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CORA GOLD

COMMERCIAL DEVELOPMENT OF THE SANONKORO GOLD PROJECT, REPUBLIC OF MALI

SCOPING STUDY

January 2020

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January 2020

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EXECUTIVE SUMMARY

Wardell Armstrong International (WAI) has been commissioned to carry out the completion of a Scoping Study for Cora Gold Limited ("Cora Gold" or "the Client") which includes sections completed and signed off by other consultants. The table below lists the consultants responsible for the respective sections of the scoping study. WAI has neither reviewed or signed off the sections produced by other consultants and cannot be held responsible or liable for contents therein.

Lead Author Responsibilities			
Scope of Work Element	**WAI**	**SRK Consulting**	**Digby Wells**
Mineral Resource Estimate		•	
Hydrology and Hydrogeology		•	
Geotechnical Analysis		•	
Mining Engineering Studies		•	
Dilution & Loss Estimates		•	
Open Pit Optimisation		•	
Initial Open Pit Design (Crests, Toes, Ramps)		•	
Initial Pushback Sequencing		•	
In-Pit Haulage Profiles		•	
Initial Waste Dump/Stockpile Designs		•	
Initial Mine Schedule & Stockpile Strategy		•	
Mining Capital & Operating Cost Estimates		•	
Metallurgical Testing	•		
Mineral Processing	•		
Tailings Storage Facility	•		
Infrastructure	•		
Environmental Assessment			•
Social, H&S Assessment			•
Economic Analysis	•		

Mineral Resource Estimate

Introduction

SRK Consulting (UK) Limited ("SRK") is an associate company of the international group holding company, SRK Consulting (Global) Limited (the "SRK Group"). SRK has been requested by Cora Gold Limited ("Cora Gold", hereinafter also referred to as the "Company" or the "Client") to prepare a Mineral Resource Estimate ("MRE") for the Sanankoro Gold Project ("Sanankoro", or the "Project") located in Mali, West Africa.

The MRE has been produced in accordance with the terms and guidelines of the Australasian Code for the Reporting of Exploration Results, Mineral Resources and Ore Reserves, the JORC Code, 2012 Edition, ("JORC" or the "JORC Code").

Project Description

The Sanankoro property lies approximately 110km south west of Bamako in southwest Mali. The property consists of five contiguous exploration permits (Sanankoro, Bokoro II, Bokoro Est, Dako and Kodiou) that encompass a total area of approximately 342km^2.

The Sanankoro property is associated with extensive artisanal gold mining activity. Shallow (typically < 15m deep) workings extend discontinuously over a distance of just over 10km, with individual workings up to 3km in length and 500 m in width.

Project Geology and Mineralisation

The Sanankoro property is underlain by a Paleoproterozoic Birimian volcano-sedimentary formation that trends NNE-SSW, controlled by regional scale shear zones. The formations comprise intercalated units of weakly metamorphosed feldspathic sandstones, siltstones and phyllites, often with a carbonaceous component.

Gold mineralisation occurs along a large surficial elevated gold anomaly of approximately 4.5 x 7.5km, an area characterised by widespread artisanal mining activity. At least three different sets of mineralised quartz veins occur. These include a prominent N-S/NNE-SSW striking set that appear to dip steeply to the east and is the principal focus of artisanal exploitation; a less prominent oblique E-W (80-100o) striking sub-vertical set; and a subordinate less continuous sub-horizontal set. As presently defined by drilling, gold mineralisation within the Project area is contained within a large mineralised corridor composed of 3 subparallel, broadly N-S striking structures known as Bokoro, Sanankoro and Selin. The first two zones can be traced from the north to the south of the Sanankoro permit, over a distance of some 15km, whereas the Selin zone can be traced from the north for a distance of about 10km before it merges with the Sanankoro zone.

The Mineral Resource presented herein in focussed on four zones of Mineralisation, namely “Zone A”, “Zone B”, “Zone B North” and “Selin”. Zone A, Zone B and Zone B North all occur along-strike on the Sanankoro Structure, whilst the Selin zone forms part of the Selin structure, to the north of Zone A, Zone B and Zone B North, and east of the Sanankoro structure.

Deep tropical weathering in the region has liberated and in parts re-mobilised the primary gold. The weathering profile consists of a thin hardcap layer that extends to depths of up to 20 (with an average depth of 5m), a deep saprolite that varies in depth from 10 – 120m (average depth of roughly 50m) and thin transitional saprock layer at the base of the saprolite.

Exploration, Drilling and Sampling

The Sanankoro Project has been subject to 3 main phases of exploration, namely by Randgold Resources Ltd (“Randgold”) in the mid-2000’s, Gold Fields Ltd (“Gold Fields”) between 2008 and 2012, and Cora Gold from 2017 to 2019. An overview of the salient exploration activities undertaken by each is provided below:

Randgold:

- Regional termite mound sampling;
- Regional and infill soil sampling; and
- A series of shallow (10 – 15m depth) vertical rotary air blast (“RAB”) drillholes on a 400m line spacing.

Gold Fields:

- Infill soil sampling on 100 x 200 or 50-100 x 400m grids;
- Ground geophysical surveying including induced polarisation (“IP”) and resistivity surveys;
- Systematic infill drilling using mainly reverse circulation (“RC”) holes on fences typically 100 m apart over much of the current Project area;
- A series of shallow (12 – 15m depth), vertical exploration air core (“AC”) or RAB holes drilled on variable grid spacings over large areas of the exploration permits; and
- Follow-up RC drilling, mainly completed on NW-SE orientated or E-W oriented lines on fences between 100-200m apart in “Zone A” and “Zone B, including deeper holes (to 180 m length) which comprised RC holes with diamond core tails.

Cora Gold:

- Termite mound sampling to supplement earlier soil geochemistry programmes completed by Randgold and Gold Fields on grid parameters that range from 400m x 100m to 200m x 100m.
- Follow-up ground IP surveys, to extend original Gold Fields ground IP coverage to the north by a further c 12.5km^2.
- A series of field bulk density programmes.
- A total of 264 drillholes across the Project area, for a total meterage of approximately 23,100m, including a combination of RC, AC, RAB and diamond (“DC”) drillholes, with diamond core tails on a small number of RC and AC holes, on 60 – 120m spaced sections, with between 1 and 5 holes per section.

Combined, drilling throughout the Sanankoro Project area completed by Randgold, Gold Fields and Cora Gold, totals approximately 78,500m of reverse circulation (“RC”), air core (“AC”), rotary air blast (“RAB”) and diamond core (“DC”) drilling, which includes approximately 2,100m of diamond core. The total length (mineralisation and waste) of the drillholes that have targeted and intersected mineralisation in the area of interest (namely Zone A, Zone B, Zone B North and Selin) is approximately 18,200m, including approximately 14,500m of AC and RC drilling, 1,800m of RAB drilling and 1,800m of diamond core. Drilling to date by all explorers has primarily targeted oxide mineralisation, although several deeper holes have intersected the sulphide mineralisation below the weathered rock.

SRK have not completed any independent checks on the logging, sampling or drill protocols put in place by Cora Gold. That said, based on information and assurances provided by Dr Jonathan Forster and Cora Gold on the drilling, sampling and sample analysis protocols employed during the Cora Gold drill campaigns, SRK considers that these are acceptable for the reporting of a Mineral Resource Estimate in line with the JORC Code (2012).

Mineral Resource Statement

In order to determine the quantities of material offering “reasonable prospects for economic extraction” by open pit mining, a pit optimisation analysis was completed on the estimated block model, based on reasonable mining assumptions. The Mineral Resource has been restricted to estimated blocks that fall inside of the resulting pit shell, which is based on a gold price of 1,700USD/oz, and reported above a cut-off grade of 0.4g/t Au for oxide material and 0.5g/t Au for sulphide material.

The Mineral Resource Statement presented herein has been classified by Mr. Martin Pittuck, who is a Corporate Consultant (Mining Geology) of SRK UK, a Member of the Institute of Materials, Minerals and Mining (MIMM), a Fellow of the Geological Society of London (FGS) and a Chartered Engineer, UK (CEng). Mr Pittuck is responsible for the preparation of the Mineral Resource Estimate and takes overall responsibility for the resource estimation work and resulting Mineral Resource Statement.

SRK UK have not completed a Competent Persons site visit to the Sanankoro Project. Dr. Jonathan Forster, CEO and Head of Exploration for Cora Gold Ltd, acts as the Competent Person responsible for the geology, drilling, sampling and exploration protocols employed on site.

Both Mr Pittuck and Dr. Forster have sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration and to the activity which they are undertaking to qualify as a Competent Person as defined in the 2012 Edition of the ‘Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves’. Both Mr Pittuck and Dr Forster consent to the inclusion in this announcement of the matters based on their information in the form and context in which it appears.

Mineral Resources that are not Mineral Reserves have no demonstrated economic viability. SRK are not aware of any factors (environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors) that have materially affected the Mineral Resource Estimate. It is uncertain is further exploration will convert Inferred Mineral Resources to higher confidence categories.

Mineral Resource Statement for the Sanankoro Project, as of 5 December 2019.				
Weathering State	**Resource Classification**	**Tonnes (Mt)**	**Au g/t**	**Contained Au (Oz)**
OXIDE	MEASURED	-	-	-
	INDICATED	-	-	--
	INFERRED	4.5	1.6	233,000
	TOTAL	**4.5**	**1.6**	**233,000**
SULPHIDE	MEASURED	-	-	-
	INDICATED	-	-	--
	INFERRED	0.5	1.8	32,000
	TOTAL	**0.5**	**1.8**	**32,000**
OXIDE + SULPHIDE	MEASURED	-	-	-
	INDICATED	-	-	--
	INFERRED	5.0	1.6	265,000
	TOTAL	**5.0**	**1.6**	**265,000**

Notes:

(1) The Inferred Mineral Resource Estimate is reported above a cut-off grade of 0.4 g/t for oxide material and 0.5 g/t for sulphide.
(2) The Mineral Resource Estimate for the Sanankoro deposit was constrained within grade based solids and within a Lerchs-Grossman optimised pit shell based on a gold price of 1,700 USD / oz and through the application of reasonable mining parameters.
(3) All figures are rounded to reflect the relative accuracy of the estimate.
(4) Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
(5) It is uncertain is further exploration will convert Inferred Mineral Resources to higher confidence categories.

The Mineral Resource is delineated by zone, and by the weathering profile in the tables below.

Mineral Resources by Zone			
Zone	**Tonnes (Mt)**	**Au g/t**	**Contained Au (Oz)**
Selin	1.9	1.8	108,000
Zone A	1.9	1.5	91,000
Zone B	0.7	2.0	47,000
Zone B North	0.5	1.1	19,000
TOTAL	**5.0**	**1.6**	**265,000**

Mineral Resources by Weathering Profile Domain			
Zone	**Tonnes (Mt)**	**Au g/t**	**Contained Au (Oz)**
Hardcap	0.4	1.3	16,000
Saprolite	3.7	1.6	191,000
Saprock	0.4	1.9	27,000
Fresh	0.5	1.8	32,000
TOTAL	**5.0**	**1.6**	**265,000**

Exploration Target

In October 2018, SRK derived an Exploration Target for the Sanankoro Project, based on the following:

- Volumetric modelling and grade interpolation of mineralisation at Zone A, Zone B, Zone B North and Selin, in addition to two other zones, namely Zone C and Selin South, altogether representing a total strike length of ~11 line km. The volumetric modelling was limited to a depth of 100 m below surface; and

- Assessment of an additional 33 line km of positive exploration results which suggests potential to discover additional mineralisation with similar thickness and grade.

SRK is unaware of any new information which materially impacts on the assumptions upon which the Exploration Target is based. For this reason, an unchanged Exploration Target for the Sanankoro Project **<u>of between 30Mt and 50Mt at a grade of between 1.0 and 1.3g/t Au</u>** is re-stated here.

For the avoidance of doubt, in respect to the Exploration Target, SRK notes:

- The potential quantity and grade as reported in respect of the Exploration Targets are conceptual in nature;
- There has been insufficient exploration to define a Mineral Resource; and
- It is uncertain if further exploration (as planned by the Company) will result in the determination of a Mineral Resource.

The <5 km total strike extent of the optimised pit shells used to constrain the Sanankoro Inferred Mineral Resource represents <15% of the total linear strike length of potential mineralised zones upon which the Exploration Target is based. It is noted that, of the approximate 1 – 2 million ounce Exploration Target range, approximately 700,000 ounces of gold are defined in the block model from which the 265,000 ounce Inferred Mineral Resource is derived (being inside the optimised pit and above cut-off grade).

Mining

Introduction

The Project comprises several distinct zones including Zone A, Zone B, Zone B North and Selin. The mining study has been completed for three production rates (0.5Mtpa,1.0Mtpa,1.5Mtpa) recognized as Case 1, Case 2 and Case 3. The main objective of the Study was to understand how the different cases compare, their potential impact on mining costs for owner and contractor operated scenarios and to support any future exploration activities. The mining study is restricted to oxide material (hardcap, saprolite and saprock) and excludes sulphide (fresh) mineralisation.

Dilution and Recovery

In order to address mining modifying factors such as mining losses and dilution, the mineral resource model (in Datamine format) has been regularised to a block size of 2.5 x 2.5 x 5m and used in pit optimisation and mine planning. A block size of 2.5 x 2.5 x 5m is considered representative of the selective mining unit size estimated for small scale mining equipment (1.9m^3 to 4m^3 bucket excavators, 24t to 40t capacity haul trucks), and requires a relatively high level of selectivity. Above a marginal cut-off of 0.4g/t Au, the dilution in all zones is estimated between 14% and 20% and recovery between 91% and 95%. The method of calculation is explained in the main text of the Report.

Pit Optimisation

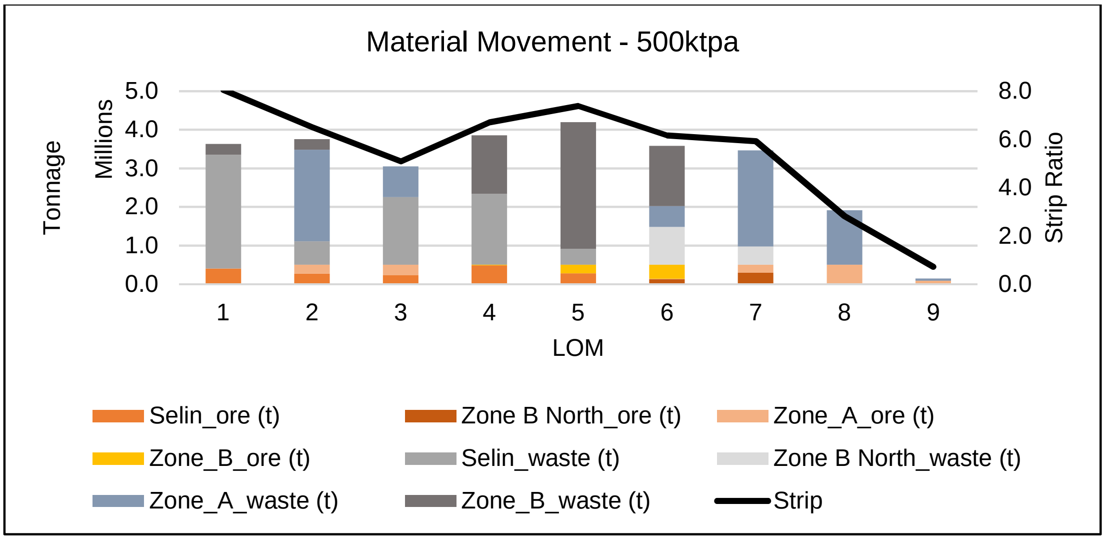
The pit optimisation was completed for a selling price of USD1,500 /oz Au. Resulting pit shells were analysed to compare how the factored metal price (Revenue Factor or "RF") affects ore tonnage, grade and strip ratio. The pit optimisation parameters are shown in the table overleaf. The optimisation parameters outlined in Table ES 1 include recoveries, costs and slope angles for fresh rock (as an alternate pit optimisation was completed on both the oxide and fresh rock for the purposes of Mineral Resource reporting), however it should be stressed that the pit optimisation employed in the mining study considered only oxide material.

Open Pit Parameters					
Parameters	**Units**	**Case 1**	**Case 2**	**Case 3**	**Comments**
Production					
Production Rate - Ore	(tpa)	500,000	1,000,000	1,500,000	Cora Gold Assumption
Geotechnical					
Overall Slope Angle - Saprolite Overall Slope Angle - Saprock Overall Slope Angle - Fresh	(˚) (˚) (˚)	34 40 42	34 40 42	34 40 42	SRK Assumption SRK Assumption SRK Assumption
Mining Factors					
Dilution	(%)	Regularised Block Model 2.5x2.5x5m			See Section 5.2 for details
Recovery	(%)	Regularised Block Model 2.5x2.5x5m			See Section 5.2for details
Processing					
Hardcap - All Zones Zone A/B (sap/saprock) Selin + Zone B North (sap/saprock) Fresh - All Zones	(%) (%) (%) (%)	80.0 95.7 92.9 80.0	80.0 95.7 92.9 80.0	80.0 95.7 92.9 80.0	WAI Assumption WAI Assumption WAI Assumption WAI Assumption
Operating Costs					
Mining Cost - Ore Saprolite Sap Rock & Fresh	 (US$/$t_{ore}$) (US$/t_{ore})	 3.50 4.00	 3.50 4.00	 3.50 4.00	SRK Assumption
Mining Cost - Waste Saprolite Saprock & Fresh	 (US$/$t_{waste}$) (US$/t_{waste})	 3.0 3.50	 3.0 3.50	 3.0 3.50	SRK Assumption
Processing - Saprolite, Saprock, Hardcap Processing - Fresh	(US$/$t_{ore}$) (US$/t_{ore})	16.2 17.0	15.5 17.0	14.7 17.0	WAI Assumption WAI Assumption
G&A	(US$m/Year) (US$/t_{ore})	1.0 2.0	2.0 2.0	3.0 2.0	WAI Assumption
Selling Cost Au	(%) (US$/oz) (US$/g)	5.0 85.0 2.5	5.0 85.0 2.5	5.0 85.0 2.5	SRK Assumption explained in Section 4.5.7 (in Appendix 2) in more detail
Metal Price					
Gold	(US$/oz) (US$/g)	1,500.0 43.8	1,500.0 43.8	1,500.0 43.8	Cora Gold Assumption
Other					
Discount Rate	(%)	10.0	10.0	10.0	SRK Assumption
Cut-Off Grade					
Marginal - Saprolite, Saprock, Hardcap	(US$/$t_{ore}$) (g/t Au)	18.2 0.4	17.5 0.4	16.7 0.4	
Marginal - Fresh	(US$/$t_{ore}$) (g/t Au)	19 0.5	19 0.5	19 0.5	

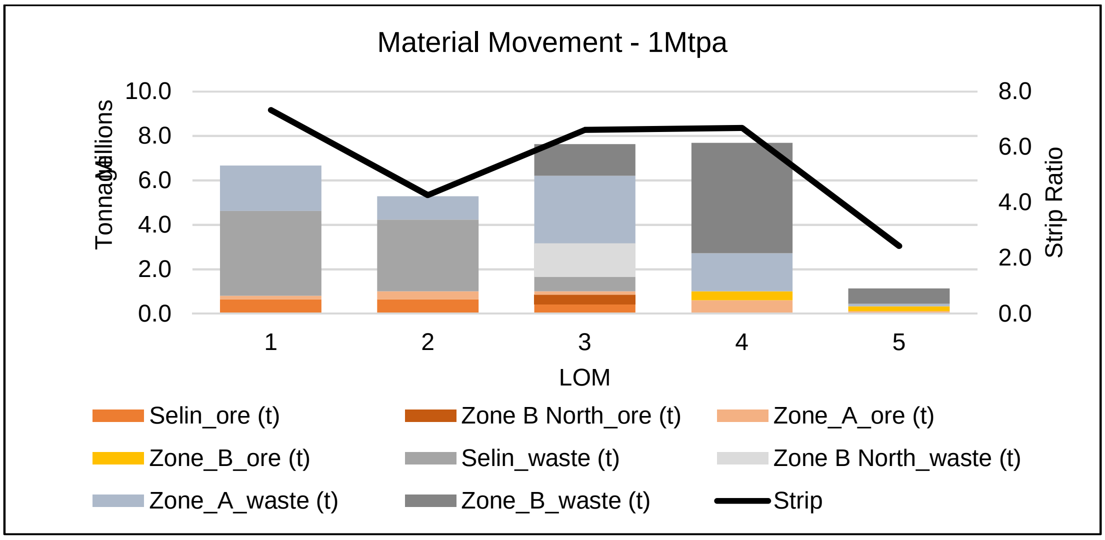
The pit optimisation results show that the mining inventory grows with increasing RF on a broadly linear basis. That said, it should be noted that the total ore tonnage is relatively sensitive to the gold price selected for the pit optimisation. The total ore tonnage inside of the USD1,300/oz pit shell (which is a more similar price to the current long-term Au price forecast) is 2.8Mt at 1.60g/t Au, whilst the total ore tonnage inside of the USD1,500/oz pit shell is 4.1Mt at 1.47g/t Au. This represents a 46% increase in ore tonnage and 35% increase in contained ounces in the USD1,500/oz pit shell, compared to the USD1,300/oz pit shell. Total rock inside the USD1,500/oz pit shell is 28.4Mt and total rock inside the USD1,300/oz pit shell is 17.0Mt. The stripping ratio is 5.9 in the USD1,500/oz pit shell and 5.1 in the USD1,300/oz pit shell. After discussions between Cora Gold and SRK, Cora Gold requested that SRK use the USD1,500/oz Au pit shell (RF=100%) for the development of the strategic schedule. This is considered acceptable at a scoping level, however the sensitivity to Au price should be carefully considered as the Project develops.

Strategic Mine Schedule

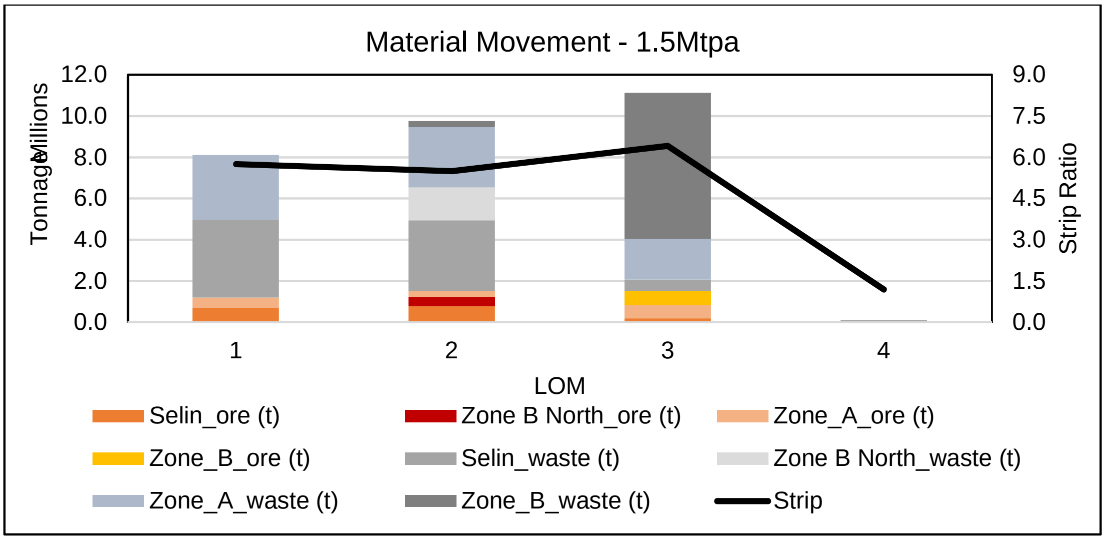
SRK has developed a strategic level mining and processing schedule for the Zone A, Zone B, Zone B North and Selin using NPVS scheduling software. The mine schedule was completed for the three production cases and has been produced in annual periods. The mining schedule for Case 1, Case 2 and Case 3 are shown in the figures below.



Case 1 – 0.5Mtpa – Material Movement



Case 2 – 1.0Mtpa – Material Movement



Case 3 – 1.5Mtpa – Material Movement

Equipment Selection

Based on an estimation of equipment productivities, production profiles and the need for a selective mining method, the selected equipment sizes are shown in the table overleaf. The quantity of each piece of equipment is dependent on the strategic schedule for each Case.

Equipment Selection					
Equipment	Material Type	Unit	Equipment Size		
			0.5Mtpa	1.0Mtpa	1.5Mtpa
Primary Backhoe	Ore	m^3 bucket width	1.9	1.9	1.9
Primary Wheel Loader	Waste	m^3 bucket width	4.2	7.1	8.4
Primary Truck	Ore	t capacity	24	24	24
Secondary Truck	Waste	t capacity	40	40	54
Primary Drill	Both	mm diameter	140	140	140

Operating Strategy

It is expected that the extraction method will be predominantly free digging, as the hardcap and saprolite weathering domains do not require blasting. Drill and blast will be required in the saprock domain.

Ore and waste will be excavated by separate fleets in order to account for a relatively high level of mining selectivity.

Based on the pit locations and the distance between the zones, it is recommended to have three Waste Rock Dumps (“WRD”). The waste rock dump tonnage schedule is reflected by the yearly waste production, but no detailed scheduling has been done for the WRDs. A stockpiling strategy has not been considered in this study.

Capital and Operating Cost Estimation

A mining cost model has been developed to assess the mining capital and operating expenditures expected for the Sanankoro Project. This cost estimation is based on both contractor mining and owner operated options as requested by the Client. All capital and operating costs have been estimated from first principles but based on SRK’s experience of open pits in Mali, or benchmarked from the 2018 Infomine cost database.

The capital cost estimation includes equipment purchase, replacement and rebuild costs, as well as mobilisation/demobilisation and site establishment costs. The results of the capital cost estimation are shown in the table overleaf. The capital cost difference between an owner operated and a contractor option, is that contractor capital does not include equipment purchase and replacement costs.

In addition to the capital cost categories, a 15% capital cost contingency is applied to both the owner operated and contractor options.

Capital Cost Estimation		
Capital Cost	**Units**	**Cost**
Primary Backhoe	(US$m)	0.9
Primary Wheel Loader	(US$m)	0.8
Primary Truck	(US$m)	0.6
Secondary Truck	(US$m)	0.8
Primary Drill	(US$m)	1.4
Track Dozer	(US$m)	0.9
Grader	(US$m)	1.0
Water Truck	(US$m)	1.4
Service Truck	(US$m)	0.1
Fuel/Lube Truck	(US$m)	0.1
Small Crane	(US$m)	0.5
Lighting Plant	(US$m)	0.5
Light Vehicle	(US$m)	0.7
Site Establishment, Mobilisation, Demobilisation	(US$m)	1.7

The operating costs are broken down into four categories including labour, maintenance, consumables and grade control. The owner operated and contractor base unit cost for these categories are the same, therefore the varying factor is a contractor premium of 25% applied to the contractor option. Similar to the capital cost estimation, a 15% operating cost contingency is applied to both the owner operated and contractor options.

The labour and consumable unit costs are provided in the tables below and overleaf.

Consumable and Grade Control Costs			
Consumables	**Source**	**Units**	**Cost**
Fuel	SRK estimate	(US$/l)	1.1
Power	SRK estimate	(US$/kWh)	0.049
Lube	SRK estimate	(US$/l)	4
AN	SRK estimate	(US$/t)	1500
Primer	SRK estimate	(US$/unit)	3.5
Detonator	SRK estimate	(US$/unit)	3
Surface Delay	SRK estimate	(US$/unit)	2
Blasting Accessories	SRK estimate	(%)	5
Sampling	SRK estimate	(US$/unit)	20
Stemming	SRK estimate	(US$/lcm)	10
Grade Control Drilling	SRK estimate	(US$M/yr)	1

Labour Costs	
Position	**Salary (USD$pa)**
Mine Operations	
Mine Manager	120,000
Superintendents	100,000
Supervisors	85,000
Trainer	75,000
Shovel Operator	40,000
Truck Operator	40,000
Loader Operator	45,000
Ancillary Operator	35,000
Driller Operator	35,000
Safety Manager	80,000
Human Resources Manager	80,000
Accountant	60,000
Security	15,000
Mine Maintenance	
Maintenance Superintendent	85,000
Maintenance Supervisor	75,000
Maintenance Planner	65,000
Maintenance Crew	25,000
Technical Services	
Chief Engineer	100,000
Chief Geologist	100,000
Senior Engineer	90,000
Senior Geologist	90,000
Planning Engineer	75,000
Mine Surveyor	40,000
Mine Geologist	60,000
Consultant (Hydro, Geotech, etc)	75,000
Administrative Assistant	25,000

The mining costs estimated in this Report are summarised in the table below.

Estimated Mining Costs					
	Scenario	**Unit**	**0.5Mtpa**	**1.0Mtpa**	**1.5Mtpa**
OPEX	Owner	(US$/t)	3.43	2.82	2.48
	Contractor	(US$/t)	4.17	3.43	3.02
CAPEX	Owner	(US$M)	19.6	32.6	31.3
	Contractor	(US$M)	3.2	5.9	6.6

Geotechnical

SRK have provided Scoping level geotechnical slope criteria for the Sanankoro project to feed into pit optimisation. The pits will be in the region of ~100m at the deepest sections and will primarily be formed within saprolite with minor saprock and fresh rock at the base of the slopes. Whilst limited geotechnical information exists for the fresh material, there is no geotechnical information for the saprolite. As such, SRK have relied on experience from developing pit slopes in other saprolite deposits to propose a range of saprolite slope angles for Sanankoro.

Several slope angles ranging from 26° to 38° were considered, with a slope angle of 34° chosen for input into pit optimisation. Within the deeper sections of the open pits, 34° can be considered steep and to achieve such an angle, high quality surface water management in addition to slope depressurisation drilling will be required to lower pore water pressure within the slope. Regardless of the success of the depressurisation programme, bench and possibly multi-bench failure may be expected as a result of remnant structure within the saprolite.

For the small sections of saprock and fresh material exposed at the toe of the slopes, SRK recommended 40° and 42° slope angles respectively. To verify the proposed Scoping level slope angles at the next project stage, geotechnical drilling, logging and sampling will be required in addition to hydrogeological testing to determine the susceptibility of the saprolite to slope depressurisation programmes.

Hydrogeology and Hydrology

SRK have completed a high-level scoping study review of the available hydrology and hydrogeological data for the Sanankoro project. These data are noted to be very limited, but have been used to inform recommendations for moving the project towards a Pre-Feasibility Study (PFS). The assumptions made are that all mining slopes will be within the saprolite formation and will need to be depressurised in order to achieve the pit slope angles defined by the geotechnical assessment.

The key hydrological risks identified relate to high intensity rainfall events resulting in either direct flooding of the pits or indirect recharging of the pit slope pore pressures; these risks should be quantified at PFS level following the installation of a site weather station and river flow gauges. The key hydrogeological risk for the project is the inability for the saprolite to remain depressurised; the hydrogeological system requires testing and conceptualisation in order to assess expected pore pressure responses to both climate and mining events. This assessment requires the establish of groundwater level monitoring and hydraulic testing within the key hydrogeological units.

Metallurgy

Wardell Armstrong International (WAI) was commissioned by Cora Gold (the Client) to undertake a programme of metallurgical testing on samples of oxide gold mineralisation from the Sanankoro deposit, Mali.

Initially, testing was undertaken using two samples of exploration drill core representing different areas of the Sanankoro deposit and consisted of a programme of testing including; head assay, coarse ore bottle roll testing and whole ore leach testing.

Subsequently, a Master Composite sample was prepared by blending the two samples. This sample was then subjected to a further programme of testing consisting of; head assay, gravity-leach testing, whole ore leach testing, agglomeration & percolation testing and column leach testing.

Phase 1

Head Assay

Detailed chemical head assay was performed on each of the samples submitted for testing. A summary of the results is given in the Table below.

Phase 1 Head Assay Results			
Element	Units	Assay	
		SD0005	SD0006
$Au_{(FA)}$	ppm	0.61	3.35
$Au_{(AR)}$	ppm	0.64	2.70
Ag	ppm	<0.5	1.2
Cu	%	0.005	0.008
Pb	%	0.003	0.002
Zn	%	0.004	0.010
Fe	%	2.31	4.35
As	%	0.046	0.028
$S_{(TOT)}$	%	0.022	0.046
$C_{(TOT)}$	%	0.027	0.13

Coarse Ore Bottle Roll Testing

Coarse ore bottle roll testing was performed to provide an indication of the maximum gold and silver recoveries achievable at coarse particle sizes, typical of those used in heap leach operations.

Testing was conducted to investigate the effect of crush size on leach response with each sample tested at three crush sizes; -20.0mm, -12.5mm and -6.3mm, in duplicate.

A summary of the average gold and silver recoveries achieved for each sample/crush size is given in the Table below.

Summary of Coarse Ore Bottle Roll Leach Test Results					
Sample	Crush Size	Reagent Consumption (kg/t)		Recovery (%)	
		Lime	Cyanide	Au	Ag
SD0005	-20.0mm	1.20	0.45	78.4	14.8
	-12.5mm	1.08	0.71	84.2	21.9
	-6.3mm	1.00	0.53	97.6	17.9
SD0006	-20.0mm	1.98	0.57	66.8	35.1
	-12.5mm	2.08	0.62	81.6	45.4
	-6.3mm	1.80	0.76	93.1	47.9

Whole Ore Leach Testing

A single agitated leach test was performed on each of the samples to determine metal recoveries achievable at fine grind sizes typical of CIL type operations.

Results are summarised in the Table below.

Phase 1, Whole Ore Cyanide Leach Test Results				
Sample	**Reagent Consumption (kg/t)**		**Recovery (%)**	
	Lime	**Cyanide**	**Au**	**Ag**
SD0005	0.72	1.35	97.4	32.7
SD0006	1.49	1.50	96.7	67.3

Phase 2

Head Assay

Following preparation of the Master Composite, a representative sub-sample was submitted for head assay for gold and silver. Results are summarised in the Table below.

Master Composite Head Assay Result		
Element	**Units**	**Assay**
$Au_{(AR)}$	ppm	2.74
Ag	ppm	0.8

Gravity-Leach Testing

Gravity-leach testing was undertaken to investigate the total amount of gold recoverable through the combination of gravity preconcentration followed by cyanide leaching of the gravity tailings.

Results of the gravity preconcentration stage are summarised in the Table below.

Master Composite Gravity Concentration Test Results					
Product	**Stage**	**Grind Size (µm)**	**Mass (%)**	**Assay, Au (ppm)**	**Recovery, Au (%)**
Concentrate	1	212	0.34	406	50.65
	2	75	0.36	167	22.37
	Total	**-**	***0.70***	***282***	***73.02***
Tailings	-	-	99.30	0.74	26.98
Feed	-	-	100.00	2.71	-

Following completion of the gravity testing, the gravity tailings were subjected to kinetic cyanide leach testing at two cyanide concentrations. Results of these tests are summarised in the Table overleaf.

Master Composite Gravity Tailings Cyanide Leach Test Results				
Cyanide Concentration (g/L)	Reagent Consumption (kg/t)		Recovery (%)	
	Lime	Cyanide	Au	Ag
1.0	0.49	1.21	94.1	34.1
0.5	0.87	0.53	92.1	36.2

Whole Ore Leach Testing

A series of whole ore leach tests were conducted to investigate the amount of gold and silver that could be recovered from the Master Composite through direct cyanide leaching at fine particle sizes.

Testing investigated two key variables; grind size and cyanide concentration. A summary of the results is given in the Table below.

Master Composite Whole Ore Cyanide Leach Test Results					
Grind Size (µm)	Cyanide Concentration (g/L)	Reagent Consumption (kg/t)		Recovery (%)	
		Lime	Cyanide	Au	Ag
150µm	1.0	0.88	1.21	95.0	58.3
125µm	1.0	1.10	1.09	98.0	62.4
106µm	1.0	1.02	1.17	95.3	53.6
75µm	1.0	1.00	0.98	98.0	59.2
75µm	0.5	1.22	0.49	97.3	56.6
75µm	0.25	1.53	0.08	92.9	62.4

Agglomeration & Percolation Testing

Agglomeration and percolation testing was performed to investigate the need to agglomerate the Master Composite with cement prior to column leaching. The sample was subjected to five percolation tests, four of which were performed on material which had been agglomerated with cement.

A summary of the results is given below.

Master Composite Percolation Test Results			
Cement Addition (kg/t)	Drainage Flowrate ($l/m^2/hr$)		
	Minimum	Maximum	Average
0	14	45	**25**
5	587	965	**780**
10	2,542	3,940	**3,136**
15	4,724	11,630	**7,208**
22.5	10,002	18,773	**12,794**

Column Leach Test

A single column leach test was conducted on the Master Composite to provide an indication of the gold and silver recoveries and leach kinetics achievable under heap leach conditions.

A 40kg sample of the Master Composite was subjected to testing for a total of 105 days (95 days under direct irrigation) using a 1.0g/L cyanide solution at a target application rate of 10l/m^2/hr. Regular samples of the pregnant leach solution (PLS) were taken in order to measure levels of gold and silver extraction after which, the solution was passed through a column containing activated carbon in order to also determine metal recoveries onto the loaded carbon.

Results of the column leach test are summarised in the Table below.

Master Composite Column Leach Test Results					
Reagent Consumption (kg/t)		Recovery to PLS (%)		Recovery to Carbon (%)	
Lime	Cyanide	Au	Ag	Au	Ag
0.12	0.62	56.0	37.0	56.3	42.9

Mineral Processing

The oxide ore samples tested are very amenable to conventional cyanide processing (CIL) with an average whole ore leach recovery of 93.5%. For the heap leach (HL) option, the coarse ore bottle roll tests indicated recoveries approaching 90% at the coarser size fractions, although the column test result using 22.5kg/t cement only produced a recovery of 55% after 90 days of leaching, although recovery was clearly continuing at the end of the test and with some evidence that more cement was required. Therefore, a conservative recovery of 70% has been assumed with the potential for higher recovery once further optimised column tests can be conducted.

A preliminary trade-off study for a 0.75Mtpa CIL or HL operation using these recoveries concluded that, with indicated capital and operating cost estimates of $61.4 million/$15.9/t and $11.4 million/$10.3/t respectively, that HL was economically the optimum processing route.

This was agreed with Cora Gold and additional capital and operating cost estimates conducted for 1.0Mtpa and 1.5Mtpa HL scenarios and used in the financial model. It appears that 1.0Mtpa is the optimum throughput for the current size of oxide resource with estimated capital and operating costs of $12.3 million and $8.8/t respectively.

The priority for further testwork is optimised column tests to confirm that recoveries of 70% or higher can be achieved and the optimum cement addition required for agglomeration.

Tailings Storage Facility

A preliminary site for a potential valley-fill TSF was identified by Cora Gold and a photograph provided for reference. The required storage capacity would be approximately 4 Mt. The scoping-level capital cost estimate for the 0.75Mtpa CIL plant option includes the TSF. However, as the HL option has been identified as the optimum process route, no TSF will be required.

Infrastructure

Water will be supplied from the nearby River Fie by a water abstraction system, located approximately 3km from site, at an estimated capital cost of $0.7 million. The pump and pipeline system will pump water to a raw water pond or tank.

A surge/event pond will also be required and included in the HL cost estimate.

Allowance has been made for upgrading approximately 30km of laterite road and two bridges from Selingue to Selefougou and then on to site, at an estimated cost of $2.5 million.

Allowance has been made for a site camp to accommodate approximately 36 people at a cost of $0.3 million. The majority of workers will be recruited and transported from local nearby villages.

Power will be supplied using a rented 1 MW diesel generator at an estimated total cost of 21 c/kWh. Allowing for the power cost already included in the operating cost estimate, an additional allowance of $0.7 million per annum has been included.

It is estimated that a total of four HL cells will be required, each accommodating 1Mt of stacked ore. Each cell will be divided into four pads, each pad holding 0.25Mt of ore. It is likely that each pad will consist of three lifts, each of 8m height. Geotechnical investigations will be required to determine the most suitable location for the cells which can be constructed in stages. Photographs of the site indicate large areas of flat terrain that appear potentially suitable.

Transport of the gold bullion smelted on site will be the responsibility of the refinery once it has left the gold room.

Environmental and Social

Cora Gold Limited (Cora Gold) is undertaking gold exploration activities associated with the Sanankoro Gold Prospect located in southern Mali. Digby Wells Environmental (Digby Wells) was appointed to undertake a Scoping Study to characterise the biophysical and socio-economic environment of the project area, provide early indication of potential environmental and social risks and determine the Terms of Reference (ToR) for the Environmental and Social Impact Assessment (ESIA) process that will be required as part of the environmental permitting process. The ESIA will be undertaken in accordance with Malian Law No. 2012-015 of 27 February 2012 (the Mining Code) and associated Decrees, among others. In addition, International Best Practice will also be considered.

Biophysical Environment and Risks

The local vegetation is predominantly distributed grasslands. Due to Artisanal and Small Scale Mining (ASM) activities throughout the project area, much of the land associated with the resource target areas, and areas identified for the processing plant and other administrative infrastructure within the project area has been disturbed. Isolated forest galleries which are associated with floodplains are also present within the project area. Furthermore, land (including floodplains and wetland areas) associated with the project area are extensively utilised for agricultural activities while nearby streams are used for ASM activities. Medium and large mammals in the Project area are rare and hunting is a practised to a limited extent. Several Nationally Protected and Red Data Plant species are expected to occur regionally.

The streams and drainage lines in the project area are predominantly ephemeral, including the Fie and Niger Rivers which traverse the west and north boundaries of the project area. Surface water resources are utilised by the communities for economic activities (agricultural and ASM) while groundwater is used for potable and domestic uses. A total of six water samples were collected upstream and downstream of the project area which indicate no issues in terms of water quality. This could be due to dilution as aesthetically there are many impacts from ASM activities. Generally groundwater quality results were found to be good.

Water management in the project design and during operations is important as these areas may intersect with various water resources. To this end, detailed floodplain determinations will be required to delineate the floodplains, as well as determine potential surface water volumes during extreme rainfall events. Groundwater modelling will also be an important task to determine the potential impact dewatering may have on surface water resources, as well as to determine potential contamination plumes from the project's waste deposits. The potential pit areas, plant and other administrative infrastructure are however largely disturbed by existing ASM activities along the targeted ore structures.

Socio-Economic Environment and Risks

The project is located within the Kangaba Cercle of the Koulikoro Region and spans over the Séléfougou and Maramadougou rural communes. A total of six rural communes are located within the project area, namely: Séléfougou, Sanankoro, Bokoro (hamlet), Sélin, Faragouagnan and Kignèlen (hamlet).

The primary economic activities in the project area comprise ASM, cultivation, livestock breeding, and limited small trade which includes the exploitation of natural resources. Agricultural activities are located within and near the communities and maintained by the respective villagers. Livestock rearing in the project area includes large and small livestock, such as cattle, sheep, goats and poultry. ASM activities are practiced throughout the regions and the population undertaking ASM activities is increasing, as well as attracting individuals from neighbouring regions and countries.

Several communities and their associated economic activities are located within 500m of potential pit areas. Communities' households, agricultural fields and ASM activities within 500m of proposed pits will result in economic and physical displacement. This, together with the expected increased influx of people into the area as a result of the presence of the project, is expected to be the key socio-economic implications.

A Resettlement Action Plan (RAP) and Livelihood Restoration Plan (LRP) will be required for the economic and physical displacement associated with project land acquisition. The RAP and LRP will need to have a clear entitlement framework to address any potential challenges as it is expected that resettlement will be widely contested due to the extent and reliance of this activity currently. ASM management, and the loss of livelihood, will have significant impacts in the area and will need to be managed carefully and in cooperation with the technical and administrative authorities.

It is recommended that baseline socio-economic surveys are undertaken in the affected communities to determine the baseline of affected communities and the extent of potential resettlement prior to any population influx. It is important to manage community expectations and potential resettlement should not be communicated until a final layout plan is complete.

Population influx is expected because of the project as individuals from surrounding regions and neighbouring countries move to the project area in search of employment. The population influx will also place additional pressure on the already stressed natural resources as well as social services and infrastructure in the project area.

Economic Analysis

WAI has performed a preliminary economic assessment of the Cora Gold open pit operations using a Discounted Cash Flow (DCF) analysis to assess the following options of the project further development at 1Mtpa (Base Case), 750ktpa (Option 1) and 1.5Mtpa (Option 2).

All options have been considered with HL processing route implementation, assuming 70% total recovery throughout mine life.

A discounted cash flow model has been developed in real US$ as of January 2020 with no escalation taken into account. With a long term fixed gold price was assumed at US$1,400/oz (base case).

Post-tax Net Present Value has been assumed at a base case 8% discount rate.

Table below provides summary of the results from all three options compared. As could be seen, project economics gain from the economy of scale and provides best NPV results for 1.5Mtpa option. Therefore, WAI recommends this option to be taken forward for further studies.

Cora Gold Financial Results Comparison (US$ M)			
Production Rates	1Mtpa	0.75 Mtpa	1.5 Mtpa
	1	2	3
Gross Revenue	**190.8**	**190.8**	**193.8**
Total Mining Operating Cost	97.4	115.7	87.8
Total Processing Cost	36.3	42.5	27.6
Production Costs	**133.7**	**158.3**	**115.4**
Marginal Revenue	**57.1**	**32.5**	**78.4**
General and Administrative Cost	6.2	6.2	6.4
Realisation Cost	0.7	0.7	0.7
Mining Royalty	5.7	5.7	5.8
Closure and Reclamation Fees	0.0	0.0	0.0
EBITDA	**44.5**	**19.9**	**65.5**
Pre-Production Capex	**19.1**	**18.0**	**20.6**
Sustaining Capex	3.0	3.9	2.1
Pre-Tax Free Cash Flow	**22.3**	**-1.9**	**42.8**
Pre-Tax NPV	**12.8**	**-5.5**	**30.9**
IRR	**32%**	**-3%**	**84%**
CIT	***3.4***	***0.0***	***8.9***
Post-Tax NPV	**7.4**	**-7.3**	**23.5**
IRR	**30%**	**-3%**	**73%**
AISC (LOM), US$/oz	1,096	1,282	942

Based on the results from a sensitivity analysis performed, Cora Gold project is mostly sensitive to change in metal prices and recovery rates, followed by change in mining operating costs and less sensitive to change in processing opex. The least sensitive parameter is capital cost.

