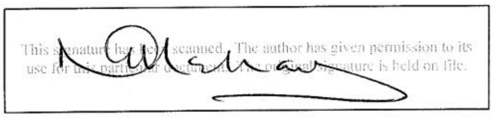
25.2 Cost of Recommendations

Table 25-1 provides a breakdown estimate of the recommended work items required to complete the Feasibility Study and undertake the initial exploration decline development to enable a bulk sample to be collected for metallurgical testwork which totals US$9.85M. This cost of the Exploration Decline and bulk sample (US$9.0M) is included in the TEM however the other items totalling (US$0.85M) are considered to be additional costs to the Company.

Table 25-1: Breakdown of Cost of Recommendations

Description	US$
Ramp permitting report	150,000
Contractor tender documents	150,000
Metallurgical testwork	50,000
Feasibility Study	500,000
Exploration Decline and bulk sample	9,000,000
Total	**9,850,000**

For and on behalf of SRK Consulting (UK) Limited



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26 REFERENCES (ITEM 27)

Allibone A., McCuaig T.C., Harris D., Etheridge M., Munroe S., Byrne D.; 2002a; Structural Controls on Gold Mineralization at the Ashanti Gold Deposit, Obuasi, Ghana; Society of Economic Geologists, Special Publication, Vol. 9, pp. 65–93.

Allibone A., Teasdale J., Cameron G., Etheridge M., Uttley P., Soboh A., Appiah-Kubi J., Adanu A., Arthur R., Mamphey J., Odoom B., Zuta J., Tsikata A., Pataye F., Famiyeh S., Lamb E.; 2002b; Timing and Structural Controls on Gold Mineralization at the Bogoso Gold Mine, Ghana, West Africa; Economic Geology, Vol. 97, pp. 949-969.

Apau D.; 2012; Structural Evolution of the Wassa Gold Deposit, South Central Ghana; Implication for Mineralization; University of Ghana, 112p (Unpublished).

Bonhomme M.;1962; Contribution à l'étude géochronologique de la plate-forme de l'Ouest Africain; Ann. Fac. Sci. Univ. Clermont-Ferrand, Géol. Minéral., No. 5, 62 p.

Bourassa Y.; 2003; Geology of the Wassa Mine Report; Golden Star Resources Ltd, 32p (Unpublished).

Bowell R.J., Foster R.P., Stanley C.J.; 1990; Telluride mineralization at Ashanti gold mine, Ghana; Mineralogical Magazine, Vol. 54, pp. 617-627.

Crow A.T.; 1952; The rocks of the Sekondi Series of the Gold Coast; Gold Coast Geological Survey, Bulletin No. 18, 68 p.

Dampare S.B., Shibata T., Asiedu D.K., Osae S., Banoeng-Yakubo B.; 2008; Geochemistry of Paleoproterozoic metavolcanic rocks from the southern Ashanti volcanic belt, Ghana: Petrogenetic and tectonic setting implications; Precambrian Research, Vol. 162, pp. 403-423.

Davis D.W., Hirdes W., Schaltegger U., Nunoo E.A.; 1994; U-Pb age constraints on deposition and provenance of Birimian and gold-bearing Tarkwaian sediments in Ghana, West Africa; Precambrian Research, Vol. 67, pp. 89-107.

Agyei Duodu J., Loh G.K., Boamah K.O., Baba M., Hirdes W., Toloczyki M., Davis D.W.; 2009; Geological Map of Ghana 1 : 1,000,000; Geological Survey of Ghana, Map 1:1M.

Eisenlohr B.N.; 1992b; Conflicting evidence on the timing of mesothermal and paleoplacer gold mineralisation in early Proterozoic rocks from southwest Ghana, West Africa; Mineralium Deposita, Vol. 27, pp. 23-29.

Eisenlohr B.N., Hirdes W.; 1992a; The structural development of the early Proterozoic Birimian and Tarkwaian rocks of southwest Ghana, West Africa; Journal of African Earth Sciences, Vol. 14, No. 3, pp. 313-325.