Project break-even gold prices and recovery rates are as following:

- Option 1Mtpa: recovery rate drops below 63%, break-even price of US$1,271/oz.
- Option 0.75Mtpa: recovery rate below 74%, break-even price of US$1,458/oz; and
- Option 1.5Mtpa: recovery rate below 56%, break-even price of US$1,108/oz.

More details on project valuation assumptions and sensitivity analysis results are given in the relevant chapter of this report.

Conclusions

Results of the Study show good initial validation of the Project’s future economic potential.

Based on the performed preliminary economic assessment, the Cora Gold project gains from economy of scale, and therefore higher production throughput results in better NPVs.

The project is mostly sensitive to change in metal price, recovery rates and mining operating costs.

The base case gold price of US$1,400/oz has been selected on a moderately optimistic side with break-even prices for the considered options as following:

- Option 1Mtpa: recovery rate drops below 63%, break-even price of US$1,271/oz.
- Option 0.75Mtpa: recovery rate below 74%, break-even price of US$1,458/oz; and
- Option 1.5Mtpa: recovery rate below 56%, break-even price of US$1,108/oz.

The project remains robust at production rates above 1Mtpa and is not viable for lower production rates using technical and economic parameters outlined in this report.

1 MINERAL RESOURCE ESTIMATE

The Mineral Resource Estimate chapter of this Scoping Study has been carried out by SRK and can be found in **Appendix 1- A MINERAL RESOURCE ESTIMATE ON THE SANANKORO GOLD PROJECT, MALI** of this report. WAI has not undertaken a review of this section and cannot be held responsible or liable for the contents therein.

2 MINING

The Mining chapter of this Scoping Study has been carried out by SRK and can be found in **Appendix 2 – A REPORT FOR THE MINING SCOPING STUDY ON THE SANANKORO GOLD PROJECT, MALI-** of this report. WAI has not undertaken a review of this section and cannot be held responsible or liable for the contents therein.

3 GEOTECHNICS

The Geotechnical chapter of this Scoping Study has been carried out by SRK and can be found in **Appendix 2-– A REPORT FOR THE MINING SCOPING STUDY ON THE SANANKORO GOLD PROJECT, MALI** of this report. WAI has not undertaken a review of this section and cannot be held responsible or liable for the contents therein.

4 HYDROGEOLOGY AND HYDROLOGY

The Hydrology and Hydrogeology chapter of this Scoping Study has been carried out by SRK and can be found in **Appendix 2-– A REPORT FOR THE MINING SCOPING STUDY ON THE SANANKORO GOLD PROJECT, MALI** of this report. WAI has not undertaken a review of this section and cannot be held responsible or liable for the contents therein.

5 METALLURGY

Wardell Armstrong International (WAI) was commissioned by Cora Gold (the Client) to undertake a programme of metallurgical testing on samples of oxide gold mineralisation from the Sanankoro deposit, Mali. Whereas a summary of these tests has been provided in this section, the full report can be found in **Appendix 3- (WAI) METALLURGICAL TESTING ON SAMPLES OF OXIDE MINERALISATION.**

Initially, testing was undertaken using two samples of exploration drill core representing different areas of the Sanankoro deposit and consisted of a programme of testing including; head assay, coarse ore bottle roll testing and whole ore leach testing.

Subsequently, a Master Composite sample was prepared by blending the two samples. This sample was then subjected to a further programme of testing consisting of; head assay, gravity-leach testing, whole ore leach testing, agglomeration & percolation testing and column leach testing.

5.1 Phase 1

5.1.1 Head Assay

Detailed chemical head assay was performed on each of the samples submitted for testing. A summary of the results is given in Table 5.1 below.

Table 5.1: Phase 1 Head Assay Results

Element	Units	Assay	
		SD0005	SD0006
$Au_{(FA)}$	ppm	0.61	3.35
$Au_{(AR)}$	ppm	0.64	2.70
Ag	ppm	<0.5	1.2
Cu	%	0.005	0.008
Pb	%	0.003	0.002
Zn	%	0.004	0.010
Fe	%	2.31	4.35
As	%	0.046	0.028
$S_{(TOT)}$	%	0.022	0.046
$C_{(TOT)}$	%	0.027	0.13

5.2 Coarse Ore Bottle Roll Testing

Coarse ore bottle roll testing was performed to provide an indication of the maximum gold and silver recoveries achievable at coarse particle sizes, typical of those used in heap leach operations.

Testing was conducted to investigate the effect of crush size on leach response with each sample tested at three crush sizes; -20.0mm, -12.5mm and -6.3mm, in duplicate.

A summary of the average gold and silver recoveries achieved for each sample/crush size is given in Table 5.2 below.

Table 5.2: Summary of Coarse Ore Bottle Roll Leach Test Results

Sample	Crush Size	Reagent Consumption (kg/t)		Recovery (%)	
		Lime	Cyanide	Au	Ag
SD0005	-20.0mm	1.20	0.45	78.4	14.8
	-12.5mm	1.08	0.71	84.2	21.9
	-6.3mm	1.00	0.53	97.6	17.9
SD0006	-20.0mm	1.98	0.57	66.8	35.1
	-12.5mm	2.08	0.62	81.6	45.4
	-6.3mm	1.80	0.76	93.1	47.9

5.3 Whole Ore Leach Testing

A single agitated leach test was performed on each of the samples to determine metal recoveries achievable at fine grind sizes typical of CIL type operations.

Results are summarised in Table 5.3 below.

Table 5.3: Phase 1, Whole Ore Cyanide Leach Test Results

Sample	Reagent Consumption (kg/t)		Recovery (%)	
	Lime	Cyanide	Au	Ag
SD0005	0.72	1.35	97.4	32.7
SD0006	1.49	1.50	96.7	67.3

5.4 Phase 2

5.4.1 Head Assay

Following preparation of the Master Composite, a representative sub-sample was submitted for head assay for gold and silver. Results are summarised in Table 5.4 below.

Table 5.4: Master Composite Head Assay Result

Element	Units	Assay
$Au_{(AR)}$	ppm	2.74
Ag	ppm	0.8

5.5 Gravity-Leach Testing

Gravity-leach testing was undertaken to investigate the total amount of gold recoverable through the combination of gravity preconcentration followed by cyanide leaching of the gravity tailings.

Results of the gravity preconcentration stage are summarised in Table 5.5 below.

Table 5.5: Master Composite Gravity Concentration Test Results

Product	Stage	Grind Size (µm)	Mass (%)	Assay, Au (ppm)	Recovery, Au (%)
Concentrate	1	212	0.34	406	50.65
	2	75	0.36	167	22.37
	Total	**-**	***0.70***	***282***	***73.02***
Tailings	-	-	99.30	0.74	26.98
Feed	-	-	100.00	2.71	-

Following completion of the gravity testing, the gravity tailings were subjected to kinetic cyanide leach testing at two cyanide concentrations. Results of these tests are summarised in Table 5.6 below,

Table 5.6: Master Composite Gravity Tailings Cyanide Leach Test Results

Cyanide Concentration (g/L)	Reagent Consumption (kg/t)		Recovery (%)	
	Lime	Cyanide	Au	Ag
1.0	0.49	1.21	94.1	34.1
0.5	0.87	0.53	92.1	36.2

5.6 Whole Ore Leach Testing

A series of whole ore leach tests were conducted to investigate the amount of gold and silver that could be recovered from the Master Composite through direct cyanide leaching at fine particle sizes.

Testing investigated two key variables; grind size and cyanide concentration. A summary of the results is given in Table 5.7 below.

Table 5.7: Master Composite Whole Ore Cyanide Leach Test Results

Grind Size (µm)	Cyanide Concentration (g/L)	Reagent Consumption (kg/t)		Recovery (%)	
		Lime	Cyanide	Au	Ag
150µm	1.0	0.88	1.21	95.0	58.3
125µm	1.0	1.10	1.09	98.0	62.4
106µm	1.0	1.02	1.17	95.3	53.6
75µm	1.0	1.00	0.98	98.0	59.2
75µm	0.5	1.22	0.49	97.3	56.6
75µm	0.25	1.53	0.08	92.9	62.4

5.7 Agglomeration & Percolation Testing

Agglomeration and percolation testing was performed to investigate the need to agglomerate the Master Composite with cement prior to column leaching. The sample was subjected to five percolation tests, four of which were performed on material which had been agglomerated with cement.

A summary of the results is given Table 5.8 below.

Table 5.8: Master Composite Percolation Test Results			
Cement Addition (kg/t)	Drainage Flowrate ($l/m^2/hr$)		
	Minimum	Maximum	Average
0	14	45	**25**
5	587	965	**780**
10	2,542	3,940	**3,136**
15	4,724	11,630	**7,208**
22.5	10,002	18,773	**12,794**

5.8 Column Leach Test

A single column leach test was conducted on the Master Composite to provide an indication of the gold and silver recoveries and leach kinetics achievable under heap leach conditions.

A 40kg sample of the Master Composite was subjected to testing for a total of 105 days (95 days under direct irrigation) using a 1.0g/L cyanide solution at a target application rate of $10l/m^2/hr$. Regular samples of the pregnant leach solution (PLS) were taken in order to measure levels of gold and silver extraction after which, the solution was passed through a column containing activated carbon in order to also determine metal recoveries onto the loaded carbon.

Results of the column leach test are summarised in Table 5.9 below.

Table 5.9: Master Composite Column Leach Test Results					
Reagent Consumption (kg/t)		Recovery to PLS (%)		Recovery to Carbon (%)	
Lime	Cyanide	Au	Ag	Au	Ag
0.12	0.62	56.0	37.0	56.3	42.9

6 MINERAL PROCESSING

6.1 Introduction

The detailed metallurgical testwork results for the three oxide ore samples are described in a separate section of this scoping study. The three oxide ore types included carbon phyllite, sandstone and zone A+B samples. The mineable oxide resource is approximately 4.06Mt @ 1.48g/t Au. Although there is the potential for sulphide ore treatment below the oxide zone, this is not included within the current scoping study.

Based on the testwork results, including limited coarse bottle roll and column testwork to evaluate heap leaching, a simple trade-off study has been included in this study to evaluate heap leaching and CIL as the potential process routes going forwards.

6.2 Recoveries

6.2.1 Carbon-in-Leach (CIL)

The optimum test results for whole ore leaching (CIL) have been selected based on a grind of 80% passing 106 microns, air sparging only and with 0.5g/l initial cyanide concentration. The improved results for oxygen sparging in terms of recovery and leach kinetics were demonstrated (albeit from a single test) but were somewhat marginal across the three ore types, although significantly better for the sandstone sample. For the same testwork conditions, an average increase for all three samples of approximately 1.1% was obtained with oxygen sparging. An oxygen plant has not been considered at this stage, but a future testwork program should fully evaluate this option. The cyanide level of 0.5g/l (500 ppm) has been assumed as a practical maximum for normal plant operation.

From the testwork results and for the optimum conditions stated above, the recoveries were 92.4%, 94.1% and 96.7% respectively for the carbon phyllite, sandstone and zone A+B samples. Applying the 99% factor for solution losses, the recoveries for use in this scoping study are:

Carbon Phyllite: 91.5%
Sandstone: 93.2%
Zone A+B: 95.7%

The average recovery for all three samples is therefore 93.5%.

There is potential for further recovery improvement through optimisation of the grind size, cyanide concentration and effect of air/O_2 on leach performance.

6.2.1.1 Evaluation of Gravity Testwork

The testwork results included gravity processing with cyanidation of the gravity tailings. The results based on recovery without intensive leaching of the concentrates indicates recoveries of 94.9%, 94.9% and 96.6% respectively for the carbon phyllite, sandstone and zone A+B samples. The average recovery is therefore 95.5%, which is approximately 2% higher than the average whole ore leach recovery. This is roughly at the margin of a gravity circuit being economically viable.

However, the results of intensive leaching of the gravity concentrates indicated recoveries of 83.5%, 95.4% and 82.2% for the carbon phyllite, sandstone and zone A+B samples respectively. Therefore, for the carbon phyllite and zone A+B samples, intensive leaching of the gravity concentrates produced poor results. The reason for this is not apparent at this stage of the testwork program.

Therefore, a gravity circuit has not been selected for the scoping study. However, the next stage of testwork should fully evaluate the gravity option and, in particular, the reasons for the low intensive leach recovery on the two samples.

6.2.2 Heap Leaching (HL)

Based on the limited testwork conducted to evaluate the heap leach option, the average coarse bottle roll testwork results on a master composite sample indicated average recoveries of approximately 83.7%, 88.5% and 93.0% for sizes of -20mm, -12.5mm and -6.3mm respectively. This would typically indicate good heap leach amenability, typical of West African saprolites, and with no significant variation in recovery between the coarser and finer sizes.

Unfortunately, the single column test conducted at -20mm and using 22.5kg/t of cement for agglomeration, indicated a recovery of only 55%, significantly lower than the bottle roll test results. However, the ore was still leaching well at the end of the test (90 days). The difference between the assayed and calculated head grade was also slightly higher than normal at 24% (typically 10-15% maximum). The leach residue also showed some evidence of agglomerate breakdown, even at 22.5kg/t cement addition, but significantly better compared to lower cement additions.

It is therefore likely that, with optimised solution irrigation rates, higher cement addition and a longer leach cycle time, higher recoveries approaching the bottle roll test recovery could be achieved.

For the purposes of this study, a heap leach recovery of 70% is assumed.

Based on the column test recovery profile and for financial modelling purposes, it is assumed that 48% of the gold is recovered after 30 days leaching, 60% after 60 days and 70% after 90 days, i.e. a three-month leach cycle.

6.3 Trade-Off Study: CIL vs HL

For the purposes of evaluating the optimum process route, a mineable resource of 4.06Mt @ 1.48g/t Au is assumed. The average CIL and HL recovery is 93.5% and 70% respectively. The plant throughput rate is assumed to be 750,000tpa (as advised by Cora Gold) for a mine life of 5.4 years.

From the WAI database of similar projects and cost models, the average capital cost for a 1Mtpa CIL plant is approximately $73 million. Applying the 6/10^{ths} rule, this calculates as $61.4 million for a 750,000tpa operation. The operating cost for similar operations, including cost models, but with various differences in the blend of ore types, power costs, labour numbers etc, is shown in Figure 6.1.

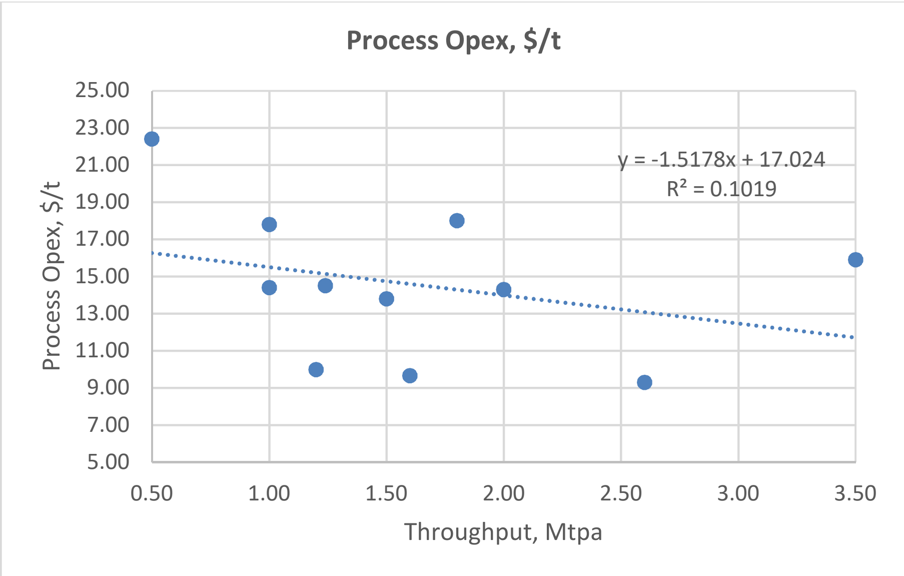


Figure 6.1: WAI Database of CIL Operating Costs

While there is clearly a wide variation in costs, for scoping study level, this indicates an approximate CIL operating cost of $15.9/t for 750,000tpa throughput. As confirmation, this agrees almost exactly with a PFS level operating cost recently determined for a similar CIL plant.

For the HL option, using the WAI database of actual operations and cost models, an approximate capital cost of $11.4 million is indicated for a 750,000tpa operation and an operating cost of $10.3/t.

Calculating a simplified comparative NPV for these two options based on gold revenue and process operating costs only, using a 10% cost of capital, a gold price of $1,500/oz and the parameters as described above, gives an NPV of $91.7 million for CIL and $108.1 million for HL.

This would indicate that heap leaching is more economic at the assumed 70% recovery. A heap leach recovery of approximately 62% would give an NPV equal to that of the CIL option, which is only slightly higher than the actual test recovery of 55%. The HL NPV is very sensitive to recovery and this requires confirmation in a future variability testwork program.

An optimistic viewpoint, based on operational experience on West African saprolite ores, is that significantly higher recoveries are often possible, as long as the agglomerate quality is satisfactory, often reliant on adding enough cement. Recoveries up to 90% are not unusual. The coarse ore bottle roll tests indicate high recoveries approaching 90% at the coarser size ranges. Therefore, an optimistic recovery forecast is approximately 80%. In this case, the NPV increases to $129.6 million.

Cora Gold has indicated that a recently provided capital cost estimate for a 1.4 Mtpa CIL plant in an adjacent property was approximately $75 million. This is slightly lower than the WAI database average and would indicate, using the 6/10^{ths} rule, a capital cost of $51.6 million, compared to approximately $61.4 million from the WAI database. This increases the NPV to $101.5 million.

Also, looking at the CIL operating cost database, three of the lowest cost operations have operating costs of around $10/t. With the lower capital cost noted above, this would further increase the NPV to $119.3 million.

These results are summarised in Table 6.1.

Table 6.1: Summary of Simplified NPV Calculations

Option	NPV, $m
CIL (WAI Database Costs)	91.7
CIL (Optimistic - Lower Capex & Opex)	119.3
HL (WAI Database Costs @70% Recovery)	108.1
HL (Optimistic - 80% Recovery)	129.6

In summary, based on the WAI database of costs and assuming a final recovery of 70% for the HL option, heap leaching is considered marginally the optimum process route compared to CIL, and this is confirmed if using an optimistic HL recovery of 80%. However, these recoveries require confirmation in a future detailed testwork program, based on the actual test recovery of 55% but bottle roll recoveries approaching 90%.

If an optimistic view is taken on the CIL operating cost (the lowest percentile costs in the database) and the lower capital cost as indicated by the Senet quotation and cost model data, then CIL becomes a realistic option compared to HL at recoveries of approximately 70%.

At this scoping study stage, both CIL and HL are viable options, but the overall numbers tend to favour HL, unless a HL recovery greater than approximately 62% cannot be achieved compared to the CIL base case.

A simplified description of both processes is defined in the following section.

6.4 Process Design

6.4.1 CIL

A simplified flowsheet for a generic CIL plant is illustrated in Figure 6.2.

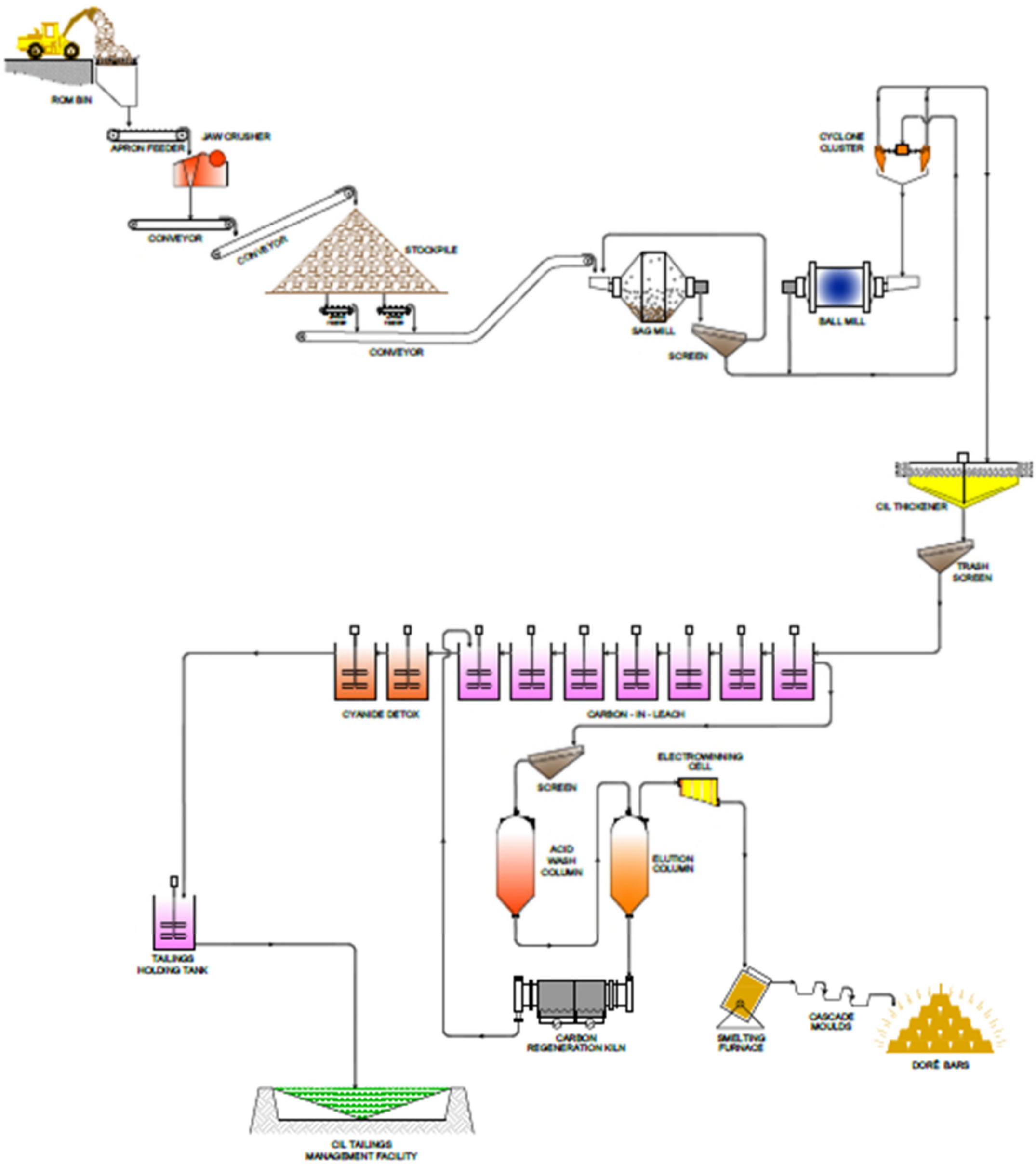


Figure 6.2: Generic Flowsheet for CIL

Ore from the ROM pad is blended as required and fed to the ROM bin, either by front-end loader or by direct tipping with trucks. The primary crusher is a jaw crusher designed to break up any large lumps of ore, although much of the fines is screened and removed prior to crushing, and the combined product is conveyed to a stockpile, normally providing about a day's worth of ore.

Ore is then fed via conveyor to the grinding circuit which, in this case, is likely to be a single variable speed Ball Mill, due to the soft oxide ore and low throughput, rather than the SAG Mill-Ball Mill combination shown (ideal for more competent ore at higher throughputs). The Ball Mill will operate in closed circuit with hydrocyclones for a product sizing of 80% passing 106 microns. The throughput is 93.8 dry t/h based on a typical plant availability of 91.3% and the design throughput of 750,000tpa. As discussed previously, there is no gravity circuit incorporated at this stage of the study.

Lime powder is added to the mill feed conveyor via a dedicated lime silo. Steel grinding media is added to the Ball Mill feed chute via overhead crane.

The average Ball Mill Work Index is 12.0kWh/t.

The cyclone overflow reports to a pre-leach thickener, as shown in the flowsheet, with the thickened underflow at approximately 50% solids reporting to a trash screen for the removal of wood chips, pieces of plastic etc prior to cyanide leaching. Alternatively, the pre-leach thickener can be removed and the cyclone overflow report direct to the trash screen. This obviously reduces capital costs and is often considered when higher pulp densities are not required. An advantage of a pre-leach thickener, as well as offering some buffer capacity, is the ability to separately control the grinding and leach circuit pulp densities, which often require different values for optimum performance.

The operating pH is approximately 10.5, controlled automatically by the lime addition to the mill feed belt.

The CIL circuit typically consists of six agitated adsorption tanks, where cyanide solution is added for an initial cyanide concentration of approximately 500 ppm in the first tank and for a total residence time of approximately 32 hours. Activated carbon is pumped counter-currently up the circuit with slurry and retained in each adsorption tank using inter-tank screens. Each adsorption tank contains approximately 10g/l carbon.

Slurry from the first adsorption tank is pumped to the loaded carbon screen and the washed loaded carbon reports to a feed hopper. The carbon is then initially batch acid washed with dilute hydrochloric acid and then eluted in a conventional pressure Zadra strip circuit (high temperature, high pressure using a caustic/cyanide solution). Barren carbon is regenerated in a kiln and returned to the last adsorption tank. Pregnant solution is passed to electrowinning cells and the loaded cathodes conventionally smelted for the production of gold bullion, which is then shipped to the contracted refinery.

Tailings slurry from the last adsorption tank passes over a safety screen to recover any carbon lost from holed inter-tank screens etc and is then pumped to a dedicated Tailings Storage Facility (TSF). Decant return water from the TSF is pumped back to the plant for process water requirements. Assuming cyanide levels in the final adsorption tank will be greater than 50 ppm, cyanide detoxification of the tailings slurry will be required.

A raw water pond or tank to receive the river abstraction water and a process water tank that will receive the TSF return water will be required. In addition, a reverse osmosis containerised potable water unit will be required to supply potable water.

6.4.2 Heap Leach (HL)

A simple schematic diagram for a generic heap leach operation is shown in Figure 6.3.

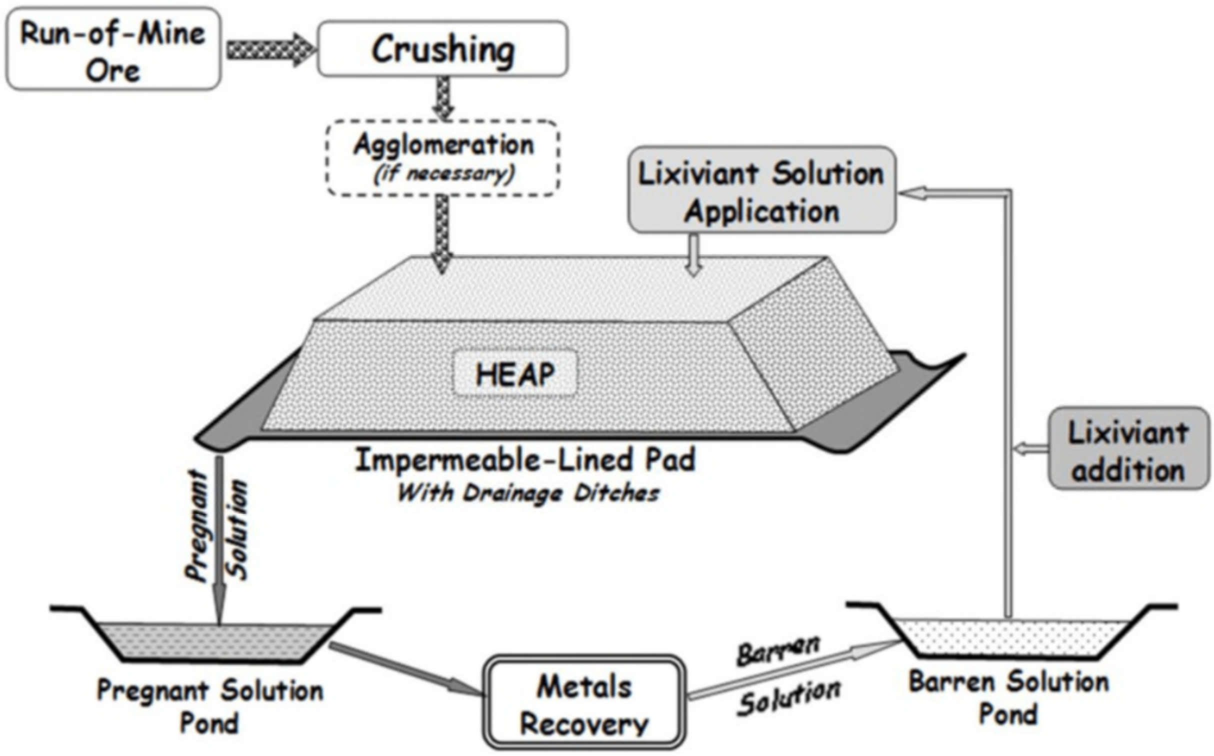


Figure 6.3: Generic Flowsheet of Heap Leach Operation

Heap leaching involves crushing the ore to the optimum size determined by testwork with agglomeration required using cement if there are significant clayey fines. In this case, an optimum crush size of approximately ½ inch is indicated from the testwork and agglomeration with cement is definitely required due to the high clayey fines content, likely exceeding 22.5kg/t. Due to the fine friable nature of the ore, an impact crusher is likely to be more practical than a jaw crusher.

The required cement is added from a storage silo direct onto the conveyor belt delivering crushed product to the agglomeration drum.

Agglomeration takes place in a dedicated agglomeration drum, rotating on rubber rollers and slightly inclined to the horizontal. Solution for agglomeration will be pumped from the barren pond, containing some cyanide which is beneficial for the agglomeration process. The moisture content of the produced agglomerates is a critical quality control measure.

The crushed and agglomerated ore is generally stacked using a system of conveyors, which can include overland conveyors, grasshopper conveyors, a horizontal feed conveyor, a horizontal index conveyor and the radial stacker itself.

After the ore is placed on the prepared pad and allowed to cure for 24 hours, it is leached with cyanide solution to recover the gold. The resulting pregnant solution gravitates to the base of the pad and exits via drain pipes to a lined solution channel from which it flows to the pregnant solution pond.

The pregnant solution is then pumped to the Carbon-In-Solution (CIC) circuit for recovery of gold onto activated carbon. The gold is then stripped from the loaded carbon using a conventional elution circuit, the carbon is regenerated and returned to the CIC circuit and the pregnant solution treated by electrowinning to recover the gold onto stainless steel cathodes. The gold sludge is then conventionally smelted to produce gold bullion. The barren solution from the CIC circuit gravitates to the barren solution pond for pumping back to the heaps as irrigation solution. Cyanide and sodium hydroxide are added to the barren solution pond as required to maintain the design pH and cyanide concentration.

The irrigation system generally uses Yelowmine or HDPE pipe with wobblers or drip emitters with pressure regulators fitted for irrigating the solution on a typical 6m x 6m pattern. The irrigation rate is typically $10l/h/m^2$.

In permanent heaps, the ore remains on the pad and the pads can either be extended or multi-lifts used to achieve the required capacity. Ideally, a nearly flat surface (typically 1%) is required, so some earthworks are usually needed, to allow gravity flow of pregnant solution to the base or sides of the heap, with the process ponds constructed downstream. An overflow/storage pond is also often required to allow for a sustained power outage (drain down of solution from the pads) and/or a heavy rainfall event.

The stages of pad construction typically involve the following: preliminary earthworks followed by a layer of compacted clay-rich soil; installation of leak detection perforated pipes; installation of the actual plastic liner, usually 1.5mm HDPE with sometimes a geotextile layer added above for protection; installation of the drain pipes, normally perforated flexible pipes and large collector pipes as required; and finally, installation of a gravel layer up to 1m thick, to provide protection and act as a permeable layer.

It is estimated that a total of four cells will be required, each accommodating 1Mt of stacked ore. Each cell will be divided into four pads, each pad holding 0.25Mt of ore. It is likely that each pad will consist of three lifts, each of 8m height.

The process pond construction is similar, with pregnant and barren solution ponds required, but normally a double layer of HDPE with leak detection between the two liners is required. In addition, a surge/event pond will be required to accommodate a storm event and/or power failure and drain down of the heaps.

6.5 Process Plant Capital & Operating Costs

As described within the trade-off study of Section 1.3, the base case capital cost for a 750,00tpa operation is approximately $61.4 million for CIL and $11.4 million for HL. The respective operating costs are $15.9/t and $10.3/t of ore.

In addition, for a 1Mtpa HL operation, the capital and operating costs are estimated as $12.3 million and $8.8/t respectively. For a 1.5Mtpa HL operation, the capital and operating costs are estimated as $12.9 million and $6.5/t of ore.

7 TAILINGS STORAGE FACILITY AND MAJOR INFRASTRUCTURE

For the CIL option, at scoping study accuracy (+/- 50%), the estimated cost of $61.4 million includes the dedicated Tailings Storage Facility (TSF). For the LOM storage capacity of 4.06 Mt and considering a conventional valley-fill tailings dam, the capital cost of the TSF is estimated at approximately $7.1 million. No TSF is required for the HL option.

For the CIL option, process water will primarily come from the TSF return water, but raw water make-up will likely be required, depending on the detailed site and process water balance (to be conducted at the next stage of study). Raw water will also be required for other areas of the site.

The source of the raw water will need to be determined, but this will most likely be from the nearby rivers and/or bore holes, if a suitable aquifer is present, and pumped to a raw water pond or tank. Depending on rainfall, run-off water can also be diverted and collected into the raw water pond. The site water balance will determine the size of raw water pond required for the design requirement.

The two main water sources available are the Fie and Niger rivers located approximately 3km and 6km respectively from site. The Niger river is the largest river. It is reported that a maximum 3% abstraction rate is permissible without a permit. Therefore, allowance must be made for a pipeline and pumping station to pump to the Raw Water Pond.

For the HL option, the site water balance will again determine the amount of overall make-up water required, allowing for precipitation and evaporation and lock-up of water within the heaps (some is released on drain down). The raw water make-up would be added to the barren solution pond. However, a surge/event pond would also be required.

Regarding access roads, there is an existing tarred road from Bamako to Selingue for about 130km. There is then a laterite road from Selingue to Selefougou for about 15km and with two bridges encountered. From Selefougou to the site, the laterite road continues for another 15km although the condition here is reportedly very poor. Therefore, allowance must be made for upgrading approximately 30km of laterite road and the accompanying two bridges.

A site camp will be required. The total labour complement, depending on the process route selected, will be approximately 94, of which 36 will be permanently based in the camp and 58 supplied and transported from two local villages, located within approximately 4km from site. Therefore, allowance should be made for a site camp to accommodate approximately 36 people.

Power will most likely be supplied from a dedicated power station using HFO or diesel generators, rather than national grid, due to the remote area. The nearest power source is actually a Selingue hydro power station about 30km from site with a reported capacity of 46 MW. However, Cora Gold report that this is unlikely to be available for site use. Therefore, rented diesel generators are the most likely option. Power costs at the nearby Yanfolila operation are reported as 21 c/kWh. Therefore, allowance must be made for the cost of installation and rental of gensets.

Transport of the gold bullion smelted on site will normally be the responsibility of the refinery once it has left the goldroom.

Cora Gold has provided two photographs, showing a prospective site for the plant (HL or CIL) and also a valley for a prospective TSF (CIL).

Photo 7.1: Prospective Site for Processing Plant

The prospective site for the process plant appears to indicate a relatively flat and level surface. Pending geotechnical investigations, this looks potentially ideal for either the HL or CIL option, with likely a minimum of earthworks required, hence significantly reducing capital costs.

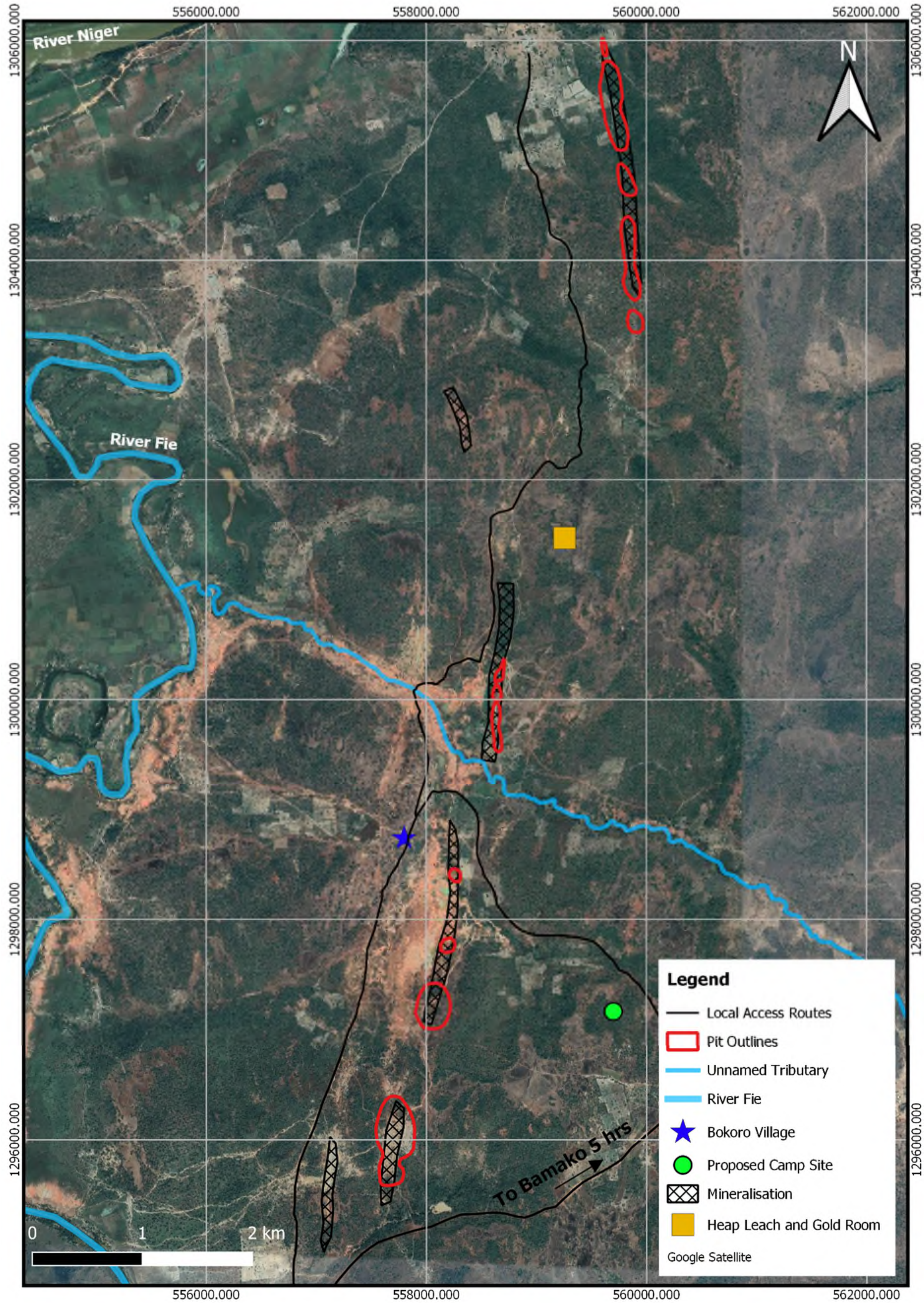


Figure 7.1: Prospective Site Plan and Infrastructure

Photo 7.2: Prospective Valley Site for a TSF

The prospective site for the TSF is not so clear from the photograph, but clearly some earthworks and bush scrubbing would be required pending geotechnical investigations to determine suitability for a conventional valley-filled TSF, not required for the HL option.

Figure 7.1 shows a schematic diagram of the site area with location of the two rivers and the nearby hydro power station.

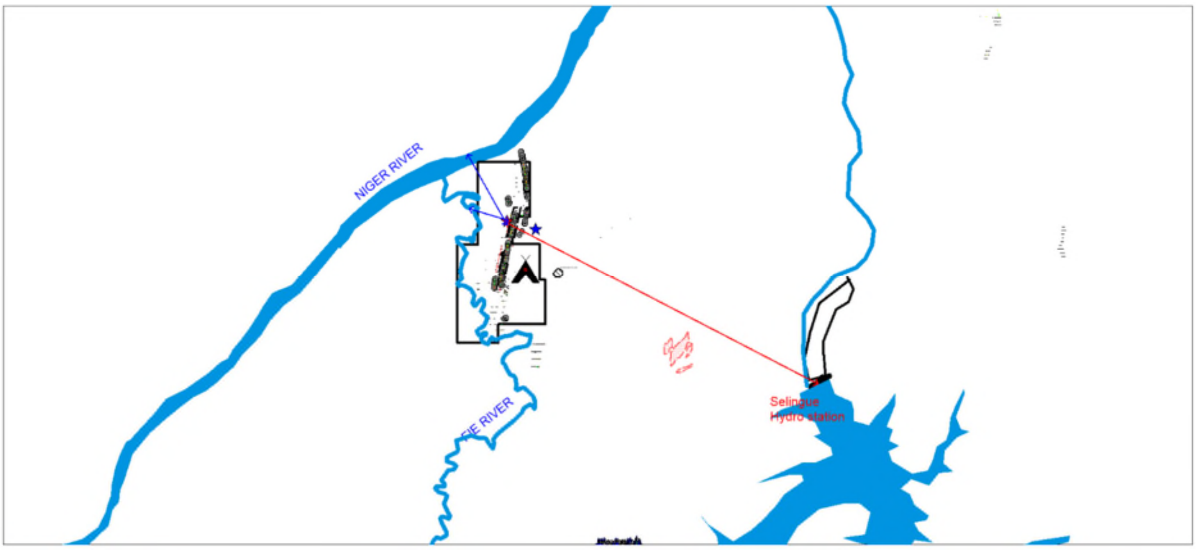


Figure 7.1: Site Location

7.1.1 Infrastructure Capital Costs

It is assumed that the water pumping station will be installed on the Fie River with 3km of pipeline required. Factoring the cost from a similar project, the capital cost is estimated at **$700,000**.

The capital cost for upgrading the 30km of laterite roads and bridges is estimated at **$2.5 million**, factored from a similar project.

The capital cost for a site camp to accommodate 36 people is estimated at **$300,000**.

It is understood that Cora Gold intends to rent the required generators to supply on-site power. It is assumed that the rental power cost is $0.21/kWh. As the power cost is already included in the process plant operating cost estimates, an additional allowance is therefore assumed for the proposed rental of the generators, roughly estimated at $0.14/kWh. For an assumed installed power of approximately 3 MW for CIL and 700 kW for heap leach and an average 80% duty, then the approximate additional annual allowance is **$2.9 million** and **$0.7 million** respectively.

7.2 Conclusions and Recommendations

The oxide ore samples tested are very amenable to conventional cyanide processing (CIL) with an average whole ore leach recovery of **93.5%**. For the heap leach (HL) option, the coarse ore bottle roll tests indicated recoveries approaching 90% at the coarser size fractions, although the column test result using 22.5kg/t cement only produced a recovery of 55% after 90 days of leaching, although recovery was clearly continuing at the end of the test and with some evidence that more cement is required. Therefore, a conservative recovery of **70%** has been assumed with the potential for higher recovery once further optimised column tests can be conducted.

The indicated process capital and operating costs for the 0.75Mtpa CIL and HL options are **$61.4 million/$15.9/t** and **$11.4 million/$10.3/t**. In addition, for a 1Mtpa HL operation, the capital and operating costs are estimated as **$12.3 million** and **$8.8/t** respectively.

For both options, the additional infrastructure capital costs are **$0.7 million** for the water abstraction pump and pipeline, **$2.5 million** for upgrading 30km of laterite road and **$0.3 million** for the site camp. For the CIL option, an additional allowance of **$2.9 million per annum** should be allowed for the rental of the gensets (assume 3 x 1MW units). For the heap leach options, an additional allowance of **$0.7 million per annum** should be allowed, assuming 1 x 1MW unit.

Based on these results and a 750,000 tpa operation for a LOM of 5.4 years (4.06Mt mineable resource at 1.48g/t Au), heap leaching provides overall the most economic processing route at significantly lower capital cost.

Recommendations for further testwork include variability testing for both the CIL and HL options. The priority is to conduct further optimised column tests to confirm that recoveries of 70% or higher can be achieved and the optimum cement addition required for agglomeration.

For the CIL option, variability testwork is required to optimise the grind, cyanide concentration and leach residence time and, in particular, the effect of oxygen sparging compared to air sparging. Further gravity testwork is also required to determine if a gravity circuit is economically beneficial and to confirm the poor intensive leach recoveries on the gravity concentrates from the carbon phyllite and zone A+B samples.

Geotechnical investigations are required for the prospective HL pads and ponds (or CIL plant and associated TSF and infrastructure).

A detailed site and process water balance is required for the selected option.

8 ENVIRONMENTAL AND SOCIAL

The Environmental and Social chapter of this Scoping Study has been carried out by Digby Wells Environmental and can be found in **Appendix 4 -ENVIRONMENTAL AND SOCIAL SCOPING STUDY FOR THE SANANKORO GOLD PROSPECT SCOPING REPORT -** of this report. WAI has not undertaken a review of this section and cannot be held liable for the contents therein.

9 ECONOMIC ANALYSIS

9.1 Key Project Assumptions

WAI has performed a preliminary economic assessment of the Cora Gold open pit operations using a Discounted Cash Flow (DCF) analysis to assess the following options of the project further development at 1Mtpa (Base Case), 750ktpa (Option 1) and 1.5Mtpa (Option 2).

All options have been considered with HL processing route, assuming 70% total recovery throughout mine life.

Summary of the key inputs and assumptions is outlined in Table 9.1 below.

Table 9.1: Summary of the Key Inputs and Assumptions

		1Mtpa	0.75Mtpa	1.5Mtpa
Mining Opex				
Mining Cost - Ore	S/t ore	3.43	4.08	3.02
Mining Cost - Waste	S/t waste	3.43	4.08	3.02
Processing Opex (HL only)	S/t ore	8.8	10.3	6.5
Reclamation	Not included in the valuation *			
G&A Cost	US$/t	1.5	1.5	1.5
Other Revenue / Expense	US$'000	0	0	0
Mining Capex (contractor)	US$'000	**2,600**	**2,410**	**3,500**
Equipment Mobilization & Establish Site Facilities	US$'000	1,700	1,576	1,900
Equipment Demobilization & Disestablish Site Facilities	US$'000	0 *	0*	0 *
Miscellaneous	US$'000	900	834	900
Contractor Premium	US$'000	0	0	700
Contingency	US$'000	0	0	0
Processing Capex	US$'000	12,300	11,400	12,900
Infrastructure capex for all options				
Water abstraction System	US$'000	700		
Access Roads	US$'000	2,500		
Site Camp	US$'000	300		
Power Rental of $700 thousands per year	US$'000	700		
Total Pre-Production Capital Cost	**US$'000**	19,100	18,010	20,600
Sustaining Capital Cost	US$'000	3,031	3,855	2,123
Total Processing Recovery Rate	%	70%		

Note:
** Due to the large Exploration Target at Sanankoro the life of mine ('LoM') is expected to be extended beyond that which is modelled here and as such Closure & Reclamation Fees in the region of US$4.2m as well as demobilisation costs have not been factored into this model'*

All-In Sustaining Cost (AISC) details are shown Table 9.2 below

Table 9.2: All In Sustaining Costs (AISC) Comparative Analysis between Various Production Rates

Production Rate Option	**1Mtpa** **1**	**750 ktpa** **2**	**1.5 Mtpa** **3**
Total Mining Operating Cost	715	849	634
Total Processing Cost	267	312	200
G&A Costs	45	45	46
Mining Royalty	42	42	42
Closure & Reclamation Fees	0	0	0
Realisation Costs (incl insurance and freight)	5	5	5
Sustaining Capital Costs	22	28	15
AISC	**1,096**	**1,282**	**942**

Production Schedules for 1Mtpa and 1.5Mtpa rates have been developed by SRK and included in the financial model for further analysis. WAI has also added a lower production rate of 0.75Mtpa to complete the analysis. WAI notes that 0.75Mtpa scenario schedule has been drafted based on the overall tons and grade, rather than optimisation results and serves for indicative purposes only.

Table 9.3 below provides three mining schedules compared between various production rates.

Table 9.3: Cora Gold Production Schedules Considered in the Financial Analysis

1Mtpa			2020	2021	2022	2023	2024	2025	2026
Rock	t	**28,390,889**		6,669,292	5,275,738	7,626,294	7,684,048	1,135,517	
Strip Ratio	W:O	5.87		7.33	4.27	6.63	6.69	2.44	
Waste	t	**24,260,238**		5,869,041	4,275,597	6,626,232	6,684,224	805,145	
Total Ore	t	**4,130,658**		800,251	1,000,163	1,000,035	999,839	330,370	
Au grade	g/t	1.47		1.59	1.62	1.17	1.36	1.91	
Au Mined	tr oz	**194,690**		**40,795**	**52,163**	**37,763**	**43,666**	**20,304**	
Au Recovery	%	70.00%		70%	70.00%	70.00%	70.00%	70.00%	
Au recovered	tr oz	**136,283**		28,557	36,514	26,434	30,566	14,213	
0.75 Mtpa			2020	2021	2022	2023	2024	2025	2026
Rock	t	28,390,839		5,154,900	5,154,900	5,154,900	5,154,900	5,154,900	2,616,339
Waste	t	24,260,181		4,404,900	4,404,900	4,404,900	4,404,900	4,404,900	2,235,681
Total Ore	t	4,130,658		750,000	750,000	750,000	750,000	750,000	380,658
Au grade	g/t	1.47		1.47	1.47	1.47	1.47	1.47	1.47
Au Mined	tr oz	**194,690**		**35,350**	**35,350**	**35,350**	**35,350**	**35,350**	**17,942**
Au Recovery	%	70.00%		70%	70.00%	70.00%	70.00%	70.00%	70.00%
Au recovered	tr oz	**136,283**		24,745	24,745	24,745	24,745	24,745	12,559
1.5Mtpa			2020	2021	2022	2023	2024	2025	2026
Rock	t	**29,078,119**		8,105,654	9,747,912	11,118,128	106,425		
Waste	t	**24,828,943**		6,905,281	8,247,867	9,617,945	57,850		
Total Ore	t	**4,249,086**		1,200,313	1,499,994	1,500,202	48,577		
Au grade	g/t	1.45		1.68	1.28	1.43	1.40		
Au Mined	tr oz	**197,753**		**64,819**	**61,754**	**68,994**	**2,186**		
Au Recovery	%	70.00%							
Au recovered	tr oz	**138,427**		45,373	43,228	48,296	1,530		

9.2 Discounted Cash Flow Model

9.2.1 Base Case Results

A discounted cash flow model has been developed in real US$ as of January 2020 with no escalation taken into account.

A long term fixed gold price was assumed at US$1,400/oz (base case).

Post-tax Net Present Value has been assumed at a base case 8% discount rate.

Corporate Income Tax was applied at 30%. Tax holiday has been included in the valuation following instruction from the Client. The tax basis has been estimated following a full payback of the initial capital cost plus additional sunk cost of US$8.2M spent for project exploration. Being sunk costs by nature, these costs have not formed part of the project capital costs. But have only been considered when calculating tax relief basis for corporate income tax.

Mining royalty rate at 3% of the project gross revenue was included in valuation.

Working capital has assumed 45 days for accounts payable and 30 days for accounts receivable and inventories.

Selling and realisation cost has been assumed by WAI at US$5/oz to cover potential cost associated with final product delivery to final customer.

A summary of the Cora Gold project cash flow analysis results compared between three options is provided in Table 9.4 below.

Project economics gain from the economy of scale and provides better NPV results at an increased production rate of 1Mtpa and 1.5Mtpa. Scenario with 0.75Mtpa production capacity resulted in negative NPV using US$1,400/oz gold price.

Therefore, WAI recommends 1.5Mtpa option to be taken forward for further studies.

Table 9.4: Cora Gold Financial Results Comparison (US$ M)			
Production Rates	**1Mtpa**	**0.75Mtpa**	**1.5Mtpa**
	1	2	3
Gross Revenue	**190.8**	**190.8**	**193.8**
Total Mining Operating Cost	97.4	115.7	87.8
Total Processing Cost	36.3	42.5	27.6
Production Costs	**133.7**	**158.3**	**115.4**
Marginal Revenue	**57.1**	**32.5**	**78.4**
General and Administrative Cost	6.2	6.2	6.4
Realisation Cost	0.7	0.7	0.7
Mining Royalty	5.7	5.7	5.8
Closure and Reclamation Fees	0.0	0.0	0.0
EBITDA	**44.5**	**19.9**	**65.5**
Pre-Production Capex	**19.1**	**18.0**	**20.6**
Sustaining Capex	3.0	3.9	2.1
Pre-Tax Free Cash Flow	**22.3**	**-1.9**	**42.8**
Pre-Tax NPV	**12.8**	**-5.5**	**30.9**
IRR	**32%**	**-3%**	**84%**
CIT	***3.4***	***0.0***	***8.9***
Post-Tax NPV	**7.4**	**-7.3**	**23.5**
IRR	**30%**	**-3%**	**73%**
AISC (LOM), US$/oz	1,096	1,282	942

Table 9.5 below provides comparative analysis of the project results operating at 1.5Mtpa production rate:

Table 9.5: Key Project Summary Result for the Selected Production rate of 1.5Mtpa			
	1,400	1,500	1,300
Gross Revenue	**193.8**	**207.6**	**180.0**
Total Mining Operating Cost	87.8	87.8	87.8
Total Processing Cost	27.6	27.6	27.6
Production Costs	115.4	115.4	115.4
Marginal Revenue	**78.4**	**92.2**	**64.5**
General and Administrative Cost	6.4	6.4	6.4
Realisation Cost	0.7	0.7	0.7
Mining Royalty	5.8	6.2	5.4
Closure and Reclamation Fees	0.0	0.0	0.0
EBITDA	65.5	78.9	52.1
Pre-Production Capex	**20.6**	**20.6**	**20.6**
Sustaining Capex	2.1	2.1	2.1
Free Cash Flow	**42.8**	**56.2**	**29.3**
Pre-Tax NPV	**30.9**	**41.5**	**20.4**
IRR	**84%**	**107%**	**60%**
CIT	***8.9***	***12.9***	***5.0***
Post-Tax NPV	**23.5**	**32.2**	**14.7**
IRR	**73%**	**91%**	**53%**
AISC (LOM)	942	945	939

9.2.2 Sensitivity Analysis

A sensitivity analysis was performed on several key parameters within the financial model to assess the impact of changes upon the Net Present Value of the project (at 8% discount rate). These parameters are as follows:

- Au Price;
- HL recovery rate;
- Project Operating Costs; and
- Project Capital Costs.

Metal prices, recovery rates and head grades form revenue cash flows of any mineral project, therefore known to be the most influential parameters.

Figure 9.1 to Figure 9.3 below demonstrate project behaviour at various Au prices and HL recovery rates within +/-25% range.

Project break-even gold prices and recovery rates are as following:

- Option 1Mtpa: recovery rate drops below 63%, break-even price of US$1,271/oz.
- Option 0.75Mtpa: recovery rate below 74%, break-even price of US$1,458/oz; and
- Option 1.5Mtpa: recovery rate below 56%, break-even price of US$1,108/oz.

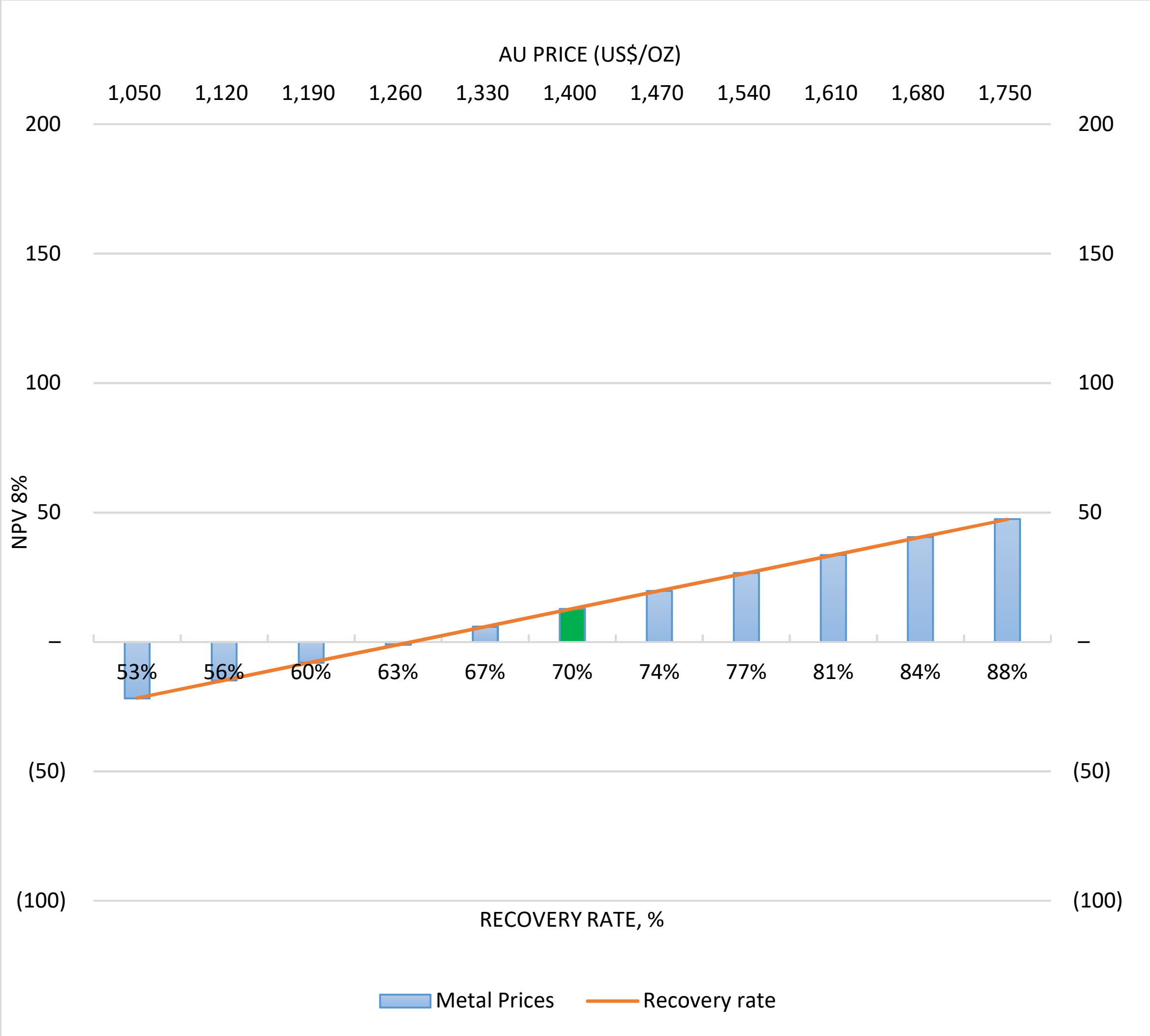


Figure 9.1: Project Sensitivity Analysis to Change in Gold Price: Option 1 (1Mtpa)

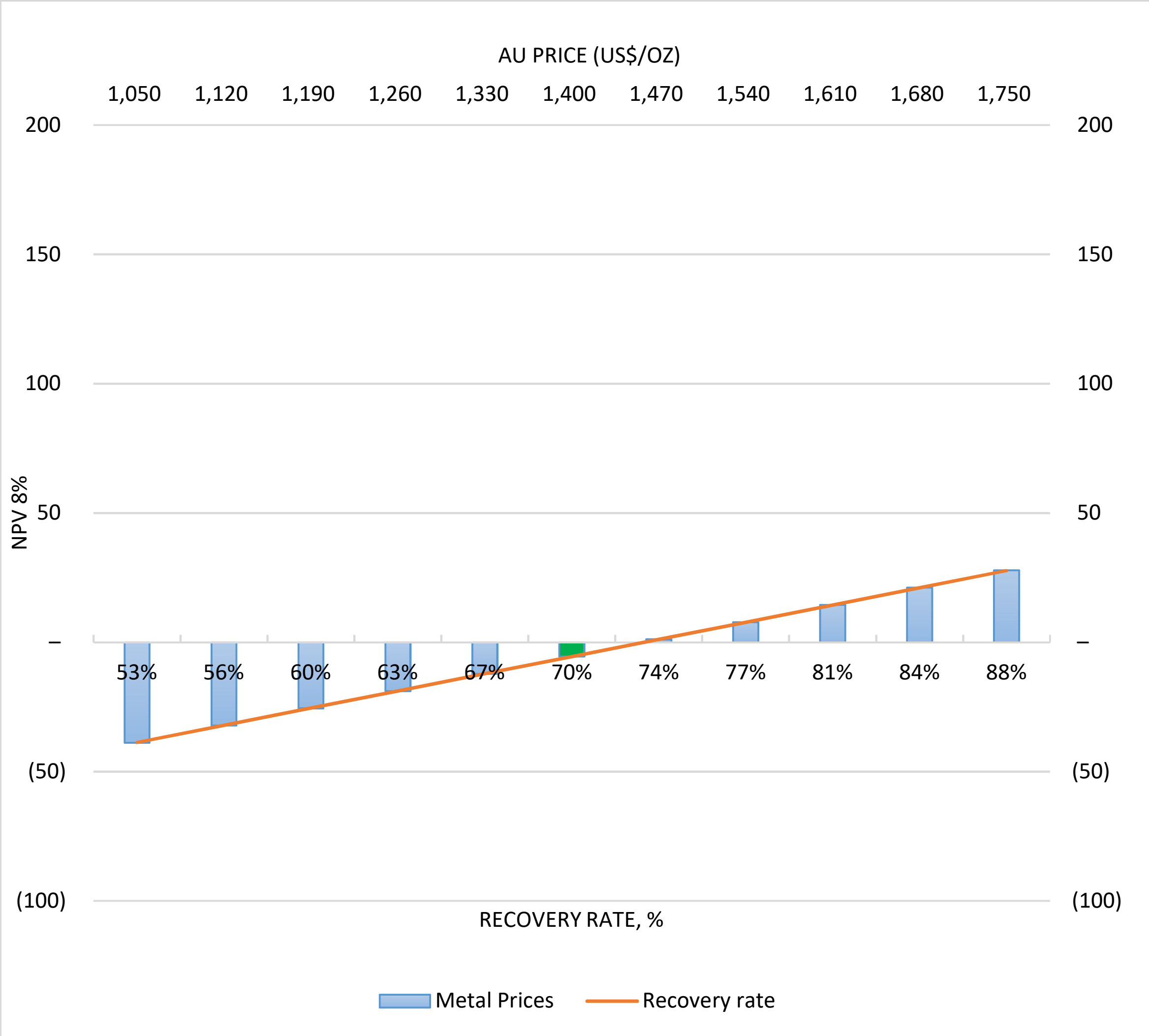


Figure 9.2: Project Sensitivity Analysis to Change in Gold Price: Option 2 (0.75Mtpa)

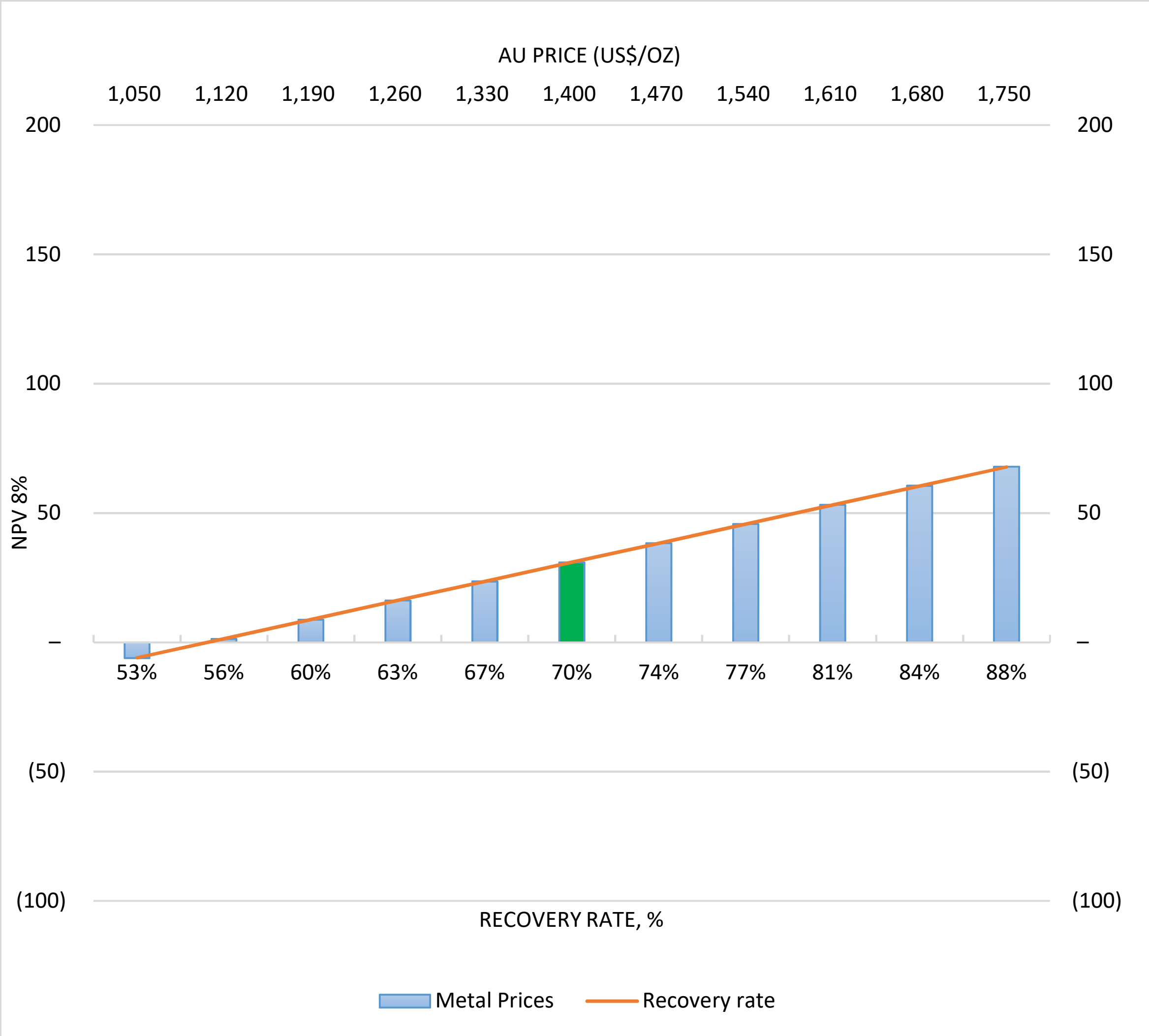


Figure 9.3: Project Sensitivity Analysis to Change in Gold Price: Option 3 (1.5Mtpa)

Figure 9.4 to Figure 9.6 below show the project sensitivity to change in project costs alongside Au price and HL recovery rate. The latter two coincide in each chart as both form revenue streams and have equal impact of the project value being changed within identical range.

Proportionally, mining operating costs have been estimated within 45%-60% of the project gross revenue (depending of the selected scenario). While HL processing opex has been estimated within 14% and 22% of the gross revenue. Therefore, following HL processing route Cora gold project is mostly sensitive to change in mining operating costs rather than processing costs. This is due to high stripping ratio.

Cora gold project is less sensitive to change in capital costs given that total capex is only 2% of the project gross revenue.

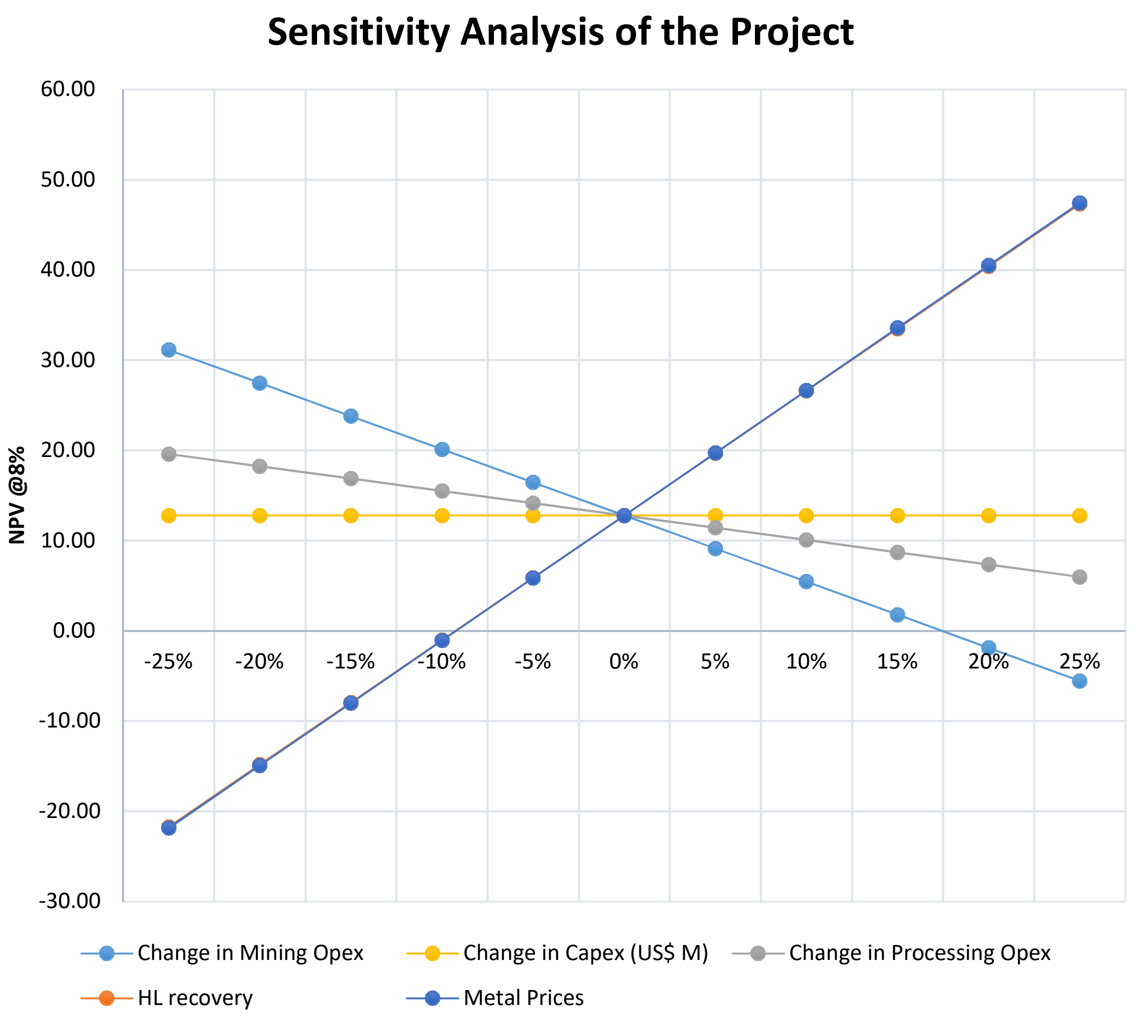


Figure 9.4: Project Sensitivity Analysis to Change in Operating and Capital Costs: Option 1 (1Mtpa)

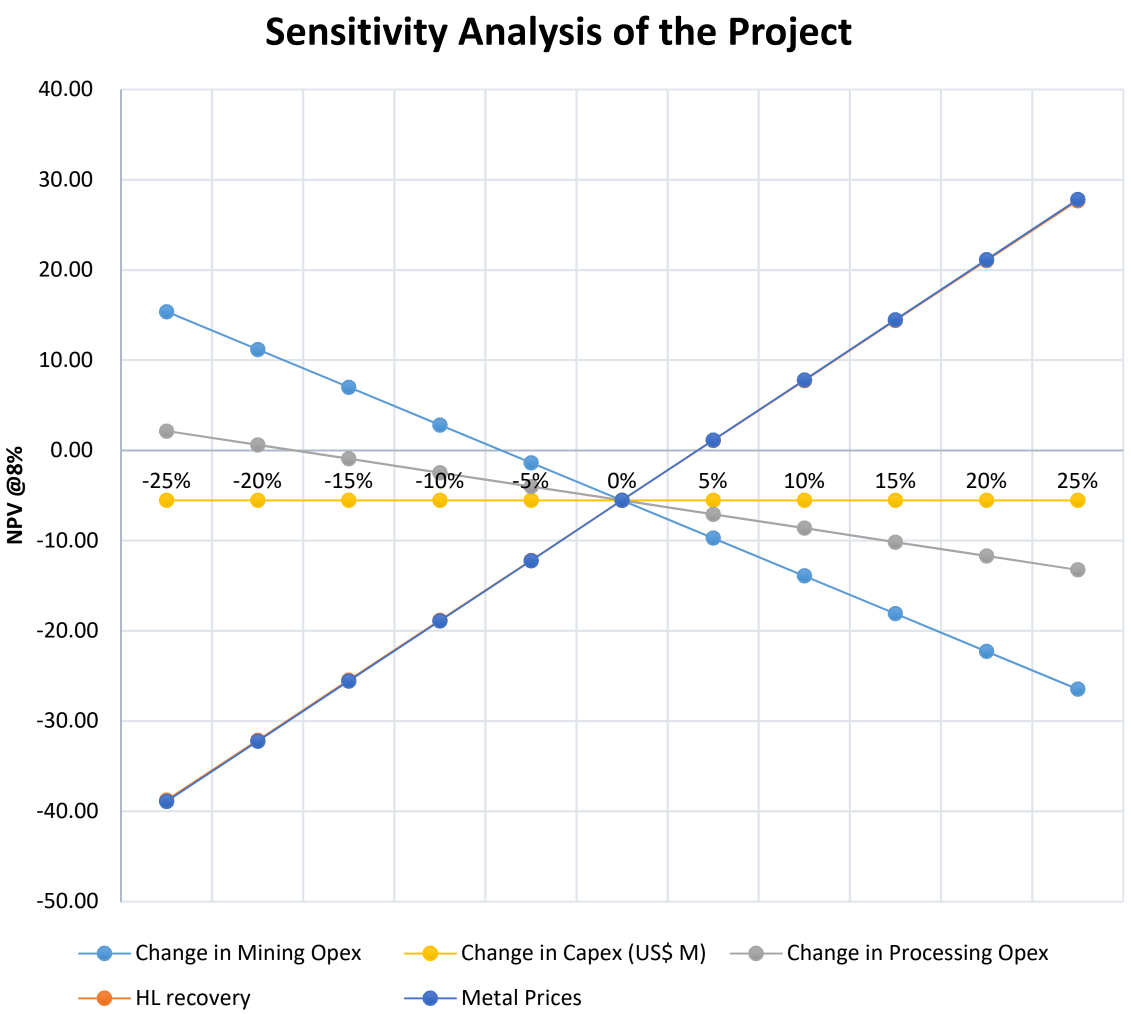


Figure 9.5: Project Sensitivity Analysis to Change in Operating and Capital Costs: Option 2 (0.75Mtpa)

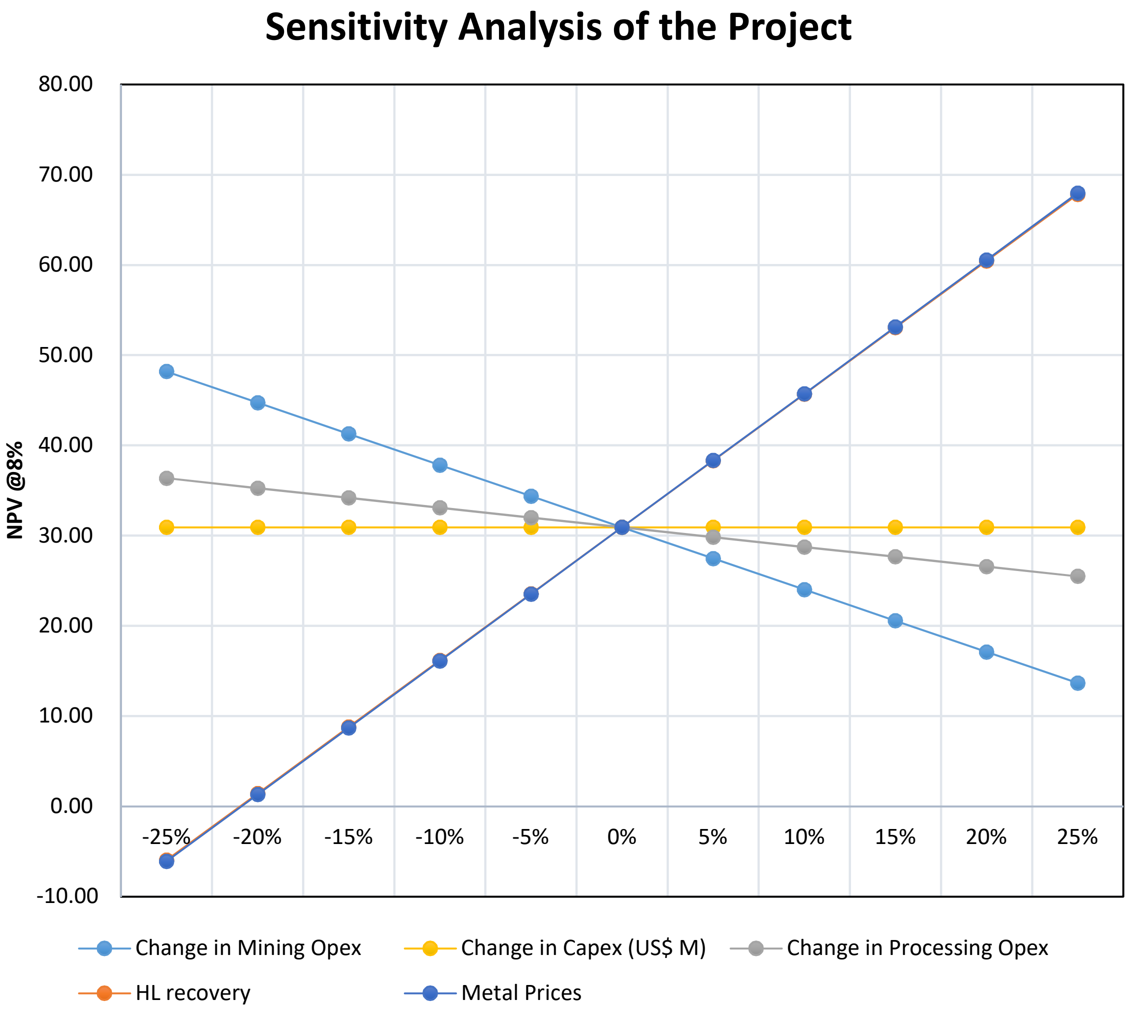


Figure 9.6: Project Sensitivity Analysis to Change in Operating and Capital Costs: Option 3 (1.5Mtpa)

10 CONCLUSIONS AND RECOMMENDATIONS

10.1 Mineral Resource Estimate

SRK has derived a Maiden Inferred Mineral Resource Estimate for the Sanankoro Project of 5.0Mt at 1.6g/t Au. The resource is primarily restricted to saprolitic oxide mineralisation, with a small contribution from fresh sulphide mineralisation. The Mineral Resource has been defined within an area tested by a reasonable volume and density of RC, AC, RAB and diamond drilling. A number of additional targets have been identified by regional exploration drilling, geochemical soil and termite sampling and mapping of artisanal outlines, which have been used to derive an Exploration Target, within which the Inferred Mineral Resource sits, of between 30Mt and 50Mt, at a grade of between 1.0g/t and 1.3g/t. Given the relatively small thickness and steep dip of the mineralised zones, the depth of the optimised pits used to constrain the Mineral Resource is limited by excessive stripping ratios at depth. SRK therefore consider that the main potential for defining additional Mineral Resources is outside of the zones that have been the primary focus of drilling to date. SRK would recommend that a key focus of the next phase of exploration would be drilling of the most prospective exploration target zones.

10.2 Mining

The dilution in all zones is estimated between 14% and 20% and recovery between 91% and 95% based on a regularised block model to a selected SMU of 2.5 x 2.5 x 5m and 0.4g/t Au cut-off grade. Based on the pit optimisation results, there are no visible or significant step changes in sensitivity in tonnes or grade around the considered prices however, fairly linear increase in the ore inventory is relatively steep. The Client and SRK selected the USD1,500/oz Au pit shell (RF=100%) for developing the mine design and strategic schedule.

The production rate of each schedule directly effects the length of the life of mine (LoM), material movement and equipment/personnel requirements. Three cases were considered. Case 1 at a production rate of 0.5Mtpa can be achieved at total mining rate up to 4.3Mtpa and contains the most balanced material movement. Case 2 and Case 3 at a production rate of 1Mpta and 1.5Mpta respectively, contain a total mining rate ranging between 5.0Mpta and 11Mpta. All cases require the same ore mining fleet that includes a minimum of 2 x 1.9m^3 excavators, 2 x 24t trucks and 1 x 140mm production drill. The waste equipment size is dependent on the production rate of each case. Relatively long distances and resulting cycle times were observed for the ore haulage, due to the plant location and number of pits located around it. It is expected that the extraction method will be predominately free digging where drill and blast will only be required in the saprock weathering domain.

This Mining Section of the Scoping Study for Sanankoro has been completed by SRK while the main Report of the Scoping Study has been developed by Wardell Armstrong consultants. From this perspective, SRK has not identified any fatal flaws at the scoping level, however SRK is not able to comment on the overall project potential as it has not been involved in developing the Technical Economic Model ("TEM").

10.2.1 Recommendations

Based on the work undertaken as part of the Scoping Study, SRK makes the following mining study related recommendations:

- Mining operating costs should be confirmed through obtaining mining contractor costs, using the strategic schedule and equipment sizing and estimated requirements from the Scoping Study;
- Open pit optimisation and pit shell selection should be repeated when material changes to the geological model or operating / processing assumptions occur;
- Run trade-off studies with civil type of trucks hauling ore to the plant and consult tyre manufacturers regarding the tyres for high temperature environment;
- Verify potential for the fresh material;
- This Mining Section of the Scoping Study for Sanankoro has been completed by SRK, while the main Report of the Scoping Study has been developed by Wardell Armstrong consultants. SRK recommends using the annual cost estimate in the TEM for the entire Project; and
- It is recommended by SRK that this scoping level mine planning should be updated when new geological information and an updated block model is available. Based on the results of the updated scoping level mine planning, if the Project is economically favourable, more detailed study (to Pre-Feasibility Level) should be undertaken to determine the mining parameters, costs and operating strategy at an increased level of detail.

10.3 Geotechnical

For the small sections of saprock and fresh material exposed at the toe of the slopes, SRK recommended 40° and 42° slope angles respectively. To verify the proposed Scoping level slope angles at the next project stage, geotechnical drilling, logging and sampling will be required in addition to hydrogeological testing to determine the susceptibility of the saprolite to slope depressurisation programmes.

10.4 Hydrology and Hydrogeology

The key hydrological risks identified relate to high intensity rainfall events resulting in either direct flooding of the pits or indirect recharging of the pit slope pore pressures; these risks should be quantified at PFS level following the installation of a site weather station and river flow gauges. The key hydrogeological risk for the project is the inability for the saprolite to remain depressurised; the hydrogeological system requires testing and conceptualisation in order to assess expected pore pressure responses to both climate and mining events. This assessment requires the establish of groundwater level monitoring and hydraulic testing within the key hydrogeological units.

10.5 Metallurgy

10.5.1 Conclusions

Phase 1 Testwork

Head Assay

- Two samples of gold mineralisation from the Sanankoro deposit, Mali were received for testing;
- Head assay of the samples showed the SD0005 sample to contain 0.61ppm Au whilst the SD0006 sample contained 3.35ppm Au based on analysis by Fire Assay. Silver grades were <0.5ppm Ag and 1.2ppm Ag respectively;
- Base metal levels within the samples were low with copper values of 0.005 – 0.008% Cu, lead values of 0.002 – 0.003% Pb and zinc values of 0.004 – 0.010% Zn; and
- Total sulphur levels ranged from 0.022 – 0.046% S(TOT) whilst total carbon levels ranged from 0.027 – 0.13% C(TOT).

Coarse Ore Bottle Roll Testing

- Coarse ore bottle roll testing to investigate the effect of crush size on leach response showed gold recoveries ranging from 78.4% (-20.0mm) to 97.6% (-6.3mm) for the SD0005 sample and 66.8% (-20.0mm) to 93.1% (-6.3mm) for the SD0006 sample;
- Silver recoveries ranged from 14-8 – 21.9% (SD0005) and 35.1 – 47.9% (SD0006);
- Analysis of the leach kinetics showed continued gold recovery from the SD0005 samples throughout the duration of testing whereas results for the SD0006 sample showed gold recoveries to have reached maximum by around the tenth day of leaching;
- Cyanide consumption in the tests conducted on the SD0005 sample averaged 0.56kg/t and were lower than those of the SD0006 sample which averaged 0.65kg/t. This increase in cyanide consumption is likely due to the higher gold content in the SD0006 sample;
- Particle size analysis of the bottle roll feed showed all of the samples to have a considerably finer particle size distribution than that intended based on the target crush size. For the SD0005 sample, D80 values ranged from 1.28mm (-20.0mm) to 0.46mm (-6.3mm) whilst for the SD0006 sample, D80 values ranged from 6.40mm
- (-20.0mm) to 1.75mm (-6.3mm);
- Subsequent particle size analysis of the bottle roll leach residues showed a reduction in particle size distribution for all of the samples tested indicating a breakup of

material during leaching. For the SD0005 sample, D80 values were observed to have reduced to 0.31 – 0.29mm whilst for the SD0006 sample, D80 values reduced to 0.74 – 0.48mm; and

- Calculation of the gold recovery by particle size showed a general trend for increased gold recovery by reduced particle size, up to 98% in selected size fractions, as would be expected given improved liberation within the finer size classes. Some discrepancies were observed with certain size fractions indicating negative gold recoveries however, this is most likely due to the change in the particle size distribution of the material during testing as previously mentioned.

Whole Ore Leach Testing

- Whole ore leach testing showed that gold recoveries of 97.4% and 96.7% could be achieved from the SD0005 and SD0006 samples respectively following 48 hours of leaching using a 2.0g/L cyanide solution; and
- Cyanide consumption during both tests was broadly similar, averaging 1.43kg/t.

Phase 2 Testwork

Head Assay

- Head assay of the Master Composite, which had been prepared by blending the SD0005 and SD0006 samples, showed it to contain 2.74ppm Au and 0.8ppm Ag.

Gravity Leach Testing

- Gravity testing using a Knelson centrifugal concentrator showed that 73.0% of the gold could be recovered from the Master Composite to a grade of 282ppm Au following two stages of processing;
- Of the gold recovered by gravity, over two thirds (50.7%) was recovered during the first stage of processing which had been performed at a grind size of 80% passing 212µm;
- Subsequent cyanide leaching of the gravity tailings showed that 94.1% of the gold remaining in the tailings could be recovered after 48 hours of leaching using a 1.0g/L cyanide solution. If the cyanide concentration was reduced to 0.5g/L, gold recovery fell slightly to 92.1%; and
- The combined recovery of gold to the gravity concentrate and gravity tailings leach solution was 98.4%. If it is assumed that 98% of the gold in the gravity concentrate can be successfully leached, overall gold recovery to the combined PLS would be in the order of 96.9%.

Whole Ore Leach Testing

- Testing of the Master Composite to investigate the effect of grind size on leach response showed gold recoveries to range from 95.0% at a grind size of 80% passing 150µm to 98.0% at grind sizes of 80% passing 125µm and 75µm. Silver recoveries ranged from 53.6% to 62.4%;
- Further testing to investigate the effect of cyanide concentration on leach response showed a consistent reduction in gold recovery with reduced cyanide concentration with values falling from 98.0% at 1.0g/L to 92.9% at 0.25g/L. No such trend was observed for silver with recoveries ranging from 56.6% (0.5g/L) to 62.4% (0.25g/L); and
- When compared against the results of the gravity-leach testing, the results indicate that higher overall gold recoveries could potentially be achieved through direct leaching of the ore without the inclusion of a gravity preconcentration stage. Further testing would however, be required to verify this conclusion.

Agglomeration & Percolation Testing

- Percolation testing showed the un-agglomerated Master Composite material to have exceptionally poor drainage characteristics with an average drainage flowrate of just 25l/m^2/hr;
- Testing of material which had been agglomerated with cement showed that a cement dosage of 22.5kg/t was required to achieve an average drainage flowrate of 12,794l/m^2/hr, above the target value of 10,000l/m^2/hr; and
- Based on the agglomeration testing, a cement dosage of approximately 19kg/t would be required to achieve the target drainage flowrate with the high cement requirement likely due to a combination of the nature of the material and the high degree of fines present in the ore.

Column Leach Testing

- Column leach testing of the Master Composite at a nominal crush size of -20.0mm achieved a gold recovery of 56.0% to the PLS after 95 days of irrigation. Overall recovery of gold to a sample of loaded carbon was 56.3%;
- Silver recovery to the PLS was 37.0% increasing to 42.9% to the loaded carbon with the increase likely attributable to the low grades of silver within the PLS resulting in some analytical variance;
- The leach kinetics for both metals showed that they continued to leach through the duration of the test and had not reached equilibrium by the time the test was stopped. It is therefore highly probable that higher gold and silver recoveries would have been achieved had the test been allowed to continue; and

- Overall, the amount of gold recovered from the column leach test was approximately 16.6% lower than that which had been indicated by the previous coarse ore bottle roll tests. This reduction in recovery may be due to the lack of material breakup (attrition) during static column testing.

10.5.2 Recommendations

Based on the testing that has been completed, Wardell Armstrong International would recommend that the following additional testwork be considered:

- Further gravity-leach testing to both confirm the amount of gold that can be recovered from the Sanankoro material by means of gravity processing and to quantify the amount of gold that can be recovered from the gravity concentrate by cyanide leaching;
- Comminution testing to determine ore hardness and energy requirements to grind the Sanankoro material to sizes suitable for whole ore / gravity-leach processing;
- Variability testing to investigate any variation in gravity and/or leach response across the deposit;
- Additional coarse ore bottle roll and column leach testing to further investigate the gold recoveries achievable through heap leaching of the ore;
- Dewatering testing to investigate the settling characteristics of the Sanankoro material post-leaching; and
- Environmental testing to characterise tailings material with respect to standard parameters such as metal solubility and acid rock drainage potential.

10.6 Processing

The oxide ore samples tested are very amenable to conventional cyanide processing (CIL) with an average whole ore leach recovery of **93.5%**. For the heap leach (HL) option, the coarse ore bottle roll tests indicated recoveries approaching 90% at the coarser size fractions, although the column test result using 22.5kg/t cement only produced a recovery of 55% after 90 days of leaching, although recovery was clearly continuing at the end of the test and with some evidence that more cement is required. Therefore, a conservative recovery of **70%** has been assumed with the potential for higher recovery once further optimised column tests can be conducted.

The indicated process capital and operating costs for the 0.75Mtpa CIL and HL options are **$61.4 million/$15.9/t** and **$11.4 million/$10.3/t**. In addition, for a 1Mtpa HL operation, the capital and operating costs are estimated as **$12.3 million** and **$8.8/t** respectively.

For both options, the additional infrastructure capital costs are **$0.7 million** for the water abstraction pump and pipeline, **$2.5 million** for upgrading 30km of laterite road and **$0.3 million** for the site camp. For the CIL option, an additional allowance of **$2.9 million per annum** should be allowed for the rental of the gensets (assume 3 x 1MW units). For the heap leach options, an additional allowance of **$0.7 million per annum** should be allowed, assuming 1 x 1MW unit.

Based on these results and a 750,000tpa operation for a LOM of 5.4 years (4.06Mt mineable resource at 1.48g/t Au), heap leaching provides overall the most economic processing route at significantly lower capital cost.

Recommendations for further testwork include variability testing for both the CIL and HL options. The priority is to conduct further optimised column tests to confirm that recoveries of 70% or higher can be achieved and the optimum cement addition required for agglomeration.

For the CIL option, variability testwork is required to optimise the grind, cyanide concentration and leach residence time and, in particular, the effect of oxygen sparging compared to air sparging. Further gravity testwork is also required to determine if a gravity circuit is economically beneficial and to confirm the poor intensive leach recoveries on the gravity concentrates from the carbon phyllite and zone A+B samples.

10.7 Tailings Storage Facility and Major Infrastructure

Geotechnical investigations are required for the prospective HL pads and ponds (or CIL plant and associated TSF and infrastructure).

A detailed site and process water balance is required for the selected option.

10.8 Environmental and Social

No immediate fatal flaws were identified for the project. However, the identified project risks will require careful planning and management. These risks and key impacts can be managed throughout the ESIA process and include:

- Economic and physical displacement;
- Population influx and the resulting impacts, including increase in ASM; and
- Water management.