Feybesse JL., Billa M., Guerrot C., Duguey E., Lescuyer JL., Milési JP., Bouchot V.; 2006; The paleoproterozoic Ghanaian province: Geodynamic model and ore controls, including regional stress modeling; Precambrian Research, Vol. 149, pp. 149-196.

Goodfellow W.D,; 2007; Mineral Deposits of Canada, A Synthesis of Major Deposit Types, District Metallogeny, the Evolution of Geological Provinces and Exploration Methods; Geological Association of Canada, Mineral Deposits Division, Special Publication No.5.

Hirdes W., Davis D.W.; 1998; First U-Pb zircon age of extrusive volcanism in the Birimian Supergroup of Ghana/West Africa; Journal of African Earth Sciences, Vol. 27, No. 2, pp. 291-294.

Hirdes W., Davis D.W., Eisenlohr B.N.; 1992; Reassessment of Proterozoic granitoid ages in Ghana on the basis of U/Pb zircon and monazite dating; Precambrian Research, Vol. 56, pp. 89-96.

HUTCHINSON D.J and DIEDRICHS M.S. 1996. Cablebolting in Underground Mines, Vancouver BiTech Publishers, pp406

Isaaks E.; 2013; Grade Estimation for the Wassa Resource Model; Independent Mineral Consultant, 21p (Unpublished).

John T., Klemb R., Hirdes W., Loh G.; 1999; The metamorphic evolution of the Paleoproterozoic (Birimian) volcanic Ashanti belt (Ghana, West Africa); Precambrian Research, Vol. 98, pp. 11-30.

Junner N.R.; 1935; Gold in the Gold Coast; Gold Coast Geological Survey, Memoir No. 4, 76 p.

Junner N.R.; 1940; Geology of the Gold Coast and Western Togoland; Gold Coast Geological Survey, Memoir No. 11, 40 p.

Kesse G.O.; 1985; The Mineral and Rock Resources of Ghana; A.A. Balkema Publishers, Book, 610 p.

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

Ledru P., Johan V., Milési J.P., Tegyey M.; 1994; Markers of the last stages of the Palaeoproterozoic collision: evidence for a 2 Ga continent involving circum-South Atlantic provinces; Precambrian Research, Vol. 69, pp. 169-191.

Leube A., Hirdes W., Mauer R., Kesse G.O.; 1990; The Early Proterozoic Birimian Supergroup of Ghana and Some Aspects of its Associated Gold Mineralization; Precambrian Research, Vol. 46, pp. 139-165.

Loh G., Hirdes W., Anani C., Davis D.W., Vetter U.; 1999; Explanatory Notes for the Geological Map of Southwest Ghana 1 : 100,000; Geologisches Jahrbuch, Reihe B, Heft 93, 150 p.

MEL (1996 a). Satellite Goldfields Limited, Wassa Hydrogeological Assessment, Progress Report for Work During First Quarter 1996. MEL Report 1031R087.

MEL (1996 b). Reinterpretation of VLF Data to Locate Zones of Preferential Groundwater Flow. MEL Report 1031R138.

MEL (1996 c). Satellite Goldfields Limited, Wassa Hydrogeological Assessment, Progress Report for Work During Third Quarter 1996. Minerex Environmental Limited, December 1996. MEL Report 1031R156.

Milesi J.P., Ledru P., Ankrah P., Johan V., Marcoux E., Vinchon Ch.; 1991; The metallogenic relationship between Birimian and Tarkwaian gold deposits in Ghana; Mineralium Deposita, Vol. 26, pp. 228-238.

Milési JP., Ledru P., Feybesse JL., Dommanget A., Marcoux E.; 1992; Early Proterozoic ore deposits and tectonics of the Birimian orogenic belt, West Africa; Precambrian Research, Vol. 58, pp. 305-344.