The project area is already largely disturbed, however, natural habitats (including potential protected species and wetland areas) exist which should be avoided as far as possible. It is recommended that the environmental and social studies are undertaken in collaboration with the engineering design and feasibility studies to feed into project decision making. The ESIA process takes approximately 12 to 16 months, depending on the level of collaboration between the respective feasibility teams. It should be noted that the above timing considers two season surveys for biotic studies (wet and dry season) but excludes potential resettlement and livelihood restoration as this process is independent to the ESIA and environmental permitting.

10.9 Economic Analysis

Results of the Study show good initial validation of the Project's future economic potential.

Based on the performed preliminary economic assessment, the Cora Gold project gains from economy of scale, and therefore higher production throughput results in better NPVs.

The project is mostly sensitive to change in metal price, recovery rates and mining operating costs.

The base case gold price of US$1,400/oz has been selected on a moderately optimistic side with break-even prices for the considered options as following:

- Option 1Mtpa: recovery rate drops below 63%, break-even price of US$1,271/oz.
- Option 0.75Mtpa: recovery rate below 74%, break-even price of US$1,458/oz; and
- Option 1.5Mtpa: recovery rate below 56%, break-even price of US$1,108/oz.

The project remains robust at production rates above 1Mtpa and is not viable for lower production rates using technical and economic parameters outlined in this report.

APPENDIX 1: A Mineral Resource Estimate on the Sanankoro Gold Project, Mali

(Completed by SRK Consulting)

A MINERAL RESOURCE ESTIMATE ON THE SANANKORO GOLD PROJECT, MALI

Prepared For

Cora Gold Ltd

Report Prepared by

srk consulting

SRK Consulting (UK) Limited

UK30681

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Date:		December 2019
Project Number:		UK30681
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EXECUTIVE SUMMARY A MINERAL RESOURCE ESTIMATE ON THE SANANKORO GOLD PROJECT, MALI

1 EXECUTIVE SUMMARY

1.1 Introduction

SRK Consulting (UK) Limited ("SRK") is an associate company of the international group holding company, SRK Consulting (Global) Limited (the "SRK Group"). SRK has been requested by Cora Gold Limited ("Cora Gold", hereinafter also referred to as the "Company" or the "Client") to prepare a Mineral Resource Estimate ("MRE") for the Sanankoro Gold Project ("Sanankoro", or the "Project") located in Mali, West Africa.

The MRE has been produced in accordance with the terms and guidelines of the Australasian Code for the Reporting of Exploration Results, Mineral Resources and Ore Reserves, the JORC Code, 2012 Edition, ("JORC" or the "JORC Code").

1.2 Project Description

The Sanankoro property lies approximately 110 km south west of Bamako in southwest Mali. The property consists of five contiguous exploration permits (Sanankoro, Bokoro II, Bokoro Est, Dako and Kodiou) that encompass a total area of approximately 342 km^2.

The Sanankoro property is associated with extensive artisanal gold mining activity. Shallow (typically < 15m deep) workings extend discontinuously over a distance of just over 10 km, with individual workings up to 3 km in length and 500 m in width.

1.3 Project Geology and Mineralisation

The Sanankoro property is underlain by a Paleoproterozoic Birimian volcano-sedimentary formation that trends NNE-SSW, controlled by regional scale shear zones. The formations comprise intercalated units of weakly metamorphosed feldspathic sandstones, siltstones and phyllites, often with a carbonaceous component.

Gold mineralisation occurs along a large surficial elevated gold anomaly of approximately 4.5 x 7.5km, an area characterised by widespread artisanal mining activity. At least three different sets of mineralised quartz veins occur. These include a prominent N-S/NNE-SSW striking set that appear to dip steeply to the east and is the principal focus of artisanal exploitation; a less prominent oblique E-W (80-100^o) striking sub-vertical set; and a subordinate less continuous sub-horizontal set. As presently defined by drilling, gold mineralisation within the Project area is contained within a large mineralised corridor composed of 3 subparallel, broadly N-S striking structures known as Bokoro, Sanankoro and Selin. The first two zones can be traced from the north to the south of the Sanankoro permit, over a distance of some 15km, whereas the Selin zone can be traced from the north for a distance of about 10km before it merges with the

Sanankoro zone.

The Mineral Resource presented herein in focussed on four zones of Mineralisation, namely “Zone A”, “Zone B”, “Zone B North” and “Selin”. Zone A, Zone B and Zone B North all occur along-strike on the Sanankoro Structure, whilst the Selin zone forms part of the Selin structure, to the north of Zone A, Zone B and Zone B North, and east of the Sanankoro structure.

Deep tropical weathering in the region has liberated and in parts re-mobilised the primary gold. The weathering profile consists of a thin hardcap layer that extends to depths of up to 20 (with an average depth of 5 m), a deep saprolite that varies in depth from 10 – 120 m (average depth of roughly 50 m) and thin transitional saprock layer at the base of the saprolite.

1.4 Exploration, Drilling and Sampling

The Sanankoro Project has been subject to 3 main phases of exploration, namely by Randgold Resources Ltd (“Randgold”) in the mid-2000’s, Gold Fields Ltd (“Gold Fields”) between 2008 and 2012, and Cora Gold from 2017 to 2019. An overview of the salient exploration activities undertaken by each is provided below:

Randgold:

- Regional termite mound sampling.
- Regional and infill soil sampling.
- A series of shallow (10 – 15 m depth) vertical rotary air blast (“RAB”) drillholes on a 400 m line spacing.

Gold Fields:

- Infill soil sampling on 100 x 200 or 50-100 x 400 m grids
- Ground geophysical surveying including induced polarisation (“IP”) and resistivity surveys
- Systematic infill drilling using mainly reverse circulation (“RC”) holes on fences typically 100 m apart over much of the current Project area
- A series of shallow (12 – 15 m depth), vertical exploration air core (“AC”) or RAB holes drilled on variable grid spacings over large areas of the exploration permits.
- Follow-up RC drilling, mainly completed on NW-SE orientated or E-W oriented lines on fences between 100-200 m apart in “Zone A” and “Zone B, including deeper holes (to 180 m length) which comprised RC holes with diamond core tails.

Cora Gold:

- Termite mound sampling to supplement earlier soil geochemistry programmes completed by Randgold and Gold Fields on grid parameters that range from 400m x 100m to 200m x 100m.
- Follow-up ground IP surveys, to extend original Gold Fields ground IP coverage to the north by a further c 12.5 km^2 .
- A series of field bulk density programmes.
- A total of 264 drillholes across the Project area, for a total meterage of approximately

23,100 m, including a combination of RC, AC, RAB and diamond (“DC”) drillholes, with diamond core tails on a small number of RC and AC holes, on 60 – 120 m spaced sections, with between 1 and 5 holes per section.

Combined, drilling throughout the Sanankoro Project area completed by Randgold, Gold Fields and Cora Gold, totals approximately 78,500 m of reverse circulation (“RC”), air core (“AC”), rotary air blast (“RAB”) and diamond core (“DC”) drilling, which includes approximately 2,100m of diamond core. The total length (mineralisation and waste) of the drillholes that have targeted and intersected mineralisation in the area of interest (namely Zone A, Zone B, Zone B North and Selin) is approximately 18,200 m, including approximately 14,500 m of AC and RC drilling, 1,800 m of RAB drilling and 1,800 m of diamond core. Drilling to date by all explorers has primarily targeted oxide mineralisation, although several deeper holes have intersected the sulphide mineralisation below the weathered rock.

SRK have not completed any independent checks on the logging, sampling or drill protocols put in place by Cora Gold. That said, based on information and assurances provided by Dr Jonathan Forster and Cora Gold on the drilling, sampling and sample analysis protocols employed during the Cora Gold drill campaigns, SRK considers that these are acceptable for the reporting of a Mineral Resource Estimate in line with the JORC Code (2012).

1.5 Data Verification

SRK have completed a series of verification checks on the Sanankoro drillhole database to determine the suitability of the data provided for use in a Mineral Resource Estimate. The checks completed include the following:

- High-level validation checks on the historic Randgold and Gold Fields drill results, including: statistical analysis of the Randgold and Gold Fields mineralised intersections inside of the mineralisation wireframes used to derive the Mineral Resource; and grade trend assessment inside mineralisation wireframes by radial basis function “RBF” interpolants, comparing the historic drillhole results with the Cora Gold drillhole results.
- Visual verification of the location and elevation of the Cora Gold drillhole collars against the historic Gold Fields and Randgold collars.
- Data validation checks on the sample database supplied by Cora Gold;
- A statistical comparison of the analytical methods employed by Cora Gold for Au assaying, namely fire assay and bottle roll analyses;
- A study on the results of the QAQC sampling programme put in place by Cora Gold, which included the use of blanks, field duplicates, repeat assays, standards and umpire lab repeats.

The results of the verification checks completed by SRK indicate that the results of both the historic and Cora Gold drilling are suitable for use in deriving a Mineral Resource Estimate for the Project. The results of the QAQC analyses undertaken by Cora Gold do not indicate any serious issues in the sample assays. Although the standards used for bottle roll analysis perform very poorly, this is considered to be most likely a result of the method used to prepare these samples, rather than indicating any problem with the analytical equipment and method.

1.6 Mineral Resource Estimate

In deriving the Mineral Resource Estimate presented herein, SRK has completed the following:

- Modelled Au mineralisation domains in 3D, based on selecting mineralised intervals that can be consistently traced across at least 3 drillholes, at an absolute minimum modelling cut-off of 0.2 g/t Au. The strike of the mineralised veins was guided between drillhole sections by the trend of IP anomaly contrasts;
- Constructed a model of the Sanankoro weathering profile, based on regolith logging of the Cora Gold drillholes to define 4 weathering domains, namely hardcap, saprolite, saprock and fresh rock;
- Composited the assays inside of the mineralisation domains to 3 m;
- Applied high grade caps per zone and broad weathering state (i.e. oxide or sulphide), based on histogram and log histogram analysis;
- Created a block model, coded and sub-blocked by the mineralisation domains and weathering domains, with parent block sizes selected based on the average drillhole spacing in each area, being roughly half the on-section drillhole spacing and with approximately 2-3 columns of blocks between sections;
- Completed a geostatistical analysis, primarily focussed on the best informed mineralisation in the Zone A domain. It was not possible to produce meaningful variogram models for other domains;
- Interpolated Au grades into the block model, based on the following:
- Interpolation completed separately for each mineralisation domain, with hard boundaries applied throughout;
- Ordinary kriging used as the interpolation method for all domains;
- Kriging variogram parameters for all domains based on the results of variography completed on Zone A. This assumes that the grade continuity in Zone A, Zone B North and Selin will be comparable to Zone A;
- Search ellipsoid, and sample number requirements adjusted per domain to reflect the data spacing in each domain. Other than Zone A, for the majority of domains an isotropic ellipse was applied due to uncertainty in the mineralisation plunge for these domains at this stage;
- Average dry density values from density determinations carried out by Cora Gold applied separately to the hardcap (2.55 g/cm3), saprolite (2.15 g/cm3), saprock (2.15g/cm3) and fresh rock (2.75 g/cm3) weathering domains;
- Visually and statistically validated the estimated block grades relative to the original sample results;
- Depleted the block model to account for artisanal mining activity; and
- Reported the Mineral Resource according to the terminology, definitions and guidelines given in the JORC Code.

Upon consideration of data quality, drillhole spacing and the interpreted continuity of grades controlled by the deposit, SRK has classified portions of the deposit model in the Inferred Mineral Resource categories. Almost all mineralisation domains at Zone A, Zone B, Zone B

North and Selin have been classified as Inferred, generally to 50 m beyond the deepest drillhole intersection on each section.

1.6.1 Mineral Resource Statement

In order to determine the quantities of material offering “reasonable prospects for economic extraction” by open pit mining, a pit optimisation analysis was completed on the estimated block model, based on reasonable mining assumptions. The Mineral Resource has been restricted to estimated blocks that fall inside of the resulting pit shell, which is based on a gold price of 1,700 USD/oz, and reported above a cut-off grade of 0.4 g/t Au for oxide material and 0.5 g/t Au for sulphide material.

The Mineral Resource Statement presented herein has been classified by Mr. Martin Pittuck, who is a Corporate Consultant (Mining Geology) of SRK UK, a Member of the Institute of Materials, Minerals and Mining (MIMM), a Fellow of the Geological Society of London (FGS) and a Chartered Engineer, UK (CEng). Mr Pittuck is responsible for the preparation of the Mineral Resource Estimate and takes overall responsibility for the resource estimation work and resulting Mineral Resource Statement.

SRK UK have not completed a Competent Persons site visit to the Sanankoro Project. Dr. Jonathan Forster, CEO and Head of Exploration for Cora Gold Ltd, acts as the Competent Person responsible for the geology, drilling, sampling and exploration protocols employed on site.

Both Mr Pittuck and Dr. Forster have sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration and to the activity which they are undertaking to qualify as a Competent Person as defined in the 2012 Edition of the ‘Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves’. Both Mr Pittuck and Dr Forster consent to the inclusion in this announcement of the matters based on their information in the form and context in which it appears.

Mineral Resources that are not Mineral Reserves have no demonstrated economic viability. SRK are not aware of any factors (environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors) that have materially affected the Mineral Resource Estimate. It is uncertain is further exploration will convert Inferred Mineral Resources to higher confidence categories.

Table ES 1: Mineral Resource Statement for the Sanankoro Project, as of 5 December 2019.

Weathering State	Resource Classification	Tonnes (Mt)	Au g/t	Contained Au (Oz)
OXIDE	MEASURED	-	-	-
	INDICATED	-	-	--
	INFERRED	4.5	1.6	233,000
	TOTAL	**4.5**	**1.6**	**233,000**
SULPHIDE	MEASURED	-	-	-
	INDICATED	-	-	--
	INFERRED	0.5	1.8	32,000
	TOTAL	**0.5**	**1.8**	**32,000**
OXIDE + SULPHIDE	MEASURED	-	-	-
	INDICATED	-	-	--
	INFERRED	5.0	1.6	265,000
	TOTAL	**5.0**	**1.6**	**265,000**

Notes:

(1) The Inferred Mineral Resource Estimate is reported above a cut-off grade of 0.4 g/t for oxide material and 0.5 g/t for sulphide.
(2) The Mineral Resource Estimate for the Sanankoro deposit was constrained within grade based solids and within a Lerchs-Grossman optimised pit shell based on a gold price of 1,700 USD / oz and through the application of reasonable mining parameters.
(3) All figures are rounded to reflect the relative accuracy of the estimate.
(4) Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
(5) It is uncertain is further exploration will convert Inferred Mineral Resources to higher confidence categories.

The Mineral Resource is delineated by zone, and by the weathering profile in Table ES 2 and Table ES 3 respectively.

Table ES 2: Mineral Resources by Zone.

Zone	Tonnes (Mt)	Au g/t	Contained Au (Oz)
Selin	1.9	1.8	108,000
Zone A	1.9	1.5	91,000
Zone B	0.7	2.0	47,000
Zone B North	0.5	1.1	19,000
TOTAL	**5.0**	**1.6**	**265,000**

Table ES 3: Mineral Resources by Weathering Profile Domain.

Zone	Tonnes (Mt)	Au g/t	Contained Au (Oz)
Hardcap	0.4	1.3	16,000
Saprolite	3.7	1.6	191,000
Saprock	0.4	1.9	27,000
Fresh	0.5	1.8	32,000
TOTAL	**5.0**	**1.6**	**265,000**

1.7 Exploration Target

In October 2018, SRK derived an Exploration Target for the Sanankoro Project, based on the following:

- Volumetric modelling and grade interpolation of mineralisation at Zone A, Zone B, Zone B North and Selin, in addition to two other zones, namely Zone C and Selin South, altogether representing a total strike length of ~11 line km. The volumetric modelling was limited to a depth of 100 m below surface.
- Assessment of an additional 33 line km of positive exploration results which suggests potential to discover additional mineralisation with similar thickness and grade.

SRK is unaware of any new information which materially impacts on the assumptions upon which the Exploration Target is based. For this reason, an unchanged Exploration Target for the Sanankoro Project of **between 30 Mt and 50 Mt at a grade of between 1.0 and 1.3 g/t Au** is re-stated here.

For the avoidance of doubt, in respect to the Exploration Target, SRK notes:

- The potential quantity and grade as reported in respect of the Exploration Targets are conceptual in nature;
- There has been insufficient exploration to define a Mineral Resource; and
- It is uncertain if further exploration (as planned by the Company) will result in the determination of a Mineral Resource.

The <5 km total strike extent of the optimised pit shells used to constrain the Sanankoro Inferred Mineral Resource represents <15% of the total linear strike length of potential mineralised zones upon which the Exploration Target is based. It is noted that, of the approximate 1 – 2 million ounce Exploration Target range, approximately 700,000 ounces of gold are defined in the block model from which the 265,000 ounce Inferred Mineral Resource is derived (being inside the optimised pit and above cut-off grade).

1.8 Conclusions

SRK has derived a Maiden Inferred Mineral Resource Estimate for the Sanankoro Project of 5.0 Mt at 1.6 g/t Au. The resource is primarily restricted to saprolitic oxide mineralisation, with a small contribution from fresh sulphide mineralisation. The Mineral Resource has been defined within an area tested by a reasonable volume and density of RC, AC, RAB and diamond drilling. A number of additional targets have been identified by regional exploration drilling, geochemical soil and termite sampling and mapping of artisanal outlines, which have been used to derive an Exploration Target, within which the Inferred Mineral Resource sits, of between 30 Mt and 50 Mt, at a grade of between 1.0 g/t and 1.3 g/t. Given the relatively small thickness and steep dip of the mineralised zones, the depth of the optimised pits used to constrain the Mineral Resource is limited by excessive stripping ratios at depth. SRK therefore consider that the main potential for defining additional Mineral Resources is outside of the zones that have been the primary focus of drilling to date. SRK would recommend that a key focus of the next phase of exploration would be drilling of the most prospective exploration target zones.

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A REPORT ON THE EXPLORATION RESULTS AND ASSOCIATED EXPLORATION TARGET FOR THE SANANKORO PROJECT, MALI

1 INTRODUCTION

1.1 Background

SRK Consulting (UK) Limited ("SRK") is an associate company of the international group holding company, SRK Consulting (Global) Limited (the "SRK Group"). SRK has been requested by Cora Gold Limited ("Cora Gold", hereinafter also referred to as the "Company" or the "Client") to prepare a Mineral Resource Estimate ("MRE") for the Sanankoro Gold Project ("Sanankoro", or the "Project") located in Mali, West Africa.

1.2 Requirement, Structure and Compliance

The reporting standard adopted for the reporting of the MRE is the Australasian Code for the Reporting of Exploration Results, Mineral Resources and Ore Reserves, The JORC Code, 2012 Edition. The JORC Code is an internationally recognised reporting code as defined by the Combined Reserves International Reporting Standards Committee (CRIRSCO).

The Mineral Resource Statement presented herein has been classified by Mr. Martin Pittuck, who is a Corporate Consultant (Mining Geology) of SRK UK, a Member of the Institute of Materials, Minerals and Mining (MIMM), a Fellow of the Geological Society of London (FGS) and a Chartered Engineer, UK (CEng). Mr Pittuck is responsible for the preparation of the Mineral Resource Estimate and takes overall responsibility for the resource estimation work and resulting Mineral Resource Statement.

Dr. Jonathan Forster, CEO and Head of Exploration for Cora Gold Ltd, acts as the Competent Person responsible for the geology, drilling and exploration protocols employed on site.

Both Mr Pittuck and Dr. Forster have sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration and to the activity which they are undertaking to qualify as a Competent Person as defined in the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves'. Both Mr Pittuck and Dr Forster consent to the inclusion in this announcement of the matters based on their information in the form and context in which it appears.

1.3 Details of Personal Inspections

SRK have not completed a Competent Persons site visit to the Sanankoro Project due to security conditions prevailing at the time. The geological interpretation of the deposit and controls on mineralisation have been developed by Cora Gold. All data upon which the Mineral Resource Estimate is based has been provided to SRK by Cora Gold, and SRK have not completed any independent checks on the logging, sampling or drill protocols put in place by Cora Gold.

1.4 Declaration, Limitations and Cautionary Statements

SRK's opinion contained herein and effective 5 December 2019 is based on information collected by SRK throughout the course of SRK's investigations, 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 the Company, and neither SRK nor any affiliate has acted as advisor to the Company, 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.

Except as specifically required by law, SRK does not assume any responsibility and will not accept any liability to any other person for any loss suffered by any such other person as a result of, arising out of, or in connection with this Technical Report or statements contained herein, required by and given solely for the purpose of complying with the mandate as outlined in this Technical Report. SRK has no reason to believe that any material facts have been withheld by the Company.

This report is intended to be read as a whole, and sections should not be read or relied upon out of context. The Technical Report contains expressions of the professional opinion of the Competent Person based upon information available at the time of preparation.

1.5 Qualifications of Consultants

SRK is an associate company of the international group holding company SRK Consulting (Global) Limited. The SRK Group comprises over 1,400 staff, offering expertise in a wide range of resource engineering disciplines with 45 offices located on six continents. The SRK Group's independence is ensured by the fact that it holds no equity in any project. This permits the SRK Group to provide its clients with conflict-free and objective recommendations on crucial judgement issues. The SRK Group has a demonstrated track record in undertaking independent assessments of resources and reserves, project evaluations and audits, Mineral Experts' Reports, Competent Persons' Reports, Mineral Resource and Ore Reserve Compliance Audits, Independent Valuation Reports and independent feasibility evaluations to bankable standards on behalf of exploration and mining companies and financial institutions worldwide. The SRK Group has also worked with a large number of major international mining companies and their projects, providing mining industry consultancy service inputs.

SRK has extensive experience of undertaking Mineral Resource Estimates for gold projects in the West Africa region, for projects at all stages of development.

2 RELIANCE ON OTHER EXPERTS

The information reviewed in preparing this report has been provided by the Company and a compilation of proprietary and publicly available information. SRK has referenced information and data sourced from reports and documents where applicable.

SRK has relied on the following previous technical reports on the Sanankoro Project, particularly in relation to the background information:

- SRK Consulting (UK) Ltd, 2018. A Report on the Exploration Results and Associated Exploration Target for the Sanankoro Project, Mali.
- SRK Exploration Services, 2017. An Independent Report on the Mineral Assets of Cora Gold Ltd in Mali and Senegal. Report prepared for Cora Gold Ltd.
- Wardell Armstrong International, 2019. Sanankoro Gold Project, Mali – Scoping Level Metallurgical Testing on Samples of Oxide Mineralisation.
- Wardell Armstrong International, 2019. Sanankoro Gold Project, Mali – Scoping Level Metallurgical Testing on Samples of Sulphide Mineralisation.

As noted in Section 1.3, SRK have not completed a Competent Persons site visit to the Sanankoro Project. SRK have relied on Cora Gold for all information relating to the geology, mineralisation controls, exploration, drilling and sampling procedures and drillhole results for the Project. SRK has not undertaken any independent verification of any of these aspects. Whilst, SRK takes overall responsibility for the Mineral Resource Estimate presented herein, Dr. Jonathan Forster, CEO and Head of Exploration for Cora Gold Ltd, acts as the Competent Person responsible for the geology, drilling and exploration protocols employed on site.

SRK has not performed an independent verification of land title and tenure as summarised in Section 3.2 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, but has relied on the Company for any land title issues.

3 PROPERTY DESCRIPTION, LOCATION AND HISTORY

3.1 Location

The Sanankoro property lies approximately 110 km south west of Bamako, predominantly within the Kangaba Cercle, Koulikoro Region in southwest Mali, although the southern-most extent extends into the Yanfolila Cercle of the Sikasso Region.

The geographical location of the Sanankoro property permits is shown in Figure 3-1.

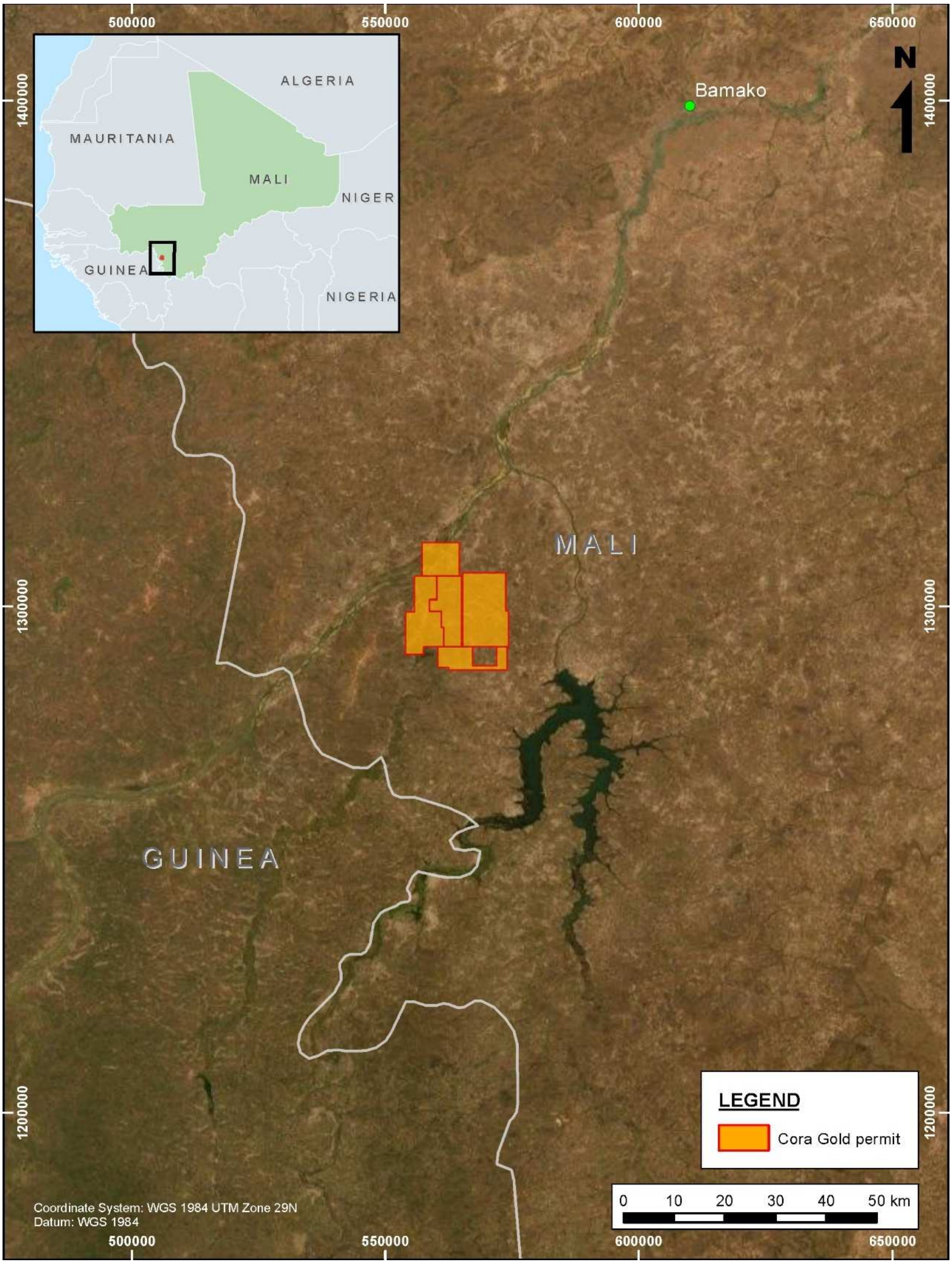


Figure 3-1: The Sanankoro property permit outlines, shown relative to satellite imagery and, inset, within the West Africa region map.

3.2 Mineral tenement and land tenure status

3.2.1 Permit Status

The Sanankoro property consists of five contiguous permits (Sanankoro, Bokoro II, Bokoro Est, Dako and Kodiou) that encompass a total area of approximately 342 km^2. Details of the permits are provided below and summarised in Table 3-1. The location and extent of the permit outlines is displayed in Figure 3-2.

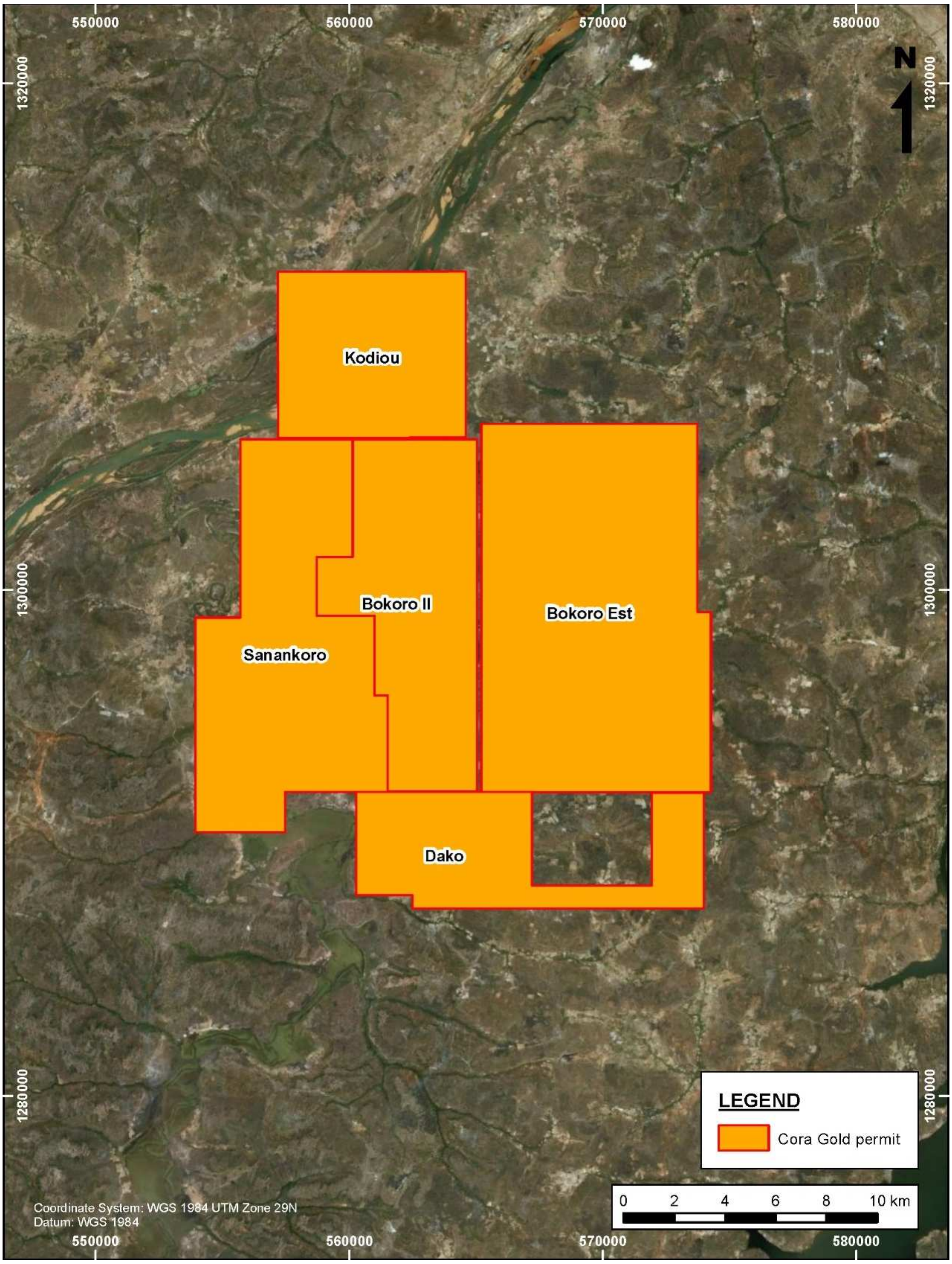


Figure 3-2: The Sanankoro Project permit outlines, shown relative to satellite imagery.

Cora Gold owns 95% of Sankarani Ressources SARL ("Sankarani") through which Cora Gold conducts its exploration in Mali. According to documentation provided by Cora Gold, the Sanankoro permit was initially granted as exploration permit (*permis de recherche*) PR 12/605 for Group 2: Precious metals (gold, silver, platinum) and industrial metals to Sankarani on 01 February 2013 for a period of three years and expired 31 January 2016 (application 2013-

0292/MM-SG). The permit was renewed by Sankarani for a period of two years and expired 31 January 2018 (application 2016-1526/MM-SG). The current exploration permit held by Sankarani (application 2018-2174/MMP-SG) was issued on the 2 July 2018 and represents the final 2-year exploration permit renewal period, being due to expire on 1 February 2020. The Company has applied for the award of a new permit over the area covered by the current Sanankoro Permit. The Company anticipates that any new permit will be issued in accordance with Mali's Mining Code and as such expects such permit to have a total life of 7 years, being an initial period of 3 years followed by two renewals for periods of 2 years each.

The Bokoro II permit was initially granted as exploration permit PR 15/769 for Group 2: Precious metals (gold, silver, platinum) and industrial metals to Sankarani on 25 August 2015 for a period of 3 years and expired on 24 August 2018 (application 2015-2957/MM-SG). The permit was renewed on 23 August 2019 (application 2019-2497/MMP-SG) and is due to expire on 25 August 2022. In accordance with the Malian Mining Code, the permit can be renewed once more for a period of two years, after the expiration of the current licence.

The Bokoro Est permit was granted as exploration permit PR 10/432 for Group 2: Precious metals (gold, silver, platinum) and industrial metals to Sankarani on 20 August 2010 for a period of 3 years and expired 19 August 2013 (application 10-2665/MM-SG). The licence was renewed twice (applications 2014-2398/MM-SG and 2015-3599/MM-SG) and expired on 19 August 2017. A new permit has since been re-issued to Sankarani on 18 September 2019 (application 2019-3057/MMP-SG) and is due to expire on 18 September 2022. The permit can be renewed twice more for periods of 2 years, after the expiration of the current licence.

The Dako permit was granted as exploration permit PR 09/392 for Group 2: Precious metals (gold, silver, platinum) and industrial metals to Gold Fields Exploration Mali SARL on 19 August 2009 for a period of 3 years (application 09-2127/MM-SG). The name of Gold Fields Exploration Mali SARL was reportedly changed to Hummingbird Exploration Mali SARL (SRK Exploration Services, 2017). The permit has since been renewed three times (applications 2012-3272/MM-SG, 2014-3478/MM-SG and 2018-4535/MMP-SG). new permit has since been re-issued to Sankarani on 31 December 2018 and is due to expire on the 31 December 2021. The permit can be renewed twice more for periods of 2 years, after the expiration of the current licence.

The Kodiou Permit was granted as an exploration permit PR 15/735 to a third party initially on 15 May 2015. The permit expires on 15 May 2022. Through a Joint Venture Agreement, Cora Gold have the option to earn up to 100% through payment of staged fees to the permit holder, totalling USD 55,000, subject to the 3rd party being paid a 1% NSR royalty of production from the permit area, with Cora given the right to buyout the 3rd party for the sum of USD 600,000.

Sankarani Ressources SARL is 95%-owned by Cora Gold Ltd. A 5% free carried interest held by a third party in the permits granted to Sankarani may be bought out for a once only US$ 1 M payment that may be made against the interest held in either of the Komana (Hummingbird Resources) or Sankarani properties. In addition, the Bokoro II, Bokoro Est and Sanankoro permits are subject to a 1 % NSR payable to a third party.

Table 3-1: Summary Table of permits in the Sanankoro property area.

Property Name	Permit Name	Permit No	Cora Gold Interest (%)	Status	Expiry Date	Area (km^2)	Comment
Sanankoro	Sanankoro	PR 12/605 2Bis	95	Exploration	01-Feb-20	84	In final renewal period. New permit application in progress.
	Bokoro Est	PR 10/432 2Bis	95	Exploration	18-Sep-22.	128	Can be renewed twice more for periods of 2 years
	Bokoro II	PR 15/769	95	Exploration	25-Aug-22	63	Can be renewed once more for a period of two years
	Dako	PR 09/392	100	Exploration	31-Dec-21	44.66	Can be renewed twice more for periods of 2 years
	Kodiou	PR 15/735	100*	Exploration	15-May-22	50	Held by 3rd party. Cora Gold have option to earn up to 100% through staged payments to 3rd party

** Earning up to 100% through payment of staged fees to permit holder totalling USD 55,000*

3.2.2 Company Description

On 13 March 2012, Cora Gold Limited ("Cora Gold") was founded by Dr Jonathan Forster and Mr Craig Banfield with the objective of exploring two gold belts in Mali, known as the Kenieba Window and the Yanfolila Gold Belt. Over the ensuing months, Cora Gold compiled a portfolio of gold exploration permits through a number of joint ventures with local partners.

In late 2013, Cora Gold was approached by a private company called Sumatran Africa regarding gold exploration permits held in the Republic of Congo (Brazzaville). In the 1990s, these permits were previously held by SAMAX Gold Inc., for whom both Dr Forster and Mr Banfield worked at that time. Discussions led to an agreement to merge both Cora Gold and Sumatran Africa. This merger was completed on 30 April 2014 when Kola Gold Limited became the parent company for the group. Through the issuance of new equity, Kola Gold subsequently raised in excess of US$ 5.8 million for the purpose of exploring its projects and for general working capital. In 2016, Cora Gold added a permit in Senegal to the mineral assets.

On 28 June 2016, Kola Gold and Hummingbird Resources PLC ("Hummingbird") entered into a Memorandum of Understanding ("MOU") with a view to amalgamating certain of Hummingbird non-core gold exploration permits in Mali together with a number of Kola Gold's permits in west Africa.

On 21 March 2017, the board of directors of Kola Gold resolved to split the group in two with Kola Gold continuing to hold permits in the Republic of Congo (Brazzaville) in central Africa and Cora Gold holding permits in Mali and Senegal in west Africa. This re-organisation was completed by a pro rata distribution-in-kind of the shares in Cora Gold held by Kola Gold to the shareholders of Kola Gold.

On 28 April 2017, the agreement to amalgamate Hummingbird's non-core gold exploration permits in Mali together with a number of Cora Gold's permits in Mali and Senegal was completed. As such, Hummingbird's subsidiary, Trochilidae Resources Ltd, became a 50% shareholder in Cora Gold. Cora Gold subsequently undertook a number of transactions which resulted in changes to its share structure. On 9 October 2017 Cora Gold's ordinary shares were admitted to trading on AIM, a market of that name operated by the London Stock Exchange plc. As a result of these transactions the Company no longer has an ultimate controlling party. As at 12 November 2019 the Company's largest shareholder was Hummingbird which held 18.00% of the total number of ordinary shares on issue and outstanding.

3.3 Physiography, Climate and Environment

Mali has a varied landscape and three distinct climatic and vegetation zones: the Saharan zone in the north; the semi-arid Sahelian Zone in the centre; and the raised savannah, or 'Sudanese' zone in the south. Northern Mali is covered by the southern extension of the Sahara Desert, and as such is arid with a hot almost rainless climate. The Sahelian zone is concentrated around the River Niger and marks the transition from desert into raised savannah.

The River Niger, which rises in the mountains of Guinea to the west, is a major lifeline to the country with much of the main agriculture and major towns, including Bamako, Mopti and Tombouktou concentrated along it. The raised savannah of the south and west parts of the country is made up of savannah type vegetation and some light forests, with a mountainous region in the far west towards the border with Senegal.

In the south, where the Sanankoro Project is located, there are two distinct seasons: a dry season lasting from mid-October to late-April, when virtually no rain falls and a rainy season from late-April to mid-October. Total annual rainfall for this region is around 1,200 mm per year, which is concentrated within these months and can impact infrastructure during this time. Temperatures are high year round (20-35°C), and peak at the end of the dry season where temperatures often exceed 40°C, particularly in the Saharan north.

The physiography of the property is typically flat-lying with shallow topography although does include several hills with elevations of up to 410 m, around 40-50 m above the surrounding plains. Drainage is moderately well developed and typically flows to the west into the Niger River. Vegetation within the property typically consists of sparse trees and bushes.

The Sanankoro property reportedly does not include any environmentally sensitive areas (for example, protected / conservation areas, forest reserves, national parks, etc.) or historical, archaeological, cultural or other heritage features (for example, monuments, grave sites, etc.) (SRK Exploration Services, 2017).

3.4 Infrastructure

There is a good network of tarred roads in and out of Bamako and an extensive network of gravel and dirt roads across the country, particularly the more populated areas in the south, although the quality of these roads is variable, especially during and following the rainy season.

A railway line connects Bamako with Kayes in the west of Mali and the port of Dakar in Senegal.

Access to Sanankoro from Bamako is via a tarred road (the Route Nationale 7) southwards to a turning just beyond Ouelessebougou and then westwards to the Selingue Dam. Beyond Selingue, the remaining route to the property is via graded tracks. By road, the journey from Bamako takes around 4.5 hours. A four-wheeled drive vehicle is required year around. It is anticipated that during the wet season some sections of the tracks between Selingue and Sanankoro would be difficult to pass.

Based upon Pleiades imagery collected on 13 January 2017 and viewed via Google Earth, the property is largely unpopulated with the only significantly-sized settlements being Selefougou in the east and the Bokoro artisanal village in the west. Agricultural development is present in the property, but mainly limited to localised subsistence farming adjacent to some of the drainage channels. The property appears to be devoid of any significant infrastructure.

3.5 Ownership History

In April 2017 Cora Gold Limited acquired the entities holding the permits which make up the Sanankoro Gold Project / Sanankoro Property along with a number of other non-core gold exploration permits all of which were held by subsidiaries of the Hummingbird Resources plc group (namely Hummingbird Exploration Mali SARL (renamed Cora Exploration Mali SARL) and Sankarani Ressources SARL). These permits were previously held Gold Fields Limited (“Gold Fields”) until they were acquired by Hummingbird Resources plc in 2014. Prior to Gold Fields it is understood that a number of these permit areas were operated by Randgold Resources Ltd (“Randgold”) which merged with Barrick Gold Corporation in 2018.

3.6 Artisanal Mining

The Sanankoro property is associated with extensive artisanal gold mining activity, mainly within the Sanankoro permit. A map of the largest workings (looking southeast), interpreted in January 2017 by SRK Exploration Services is provided in Figure 3-3. As delineated using satellite imagery viewed via Google Earth, the discontinuous open-pit workings extend over a distance of just over 10 km, with individual workings up to 3 km in length and 500 m in width. The open-pit workings are typically not very deep (< 15 m) which appears to be due to the instability of the regolith. However, vertical shafts are common in the base of the open-pits, locally extending the depth of the workings by up to a further 5-10 m.

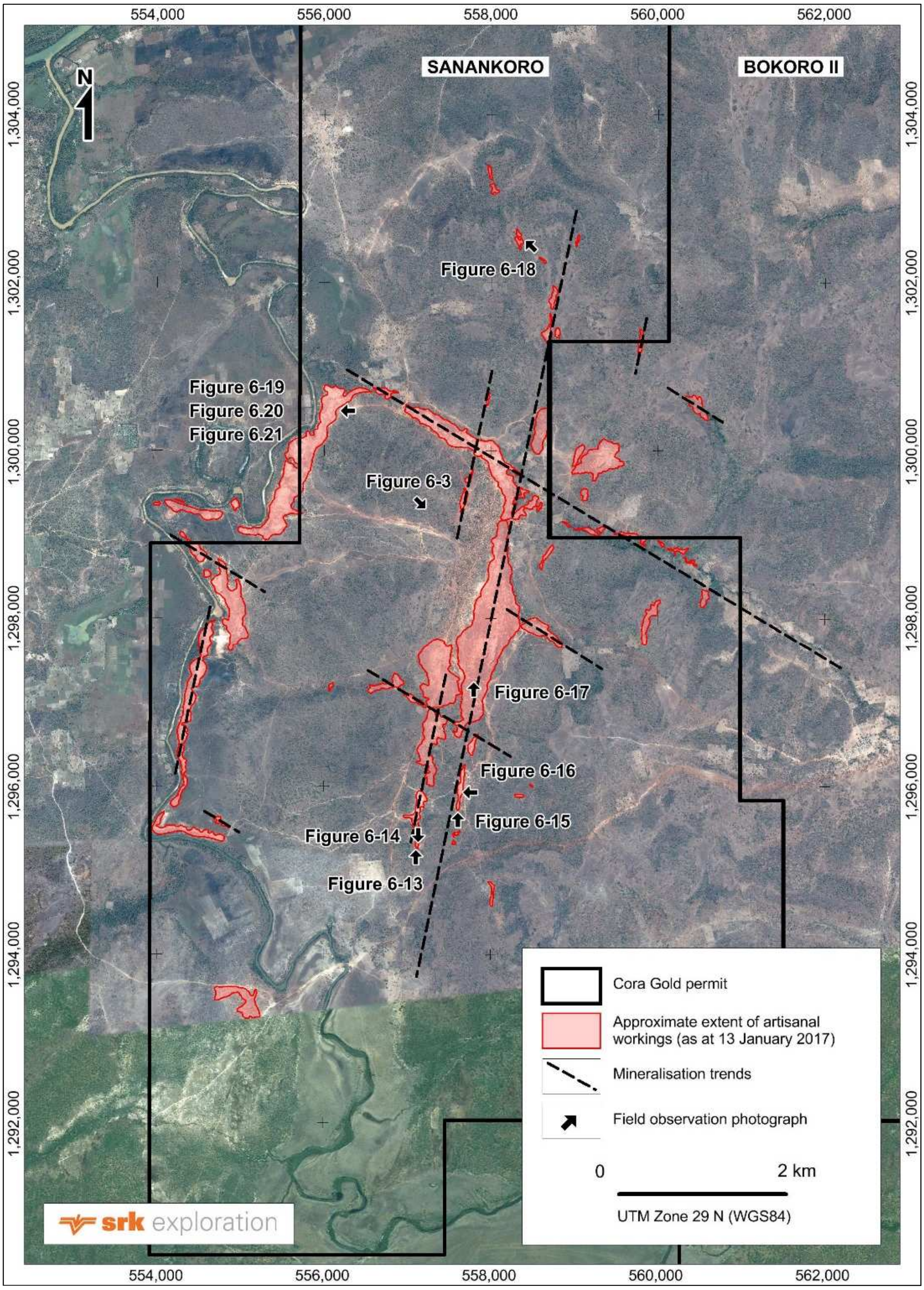


Figure 3-3: Sanankoro artisanal mining activity, as mapped by SRK Exploration Services, from Google Earth satellite imagery (SRK Exploration Services, 2017). Note that the section figure references within the image are not relevant to this report and not included.

The workings have exploited both the pedolith and saprolith. The shallower workings appear to have preferentially focused on exploiting the base of the mottled zone directly above the pallid zone and the deeper workings exploit the saprolite.

Figure 3-4 shows an example of one of the open-pits at the southerly extent of the workings visited by SRK Exploration Services in 2017. It consisted of a linear N-S orientated excavation up to 30 m wide, 200 m long and up to 20 m deep. The open-pit walls were also degraded and collapsed in places, potentially explaining the limited depth of the workings. The pit confirmed the satellite imagery interpretation that the artisanal are mainly exploiting the ferruginous, mottled and pallid zones of the pedolith and into the underlying saprolite.

Figure 3-4: North-facing view of artisanal workings at UTM29P 557100 E, 1292275 W (SRK Exploration Services, 2017).

3.7 Historical Estimates

The Mineral Resource Estimate described in this report is a maiden resource. To SRK's knowledge, no Mineral Resources have previously been declared for the Project. That said, in October 2018, SRK Consulting derived an Exploration Target for the Sanankoro Project of between 30 Mt and 50 Mt at a grade of between 1.0g/t and 1.3g/t Au.

4 GEOLOGICAL SETTING AND MINERLISATION

4.1 Geology of the West African Craton

The West African Craton comprises two major Archean to Paleoproterozoic terranes: The Man Shield (which covers Sierra Leone, Liberia, Cote d'Ivoire, Ghana, Burkina Faso, the eastern parts of Guinea and Senegal, southern Mali and southwestern Niger); and the Reguibat Shield in Mauritania (Figure 4-1). In the Man Shield, the Archean basement is only exposed in Liberia and Sierra Leone, where the rocks are highly metamorphosed gneisses with discontinuous greenstone belts. The remainder of the Shield is made up of Paleoproterozoic terrane referred to as the Birimian, which represent a series of large sedimentary basin deposits and linear or arcuate volcanic belts that were accreted during the Eburnean Orogeny around 2.1-1.0 Ga. This orogen was accompanied by the emplacement of extensive granitoid plutons. The metamorphic grade within the Paleoproterozoic rocks is generally low, except along some subsequent transcurrent fault zones. In Mali and eastern Senegal, the Birimian rocks are exposed in two areas: a wide area in the Bougouni region in the south of the country; and as an inlier referred to as the 'Kedougou-Kenieba window' present in the far west of the country.

Mali is situated on two of the major structural units of Archean-Paleoproterozoic basement that make up northwest Africa: The West African Craton in the west of the country, which hosts gold mineralisation, and the Tuareg Shield in the east. These two crustal blocks collided at the end of the Precambrian, and the suture zone, a roughly north south trending belt, is located to the west of the Adrar des Iforas Mountains, in eastern Mali. In between the outcrops of these basement blocks, two thirds of the country is covered by sediments of the Upper Proterozoic and Palaeozoic Taoudeni Basin, which are comprised mainly of sandstones. With the exception of the Adrar des Iforas Mountains, there is very little outcrop, with most of the country being covered by aeolian sand deposits in the north and tropically weathered regolith in the south.

The Tuareg Shield covers parts of Mali, Niger and Algeria. It is mainly composed of Archean or Paleoproterozoic terranes and Neoproterozoic Terranes that amalgamated during convergence of the West African Craton and Saharan mega-craton during the Pan African Orogeny.

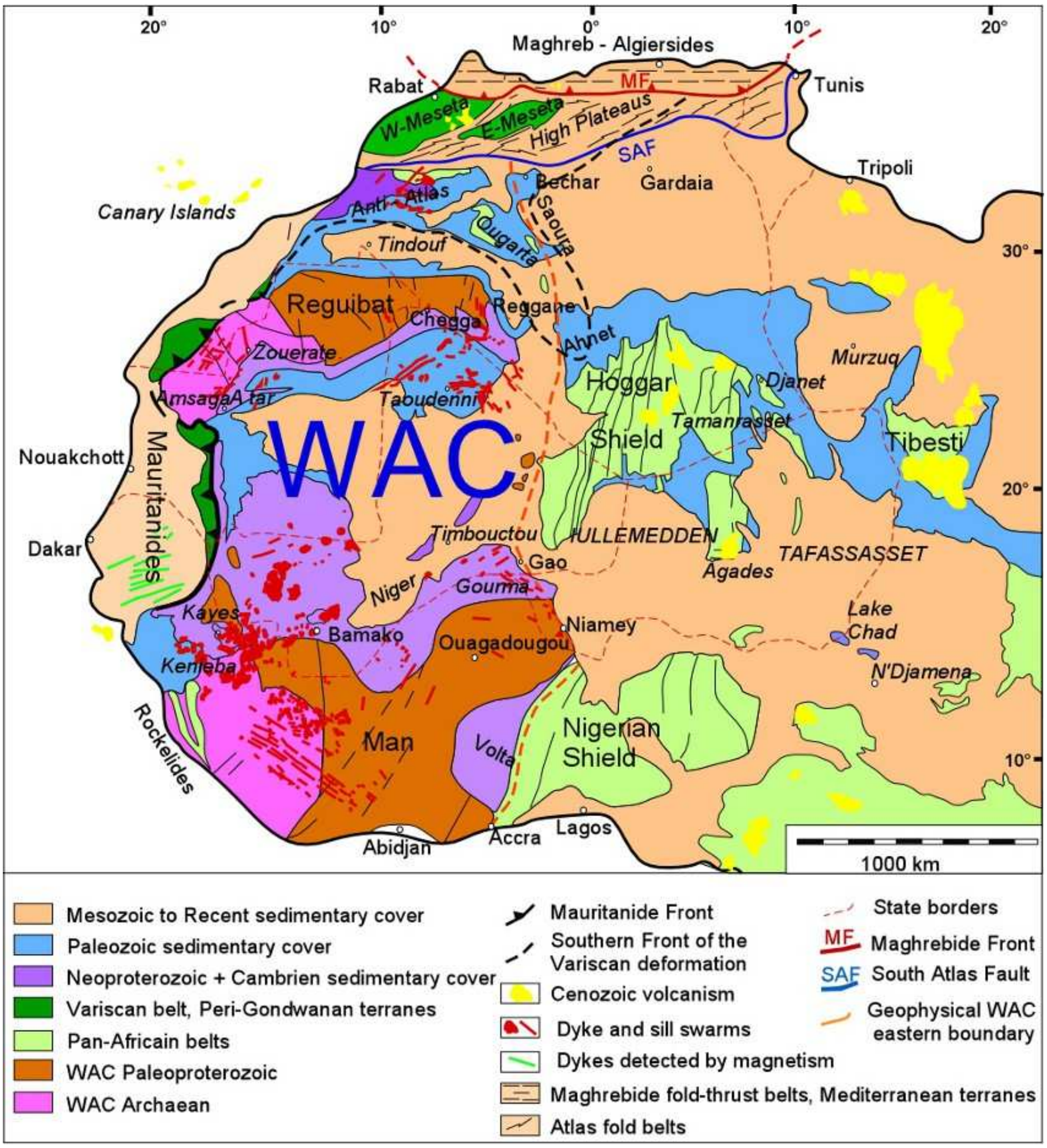


Figure 4-1: Geology of the West African Craton (Ennih and Liégeois, 2008).

4.2 The Birimian of West Africa

4.2.1 Lithology

The Birimian rocks of the West African craton are made up of an alternation of sedimentary belts and volcanic sequences intruded by large granitoid bodies which outcrop in north-south to northeast-southwest trending belts which extend for tens or hundreds of kilometres. The Birimian can be divided into two major units. The Lower Birimian, or B1 group, is made up of a basal unit of basic volcanic rocks, locally preserved in the Côte d'Ivoire; flysche deposits of sandstones, schists, metagreywackes and metapelites with intercalations of volcano-sedimentary rocks common in southern Mali, and an upper carbonate sequence, well developed in Guinea, Senegal and western Mali. The B1 basinal sequence is also known as the Dialé-Daléma Supergroup.

The Upper Birimian, or B2 group, comprises a sequence of bimodal, tholeiitic and calc-alkaline volcanic belts, metamorphosed to schists and amphibolites (greenstones), intrusive granitoid plutons and fluvio-deltaic formations that include gold-bearing sandstones and the Tarkwain conglomerates of Ghana. The B2 volcano-sedimentary units, also known as the Mako Supergroup, are more common in the Kedougou-Kenieba window than in the Bougouni region.

The Sanankoro Project is located within these Birimian terranes.

4.2.2 Structural Geology

The structural and sedimentological evolution of the Birimian of the West African craton during the Eburnean orogeny can be summarised as follows (after Milési, et al., 1992 and Sylla, et al., 2016):

1. Deposition of the largely sedimentary B1 Lower Birimian, with some basic volcanism and tholeiitic intercalations;
2. Regional deformation (D1) at around 2.1 Ga, attributed to collisional tectonics, which thrust the Paleoproterozoic terrane into contact with the Archean nuclei of the craton. This formed isoclinal folding within the B1 sediments and is associated with greenschist-facies metamorphism, with foliation (S1) roughly parallel to bedding;
3. Deposition of the largely volcaniclastic B2 Upper Birimian with some clastic basin infills.
4. Emplacement of basic to granodioritic plutons;
5. A major phase of transcurrent tectonics (D2) affecting the entire Birimian, imparting a series of N-S to NNE-SSW trending sinistral strike-slip faults, with an associated S2 schistosity;
6. A further episode of transcurrent deformation (D3) which formed a series of NE-SW striking strike-slip faults.

4.2.3 Mineralisation

Exploration for gold in West Africa was traditionally focussed on shear-hosted quartz veins. However, as modern exploration has developed, a wide range of genetic types of mineralisation have been described. These were initially documented by Milési et al. (1992) and fall into three principal types:

1. Pre-orogenic: pre-D1 mineralisation, including the stratiform Au deposits hosted within tourmalinised sandstones (Type 1 Au);
2. Syn-orogenic: post D1 to syn-D2 mineralisation within tholeiitic volcanic troughs (Type 2 Au) and Tarkwain auriferous placers in conglomerates (Type 3 Au);
3. Late-orogenic: late D2 to D3 mineralisation, with mesothermal Au deposits (gold and auriferous arsenopyrite bearing quartz veins - Type 4 Au) and gold bearing quartz veins associated with traces of polymetallic sulphides bearing Cu, Pb, Zn, Ag and Bi (Type 5 Au).

This list is not exhaustive and other mineralisation types include high-level epithermal, skarn and contact deposits, thrust-faulted occurrences, vein stockworks, intrusive disseminated and paleoplacer deposits.

Supergene enrichment of the orogenic gold lodes is economically important in the northern parts of the West Mali gold belt, involving karstification of mineralised limestones (Lawrence, et al., 2016).

Gold mineralisation in Mali is confined to the two areas of Birimian terrane as previously discussed. These are described further below:

Kedougou-Kenieba

This gold province is hosted within greenschist metamorphosed siliciclastic and carbonate sedimentary rocks of the upper B1 Birimian in Mali and within the volcanic-dominated greenstones of the Mako Supergroup or B2 further west into Senegal. Mineralisation is linked to higher order shears and folds related to the Senegal-Mali Shear Zone and the Main Transcurrent Zone (MTZ). The accretionary orogenic setting and timing (strike-slip deformation; post peak metamorphism), structural paragenesis and deposit geometry (steep, tabular ore bodies) are typical of orogenic gold mineralisation (predominantly Type 4). However, alteration assemblages and ore fluid compositions vary considerably suggesting a range of mineralising source fluids (magmatic, evaporitic and regional metamorphic) contributing to gold mineralisation in the region (Lawrence, et al., 2016).

Major deposits in the Kedougou-Kenieba gold province include the Yatela and Sadiola mines (AngloGold Ashanti) in the northern part of the district, Loulu and Gounkoto (Randgold Resources), Tabakoto-Kofi (Endeavour Mining), Fekola (B2 Gold) in the southern part, and Sabodala (Teranga), Massawa (Randgold Resources) and Petowal (Toro Gold) in Senegal.

Bougouni

Mineralisation in the western part of the Bougouni region is generally within the B1 units or along the structural contact between B1 and B2 units. The most abundant type of gold mineralisation is of the late-orogenic Type 4 and 5 Au mineralisation. Type 4 Au mineralisation is characterised by auriferous arsenopyrite, which is particularly common in Ghana, for example at Ashanti, and are commonly hosted within shear zones. Type 5 Au is characterised as mesothermal gold-bearing quartz veins with a variety of other metals such as Cu, Zn, Ag and Bi and is present at the Kalana mine in southern Mali. Both types of mineralisation are structurally controlled by the N-S and NE-SW D2 and D3 fault structures.

Further east, Type 2 Au mineralisation hosted by mafic tholeiitic volcanics is present at the Sayama mine, with the mineralisation controlled by D2-D3 faults. The Morila gold deposit (Randgold Resources) is classified as a reduced intrusion-related gold system, in which strata-bound Au–As–Sb–Bi–(W–Te) mineralisation formed early in the Eburnean orogenic cycle (syn-metamorphic) with spatial and genetic links to syn-orogenic granodiorites and leucogranites (Lawrence et al, 2016).

4.3 Sanankoro Property Geology and Mineralisation

4.3.1 Geology

The Sanankoro property is underlain by a Paleoproterozoic Birimian volcano-sedimentary formation that trends NNE-SSW, controlled by regional scale shear zones. The formations comprise intercalated units of weakly metamorphosed feldspathic sandstones, siltstones and phyllites, often with a carbonaceous component. Volcanoclastic sediments/ tuffs of acidic to intermediate composition occur within the sedimentary package in conjunction with both mafic (dioritic-gabbroic composition) and felsic igneous intrusive units locally incorporated.

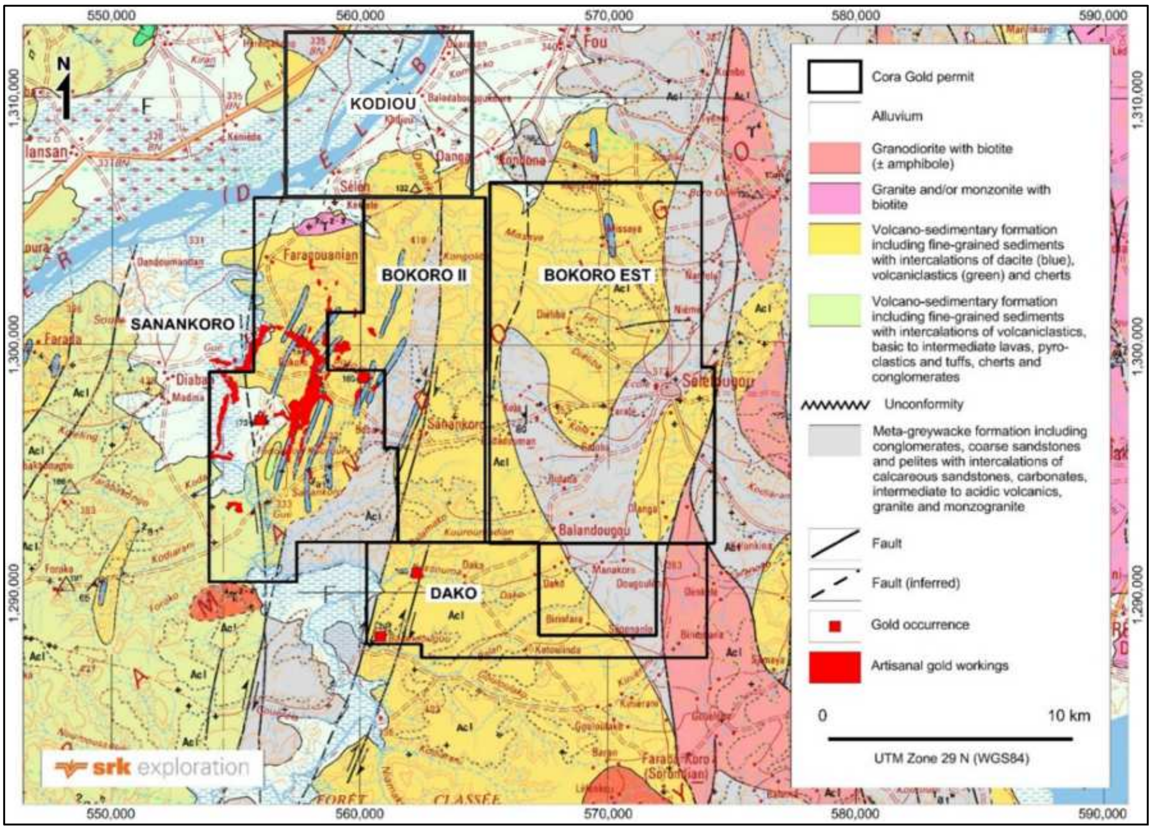


Figure 4-2: Sanankoro geological map (after PCGBM, 2006).

4.3.2 Mineralisation

Gold mineralisation occurs along a large surficial elevated gold anomaly (>50 ppb Au) of approximately 4.5 x 7.5km, an area characterised by widespread artisanal mining activity. An oblique image of the largest workings (looking southeast) is provided in Figure 4-3. The observed imagery indicates that artisanal miners appear to be exploiting alluvial and eluvial ferruginous and kaolinitic regolith material.

Given the approximate extents of the artisanal gold workings, two conspicuous trends are evident. Most of the larger workings are elongate in a NNE-SSW orientation (approximately 010^o), a trend that is consistent with regional structures and gold mineralised zones in Mali. Oblique to this is a SE-NW trend (approximately 120^o), along which artisanal workings are preferentially elongated.

Structurally, the property includes mapped and inferred linear and curvilinear N-S and NE-SW orientated faults, with most annotated as being associated with dextral movement.

The dominant form of structural development is shear / thrust fronts with secondary internal shear zones and local folding, most of which are now steeply dipping. Gold mineralisation broadly occurs within planar zones that dip steeply to the east at approximately 70º. However, given the apparent structural control on mineralisation, this represents a generalisation and localised variations and complexities will inevitably occur.

At least three different sets of mineralised quartz veins occur. These include a prominent N-S/NNE-SSW striking set that appear to dip steeply to the east and is the principal focus of artisanal exploitation; a less prominent oblique E-W (80-100º) striking sub-vertical set; and a subordinate less continuous sub-horizontal set. All three sets are typically ferruginous and the adjacent wallrock includes remnant sulphides. According to the artisanal miners, the N-S/NNE-SSW set contains the most gold and the sub-horizontal set containing the least.

Figure 4-3: Oblique, southeast-facing view of 2017 Google Earth satellite imagery, showing the artisanal workings on the Sanankoro project (from SRK Exploration Services, 2017).

Gold mineralisation at the Sanankoro project delineated through drilling, is observed along a large mineralised corridor composed of 3 subparallel structures known as Bokoro, Sanankoro and Selin (Figure 4-4).

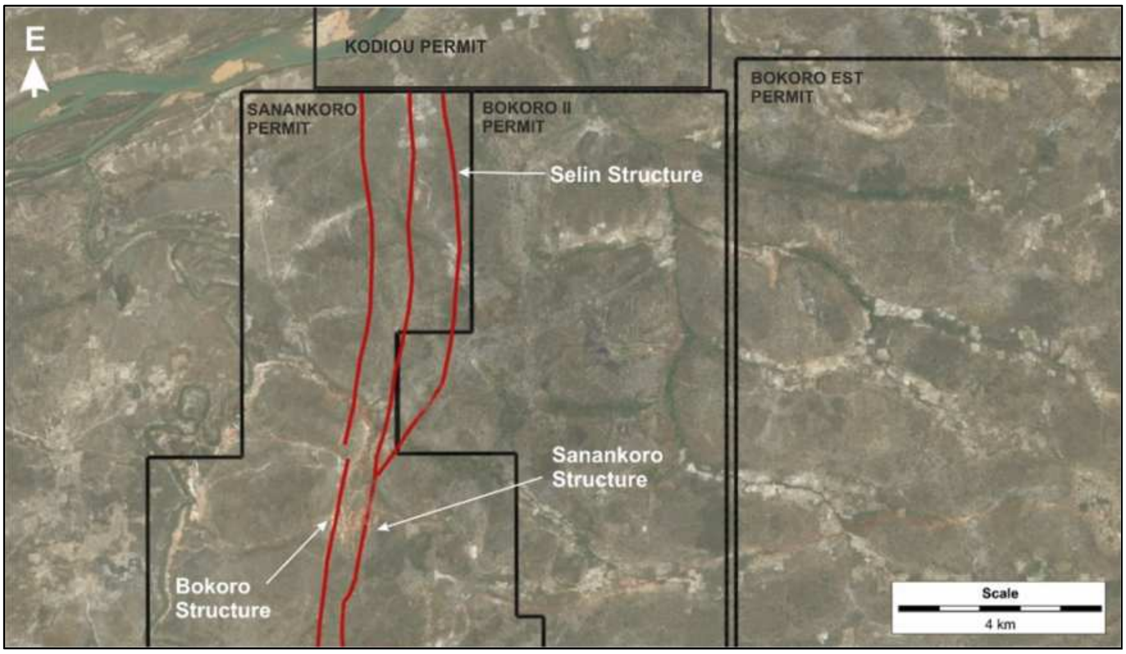


Figure 4-4: The principal gold-bearing structures identified by Cora Gold.

The first two zones can be traced from the north to the south of the Sanankoro permit, over a distance of some 15km, whereas the Selin zone can be traced from the north for a distance of about 10km before it merges with the Sanankoro zone. The occurrence and strike extent of these structure are confirmed by historical and recent ground geophysics.

Sanankoro Structure

The Sanankoro structure has been divided by Cora Gold into three main delineated zones, namely “Zone A”, “Zone B” and “Zone B North”.

The geology of Zone A, Zone B and Zone B North is relatively consistent along strike, being characterised by a steeply dipping sedimentary package that includes sandstones, siltstones and mudstones. A coarse-grained volcanic tuff is prominent in the south and central part of the structure, along with the recognition of a footwall shear zone demarcated by sheared carbonaceous phyllite along which a felsite dyke has been intruded.

Gold mineralisation can be seen in core and from mapping of the excavated pit to be associated with steeply dipping quartz vein sets that variably strike NNE and approximately E-W, along with subsidiary low angle quartz veins that dip to both the east and west. The sub vertical layering and quartz vein sets seen in the pits are also seen to “roll” along open folds with axial planes at a low angle, locally giving the layers a steep dip to the west, as well as the more usual steep dip to the east.

Selin Structure

The lithology along the Selin structure is defined by a package (from hanging to footwall) of a mix of sandstones/siltstones, followed by a 30-40m wide zone of sandstone, which overlies a footwall phyllite, which is interpreted to be carbonaceous. A mafic igneous unit lies within the carbonaceous phyllite, both of which are strongly sheared with alteration of the mafic unit including haematite, sericite and carbonate. A zone of quartz veining (interpreted from drilling results to be sub-vertically dipping), appears to be ubiquitous, and usually constrained within the mafic unit along the structure, with widths often around 30 – 40 m.

The main zone of mineralisation along the Selin structure delineated to date by inclined air core and rotary air blast drill fences is "Target 1", a >3 km long, N-S / NNW-SSE oriented sub-vertically dipping zone, which typically comprises two parallel mineralised structures.

Bokoro Structure

The main zone of mineralisation currently delineated along the Bokoro Structure, is the ~1 km strike length, steeply E/ESE dipping Zone C. The southern end of Zone C is characterised by a main coarse sandstone unit of approximately 5 - 8 m thickness, lying within a sequence of siltstones and phyllites; the latter may locally be carbonaceous and appears to form a noticeable footwall unit. In the centre of Zone C, the sandstone units appear to be reduced / absent, where siltstones / phyllites are more prevalent, whilst the sandstone is observed in the north as multiple layers, possibly indicative of a synformal fold nose. Mineralised quartz veins can be seen to be preferentially associated with the principle sandstone unit in the south, although this relationship is difficult to follow further north. A coalescing of the quartz zones appears to occur in the vicinity of the interpreted fold nose in the north.

4.3.3 Preliminary Genetic Model

Cora Gold have established a preliminary geological model that involves the rotation of the host Birimian sedimentary sequence (comprising interbedded volcanic tuffs and mafic unit, sandstones, siltstones and mudstone) into a N-S orientated sub vertical geometry. The package is repeated by regional-scale, steeply east-dipping reverse faults / thrusts, with associated tight to isoclinal folding. The faulting /shearing provided a focus for the development of extensive zones of quartz veining, iron carbonate and pyrite alteration in association with the gold mineralisation.

The deep tropical weathering in the region has liberated and in parts re-mobilised the primary gold to depths of 40 - 100 m or more.

5 EXPLORATION

5.1 Historic Exploration (2000's – 2012)

Most of the historical exploration activities on the Sanankoro property were completed between mid-2000s and 2012 and included soil sampling, termite mound sampling, ground geophysical surveying (induced polarisation (IP), resistivity and potentially magnetics), trenching, drilling and associated sampling.

5.1.1 Exploration by Randgold Resources

In the mid-2000s, Randgold Resources Ltd ("Randgold") completed a regional soil and termite mound sampling programme that encompassed the Sanankoro property. Termite mound samples were collected on a 200 x 500 m grid, with each second line having a soil sample collected at the same location as the termite mound sample. The collected samples were screened in the field to -1 mm and analysed for gold-only, with results reported in parts per million (ppm) and a detection limit of 0.003 ppm Au, suggesting analysis by fire assay.

Randgold followed-up the reconnaissance programme with more detailed soil sampling over the central part of the Sanankoro permit on a 100 x 200 m grid that covered an area of around 4 x 5 km. The results of the sampling confirmed the presence of a large geochemical anomaly approximately 5 km in length.

This was further followed-up with a series of shallow vertical auger holes (vertical, 12-15 m depth) across the centre of the anomaly, (400 m line spacing) and then infilled over about 2.5 km strike length with a series of rotary air blast ("RAB") fences set about 400 m apart.

5.1.2 Exploration by Gold Fields

In about 2008-09, Gold Fields Ltd ("Gold Fields") commenced exploration on the Sanankoro property. The Gold Fields programme continued from where Randgold had stopped, and comprised further drilling in the Sanankoro permit, as well as infill soil sampling (100 x 200 m grids) in two blocks of about 3 x 8-10 km on the Bokoro II and Bokoro Est permits, and at the eastern end of the Dako permit (on a 50-100 x 400 m grid). Gold Fields also completed ground geophysical surveying including induced polarisation ("IP") and resistivity surveys

The consolidated soil sampling dataset from all permits includes more than 8,300 samples with geochemical results ranging from 0.5 to 4,875 ppb Au (Figure 5-1). The results clearly delineate a large elevated gold anomaly (> 50 ppb Au) approximately 4.5 x 7.5 km within the Sanankoro permit.

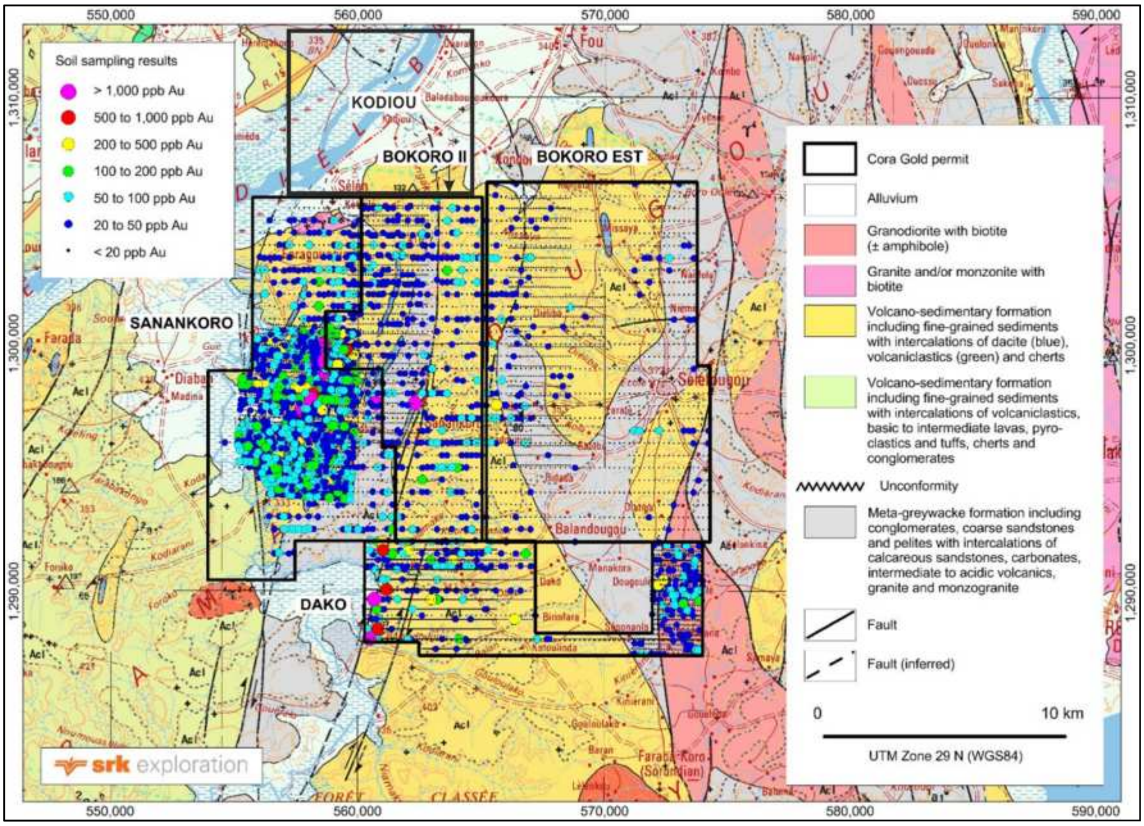


Figure 5-1: Gold Fields Sanankoro soil sampling results (SRK Exploration Services, 2017).

Subsequent to the soil sampling campaign described above, Gold Fields completed a drilling programme with three main objectives:

i) To determine the gold potential of the central part of the Sanankoro permit;

ii) To assess along strike extension to the north and south of the Sanankoro geochemical anomaly and;

iii) To undertake first-pass reconnaissance drilling on the Bokoro Est and Dako projects.

The first objective involved systematic infill drilling using mainly reverse circulation ("RC") holes on fences typically 100 m apart over the southern part of the central area, and fences between 100-200 m apart at the northern end of the central area. It also included follow-up RAB, air core ("AC") and some RC + core tail drilling.

The second objective involved using either AC or RAB to drill vertical holes to depths of 12-15 m, spaced 100m apart along fences 400 m apart and up to 3-4 km in length. The bottom sample was analysed for gold. This was reportedly designed to provide information on regional geology and identify areas of anomalous gold. This was the first pass methodology to look at the north and south extensions to the Sanankoro structure.

Within this grid, a series of inclined RAB holes with typical length about 50 m were then drilled on 400 m fence spacing with collar intervals of about 25 m to follow-up perceived mineralised structures.

The third objective of first-pass reconnaissance drilling on the Bokoro Est and Dako permits involved a similar approach to that used at Sanankoro, but the anomalous bottom samples at the Bokoro Est permit were never systematically followed-up.

At the Dako permit, reconnaissance drilling commenced directly with inclined air core holes set at 40 m collar intervals on fences 620 m apart, with hole lengths of 40-60 m.

The different stages of drilling completed by Gold Fields are provided in Figure 5-2.

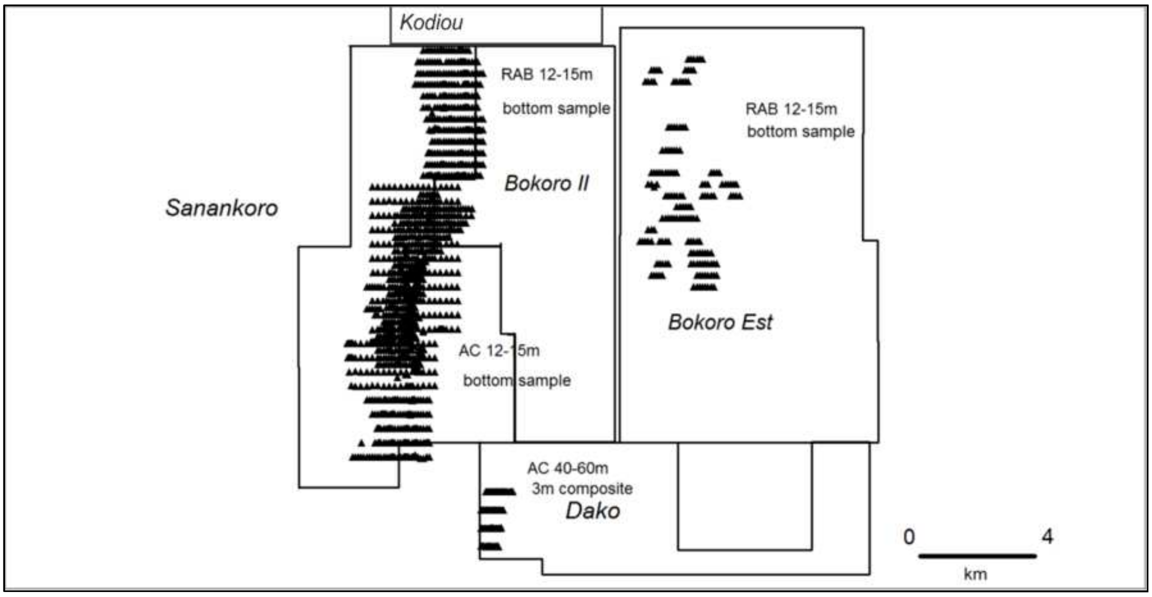


Figure 5-2: Sanankoro historic drillhole coverage (after Cora Gold, 2017).

To follow-up on the RAB drilling programmes of Randgold and Gold Fields, an additional two phases of RC drilling were completed by Gold Fields, namely the “BRC” and “GBRC” programmes. The “BRC” series of RC holes, were mainly completed on NW-SE orientated lines, presumably in the belief that gold structures were trending NE. This was followed on E-W lines by the GBRC series, which were set on fences between 100-200 m apart in Zone A and Zone B. The GBRC series included deeper holes (to 180 m length) which comprised RC holes with diamond core tails.

The total historic drilling completed on the Sanankoro property by Randgold and Gold Fields is summarised in Table 5-1. The locations of the Randgold and Gold Fields drillhole collars are displayed in Figure 5-3.

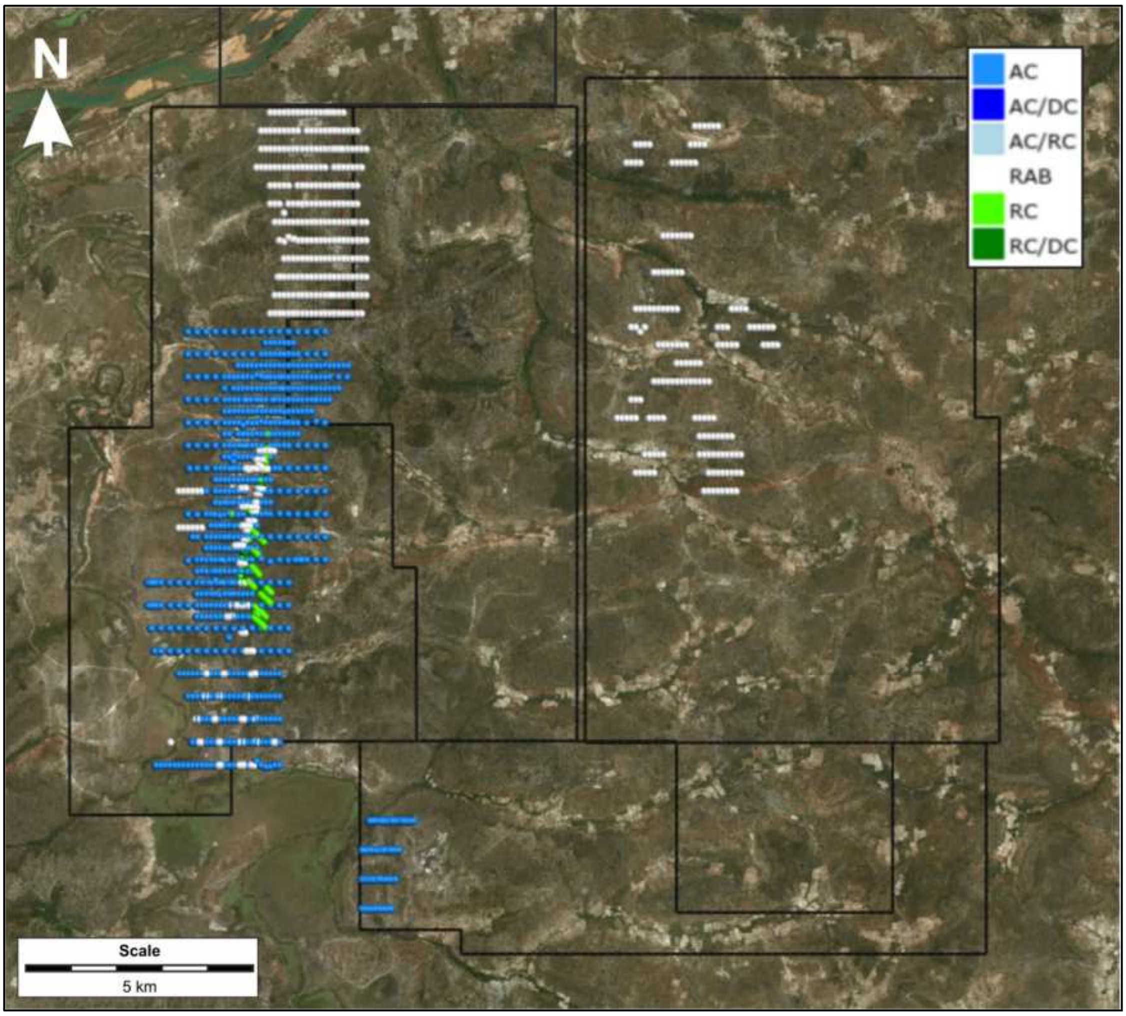


Figure 5-3: All Randgold and Gold Fields drillhole collars, shown relative to the Sanankoro Project permit outlines and the Google Earth satellite imagery.

In total, Randgold and Gold Fields completed 1,798 drillholes for approximately 57,500 m across the Sanankoro Project area. This includes approximately 32,800 m of reverse circulation (“RC”) and air core (“AC”) drilling, in addition to 23,700 m of rotary air blast (“RAB”) drilling. A small number of reverse circulation and air core holes were finished with a diamond core tail. The total length diamond core drilling completed by Randgold and Gold Fields is 910 m. Summary drillhole length statistics by drillhole type for the Randgold and Goldfield campaigns are provided in Table 5-1.

Table 5-1: Randgold and Gold Fields drillhole types and length statistics.

Drilling Type	No. of Holes	Total Meterage (m)	Minimum Length (m)	Maximum Length (m)	Average Length (m)
Air core (AC) + reverse circulation (RC)*	1,007	32,840	6	150	32
Rotary air blast (RAB)	775	23,700	5	87	31
Diamond core tail on AC/RC holes**	16	910	11	104	61

** Total Meterage Minimum, maximum and average air core and reverse circulation lengths include reverse circulation and air core pre-collars.*

***Total Meterage Minimum, maximum and average diamond tail lengths relate to the length of the diamond portion of the holes and exclude the AC / RC pre-collars.*

The drilling completed by Randgold and Gold Fields delineated a mineralised zone referred to as the Central Zone that consists of two sub-zones (Zone A and Zone B – as described in Section 4.3.2) and also what appears to be two parallel mineralised structures spaced 600-700 m apart that extend through the length of the Sanankoro permit. This creates what appears to be two sub-parallel curvilinear mineralised structures 600 to 700 m apart that extend north from the Central Zone for a distance of approximately 11 km. In the Central Zone the structure trends NNE-SSW but appears to inflect towards the NNW-SSE in the north.

Cora Gold interpreted this inflection to be associated with the preferential occurrence of gold mineralisation. Given the orientation of the inflection relative to the dextral sense of movement shown on structures on the published geological mapping, this may have acted as a releasing bend and dilation zone providing a favourable location for the deposition of quartz vein-hosted gold mineralisation. However, the inflection could also represent offsets to the main structures caused by the apparent presence of oblique cross-cutting structures.

Despite the seemingly widespread drilling across the Sanankoro geochemical anomaly, it is considered important to note that many of the drillholes are shallow rotary air blast and air core holes that are not of sufficient depth to fully test the subsurface.

5.2 SRK Note on Randgold and Gold Fields Drilling

Limited information is available to SRK on the drilling, sampling and assaying methods employed during the Randgold and Gold Fields exploration campaigns. Validation checks completed on the Randgold and Gold Fields drilling results are presented in Section 6.2, which indicate that the historic Randgold and Gold Fields drillhole data is sufficiently robust for the use in the derivation of an Inferred Mineral Resource.

5.3 Cora Gold Exploration Activities

5.3.1 Summary of Cora Gold Exploration to Date

Coral Gold commenced exploration on the Sanankoro Project in Quarter 2 2017 and has subsequently completed detailed geological and regolith mapping across an area of some 120 km^2 at a scale of mainly 1:2000 over both the Sanankoro and Bokoro II permits, although local areas more distant from the primary structural corridor were covered at a scale of 1:5000. This work was supported by termite mound sampling and mapping of artisanal workings.

Termite Mound Sampling

The sampling is used to supplement earlier systematic soil geochemistry programmes completed by Rand Gold resources and Gold Fields on grid parameters that range from 400m x 100m to 200m x 100m. The extensive development of ferricrete and transported regolith reduces the efficiency of the soil responses. The sampling involves the collection of a standard weight of material from cathedral and intermediate sized termite mounds. Based on the fact that termites excavate material from depths of many metres from surface in order to reach the water table. Material brought to the surface and deposited in the termite mounds may include gold grains if the termites have descended into a mineralised structure.

Termite mound sampling is a widely used method which has the ability to provide a sampling medium derived from beneath transported materials within the regolith, and can assist in locating primary gold mineralised structures.

The sample is obtained by using a geological hammer to collect a channel sample from the mound tip to its base at the four quadrants of the compass, as well as collect a horizontal channel sample around the circumference of the mound base. The samples are homogenised together, and then split to produce a single sample of 3 kg weight which is returned to the field camp where it is panned, and the number of visible gold grains counted. As such, the sample provides a semi quantitative indication of gold content within the mound, which when considered with the results from adjacent termite mounds, and the earlier soil geochemistry programme can guide subsequent exploration drilling programmes.

Geophysical Surveying

A ground Induced Polarisation (“IP”) survey (gradient array) had previously been completed by Gold Fields (unknown date) over an area of about 8 km^2 along a 1.5 km wide corridor that ran from south of Zone A through to the Zone B North and covered the Sanankoro and Bokoro structures. It is believed that the data was collected on lines 100m apart with stations at 25m along each line. A gradient array using the same grid dimensions was undertaken by Cora Gold which extended the ground IP coverage to the north by a further c 12.5 km^2 in Q1 2018 . The IP survey extends coverage of the Sanankoro, Bokoro and Selin structures by about 6 kms to the north of the Gold Fields survey.

Together the two surveys provide good quality map images that have been processed to show conductivity, resistivity and chargeability anomalies, which represent contrasting lithologies and potentially mineralised geological structures.

Field Density Measurements

During the most recent exploration campaign, Cora Gold completed a series of field bulk density programmes to help to better quantify expected densities within the oxidised portion of the deposit. This consisted of the following:

Saprolite density determinations using small pit excavations:

In total Cora completed 7 bulk density determinations of the saprolite at the base of the artisanal pits in Zone A and Zone C, by the following methodology:

1. A top surface area is prepared by removing all surface detritus and levelled to allow a pit to be prepared.
2. GPS easting, northing and elevation values should be taken at the centre of the levelled area and recorded.
3. A small pit is excavated by hand
4. The excavated pit is carefully filled with sand to the point of overflow and the volume of sand recorded.
5. The extracted material is dried over a covered oven heated by fire for at least 12 hours and subsequently weighed.
6. Bulk density is simply calculated as weight / volume.

Grab samples of saprolitic rock:

Cora Gold completed 32 bulk density determinations on grab samples of saprolitic mineralisation, taken from the base of the artisanal workings. The density of these samples was calculated using the water immersion method. This methodology involves weighing samples in air and water using a balance. Firstly, each sample was weighed in air. All samples were then wrapped in a very thin layer of plastic (cling film or glad wrap) in order to prevent air in any pores escaping from the sample, and subsequently weighed in both air and water, using a balance with top and modified under-slung measuring capabilities. Bulk density was calculated using the formula as *"W1/(W1/(W2-W3)"*, where *W1* is the dry weight, *W2* is the weight in water and *W3* is the weight of the cling film.

Grab samples of hardcap

Bulk density determinations were completed on a total of 6 grab samples, comprising 2 from each of Zone A, Zone B and Zone C. The density of these samples was calculated using the water immersion method, similar to the methodology described for the grab samples in the saprolite. It was not deemed necessary to wrap the hard cap samples in plastic prior to weighing in water. Density was simply calculated as *"W1/(W1/(W2)",* where *W1* is the dry weight, and *W2* is the weight in water.

The location of all field density samples is displayed in Figure 5-4.