Mumin A.H., Fleet M.E.; 1995; Evolution of gold mineralization in the Ashanti Gold Belt, Ghana: evidence from carbonate compositions and parageneses; Mineralogy and Petrology, Vol. 55, pp. 265-280.

Mumin A.H., Fleet M.E., Chryssoulis S.L.; 1994; "Gold mineralization in As-rich mesothermal gold ores of the Bogosu-Prestea mining distric of the Ashanti Gold Belt, Ghana: remobilization of ""invisible"" gold"; Mineralium Deposita, Vol. 29, pp. 445-460.

Nicholas D.E. (1981). Method Selection – A Numerical Approach. Design and Operation of Caving and Sub-Level Stoping Mines. New York. AIME Chapter 4.

Oberthür T., Vetter U., Davis D.W., Amanor J.A.; 1998; Age constraints on gold mineralization and Paleoproterozoic crustal evolution in the Ashanti belt of southern Ghana; Precambrian Research, Vol. 89, pp. 129-143.

Oberthür T., Vetter U., Schmidt Mumm A., Weiser T., Amanor J.A., Gyapong W.A., Kumi R., Blenkinsop T.G.; 1994; The Ashanti Gold Mine at Obuasi, Ghana: Mineralogical, Geochemical, Stable Isotope and Fluid Inclusion Studies on the Metallogenesis of the Deposit; Geologisches Jahrbuch, D 100, pp. 31-129.

Oberthür T., Weiser T., Amanor J.A., Chryssoulis S.L.; 1997; Mineralogical siting and distribution of gold in quartz veins and sulfide ores of the Ashanti mine and other deposits in the Ashanti belt of Ghana: genetic implications; Mineralium Deposita, Vol. 32, pp. 2-15.

Perrouty S., Aillères L., Jessell M.W., Baratoux L., Bourassa Y., Crawford B.,; 2012; Revised Eburnean geodynamic evolution of the gold-rich southern Ashanti Belt, Ghana, with new field and geophysical evidence of pre-Tarkwaian deformations, Vol. 204-205, pp. 12-39.

Perrouty S., Jessell M.W., Aillères L., Apau D., Velasquez G., Siebenaller L., Miller J., Bourassa Y., Beziat d., Baratoux L., 2012; Tectonic Context of Eoeburnean Gold Mineralisation in Wassa mine, Southwest Ghana, unpublished.

Pigois JP., Groves D.I., Fletcher I.R., McNaughton N.J., Snee L.W.; 2003; Age constraints on Tarkwaian palaeoplacer and lode-gold formation in the Tarkwa-Damang district, SW Ghana; Mineralium Deposita, Vol. 38, pp. 695-714.

SRK Consulting (UK) Limited 2013. NI 43-101 Technical Report on Mineral Resources and Mineral Reserves Golden Star Resources Ltd, Wassa Gold Mine, Ghana Effective Date 31st December 2012.

Tunks A.J., Selley D., Rogers J.R., Brabham G.; 2004; Vein mineralization at the Damang Gold Mine, Ghana: controls on mineralization; Journal of Structural Geology, Vol. 26, pp. 1257-1273.

Whitelaw O.A.L.; 1929; The Geological and Mining Features of the Tarkwa-Abosso Goldfield; Gold Coast Geological Survey, Memoir No. 1, 46 p.

27 CERTIFICATES AND CONSENTS

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SCHEDULE "A"

CONSENT OF QUALIFIED PERSON

British Columbia Securities Commission
Alberta Securities Commission
Saskatchewan Financial Services Commission
The Manitoba Securities Commission
Ontario Securities Commission
New Brunswick Securities Commission
Nova Scotia Securities Commission
Deputy Registrar, Securities Division, Prince Edward Island
Director of Securities, Department of Government Services and Lands,
Newfoundland and Labrador
Toronto Stock Exchange

Re: Technical Report titled "NI 43-101 Technical Report on a Preliminary Economic Assessment of the Wassa Open Pit Mine and Underground Project in Ghana" with an effective date of October 30, 2014

I, Michael J Beare, BSc, CEng, MIOM3 consent to the public filing of the technical report titled "NI 43-101 Technical Report on a Preliminary Economic Assessment of the Wassa Open Pit Mine and Underground Project in Ghana" with an effective date of 30, October 2014 (the "**Technical Report**") by Golden Star Resources Ltd. ("**Golden Star**").