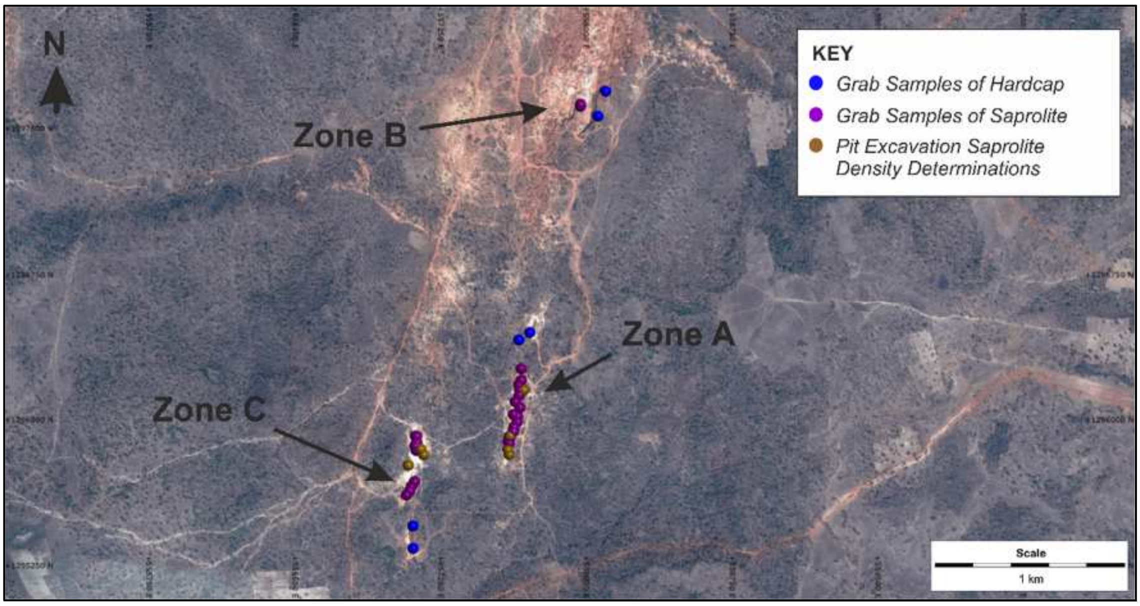


Figure 5-4: The location of the field density measurements completed by Cora Gold, shown in relation to Google Earth satellite imagery.

5.4 Cora Gold Drilling

5.4.1 Overview

Between December 2017 to September 2019, Cora Gold have completed a total of 264 drillholes across the Project area, for a total meterage of approximately 23,100 m. Drilling completed by Cora Gold includes a combination of reverse circulation ("RC"), air core ("AC") rotary air blast ("RAB") and diamond ("DC") drillholes, with diamond core tails on a small number of RC and AC holes, and a single dedicated diamond drillhole. Specifically, the Cora Gold drilling comprises 236 reverse circulation and air core holes, 17 rotary air blast holes, 5 reverse circulation tails on air core holes, 10 reverse circulation and air core holes with diamond tails and 1 dedicated diamond hole (Table 5-2).

Table 5-2: Cora Gold drillhole types and length statistics.

Drilling Type	No. of Holes	Total Meterage (m)	Minimum Length (m)	Maximum Length (m)	Average Length (m)
Air core (AC) + reverse circulation (RC)*	236	21,600	9	141	88
Rotary air blast (RAB)	17	340	20	20	20
Diamond core tail on AC/RC holes**	10	1,100	46	164	109
Diamond (DC)	1	70	70	70	70

** Total Meterage Minimum, maximum and average air core and reverse circulation lengths include reverse circulation and air core pre-collars.*

***Total Meterage Minimum, maximum and average diamond tail lengths relate to the length of the diamond portion of the holes and exclude the AC / RC pre-collars.*

For holes with diamond core tails, the total length of the reverse circulation and air core pre collars is 550 m, with diamond core drilling commencing at depths of between 22 m and 78 m.

The first phase of exploration drilling by Cora Gold started in December 2017 and primarily consisted of reconnaissance drilling on a 160 to 320m fence spacing over a strike length of more than 10km. This drilling program was completed by May 2018 with 135 drill holes drilled for a total of approximately 13,000 m, including approximately 12,500 m of air core ("AC") and reverse circulation ("RC") drilling, and 500m of diamond core. The drill holes varied in depth between 32m and 200m, with an average of drilling depth of 95m.

The two zones of gold mineralisation (Zones A and B) that had been the focus of the Gold Fields drill campaign (as described in Section 5.1) were subject to a small amount of check RC and core drilling by Cora but did not feature significantly in the 2018 programme which was designed primarily to commence the process of making new discoveries within the Selin Structure, the Bokoro Structure and along-strike of Zones A and B on the Sanankoro Structure.

A second phase of exploration drilling was completed by Cora Gold between December 2018 and June 2019. This second Cora drill campaign comprised primarily air core and reverse circulation holes, with a small number of diamond tails completed on reverse circulation collars. Drilling was primarily focussed on the following:

- Infill drilling on the mineralised Selin structure;
- Step-out drilling on the Selin structure to the north of the previously delineated mineralisation;
- A smaller number of infill drillholes at Zone A, Zone B and Zone B North
- Infill drilling and step out drilling to the north of the previously defined Target 1S / Selin South mineralised structure
- A small number of exploration drillholes at the "excavator" structure, which lies on the Bokoro trend highlighted in Figure 4-4
- A small programme of RAB drilling in the far south of the project area to follow up on a previously identified exploration target

Across the two Cora Gold drill campaigns, drilling has typically been completed on 60 – 120 m spaced sections, with between 1 and 5 holes per section. Drilling was primarily focussed on the oxidised mineralisation in the hardcap, saprolite and saprock. A smaller number of deeper holes targeted the sulphide mineralisation at depth, with roughly 50 holes intersecting sulphide mineralisation, to a maximum depth of approximately 200 m below surface.

The locations of the Cora Gold collars are displayed in Figure 5-5.

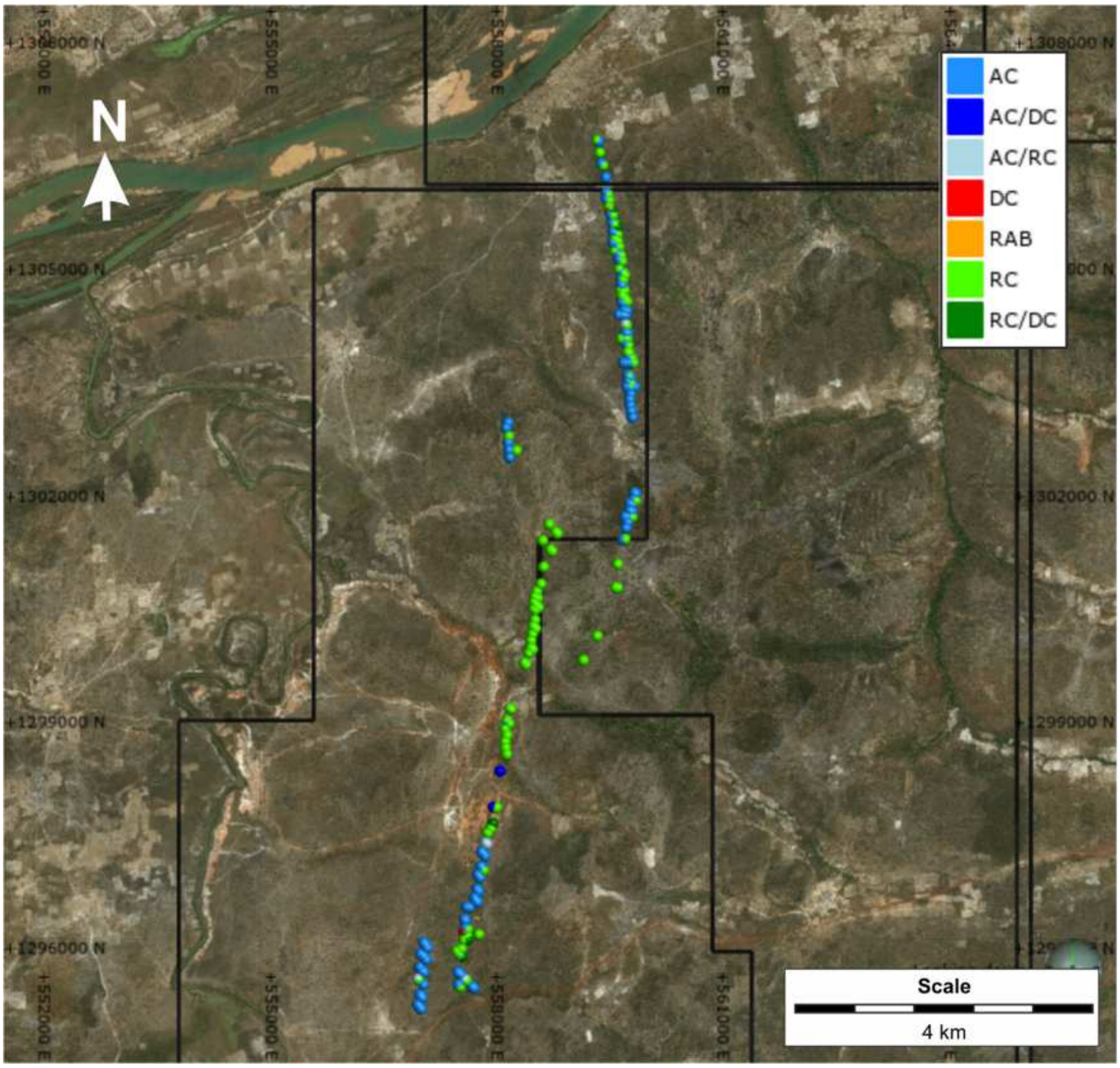


Figure 5-5: Map of the Cora Gold drillhole collars, shown relative to the Cora Gold permit outlines and Google Earth satellite imagery.

Both AC/RC and Diamond drilling was completed by Target Drilling using a multi-purpose KL 900 truck mounted RC/core drill rig with a 350 psi / 1150 cfm compressor and 6m runs. From Q2 2019 a booster was used in support of the rig, where deeper holes were required. A HQ3 drill core diameter was employed in unconsolidated ground, with HQ core collected in solid, fresh rock. The HQ3 core was drilled in 1.5 m runs and the HQ core drilled in 3 m runs.

5.4.2 Collar Survey

All Cora drillhole collars were surveyed with standard hand-held GPS equipment and later confirmed by a contractor, who re-surveyed the collars with Differential Global Positioning System “DGPS” to allow greater control and accuracy of the drill collars to be established.

5.4.3 Downhole Surveys

Downhole surveying has not been completed on the majority of the Cora drillholes. Specifically, downhole surveying is restricted to a total of 45 of the 264 Cora Gold holes, limited to the diamond core drilling and some of the deeper reverse circulation holes. For these holes downhole surveying is generally completed at 40 – 60 m increments, using a Reflex EZ-TRAC downhole survey tool. The reverse circulation, air core and rotary air blast hole that are not downhole surveyed range in length from 9 m to 140 m, with an average length of 80 m. For these holes, the hole dip and azimuth are derived from the measured hole dip and azimuth taken at the drillhole collar. It is noted that, for those holes on which downhole surveying has been completed, the downhole deviation in hole dip and azimuth is generally considered minimal, and that visual assessment of the 3D location of mineralised intercepts in un-surveyed holes typically indicates a reasonable consistency with mineralised intercepts in nearby holes on which downhole surveying has been completed. That said, SRK would recommend that in future drill campaigns, downhole surveying is completed on all holes that exceed a length of around 50 m.

Downhole structural orientation has been completed on the diamond core tails of 5 of the Cora Gold holes. Using an ACT III H/H3 survey tool. Core orientation is performed on core that is competent and in which the core orientation line can be confidently traced across pieces of core.

5.4.4 Logging and Sampling Procedure Overview

Cora Gold has put in place a logical logging and sampling procedure to guide the on-site staff through the technical process. This aims to ensure a consistent methodology for the process of submitting the samples for external laboratory analysis.

At the drill site, the drill core from diamond drilling is packed in metal core trays, with wooden blocks separating each core run. The recording of core recovery, geotechnical and structural logging takes place at the drill site in order to avoid any the inclusion of artificial fractures induced during transport in the database. After completing the geotechnical and structural logs, the core is transported to the main field camp for descriptive logging, density determinations, further structural / alteration measurements and photographing.

All RC and AC chip logging and sampling was completed at the drill site under the supervision of the Cora Gold geology team.

5.4.5 Sample Recovery

Both total core recovery (“TCR”) and solid core recovery (“SCR”) are recorded by Cora Gold for all diamond core into a geotechnical log sheet. This is recorded at intervals of variable length (typically between 0.5m and 3m), with interval lengths determined on the basis of lengths of core with similar geotechnical characteristics. Total core recovery is generally good, with an average recovery of approximately 92%. 100% core recovery is achieved for more than 90% of the geotechnical intervals with roughly 70% of geotechnical intervals having a recorded total core recovery of and greater than 90%. Only a small length of intervals (<4%) have a recovery of <50% and these are mostly smaller intervals with lengths of 1m or less.

Core recovery data is available for 11 of the diamond tails completed on the historic Gold Fields holes. Total Core Recovery was recorded in geotechnical log sheets, in intervals of variable length, typically between 1 m and 4 m, with an average length of 2 m. Core Recovery from the Gold Fields diamond holes is lower than the Cora Gold drilling, at an average of 83%.

AC/RC samples are weighed after each run at the rig site. Samples are generally dry. If wet samples are encountered over a 6m run then the hole is stopped. No reconciliation between theoretical and actual recovery has yet been made although Cora Gold consider recovery to be generally good.

5.4.6 Geological and Geotechnical Logging

Cora Gold geologists complete geological logging on all reverse circulation, air core and diamond holes. Geological logging has not been completed on the small number of RAB exploration holes. All logging is done in hard copy and later transferred into excel spreadsheets.

Cora maintain a detailed geological log of the RC and AC chips. Information recorded includes rock type, regolith type, the type and intensity of sulphide mineralisation, the type and intensity of veining, the style and intensity of alteration, colour, and a geological description. All logging and sampling of RC and AC chips is completed at the drill site. In addition, a 1 kg sample per metre, or combined into a 3m run, is washed and panned on site with the number of spots and/or grains of gold counted and recorded.

Diamond core is geologically logged into a separate bespoke spreadsheet, which includes similar information to the RC and AC chip logging, but with additional detail on the thickness and style of veining, and the texture and distribution of sulphide mineralisation and alteration.

A basic mechanical geotechnical log is maintained for all diamond core. This includes the recording of core recovery data, as described in Section 5.4.5, in addition to rock strength, the number of joints in 30° alpha angle buckets, joint roughness, infill mineralogy, the pervasiveness of alteration associated with jointing and the calculation of rock quality designation (“RQD”) from solid core recovery.

Structural geotechnical logging has been completed on 3 of the structurally oriented diamond tails. This includes the recording of alpha and beta orientation data for individual joints, with associated micro roughness, infill type and infill thickness for each joint. A flexible sleeve is used for the calculation of alpha and beta angles.

In addition to geotechnical structural data, Cora Gold geologists have also recorded dips and dip directions for a small number of quartz veins in the structurally oriented diamond core. This includes the capture of complimentary data including vein thickness and the style of veining (e.g. planar, irregular etc.). This data was taken by the on-site geologists, directly with a compass clinometer, with the core correctly orientated according to the drillhole survey using a core orientation frame.

The geotechnical and structural logging described was completed at the drill site, whilst the geological logging was undertaken at the field camp where the core was also photographed.

5.4.7 Density Determinations

The on-site geology team completed density determinations on the diamond drill core, at variably spaced intervals on average 4 m apart. Measurements were completed on whole core pieces, normally 10-15 cm in length. Two methods for bulk density determination were carried out on each piece of core, as described below. For both methods, all samples were thoroughly dried using an oven prior to any analysis.

Water Immersion:

This methodology involves weighing samples in air and water using a balance. Firstly, each sample was weighed in air. All samples were then wrapped in a very thin layer of plastic (cling film or glad wrap) in order to prevent air in any pores escaping from the sample, and subsequently weighed in both air and water, using a balance with top and modified under-slung measuring capabilities. Bulk density was calculated using the formula as *"W1/(W1/(W2-W3)"*, where *W1* is the dry weight, *W2* is the weight in water and *W3* is the weight of the cling film.

Volumetric Determination:

This method involves first measuring the dry weight of the sample, and then calculating the volume of the sample to derive a density. To calculate the volume of the core samples, length was derived by measuring the length of the core with a measuring tape along 3 different parts of the drill core perimeter. These were then divided to derive an average length value. This was multiplied by the drill core radius (which was assumed to be constant and given as 3 cm for HQ3 diameter core and 3.15 cm for HQ core) and further multiplied by π to derive volume. Density can then be simply calculated as *weight / volume.*

5.4.8 Sampling Procedure

Reverse Circulation and Air Core Drilling

For all RC and AC holes, samples are collected at each metre, from the cyclone, into a 50kg plastic bag. The sample is then immediately weighed.

A Rifle Splitter (provided by the drilling company), is then used to homogenise and split the material collected at each metre (the material is passed through the splitter twice), with approximately 1/8 of the material taken for sampling. The remaining material (reject) is then returned to the 50kg sample bag and is used for gold panning and logging (as described in Section 5.4.6).

Depending on the results of logging and/or panning, the 1/8 samples are either composited to a 3 m composite sample or retained as a 1 m sample. To prepare samples for shipment to the analytical laboratory, the final 1/8 samples are homogenised further, by passing through a Gilson Porta Splitter (model SP2). For the 3 m composite samples, each 1 m sample is passed through the Gilson splitter and split into two, with one half stored as a field duplicate. The 3 m composite sample is then passed through the splitter again to homogenise it. For the 1 m samples, the sample is passed through the Gilson splitter, with one half taken for sampling and the other stored as a field duplicate.

As part of the sampling procedure, the following QAQC sample insertion protocol is practiced:

- Submission of standards within drill sample batches at a frequency of 1 in 20 (5%)
- Submission of blanks, prepared on site, within drill sample batches at a frequency of 1 in 20 (5%)
- Submission of field duplicates within drill sample batches at a frequency of 1 in 20 (5%)

Diamond Drilling

Prior to sampling, the diamond core was split using a diamond core saw. The core was split vertically down the core axis, normal to the foliation/bedding to produce two identically sized sections of half core. In structurally oriented holes, in order to preserve the orientation lines for further structural measurements, the half core with the orientation line preserved was retained and the other half sampled.

Sample intervals were determined by the geologist and kept at regular 1m intervals except were lithological or mineralised contacts were encountered. In saprolite or crumbly zones, the core was cut with a knife in the box and the one half sampled with a plastic spoon. Half core samples were collected in numbered sample bags, sealed and stored prior to transport to the analytical laboratory. The samples were arranged in batches of 20 samples that included 1 standard and 1 blank.

5.4.9 Sample Storage

Duplicate AC/RC samples, each weighing about 3kg, are individually stored in numbered plastic bags with samples for each hole, combined into a labelled rice sack prior to storage within a safe area of the field camp. Core is stored in metal core trays within a covered core storage area, which also houses chip trays for each AC/RC hole.

5.4.10 Sample Shipment and Chain of Custody

Since the RC and AC chips are sampled at the drill site, a tracking form is filled in and is signed by both the geologist on site and the driver of the vehicle transporting the samples. Once arrived at the camp, the samples are received by the camp manager, who will also sign and file the tracking form.

Both the RC and RC chip samples, and the diamond core samples are sent from the field camp to Bamako, where they are directed onwards to either to SGS Ouagadougou or SGS Bamako. Transportation of the samples from the field camp to Bamako is by vehicle. Another a tracking form is filled in by the geologist on site and signed by both the geologist and the driver. A copy of the form is given to the driver (to be handed to the administrator in the Bamako office) and is also emailed to the office in Bamako on the day of departure. Once the samples have arrived at Bamako, they are inspected and (if relevant) air transport information is completed, before being sent to the lab.

5.4.11 Sample Preparation and Analysis

At the outset of Cora Gold's exploration programme in Q4 2017/Q1 2018 at Sanankoro, the oxide samples from the first round of drilling were sent to the SGS laboratory in Bamako, Mali for 50 g fire assay. This has remained the case for sulphide drill samples.

During this first phase of drilling, panning of the drill sample at the rig side confirmed that coarse gold (>100 micron) was a regular feature of the mineralisation style. As a consequence, on receipt of this initial batch of oxide samples analysed by 50 g fire assay, a selection of duplicate oxide samples were sent for 2 kg bottle roll analysis from which it was concluded that a more representative assay was derived by analysis of the larger oxide sample.

It was therefore decided to preferentially use cyanide bottle roll as the analytical technique for determining the gold content for all subsequent oxide drill samples. The bottle roll technique allows for a substantially larger sample (2 kg) to be analysed, compared to 50 g fire assay, thereby providing the opportunity for a more representative gold analysis. In the Q2/2018 drill programme 2 kg samples were split at the rig side for dispatch to the laboratory, but from Q1 2019 this was increased to a 4 kg sample to further improve the representative nature of the sample. A description of the sample preparation and analytical procedures for both the fire assay and bottle roll analyses is provided below.

Fire Assay

All samples analysed by fire assay alone were completed at the independent SGS laboratory facility in Bamako, Mali. 1 kg samples are collected at the drill rig site, following a process of homogenisation and splitting by riffle splitter. On arrival at the laboratory, the sample is received, checked against the dispatch notes and entered into the laboratory tracking system prior to being weighed and then dried in an oven at 105 +/- 10˚c. The entire 1kg dry sample is then crushed with 75% < 2mm followed by pulverisation with 85% < 75 micron. Once pulverised, the 1 kg sample is split, and a pulp sample is collected for Au 50g FA analysis and the reject stored.

The fire assay analysis comprises the pulverized samples being weighed and mixed with flux and fused using lead oxide at 1,100°C, followed by cupellation of the resulting lead button (dore bead). The bead is digested using 1:1 HNO_3 and HCl and the resulting solution is submitted for analysis. The digested sample solution is aspirated into a Flame Atomic Absorption Spectrometer (AAS), aerosolized, and mixed with combustible gas, acetylene and air. The mixture is ignited in a flame with temperatures ranging between 2,100 and 2,800°C. During combustion, atoms of gold in the sample are reduced to free, unexcited ground state atoms, which absorb light. Light of the appropriate wavelength is supplied and the amount of light absorbed can be measured against a standard curve.

The results of the fire assays are transferred online to the Laboratory Information Management System (SLIM) which has in place a secure audit trail. Results are reported in ppm with a lower detection limit of 0.01 and an upper detection limit of 100ppm.

Bottle Roll

For bottle roll analysis, 4 kg samples (2 kg during earlier phases of drilling) were collected at the drill rig site, following a process of homogenisation and splitting by riffle splitter. The samples are sent to the independent SGS laboratory facility in Ouagadougou, Burkina Faso. On arrival at the laboratory, the sample is received, checked against the despatch notes and entered into the laboratory tracking system prior to being weighed and then dried in an oven at 105 +/- 10ºc. The entire 4 kg (2 kg during earlier phases of drilling) dry sample is then crushed with 75% < 2 mm followed by pulverisation with 85% < 75 micron. Once pulverised, the 4 kg sample is split into a 2 kg sub sample and the reject stored.

The 2 kg sample is placed into a bottle, with 5 leachwell tablets and 10gm of lime along with 4000ml of water. The bottle is capped, shaken thoroughly to mix the ingredients prior to being placed on a roller for 12 hours. At the end of the rolling period, the bottle is removed from the roller and allowed to settle. After settling, the clear solution is decanted into disposable cups from which a 40 ml aliquot is collected into a culture tube, where 0.5ml of 0.25% NaCN solution and 4 ml of DBK is added. The solution is shaken vigorously for 1 minute before being analysed for gold by AAS with a detection limit of 0.01 ppm.

Where sample analysis is 0.5 ppm or better, the residue from the settled bottle roll is collected and analysed by 50 g fire assay, to enable a total gold assay of the sample to be calculated.

5.4.12 Database management

Data is collected in the field on paper log sheets which are stored in files, with all data transferred into excel spreadsheets. This is reviewed by the geologist at the site prior to forwarding to the database manager based in the UK. The data is verified with queries returned to the field where necessary, prior to being saved into a project specific Access database in the UK where standard back up procedures are maintained.

6 DATA VERIFICATION

6.1 SRK Site Visit

SRK have not completed a Competent Persons site visit to the Sanankoro Project. The geological interpretation of the deposit and controls on mineralisation have been developed by Cora Gold. All data upon which the Mineral Resource Estimate is based has been provided to SRK by Cora Gold, and SRK have not completed any independent checks on the logging, sampling or drill protocols put in place by Cora Gold.

6.2 Validation of Historic Assay data

Cora Gold have not undertaken any direct twinning of the historic Randgold or Gold Fields drillholes, as Cora has elected to drill on an azimuth to the NW to capture information from both the northerly and easterly quartz vein sets, whilst previous exploration was drilled solely on a western azimuth.

In the absence of twin drillholes to validate the historic drillhole data, SRK have completed a high level statistical and visual validation comparison of the historic drillhole data with the Cora Gold drilling to determine the suitability of the historic drillhole data for its use in deriving a Mineral Resource Estimate.

Only minimal historic drilling was completed on Zone B North and Selin, precluding a meaningful comparison of the results of the these holes with the Cora Gold drilling. For this reason, the validation completed by SRK has been restricted to drilling of Zone A and Zone B.

SRK have completed a Q-Q plot analysis of the mineralised intersections inside of the mineralisation wireframes described in Section 8.2.2, comparing the Randgold and Gold Fields assays with the Cora Gold assays (both composited to 3 m in order to provide a like-for-like comparison). A Q-Q plot is a scatterplot of two sets of quantiles against one another. If both sets of quantiles have the same grade distribution then the quantile line should be straight, and roughly 1:1. Q-Q plots of the historic assays against the Cora Gold assays, inside of the mineralisation wireframes, are provided separately for Zone A and Zone B in Figure 6-1. The Q-Q plot for Zone B generally indicates the Cora Gold and historic drillholes have similar grade distributions. However, the Q-Q plot for Zone A shows that, above ~ 1 g/t Au, the distribution of assays in the Cora Gold holes is higher grade than the historic drilling. This can, at least in part, be attributed to the spatial arrangement of the Cora Gold holes in Zone A, which are clustered in one of the higher grade portions of this zone (see Figure 6-2).

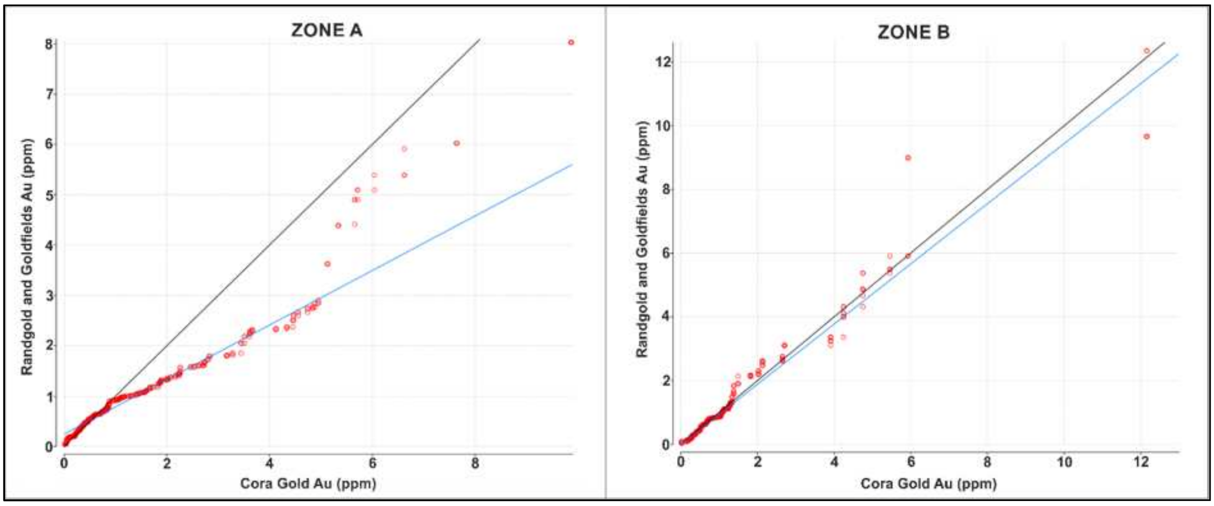


Figure 6-1: Q-Q plots of the Cora Gold Au assays against the Randgold and Gold Fields Au assays, inside the mineralisation wireframes at Zone A and Zone B. The 1:1 correlation is shown in black and the quantile correlation shown in blue.

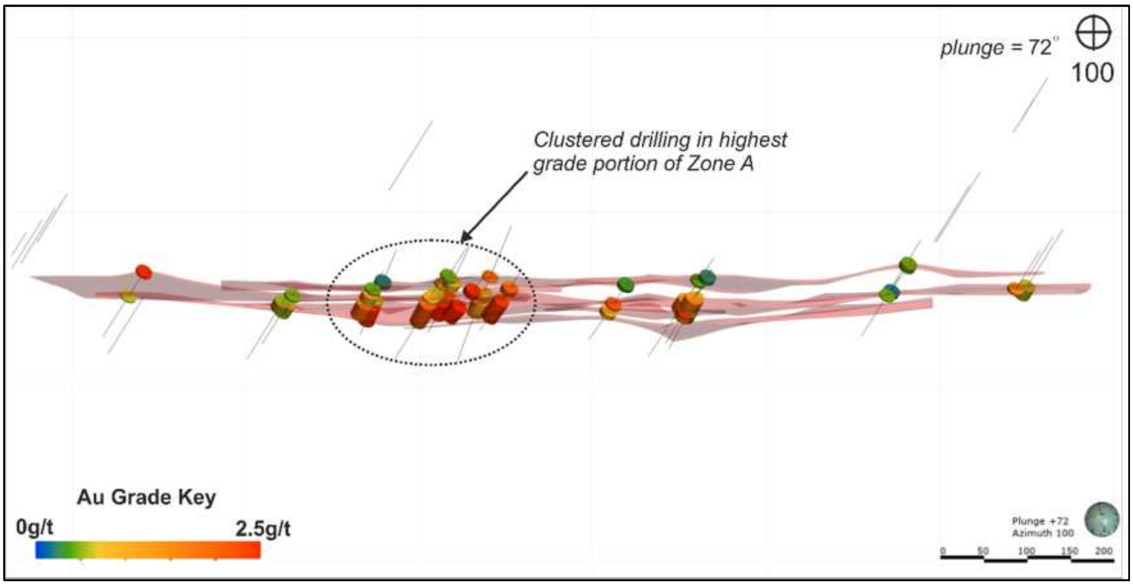


Figure 6-2: View looking down-dip of the modelled mineralised structures at Zone A, showing the Cora Gold drillholes, with intersections inside the mineralisation wireframes coloured by Au grade.

In order to provide an comparison, unaffected by clustering, of the Cora Gold drilling with the historic drilling in Zone A, SRK completed a basic interpolation of the assay data inside of the largest mineralised structure in Zone A (Zone A 2, as described in Section 8.2.2), using the Leapfrog Geo radial basis function ("RBF"). Two interpolations were undertaken, one on only the Cora Gold drillhole assays and one on only the Randgold and Gold Fields assays. Figure 6-3 displays the Zone A 2 mineralisation wireframe coloured by the Cora Gold assay RBF interpolant, whilst Figure 6-4 displays the Zone A 2 mineralisation wireframe coloured by the historic assay RBF interpolant. The drillhole intervals displayed alongside the Cora Gold interpolant in Figure 6-3 are the historic Randgold and Gold Fields drillhole intersections (composited to a single composite per intersection for visualisation purposes). Correspondingly, the drillhole intervals displayed alongside the historic drillhole interpolant in Figure 6-4 are the

Cora Gold intersections. The RBF interpolants of the two datasets show that both drillhole phases are characterised by similar spatial distributions of Au mineralisation. For the most part, the Cora Gold intersections correlate well with the RBF interpolant of the historic grades and vice versa when comparing the historic intersections with the Cora Gold interpolant. The only major exception to this rule is in the south of the structure, where the historic interpolant includes a significant high grade patch, absent from the Cora Gold interpolant. However, this is in the most sparsely drilled area of Zone A 2 and is largely based on a single high grade intersection in the Randgold and Gold Fields drilling.

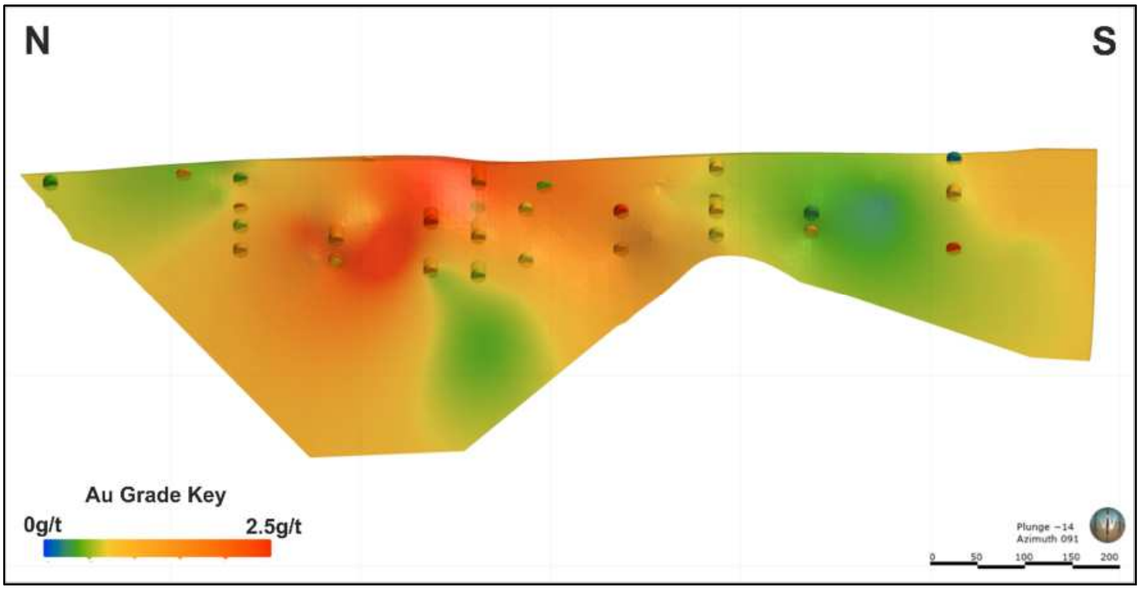


Figure 6-3: An east-facing long section of the Zone A 2 mineralisation wireframe, coloured by an RBF interpolant of the corresponding Cora Gold drillhole grades, shown alongside the historic drillhole intersections.

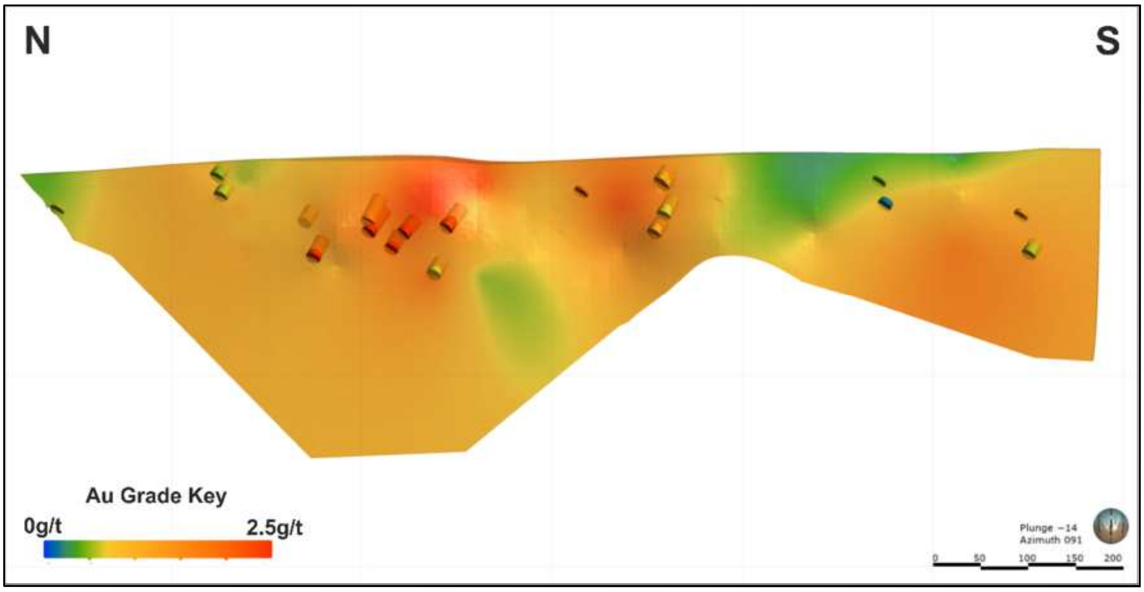


Figure 6-4: An east-facing long section of the Zone A 2 mineralisation wireframe, coloured by an RBF interpolant of the corresponding historic drillhole grades, shown alongside the Cora Gold drillhole intersections.

Visual analysis of the mineralisation wireframes described in Section 8.2.2, shows no obvious pinching or swelling associated with either the Cora Gold intersections or the historic drillhole intersections.

Based on the validation checks described above, SRK consider that, coupled with recent Cora Gold drilling, the historic Randgold and Gold Fields drillhole data is sufficiently robust for the use in the derivation of an Inferred Mineral Resource. The validation checks undertaken by SRK are inexact, and no direct comparison of the Cora Gold drilling with the historic drilling has been possible at this stage. SRK would recommend that Cora Gold complete a programme of twin validation drilling on a selection of historic drillholes as part of the next phase of drilling.

6.3 Collar validation

Upon receipt of the final drillhole database from Cora Gold, SRK completed a visual verification of the spatial location of the Cora Gold drillhole collars against the historic Gold Fields and Randgold collars. The result of this exercise highlighted some local discrepancies between the elevation of the Cora Gold collars (excluding those that were collared in the artisanal workings) and the historic collars. Specifically, when comparing close-spaced Cora Gold collars with historic Randgold and Goldfield collars, the historic collars often have an erratic distribution of elevation values and, on average are higher in elevation that the Cora Gold collars.

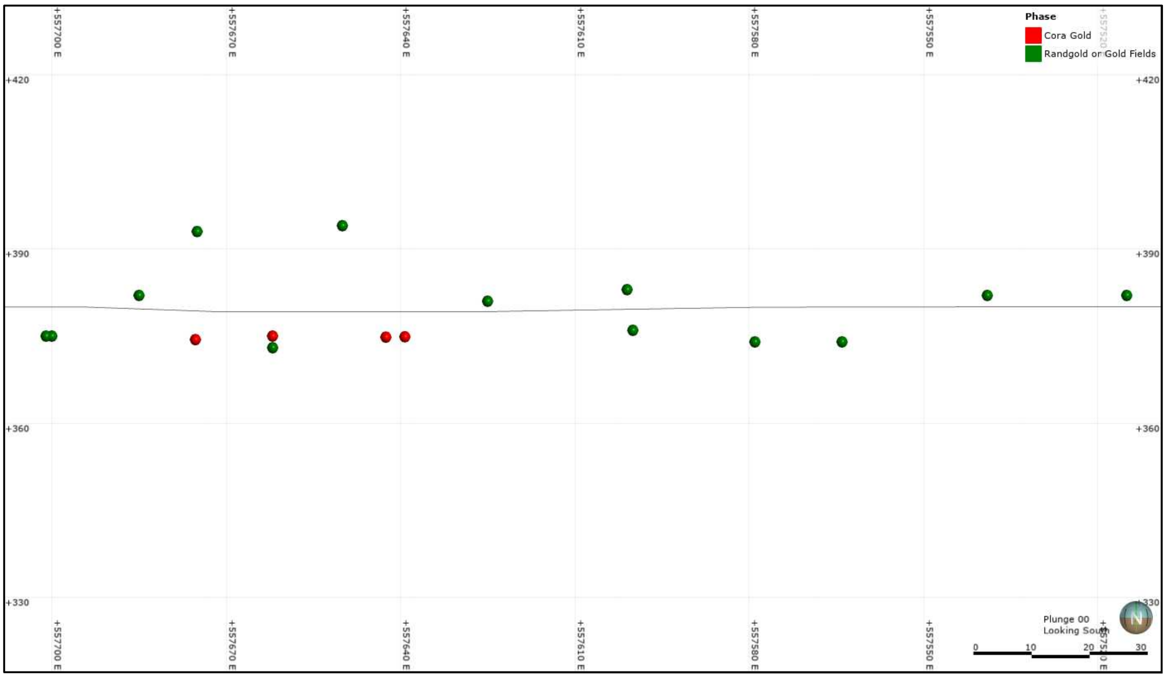


Figure 6-5: An E-W Section through Zone A, showing the elevation of the Cora Gold collars (in red) and the Randgold and Gold Fields collars (both in green). A topography surface derived from ASTER digital elevation data is displayed as a black trace.

Table 6-1 displays the average difference in elevation between the both the Cora Gold collars and the historic collars when compared to a topography surface generated from ASTER digital elevation data. This confirms that, on average, the historic collar coordinates have a significantly higher z value than the Cora Gold collars.

Table 6-1: Elevation differences between the ASTER topography surface and the Cora Gold and historic collars.

Collar Type	Average Elevation	Average Elevation of collar points projected onto ASTER surface	Average difference in elevation between collars and ASTER	Average difference in elevation between collars and ASTER (+ or -)
Cora Gold Collars	360.4m	365.8m	-5.5m	5.7m
Historic Randgold / Gold Fields Collars	366.7m	364.3m	+2.5m	6.4m

The Cora Gold collars were surveyed using a differential global positioning system "DGPS", which generally provide highly accurate elevation values. Added to this, visual assessment of the Cora Gold collars does not highlight the same degree of local deviation in elevation as the Randgold and Goldfield collars. For this reason, for the purposes of mineralisation modelling, it was decided to generate a topography surface directly from the Cora Gold collars, but retaining the trend of the ASTER digital terrane model between collar points, and to move the historic collars to the elevation of this surface at their respective x-y locations.

6.4 Validation of Final Sampling Database

SRK completed a phase of data validation on the digital sample database supplied by the Company which included, but was not limited to the following:

- Search for sample overlaps or errors in the length fields
- Search for duplicate samples
- Search for erroneous, anomalous or absent Au assay values
- Check for logging and sample intervals that exclude the hole maximum depth outlined in the collar file
- Visual inspection of hole traces to check for potentially spurious downhole surveys

No material issues were noted in the final sample database. All errors or inconsistencies or errors identified were flagged with the Cora Gold geology team, who checked the digital database against the original log sheets or assay certificates. SRK then made any necessary changes to the digital database to be used in the estimate depending on the result of the checks carried out by Cora Gold. To prevent the smoothing of higher grades in un-sampled intervals, SRK has replaced all absent or negative Au assay values with a low grade background value of 0.001 ppm.

6.5 Comparison of Fire Assay and Bottle Roll Analyses

As described in Section 5.4.11, the gold values in the Sanankoro assay database are a combination of fire assays and bottle roll analyses. Bottle roll analysis was Cora Gold's preferred analysis method used for samples taken from the oxide profile, with fire assay being employed primarily for sulphide or transitional samples.

A total of 113 intervals have been analysed by both fire assay and bottle roll. All fire assays were undertaken at SGS Bamako, whilst all bottle roll analyses were completed at SGS Ouagadougou. The samples with both fire assay and bottle roll analyses are a combination of the following:

- Lab reject pulps from 7 of the first Cora Gold RC holes, initially analysed by fire assay and subsequently re-analysed by bottle roll;
- Lab reject pulps from a single RC hole initially analyses by bottle roll and subsequently re-analysed by fire assay;
- A single diamond hole, for which ½ core was submitted for metallurgical testwork, ¼ core submitted for bottle roll and 1/8 core submitted for fire assay;
- A single diamond hole for which lab reject pulps from samples originally analysed by bottle roll were submitted for fire assay.

Scatterplot analysis of intervals analysed by both fire assay and bottle roll are provided in Figure 6-6 to Figure 6-8. In general, the scatterplots show an acceptable level of correlation between the fire assay and bottle roll analyses. Typically, at lower grades of less than around 2 ppm, the fire assays under-report gold grade relative to the bottle roll analyses. Between 2 ppm and 10ppm the distribution around the 1-1 correlation is more erratic, but the majority of samples report higher grades in the fire assay results. At very high grades in excess of 10 ppm, the bottle roll analyses typically return higher grades than the fire assays. Overall, the squared correlation coefficient between the bottle roll analyses and fire assays is deemed reasonable for a gold project of this nature. On this basis it is considered appropriate to utilise both fire assay and bottle roll assay results in deriving a Mineral Resource Estimate for the Sanankoro Project.

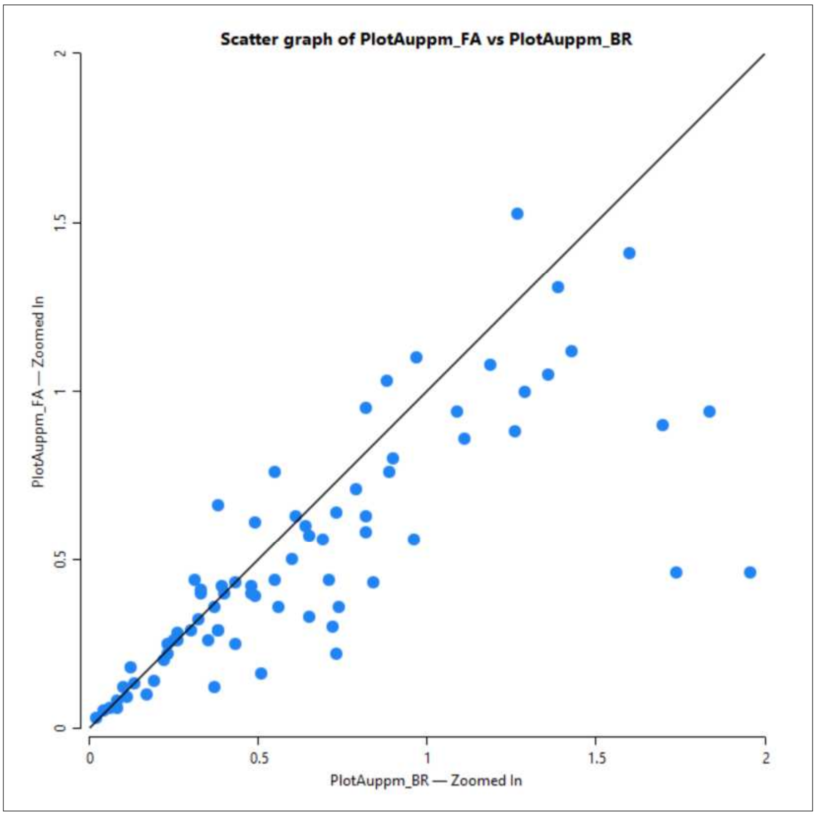


Figure 6-6: Scatterplot of bottle roll analyses (X axis) against fire assays (Y axis) clipped to 2 ppm.

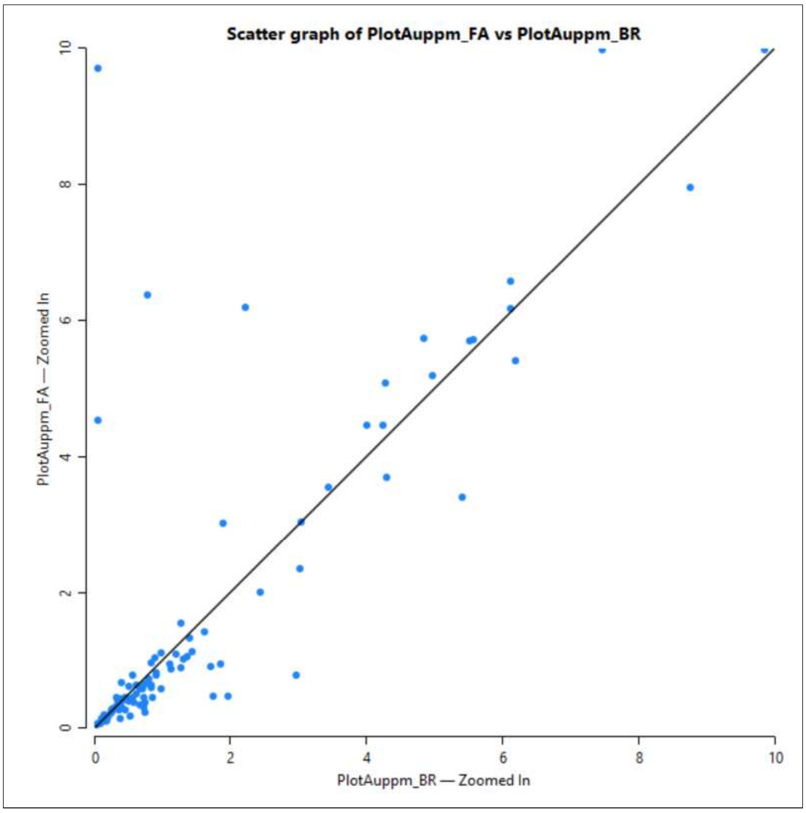


Figure 6-7: Scatterplot of bottle roll analyses (X axis) against fire assays (Y axis) clipped to 10 ppm.

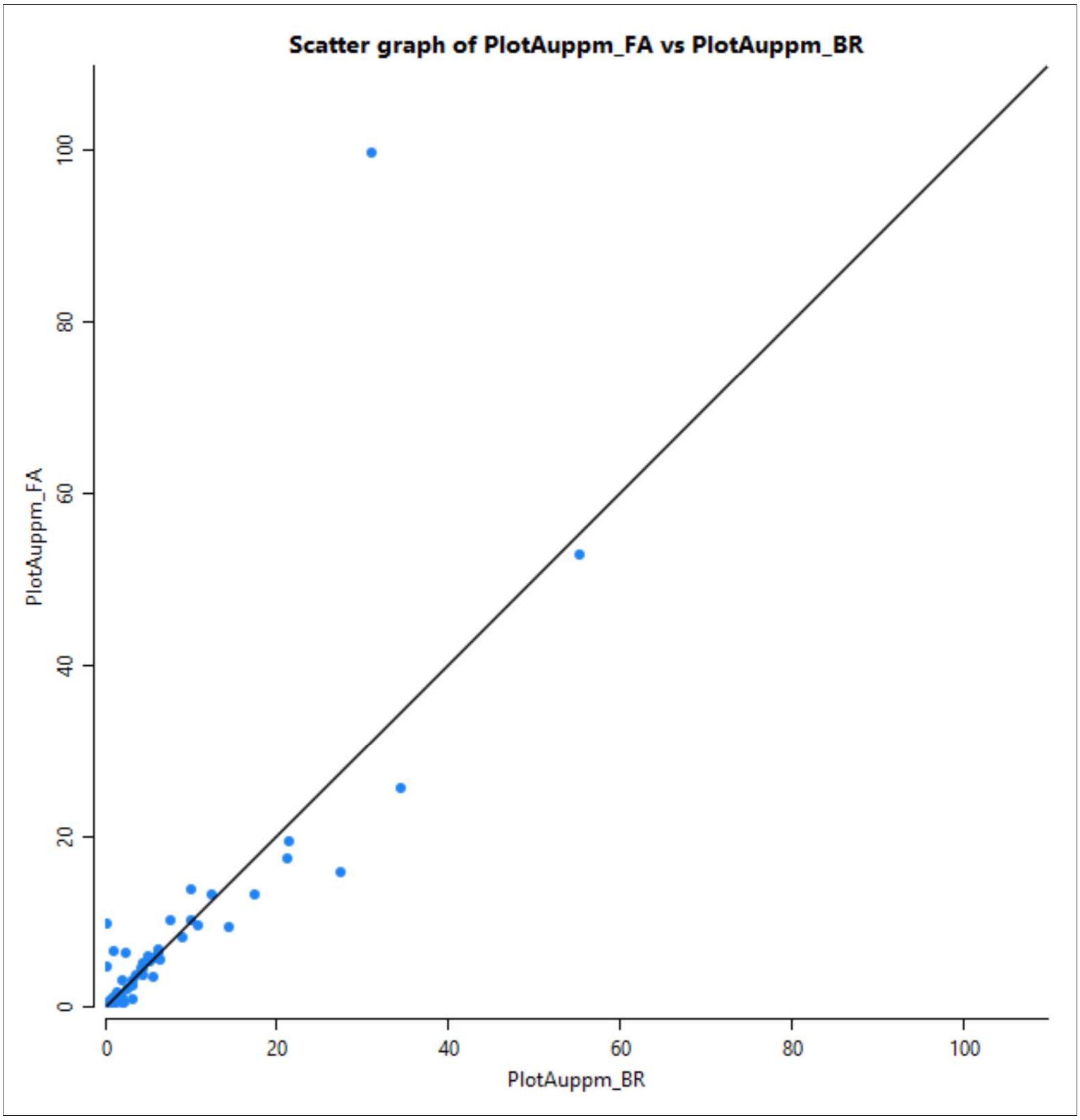


Figure 6-8: Scatterplot of all bottle roll analyses (X axis) against fire assays (Y axis).

6.6 Quality Control

6.6.1 Introduction

Cora Gold have undertaken routine QAQC checks on the Company's drillhole assays, including the following:

- Insertion of blank samples into the sample stream at a frequency of approximately 1 in 20 (5%).
- Insertion of field duplicates into the sample stream, commencing part way through the Cora Gold drilling at a frequency of 1 in 20 (5%). Average frequency across all Cora Gold drill phases is equivalent to approximately 2.5%. Note that field duplicates were only undertaken on the RC / AC drilling. No field duplicate analysis has been completed for the diamond drill core samples.

- Repeat assays conducted on the pulverised RC rejects for bottle roll analyses, at an equivalent insertion rate of approximately 3.5%.
- A combination of various standard samples analysed at a frequency of approximately 1 in 20 (5%). Standard samples include CRM's for fire assay QAQC, and larger bespoke standards for bottle roll QAQC.
- A small number of duplicate check assays completed at ALS Shannon in Ireland.

The results of the QAQC analyses are described in the following sections. It should be noted that these only relate to QAC completed on the Cora Gold drilling. SRK has not been provided with any the results of any historic QAQC completed on the Randgold or Gold Fields drilling.

6.6.2 Blanks

A total of 590 blank analyses have been completed by SGS Ouagadougou by bottle roll. The material used for blank analyses was a barren Upper Proterozoic sandstone. Almost all of the blank analyses returned suitably low Au grades (Figure 6-9 and Figure 6-10). Specifically, >88% of samples analysed returned Au values below the detection limit of 0.01 ppm, whilst all but 3 samples returned Au grades <0.025 ppm. 3 samples returned values of concern (namely 0.1 ppm, 2.55 ppm and 8.74 ppm) that either highlight contamination of these individual samples, or sample swap errors which is a minor concern worthy of further investigation.. Overall, the results of the blank QAQC do not indicate any systematic contamination at material grades that would impact on the quality of the Mineral Resource Estimate.

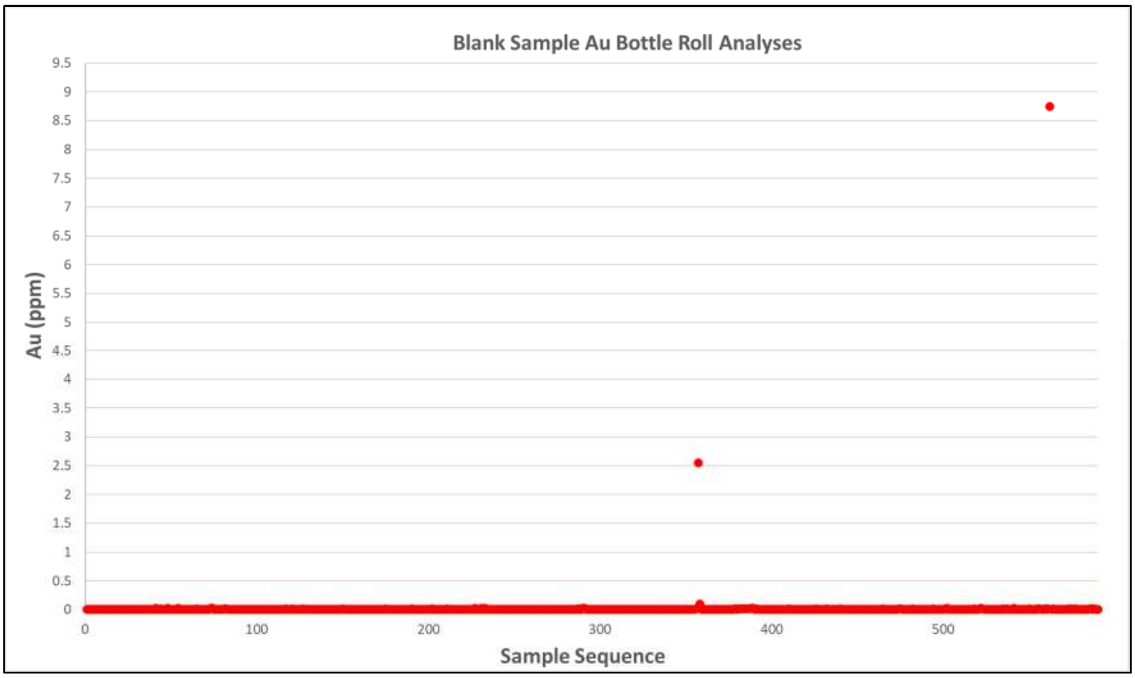


Figure 6-9: Blank sample Au bottle roll results..

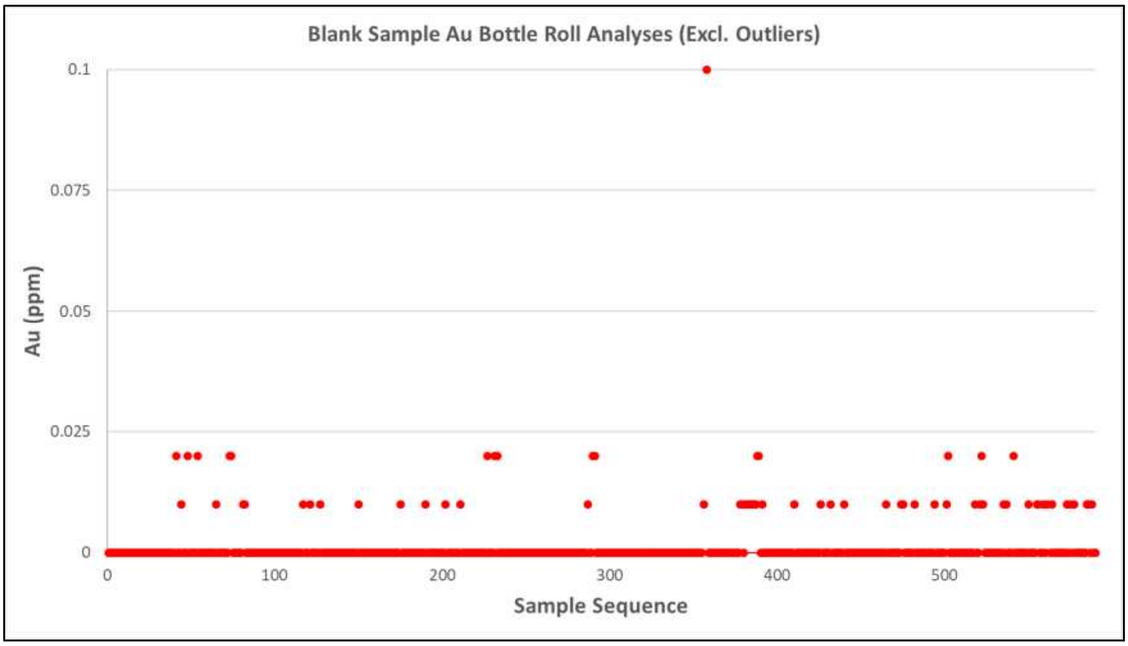


Figure 6-10: Blank sample Au bottle roll results (excluding outliers).

No blank QAQC has been completed on the fire assays at SGS Bamako. SRK recommend that blank analysis is completed on the fire assays in any future drill programmes.

6.6.3 Field Duplicates

In total, 364 duplicate samples have been submitted for analysis, this being approximately 2.5% of the samples submitted for assay (including blanks and standards). Of these samples, 302 relate to samples analysed by bottle roll and 62 relate to samples analysed by fire assay. Figure 6-11 shows the results of the field duplicate samples against the original samples for bottle roll analyses, whilst Figure 6-12 shows the results of the field duplicate samples against the original samples for fire assay analyses. Both plots are limited to samples with original and duplicate assay results <5 g/t. Figure 6-13 displays the field duplicate analyses against the original analyses for all samples (including both bottle roll and fire assay), whilst Figure 6-14 displays the same data, limited to samples with original and duplicate assay results <2 g/t.

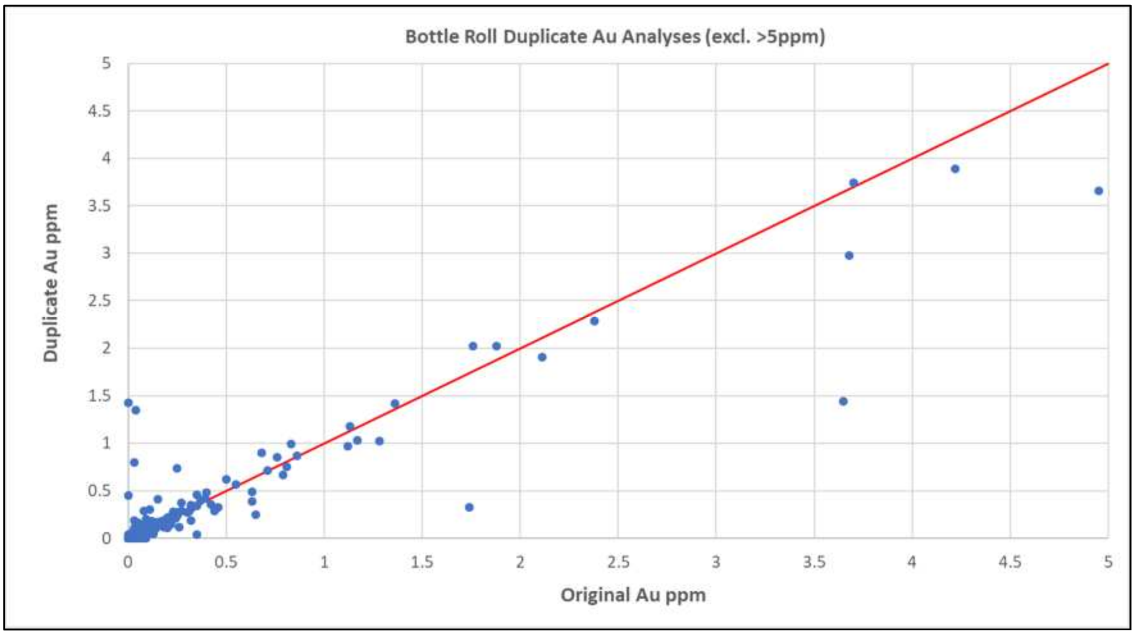


Figure 6-11: Field duplicate v original Au bottle roll analyses, filtered below 5 ppm.

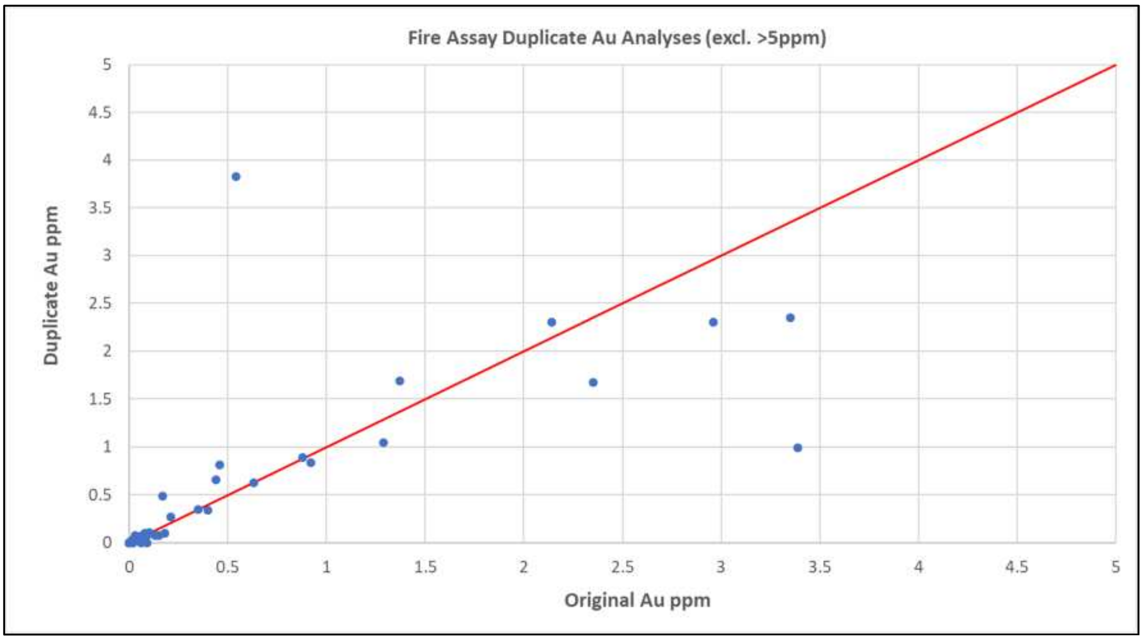


Figure 6-12: Field duplicate v original Au fire assays, filtered below 5 ppm.

In general, the correlation between duplicate and original samples is considered to be reasonable, both for the fire assay and bottle roll analyses. The degree of spread in correlation between samples is quite large, especially at higher grades, but not unexpected for such a gold deposit. A small number of samples with original or duplicate grades in excess of 0.4 g/t with corresponding barren original grades <0.05 g/t, and a single sample with a duplicate value of ~90 g/t and an original sample grade of ~2 g/t are of some concern, but not considered to materially impact on the confidence in the global estimate at this stage. It is noted that the 90 g/t duplicate sample is a fire assay duplicate. It is therefore considered feasible that, in this instance, the significant differnce between the duplicate and original assay results could be a function of the nugget effect.

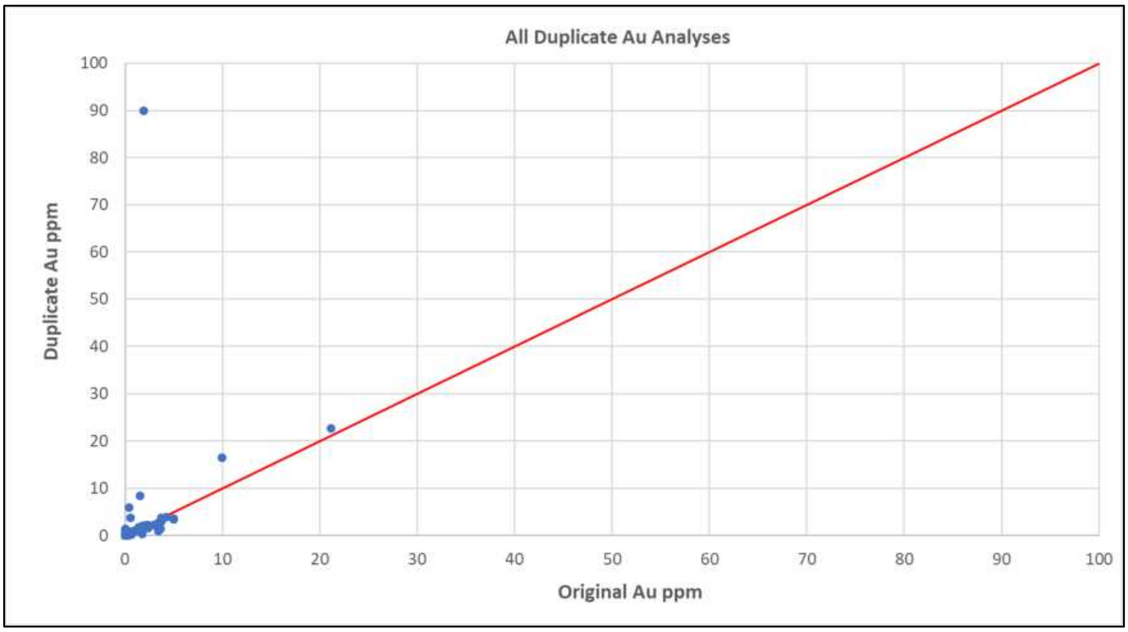


Figure 6-13: Field duplicate v original assays, for both fire assay and bottle roll.

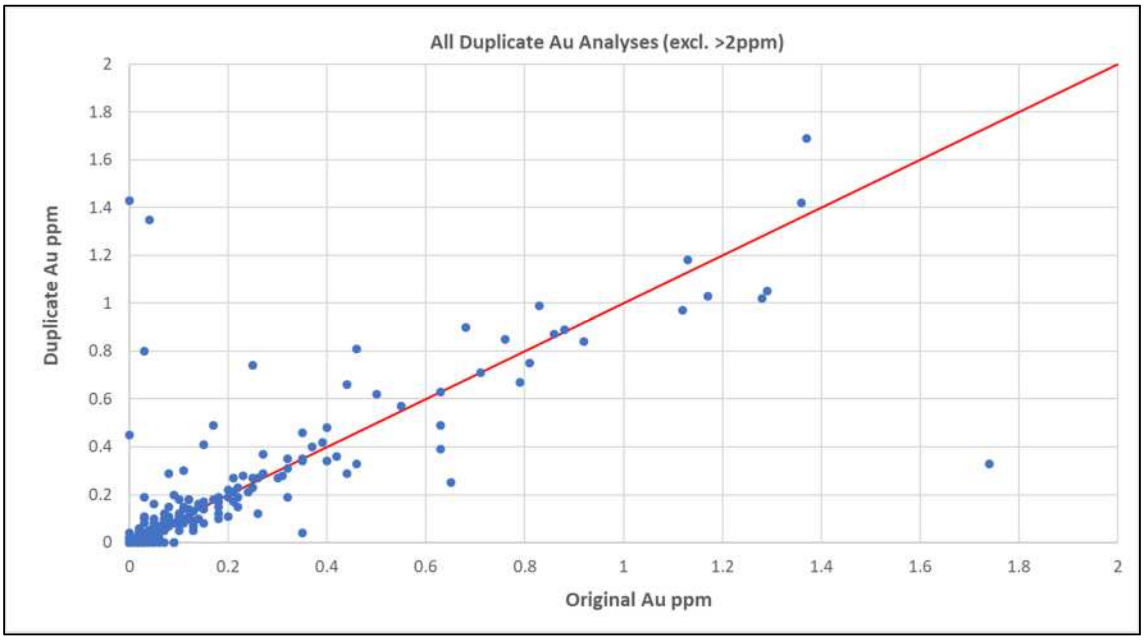


Figure 6-14: Field duplicate v original assays, for both fire assay and bottle roll, filtered below 2 ppm Au.

6.6.4 Repeat assays

During 2018, Cora Gold requested SGS Ouagadougou complete a programme of repeat analyses by bottle roll on the pulverised RC rejects from the original bottle roll standards. In total, 388 repeat assays were completed from samples in 28 RC holes. The results of the repeat assays, plotted against the original analyses, are displayed in Figure 6-15 and excluding high grade samples >10 ppm Au in Figure 6-16.

The correlation between original and repeat assays is generally good, indicating acceptable performance of the bottle roll analyses.

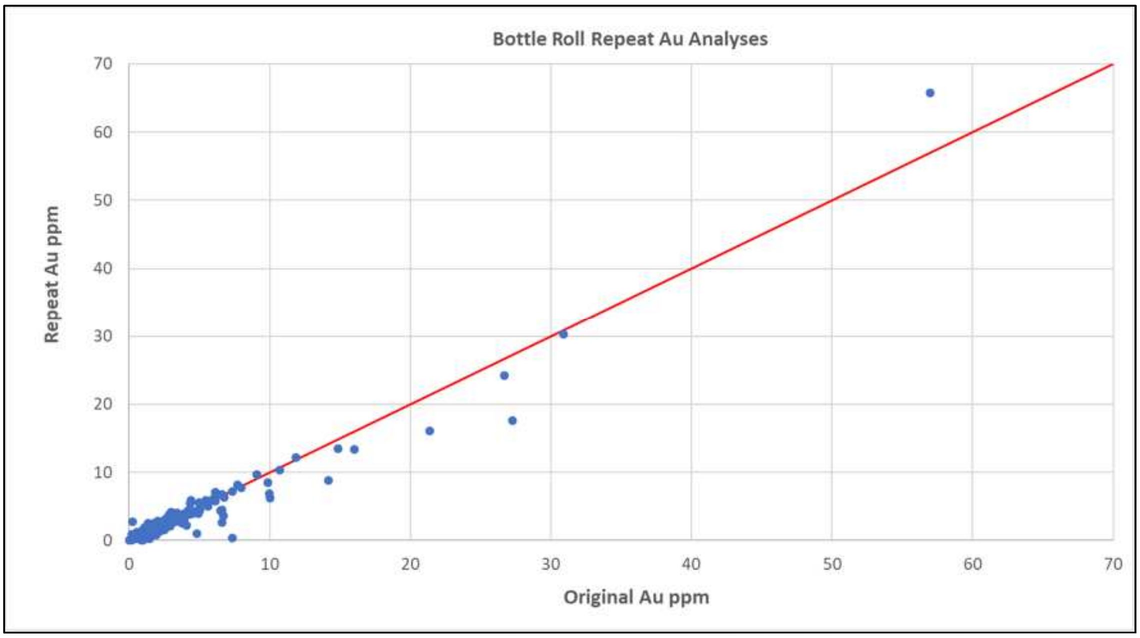


Figure 6-15: Original v repeat bottle roll analyses.

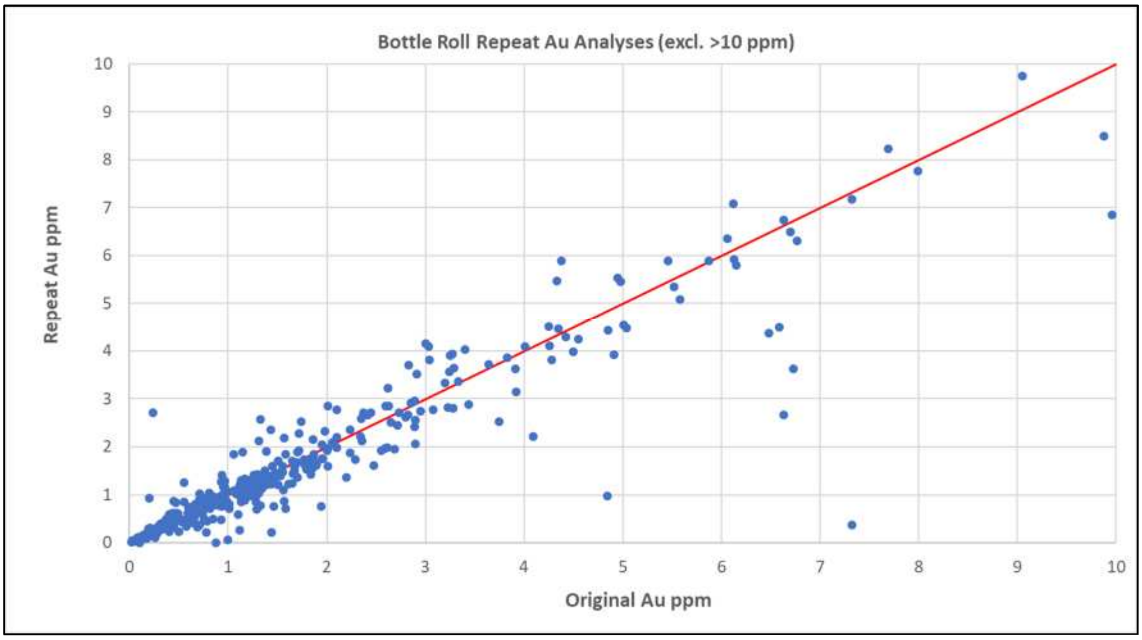


Figure 6-16: Original v repeat bottle roll analyses, filtered below 10 ppm Au.

6.6.5 Standards

Fire Assay Standards

Cora Gold have completed fire assay standard analyses on a total of 263 samples, across 4 different Certified Reference Materials ("CRM"). This equates to approximately 6% of all fire assay analyses completed by Cora Gold. A list of the CRMs employed by Cora Gold is provided in Table 6-2. The CRMs are all prepared at certified by ROCKLABS. All fire assay analyses were completed at SGS Bamako. Plots of the fire assay results for each CRM as a percentage of the certified value are presented in Figure 6-17 to Figure 6-21.

Table 6-2: CRM grades and number of analyses undertaken.

CRM	Certified Grade (Au ppm)	Number of Analyses Completed
OxL118	5.828	85
OxE143	0.621	65
OxJ120	2.365	64
OxG103	1.019	49

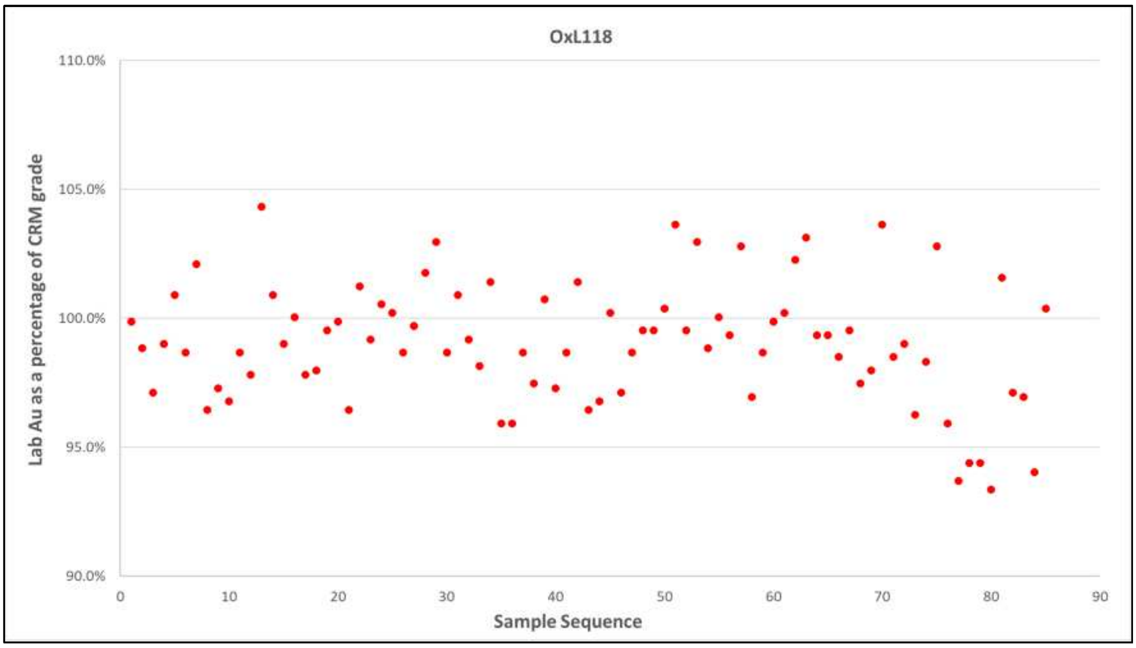


Figure 6-17: Results for OxL118, presented as a percentage of the certified Au grade.

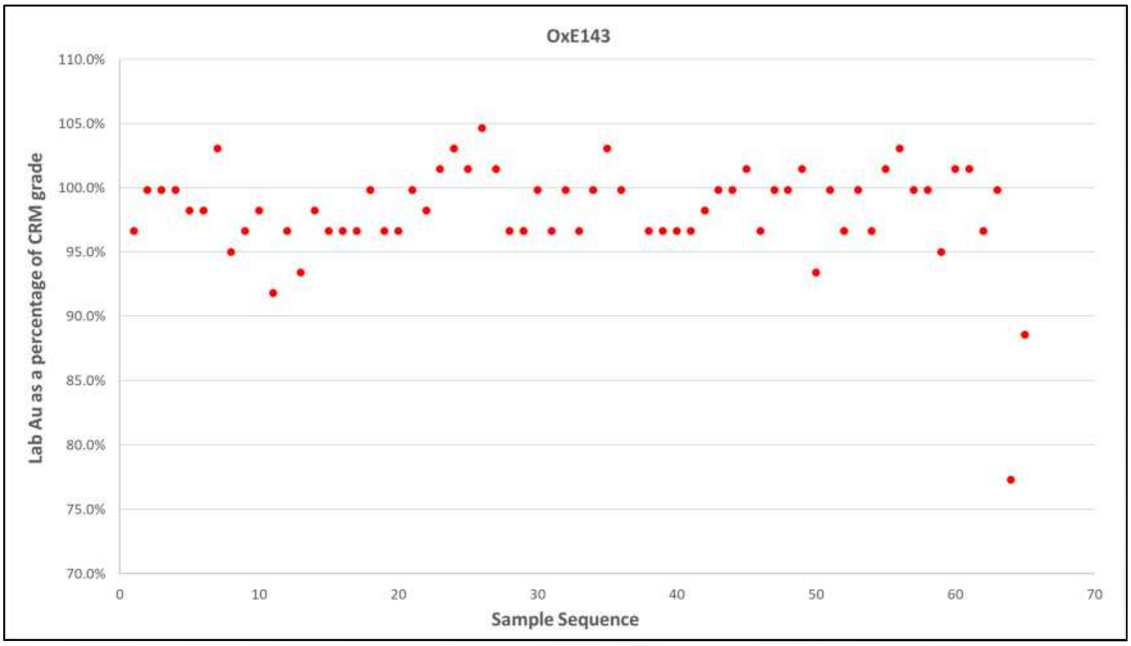


Figure 6-18: Results for OxE143, presented as a percentage of the certified Au grade.

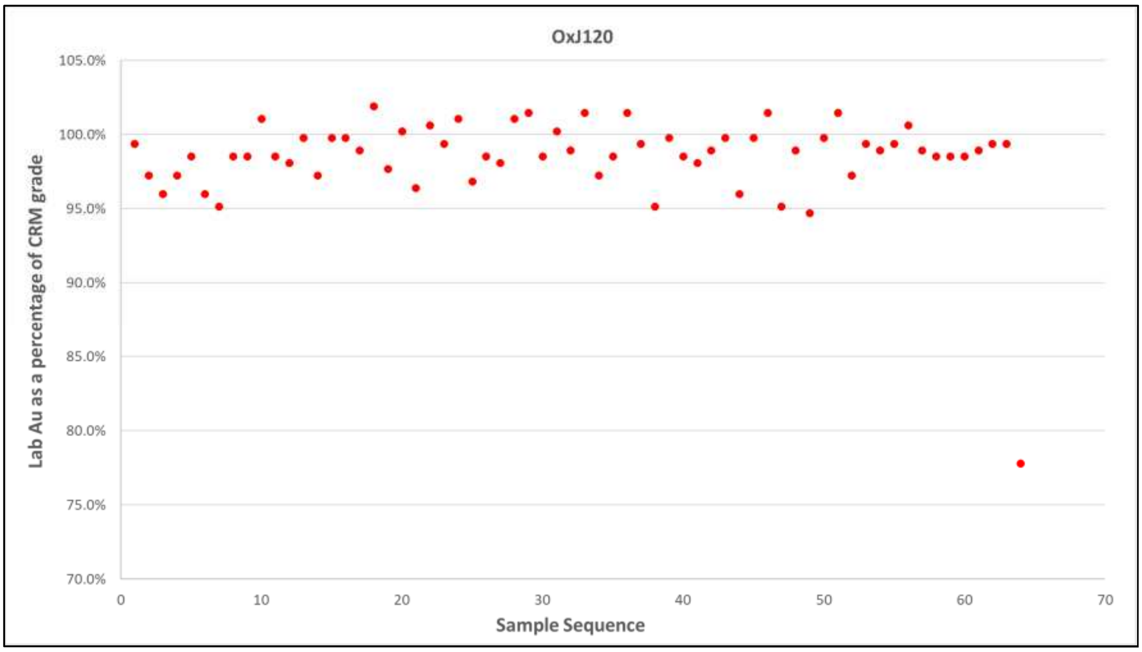


Figure 6-19: Results for OxJ120, presented as a percentage of the certified Au grade.

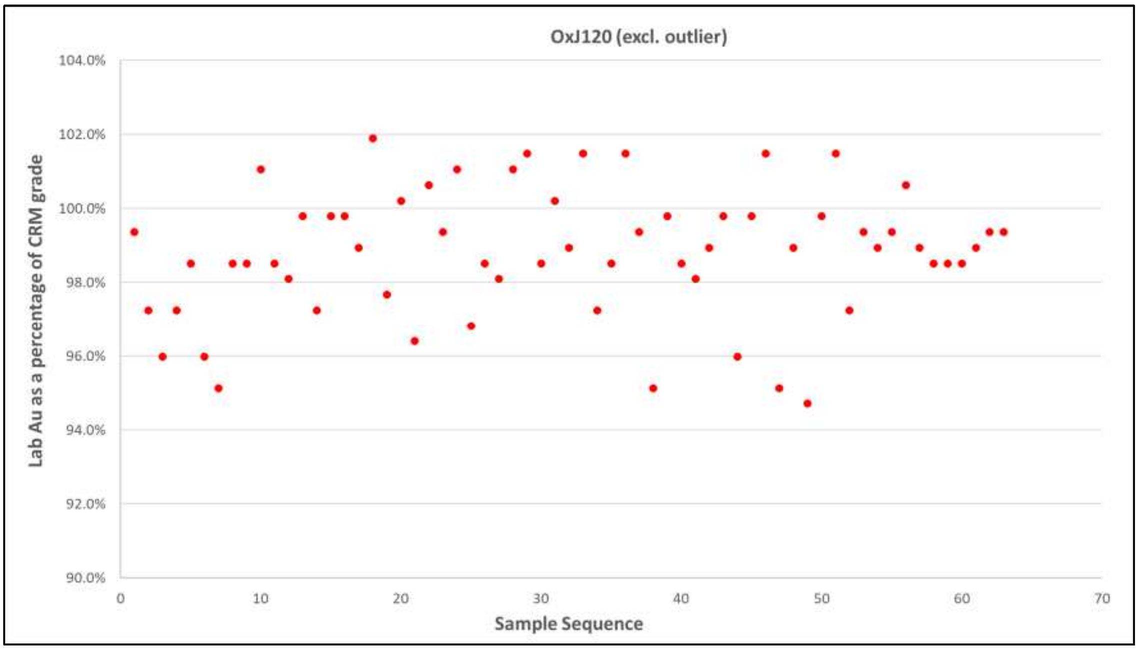


Figure 6-20: Results for OxJ120, excluding a single outlier, presented as a percentage of the certified Au grade.

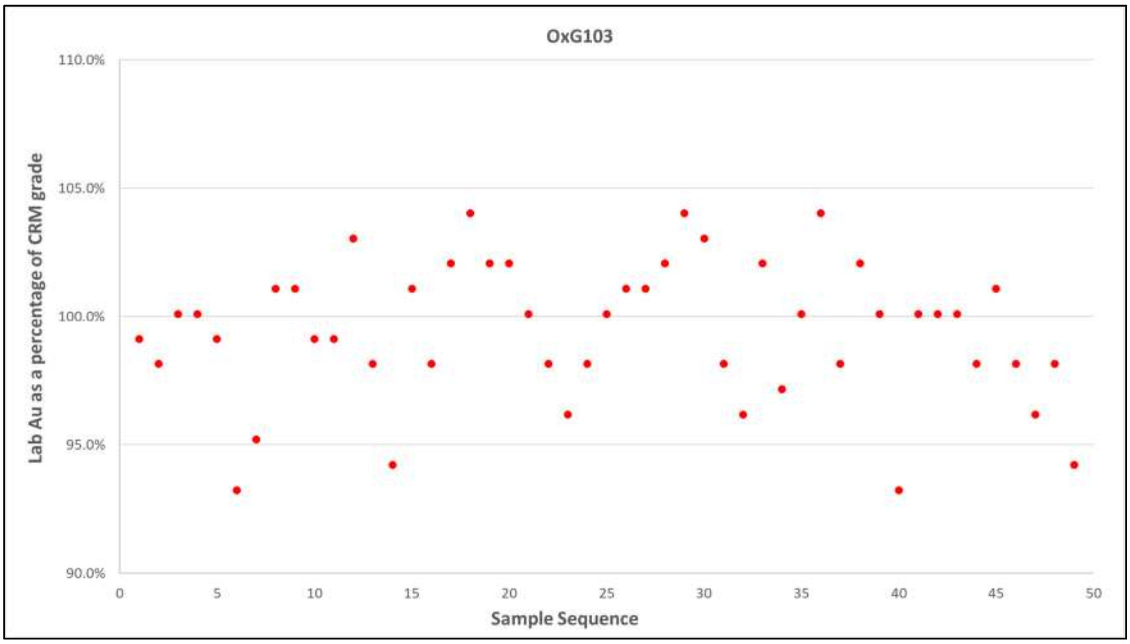


Figure 6-21: Results for OxG103, presented as a percentage of the certified Au grade.

In general, the variation in Au grade exhibited for all CRMs is considered acceptable. That said, the results for all 4 CRMs are, on average, lower grade than the ROCKLABS certified grades. Specifically, the average grades returned by SGS Bamako are 1.0% lower than the certified grade for OxL118, 1.8% lower than the certified grade for OxE143, 1.5% lower than the certified grade for OxJ120 and 0.6% lower than the certified grade for OxG103. This systematic under reporting of Au grade for all CRMs, suggests that the SGS fire assays may be marginally under-analysing Au. At this stage a potential 1-2% under-reporting of Au grade from fire assays is not considered material to the Mineral Resource Estimate. It is noted that 2 latest samples OxE143 samples to be analyses and the latest OxJ120 sample to be analyses all significantly under-report Au grade. This should be investigated by Cora Gold, as it may indicate localised short term contamination.

SRK note that only the OxL118 CRM was used for fire assay standard analyses during the most recent Cora Gold drill campaign. SRK would recommend that a combination of all four CRMs be used in future drill campaigns in order to provide a more representative range of grades to undertake standard analyses on.

Bottle Roll Standards

In order to complete standard QAQC analyses on the bottle roll assays, Cora Gold requested SGS Ouagadougou to prepare custom standard samples of sufficient size on which to complete bottle roll analysis. The custom standard samples prepared range in weight from approximately 0.5 – 1 kg, and comprise two sets of standards created by differing approaches:

a) Mixing of 1g gold pills of known grade, provided by Geostats Pty Ltd (“Geostats”), with blank material of between approximately 0.5 kg and 1 kg in weight.

b) Mixing of 50g CRM samples of known grade, provided by both ROCKLABS and Geostats, with blank material of between approximately 0.5 kg and 1 kg in weight.

In both cases, the blank material mixed with the gold pills or CRMs is the same barren Proterozoic sandstone used for the blank analyses described in Section 6.6.2. The expected grade of each standard sample was calculated as "*((W1 / W2) * G)*" where *W1* is the weight of the gold pill or CRM, *W2* is the weight of the blank sample, and *G* is the grade of the gold pill or CRM.

In total standard QAQC analyses have been completed on 323 custom bottle roll standards, which equates to approximately 4.5% of all bottle roll analyses completed by Cora Gold. The results of the bottle roll analyses completed on these custom standard samples are discussed separately for the "gold pill" standards and the "CRM" standards below.

Gold Pills

Cora Gold have completed bottle roll standard analyses using samples generated by mixing of blank material with 5 different Geostats 1g gold pills, for a total of 220 custom standard samples. For the most part, the weight of blank material mixed with the gold pills was approximately 1 kg, however, during the first Cora Gold drill campaign, the weight of blank material used was much more variable. Because the weight of blank material mixed with the gold pills varies, the expected grades of each of the standards varies accordingly. The certified grade of each of the gold pills used, plus the average, minimum and maximum grades of each of the resulting groups of standards is provided in Table 6-3.

Table 6-3: The average, minimum and maximum expected grades for each set of custom gold pill bottle roll standards and the number of analyses completed.

Gold Pill	Gold Pill Certified Grade (ppm)	Average Expected Grade of Standard	Minimum Expected Grade of Standard	Maximum Expected Grade of Standard	Number of Analyses Completed
GAP-01	3,237	4.13	2.85	6.63	52
GAP-02	1,025	1.48	0.99	2.60	41
GAP-03	10,000	14.45	9.78	26.76	45
GAP-04	2,117	2.43	2.06	3.73	24
GAP-05	5,429	5.49	4.46	10.5	58

Plots of the bottle roll results for each set of bottle roll standards, as a percentage of the expected value for each sample, are presented in Figure 6-22 to Figure 6-26. Note that, to attempt to improve homogenisation of the samples, during the 2019 drill campaign Cora Gold requested that the gold pills were pulverised before mixing with the blank material. The samples for which the gold pill was pulverised before mixing with the blank material are plotted in red. All other results are plotted in blue.