I also consent to any extracts from or a summary of the Technical Report in the press release titled "Golden Star's PEA of Wassa Mine Indicates NPV of $350 M Cash flows and Ecobank facilities fund this project" dated September 15, 2014 (the "**Press Release**") of Golden Star.

I certify that I have read the Press Release filed by Golden Star and that it fairly and accurately represents the information in the Technical Report for which I am responsible.

Dated this 30th day of October, 2014

Mr Michael Beare

Corporate Consultant (Mining Engineering),

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SCHEDULE "A"

CONSENT OF QUALIFIED PERSON

British Columbia Securities Commission
Alberta Securities Commission
Saskatchewan Financial Services Commission
The Manitoba Securities Commission
Ontario Securities Commission
New Brunswick Securities Commission
Nova Scotia Securities Commission
Deputy Registrar, Securities Division, Prince Edward Island
Director of Securities, Department of Government Services and Lands, Newfoundland and Labrador
Toronto Stock Exchange

Re: Technical Report titled "NI 43-101 Technical Report on a Preliminary Economic Assessment of the Wassa Open Pit Mine and Underground Project in Ghana" with an effective date of October 30, 2014

I, Neil Marshall, BSc (Hons), MSc, CEng, MIOM3 consent to the public filing of the technical report titled "NI 43-101 Technical Report on a Preliminary Economic Assessment of the Wassa Open Pit Mine and Underground Project in Ghana" with an effective date of 30, October 2014 (the "**Technical Report**") by Golden Star Resources Ltd. ("**Golden Star**").

I also consent to any extracts from or a summary of the Technical Report in the press release titled "Golden Star's PEA of Wassa Mine Indicates NPV of $350 M Cash flows and Ecobank facilities fund this project" dated September 15, 2014 (the "**Press Release**") of Golden Star.

I certify that I have read the Press Release filed by Golden Star and that it fairly and accurately represents the information in the Technical Report for which I am responsible.

Dated this 30th day of October, 2014

This signature has been scanned. The author has given permission to its use for this particular document. The original signature is held on file.

Neil Marshall, BSc (Hons), MSc, CEng, MIOM3
Corporate Consultant (Geotechnical Engineering)

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CONSENT OF QUALIFIED PERSON

British Columbia Securities Commission
Alberta Securities Commission
Saskatchewan Financial Services Commission
The Manitoba Securities Commission
Ontario Securities Commission
New Brunswick Securities Commission
Nova Scotia Securities Commission
Deputy Registrar, Securities Division, Prince Edward Island
Director of Securities, Department of Government Services and Lands,
Newfoundland and Labrador
Toronto Stock Exchange

Re: Technical Report titled "NI 43-101 Technical Report on a Preliminary Economic Assessment of the Wassa Open Pit Mine and Underground Project in Ghana" with an effective date of October 30, 2014

I, Dr John Willis, Ph.D., BE (Met) consent to the public filing of the technical report titled "NI 43-101 Technical Report on a Preliminary Economic Assessment of the Wassa Open Pit Mine and Underground Project in Ghana" with an effective date of 30, October 2014 (the "**Technical Report**") by Golden Star Resources Ltd. ("**Golden Star**").

I also consent to any extracts from or a summary of the Technical Report in the press release titled "Golden Star's PEA of Wassa Mine Indicates NPV of $350 M Cash flows and Ecobank facilities fund this project" dated September 15, 2014 (the "**Press Release**") of Golden Star.

I certify that I have read the Press Release filed by Golden Star and that it fairly and accurately represents the information in the Technical Report for which I am responsible.

Dated this 30th day of October, 2014

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Dr John Willis, Ph.D., BE (Met)

Principal Consultant (Mineral Processing),

GOLDEN STAR

Golden Star Exploration Limited

SCHEDULE "A"
CONSENT OF QUALIFIED PERSON

British Columbia Securities Commission
Alberta Securities Commission
Saskatchewan Financial Services Commission
The Manitoba Securities Commission
Ontario Securities Commission
New Brunswick Securities Commission
Nova Scotia Securities Commission
Deputy Registrar, Securities Division, Prince Edward Island
Director of Securities, Department of Government Services and Lands,
 Newfoundland and Labrador
Toronto Stock Exchange

Re: Technical Report titled "NI 43-101 Technical Report on a Preliminary Economic Assessment of the Wassa Open Pit Mine and Underground Project in Ghana" with an effective date of October 17, 2014

I, S, Mitchel Wasel consent to the public filing of the technical report titled "NI 43-101 Technical Report on a Preliminary Economic Assessment of the Wassa Open Pit Mine and Underground Project in Ghana" with an effective date of 17, October 2014 (the "**Technical Report**") by Golden Star Resources Ltd. ("**Golden Star**").

I also consent to any extracts from or a summary of the Technical Report in the press release titled "Golden Star's PEA of Wassa Mine Indicates NPV of $350 M Cash flows and Ecobank facilities fund this project" dated September 15, 2014 (the "**Press Release**") of Golden Star.

I certify that I have read the Press Release filed by Golden Star and that it fairly and accurately represents the information in the Technical Report for which I am responsible.

Dated this 17th day of October, 2014

S. Mitchel Wasel

Vice President Exploration

Golden Star Resources Ltd

CHARTERED PROFESSIONAL
AusIMM
THE MINERALS INSTITUTE
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GOLDEN STAR

Golden Star Exploration Limited

To accompany the report entitled: NI 43-101 TECHNICAL REPORT ON A PRELIMINARY ECONOMIC ASSESSMENT OF THE WASSA OPEN PIT MINE AND UNDERGROUND PROJECT IN GHANA, effective date 17 October 2014. I, S. Mitchel Wasel, BSc Geology, AusIMM(CP) residing at 23 Hood Street, Takoradi, Western Region Ghana, do hereby certify that:

1) I am Vice President Exploration for Golden Star Resources Ltd with an office at 150 King Street West, Sun Life Financial Tower Suite 1200, Toronto, Ontario, Canada.

2) I am a graduate of the University of Alberta, with a Bachelor of Science degree (BSc) in Geology graduating in 1988. I have practiced my profession continuously since 1988. I have been employed by Golden Star Resources since May 1993 during which time I have been directly involved in Annual Resource estimations for over 10 of the companies gold deposits.

3) I am Member of the Australasian Institute of Mining & Metallurgy and a Chartered Professional AusIMM(CP). My membership number is 209098 and I have been an active member since 2000.

4) I frequently (Last visit July 2014) visit the Wassa Mine site as part of my duties with Golden Star Resources and was responsible for the preparation and compilation of the Geology and Resource sections of the technical report titled, NI 43-101 TECHNICAL REPORT ON A PRELIMINARY ECONOMIC ASSESSMENT OF THE WASSA OPEN PIT MINE AND UNDERGROUND PROJECT IN GHANA, and dated 17 October, 2014.

5) I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of National Instrument 43-101;

6) I am responsible for Sections 6 to 13, of the report;

7) I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith;

8) The WASSA PEA Geology and Resource sections were completed using CIM "Best practices" and Canadian Securities Administrators National Instrument 43-101 guidelines;

9) That, as of the date of this technical report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading; and

10) I consent to the filing of the technical report with any stock exchange and other regulatory authority and any publication for regulatory purposes, including electronic publication in the public company files on their websites accessible to the public of extracts from the technical report.

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Golden Star Exploration Limited

S. Mitchel Wasel

Golden Star Resources Ltd

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Toronto, Ontario, Canada, October 30, 2014

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