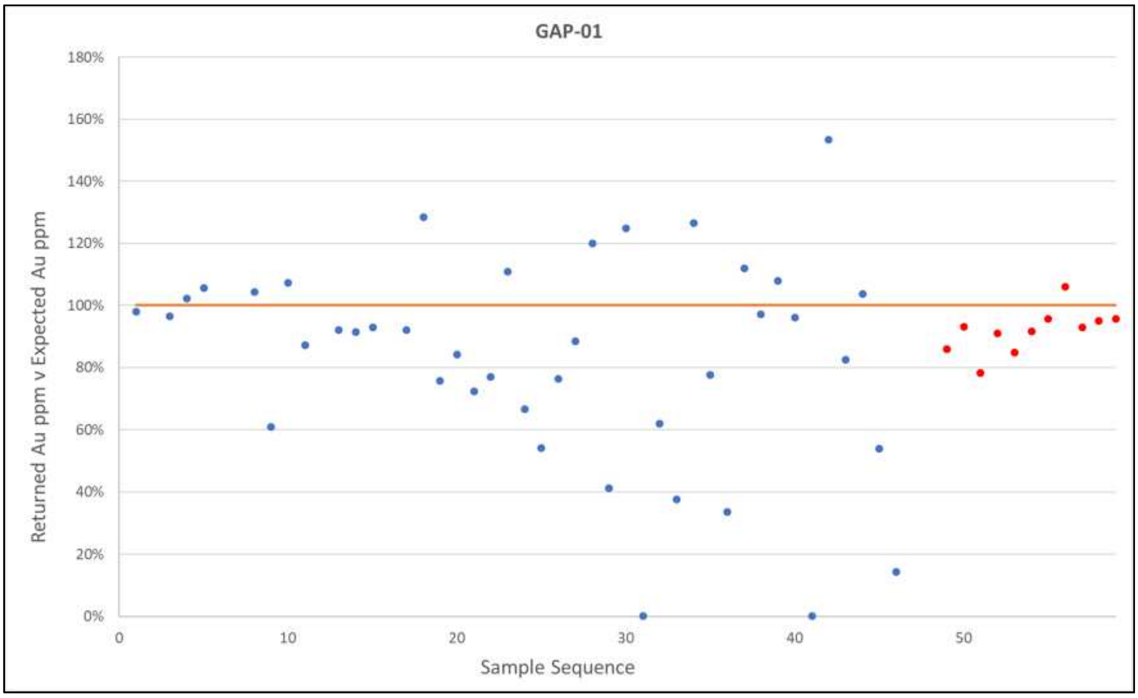


Figure 6-22: Results for bottle roll standards prepared using GAP-01, plotted as a percentage of the expected Au grade.

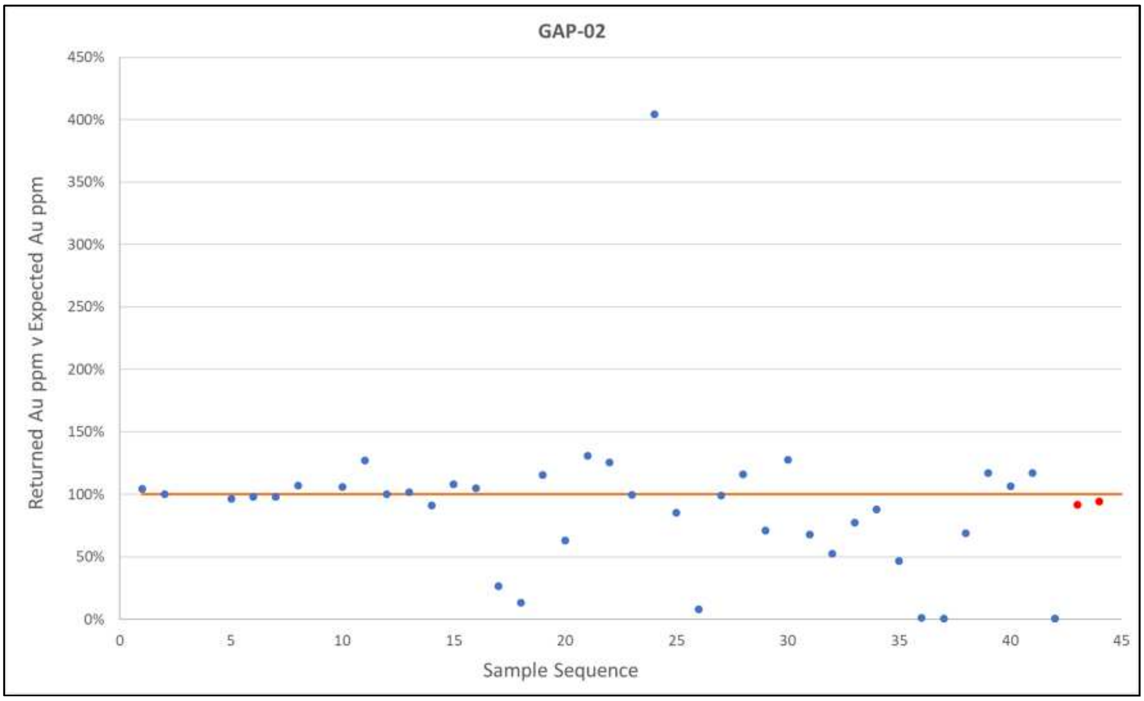


Figure 6-23: Results for bottle roll standards prepared using GAP-02, plotted as a percentage of the expected Au grade.

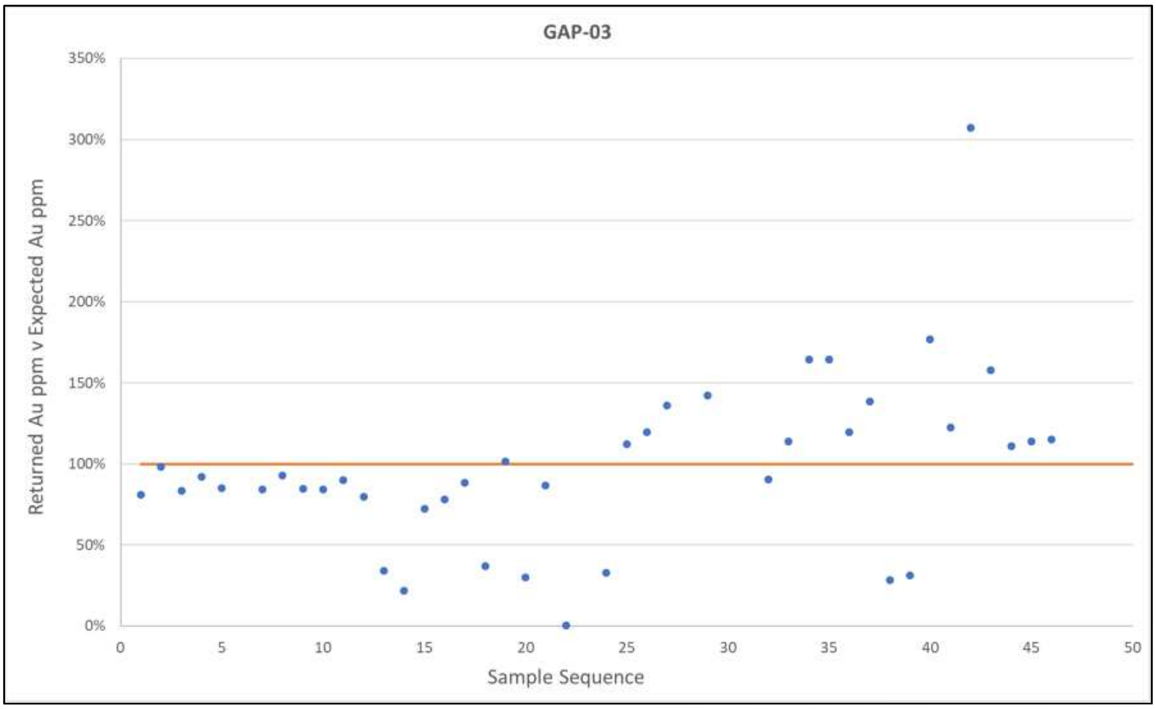


Figure 6-24: Results for bottle roll standards prepared using GAP-03, plotted as a percentage of the expected Au grade.

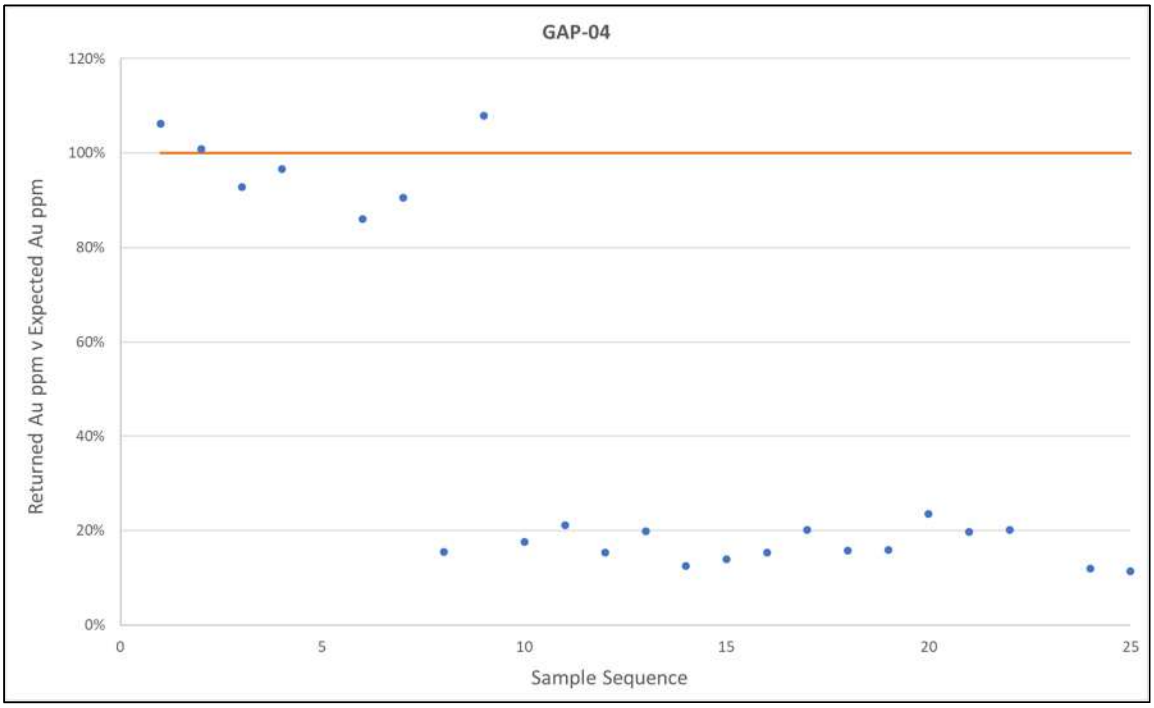


Figure 6-25: Results for bottle roll standards prepared using GAP-04, plotted as a percentage of the expected Au grade.

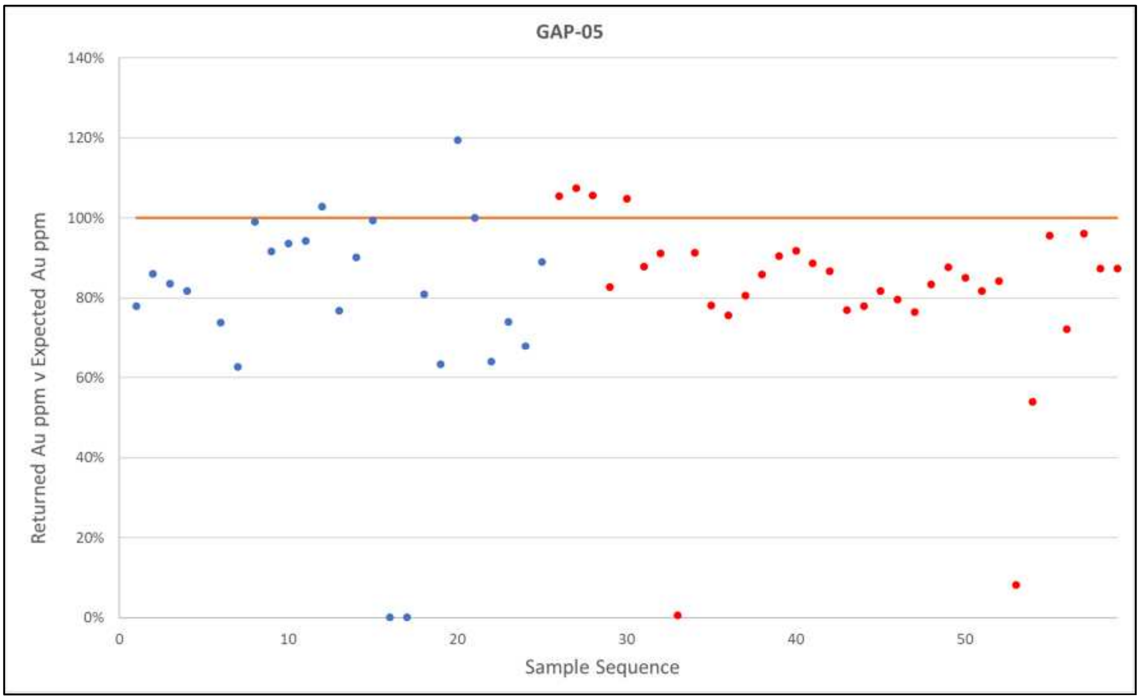


Figure 6-26: Results for bottle roll standards prepared using GAP-05, plotted as a percentage of the expected Au grade.

Figure 6-22 to Figure 6-26 demonstrate that the gold pill bottle roll standards perform extremely poorly. For the most part, the results are highly erratic and show limited or no correspondence to the expected grades. Also of concern is a relatively consistent cluster of returned grades for GAP-04 grades around 0.3 - 0.4 g/t Au, which are notably consistent, but have no correspondence to the expected grades for these samples, which are all approximately 2g/t. At present the cause of this systematic return of grades in the 0.3 – 0.4 g/t range for GAP-04, all of which relate to samples analysed in 2018, is unclear and unexplained. SRK strongly recommend that Cora Gold investigate this issue, which could have a number of causes, including incorrect gold pill selection or mis-documentation of the weight of the blank material, in order to rule out more serious sample swap or database issues.

The erratic distribution of the gold pill bottle roll standards is discussed in more detail in Section 0.

CRM Bottle Roll Standards

Cora Gold have completed bottle roll standard analyses, using samples generated by mixing of blank material with 3 different 50g CRM samples (2 Geostats CRMs and 1 ROCKLABS CRM), for a total of 103 custom standard samples. The certified grade of each of the 50g CRM samples used, plus the average, minimum and maximum grades of each of the resulting groups of standards is provided in Table 6-4.

Table 6-4: The average, minimum and maximum expected grades for each set of custom CRM bottle roll standards and the number of analyses completed.

CRM	CRM Certified Grade (ppm)	Average Expected Grade of Standard	Minimum Expected Grade of Standard	Maximum Expected Grade of Standard	Number of Analyses Completed
OxL118	5.828	0.27	0.23	0.28	49
G306-3	8.66	0.42	0.42	0.43	17
G915-4	9.16	0.44	0.40	0.45	37

Plots of the bottle roll results for each set of bottle roll standards, as a percentage of the expected value for each sample, are presented in Figure 6-27 to Figure 6-31.

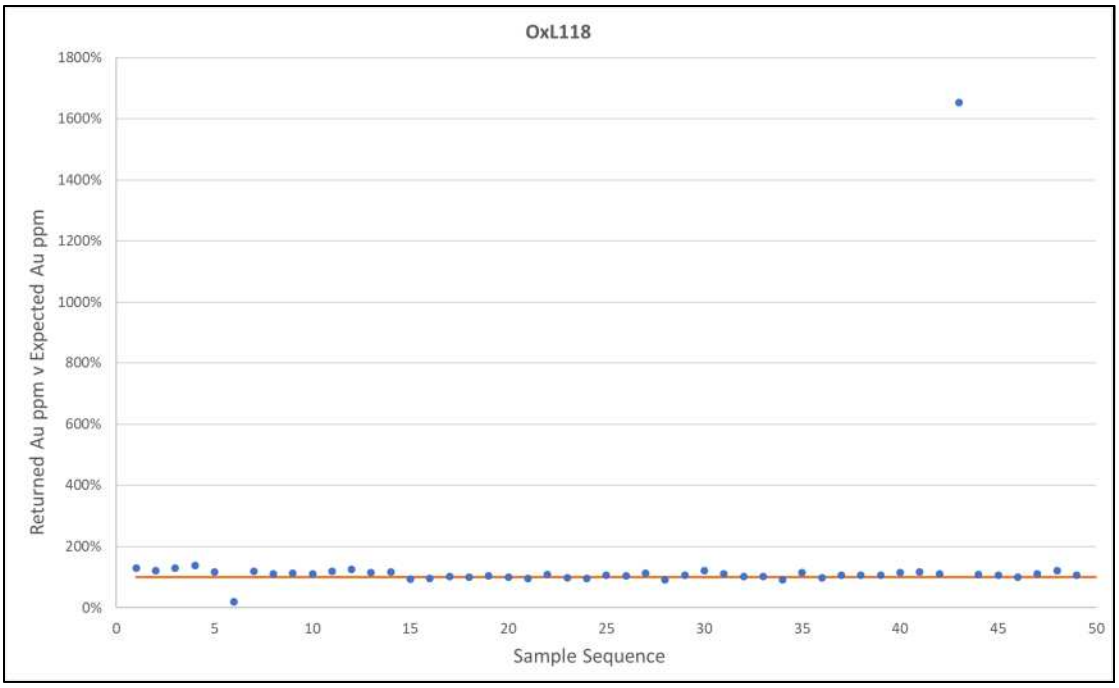


Figure 6-27: Results for bottle roll standards prepared using CRM OxL118, plotted as a percentage of the expected Au grade.

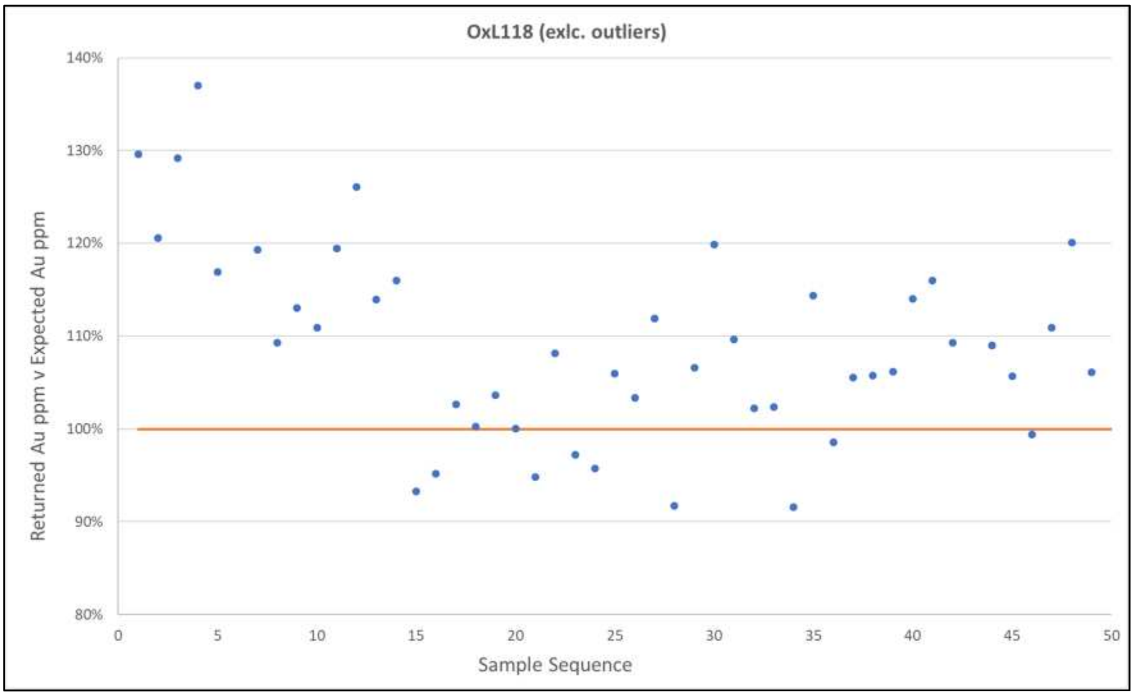


Figure 6-28: Results for bottle roll standards prepared using CRM OxL118 plotted as a percentage of the expected Au grade. Out-lying values are removed.

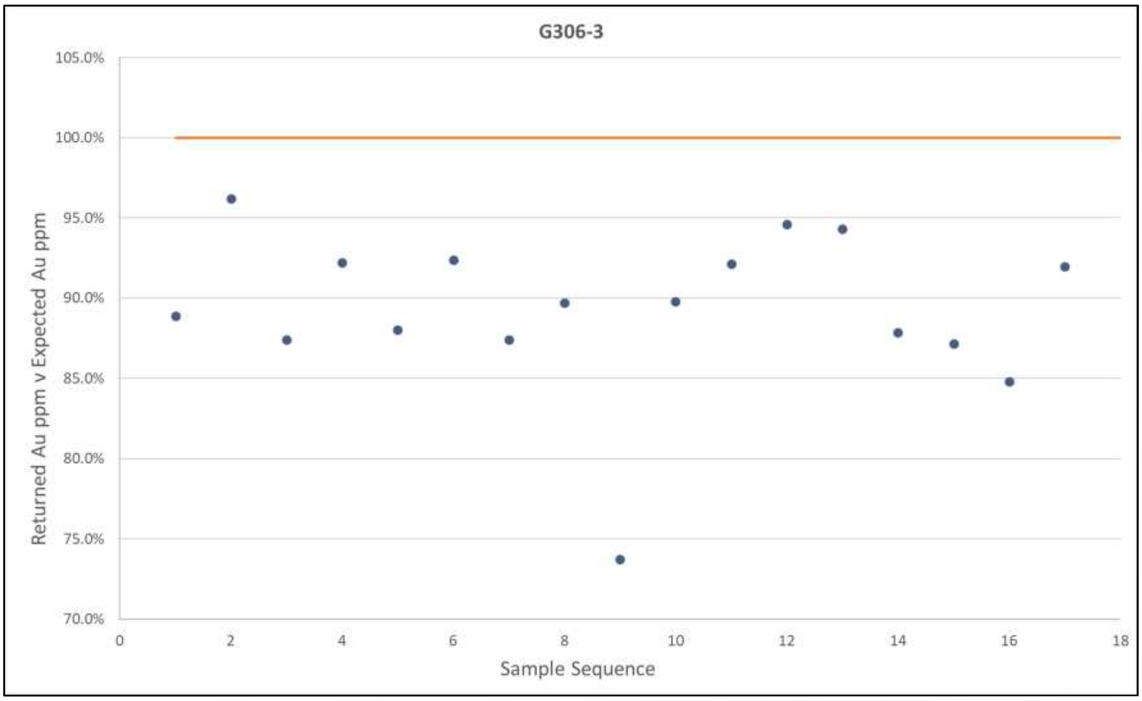


Figure 6-29: Results for bottle roll standards prepared using CRM G306-3, plotted as a percentage of the expected Au grade.

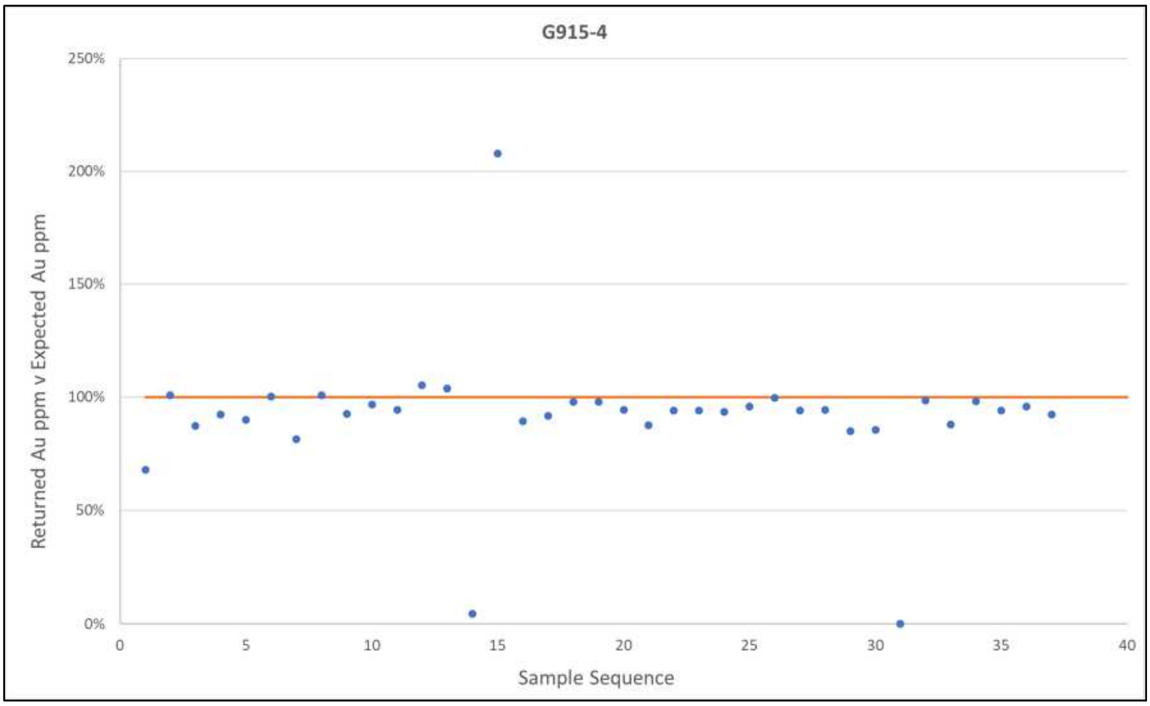


Figure 6-30: Results for bottle roll standards prepared using CRM G915-4, plotted as a percentage of the expected Au grade.

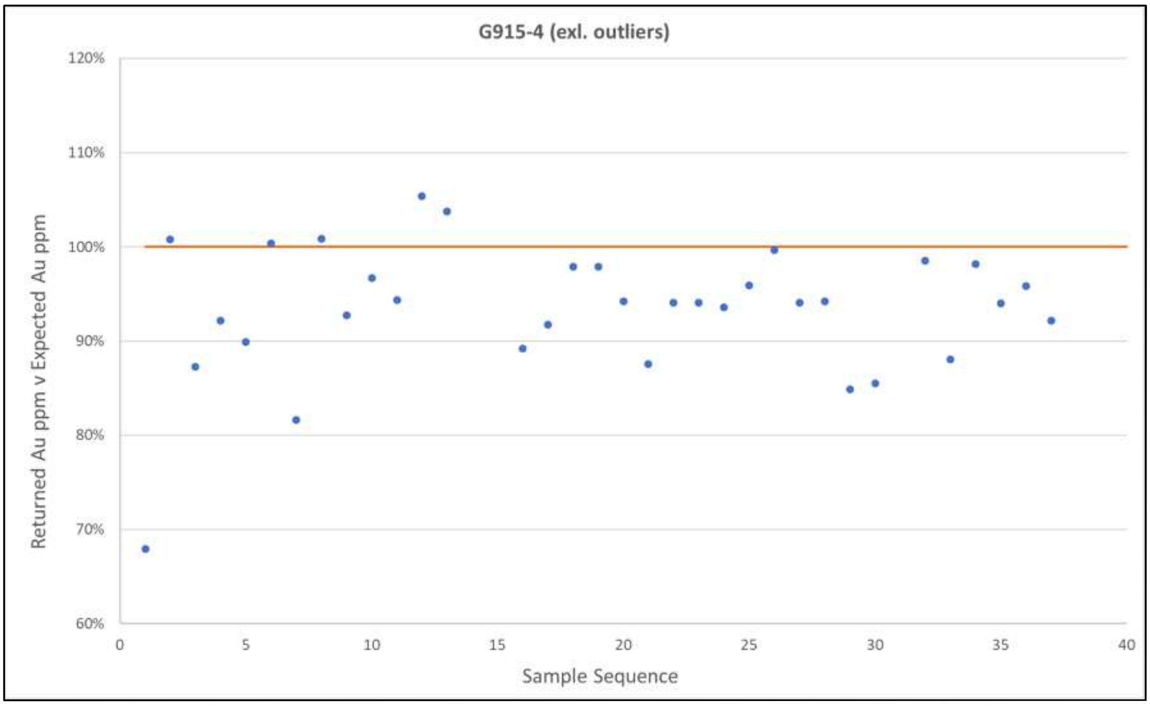


Figure 6-31: Results for bottle roll standards prepared using CRM OxL118 plotted as a percentage of the expected Au grade. Out-lying values are removed.

In general, the results of the analyses on the CRM bottle roll standards suggest relatively poor performance of these samples. The variation in returned Au values is considered large, and the G915-4 and G306-3 standards both consistently under-report expected Au grades, whilst the OxL118 standard consistently over-reports expected Au grades. Excluding outliers, the average

% difference in average grades of the standard analyses, relative to the expected grades, and the standard deviation of the % difference in grade (between actual and expected results) for each set of standards is provided below:

- OxL118 – average of 9% increase in actual grades compared to expected grades. Standard deviation of 10%
- G306-3 – average of 11% decrease in actual grades compared to expected grades. Standard deviation of 5%
- G915-4 – average of 7% decrease in actual grades compared to expected grades. Standard deviation of 7%

A total of 5 of the results across the 3 standards (2 from OxL118 and 3 from G915-4) have anomalous results (see Figure 6-27 and Figure 6-30), that differ significantly from both the expected assay value and the range of values returned for the other standard samples. These are likely the result of either mixing of incorrect CRMs, mis-labelling of samples, sample swap error or data input errors. At approximately 5% of all CRM bottle roll samples, this is considered to be a relatively large number of samples on which to record anomalous / erroneous results.

Despite the above, notably, the performance of the CRM bottle roll standards reflects a marked improvement on the gold pill bottle roll standards. The overall performance of the bottle roll standard analyses, and considerations for classification on the Mineral Resource, are discussed in more detail in Section 0.

6.6.6 Umpire Lab

Cora Gold have submitted a total of 76 samples from 5 RC holes for check assay by bottle roll to the ALS Shannon analytical facility in Ireland.

The results of the ALS repeat analyses are plotted against the original SGS analyses in Figure 6-32 and Figure 6-33.

The results show a strong correlation between the original and repeat assays, with very few outliers. The correlation coefficient is 0.99. Scatter is broadly even each side of the 1:1 correlation line, although, on average the SGS bottle rolls return slightly higher Au values at grades of <2 g/t, whilst the ALS bottle rolls return slightly higher Au values at grades of >2 g/t.

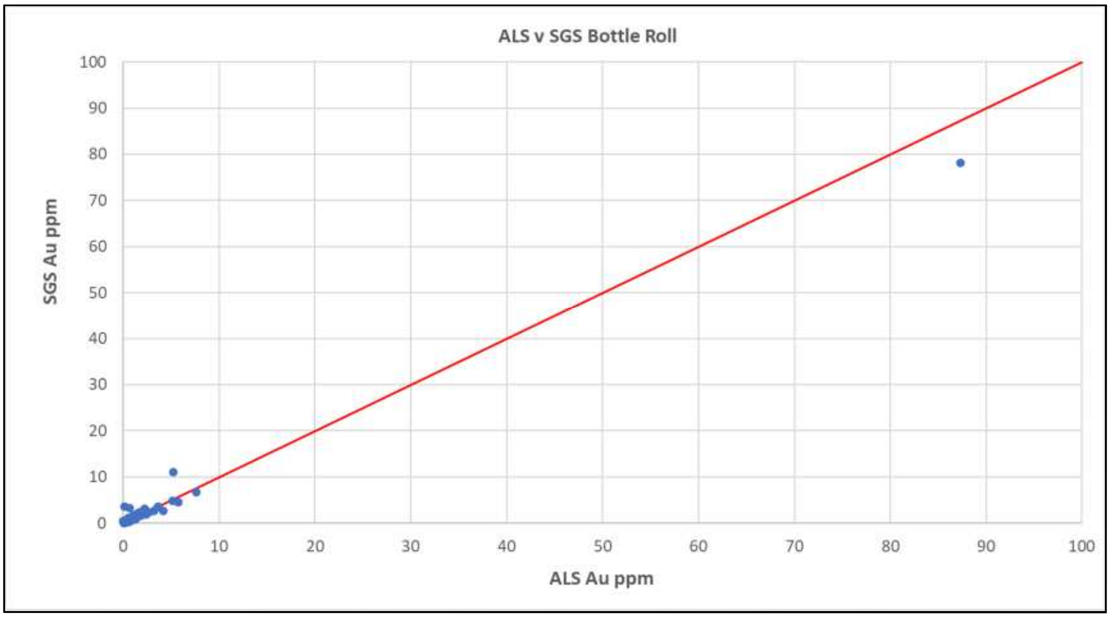


Figure 6-32: ALS repeat v SGS original bottle roll analyses.

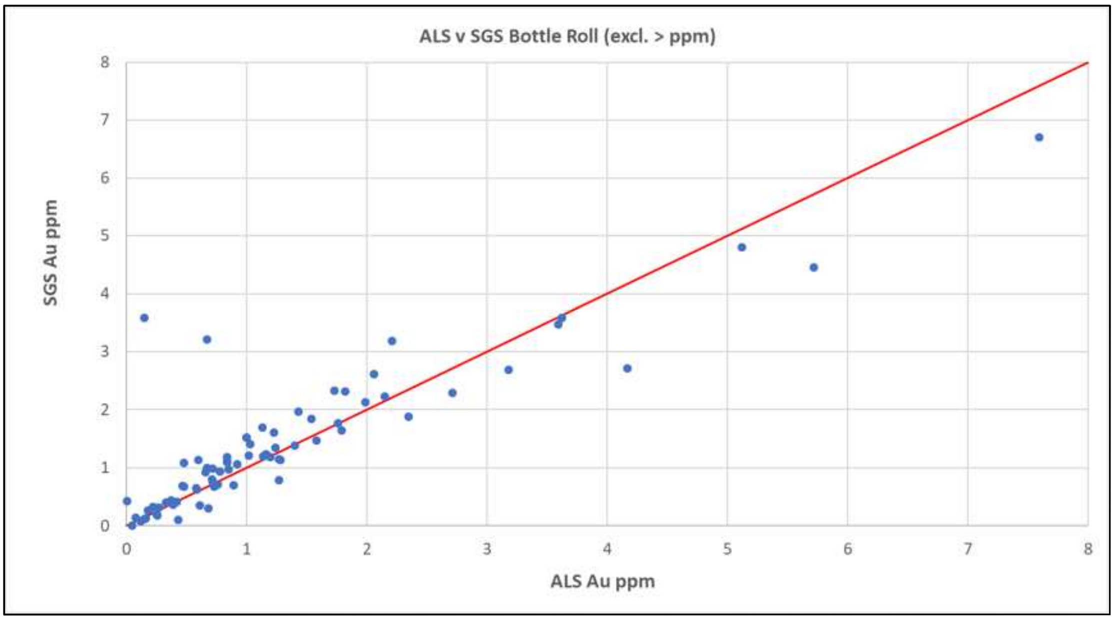


Figure 6-33: ALS repeat v SGS original bottle roll analyses excluding values > 8 ppm.

6.6.7 SRK Comment

As outlined in Section 6.6.5, the custom bottle roll standards used by Cora Gold perform very poorly. The bottle roll standards created by mixing 1g gold pills with blank material show particularly erratic results, some of which seem to have little correlation with the expected grades. It is noted that the attempt to improve the performance of this standard by pulverising the gold pills prior to mixing with the blank material has had mixed results, perhaps indicating an inconsistent degree of pulverisation of the pills. For GAP-01, the results of the standards after pulverising the gold pills are more consistent and closer to the expected grades than for the standard samples for which the gold pill was not pulverised prior to mixing with blank material. However, for GAP-05, the pulverisation of the gold pill appears to have had no impact on the performance of the standard.

Although the bottle roll standards created by mixing 50g CRM samples with blank material also perform relatively poorly, it is noted that the results of the CRM bottle roll standards reflect a marked improvement on the gold pill bottle roll standards. The distribution of returned Au grades for these standards is relatively widespread, but does show a degree of correspondence with the expected grades.

Both the field duplicate and repeat assay bottle rolls QAQC analyses completed by Cora Gold show reasonable repeatability of results. Significantly, the umpire lab bottle roll checks completed at ALS Shannon in Ireland also show a good correlation with the SGS Ouagadougou results. This suggests that the poor performance of the bottle roll standards is likely a result of the method used to prepare these samples, rather than indicating any fatal flaw in the analytical equipment at SGS Ouagadougou. The improved performance of the CRM bottle roll standards, compared to the gold pill bottle roll standards implies that the extremely poor performance of the gold pill bottle roll standards may be due to insufficient homogenisation of the pill into the blank prior to analysis. Note that this does not explain the systematic return of grades in the 0.3 – 0.4 g/t range for GAP-04, discussed in Section 6.6.5, which SRK recommend that Cora Gold investigate as a matter of priority. Peculiarly, of the 3 CRM standard sample groups analysed, 2 (G915-4 and G306-3) return results consistently lower grade than the expected Au grade, with the other (OxL118) returning results consistently higher grade than the expected Au grade. This despite all 3 CRM standard groups having very similar expected grade ranges. At present, it is therefore difficult to make any conclusions on the accuracy of the SGS bottle roll analyses on the basis of the standard QAQC analysis. Given that the under- and over-reporting of grades are relatively consistent within the standard groups, SRK would suggest that, again, this is more likely to be a function of the methodology used to generate the standards, rather than a bias in the analytical results.

At this stage, the performance of the bottle roll standards is not considered a fatal flaw that would prevent the reporting of a Mineral Resource Estimate for the project. That said, it is strongly recommended that Cora Gold rectify this issue in future QAQC programmes.

For the next round of drilling, SRK would suggest that Cora Gold continue to undertake QAQC standard checks on the bottle rolls. However, given the performance of the 1g pill custom standards it is recommended that Cora Gold cease using this approach to create standard samples. Rather, it is recommended that Cora Gold continue with the approach of generating custom standard samples by mixing 50g CRM samples with blank material, but rather than using a single 50g CRM, it is suggested that Cora Gold assess the performance of custom standards made by mixing a greater proportion of CRM (2 – 4 50g samples) with blank material, to increase the proportion of CRM in the standard. It is also recommended that the range of expected gold values from the custom standard sample sets is increased. At present, the average expected grade of the 3 CRM custom bottle roll standard groups are all within 0.2 g/t, at 0.27 g/t, 0.42 g/t and 0.44 g/t for OxL118, G306-3 and G915-4 respectively.

Aside from the bottle roll standards, the other QAQC checks undertaken by both the bottle roll and fire assay do not indicate any serious issues in the sample preparation or analytical equipment. No blank QAQC has been completed on the fire assays at SGS Bamako. SRK recommend that blank analysis is completed on the fire assays in any future drill programmes.

7 MINERAL PROCESSING AND METALLURGICAL TESTWORK

7.1 Introduction

Cora Gold commissioned Wardell Armstrong International (“WAI”) to undertake a programme of initial metallurgical testwork on the Sanankoro oxide mineralisation, in addition to very preliminary metallurgical testing of the sulphide mineralisation.

An overview of the results of metallurgical testwork undertaken on both the oxide and sulphide mineralisation is provided in the following sections. This is largely taken directly from the Wardell Armstrong metallurgical testwork reports named in Section 2 (Wardell Armstrong, 2019a and Wardell Armstrong 2019b).

7.2 Oxide Testwork

Oxide testwork was completed by WAI on 187 samples of RC drill chips, which were used to prepare 3 composite samples of different lithological characteristics, namely carbonaceous phyllite and sandstone, both relating to samples drilled at Selin, and a “Zone A + B” composite sample. The grade of gold in the samples submitted for testing, as determined by screened metallics analysis, ranged from 1.47ppm Au for the Zone A+B sample to 1.93ppm Au for the Sandstone sample. The grade of gold in the Carb Phyllite sample was 1.89ppm Au. A separate 37 intervals of diamond core were used to prepare a single composite sample of oxide mineralisation for comminution testing.

Comminution Testwork

- The Bond Crusher Work Index value of the oxide mineralisation was 1.68kWh/t, classifying the material as “very easy” with respect to crushability. It should however, be noted that testing was undertaken on “non-standard” particles and the results should therefore be considered as indicative only;
- Bond Abrasion testing showed the oxide ore to have an abrasion index value of 0.0692 which is classified as “non-abrasive” under standard classification criteria;
- The Bond Rod Mill Work index value of the oxide ore was 6.63kWh/t which is classified as “very soft” with respect to coarse ore grindability; and
- The Bond Ball Mill Work index of the oxide ore was 12.01kWh/t which is classified as “Medium" with respect to fine ore grindability.

Gravity Testwork

- Bulk two-stage gravity testing achieved gold recoveries of 39.74% from the Sandstone sample, 49.14% from the Carb Phyllite sample and 68.99% from the Zone A+B sample;
- Intensive leaching of the gravity concentrates achieved gold recoveries of 75.6 and 76.5% for the Carb Phyllite and Zone A+B samples increasing to 99.3% for the Sandstone sample;
- Cyanide leaching of the gravity tailings achieved gold recoveries of 90.9 – 91.0% (Carb Phyllite), 92.8 – 93.6% (Sandstone) and 93.3 – 94.7% (Zone A+B); and
- The overall gold recoveries from the combined gravity and leach stages were 82.16% (Zone A+B), 83.47% (Carb Phyllite) and 95.39% (Sandstone).

Whole Ore Leach Testwork

- Whole ore leach testing of the Carb Phyllite sample showed gold recoveries ranging from 91.6 – 95.7% after 48 hours of leaching. Silver recoveries ranged from 34.9 – 55.0%;
- Gold recoveries from the Sandstone sample ranged from 91.3 – 97.1%, whilst gold recoveries from the Zone A+B sample ranged from 92.5 – 97.1%;
- For all three samples, the maximum gold leach recovery was achieved when leached using O2 as the sparging gas. When sparged with air, the reduction in gold recovery ranged from 0.2% (Zone A+B) to 1.0% (Sandstone).

Dewatering Testwork

- Static settling testing, which was performed on a composite of the Carb Phyllite, Sandstone and Zone A+B leach residues which had been blended in equal proportion, showed the flocculant Magnafloc 366 to provide the best settling characteristics with respect to initial settling velocity and supernatant quality;
- Flocculant dosage testing showed the highest final solids concentration to have been achieved when using 30-45g/t flocculant with the 45g/t flocculant achieving a slightly higher initial settling rate; and
- Feed density testing showed an optimum thickener unit area of 0.08 m2/t/hr to have been achieved when treating the tailings using 30g/t flocculant at a feed solids concentration of 20% w/w.

Environmental Testwork

- ABA testing, conducted on a composite sample of the Carb Phyllite, Sandstone and Zone A+B leach residue, showed the material to be non-acid generating with a neutralisation potential ratio (NPR) of 10.67;
- NAG testing confirmed that the material was not acid generating with a NAG pH of 10.0;
- TCLP testing did not show elevated levels of any potentially hazardous elements within the leachate solution; and
- Detailed chemical analysis of the oxide leach residue showed it to contain 476ppm As, 0.28ppm Bi, 0.55ppm Cd and 0.025ppm Hg.

7.3 Sulphide Testwork

Sulphide testwork was completed by WAI on 36 samples of RC drill chips, which were blended to prepare 2 samples of oxide mineralisation of different lithological characteristics, namely carbonaceous phyllite and volcanic. The grade of gold in the samples, as determined by screened metallics analysis, ranged from 1.70ppm Au for the Carb Phyllite ore type to 5.22ppm Au for the Volcanic ore type. A separate 10 intervals of diamond drill core were used to prepare a single composite sample representative of sulphide mineralisation for comminution testing.

Comminution Testwork

- Bond Ball Mil Work Index testing, conducted on a composite sample of the sulphide mineralisation, showed the material to have a work index value of 13.87 kWh/t. Using standard criteria, this would classify the material as “Medium” with respect to grindability.

Gravity Testwork

- Gravity testing of a composite of the two sulphide ore types, blended in equal proportion, showed that 68.2% of the gold present could be recovered following two stages of gravity concentration. Of the gold recovered, 50.5% was recovered during the first stage of separation at a grind size of 80% passing 212µm with the remaining 17.7% recovered at the finer grind size of 80% passing 75µm;
- Cyanide leach testing of the gravity tailings showed poor gold recovery with just 9.5% of the gold recovered after 48 hours when leached using a 0.5g/L cyanide solution increasing to 11.0% when leached using a 1.0g/L solution; and
- Subsequent analysis of the leach kinetics for both tests indicated the possibility of preg-robbing of gold from solution during leaching with gold recoveries having reached a maximum of 47.5-50.2% after two hours of leaching before reducing to the final levels reported.

Whole Ore Leach Testwork

- Whole ore leach testing of the Carb Phyllite ore type achieved gold recoveries of 3.9% after 48 hours of leaching whilst recoveries from the Volcanic ore type were higher at 55.2%; and
- As with the gravity tailings leaches, analysis of the leach kinetics for both tests indicated the possibility of preg-robbing during leaching with gold recoveries having reached maximums of 17.0% for the Carb Phyllite ore after two hours of leaching and 73.2% for the Volcanic ore after eight hours of leaching before again reducing to the final values reported.

Diagnostic Leach Testwork

- Diagnostic leach testing, undertaken to determine the deportment of gold within each of the samples with respect to host mineralisation, indicated that approximately 58.2% of the gold present in the Carb Phyllite material was associated with carbonaceous material with a further 32.4% encapsulated or locked within silicates. Of the remaining 9.4% gold, 5.6% was associated with sulphide minerals; and
- Testing of the Volcanic ore type, which contained a higher proportion of cyanide recoverable gold, showed that 21.2% of the gold present was encapsulated or associated with silicates with a further 18.7% associated with carbonaceous minerals.
- Just 4.5% of the gold was associated with sulphide minerals.

8 MINERAL RESOURCE ESTIMATE

8.1 Introduction

All resource domain modelling, block modelling and grade interpolation was completed in Seequent Leapfrog Geo 4.5. The resource estimation methodology involved the following procedures:

- Construction of resource domain mineralisation wireframes;
- Data conditioning (compositing and capping review);
- Statistical and geostatistical analysis;
- Block modelling and grade interpolation;
- Resource classification and validation; and
- Preparation of the Mineral Resource Statement, including the construction of grade-tonnage curves.

8.2 Resource Domain Modelling

8.2.1 Topography

The mineralisation and weathering domain models described in Sections 8.2.2 and 0 are constructed below a topographic surface, generated from ASTER survey data, which has been variably offset in the Z, to be locally consistent with the elevation of the Cora Gold collars. The SWM collars, for which there is a lower confidence in the precision of the collar survey elevation (see Section 6.3), have been snapped to the elevation of this offset ASTER topography surface. The final topography surface used in limiting the geology and mineralisation wireframes is displayed in Figure 8-1.

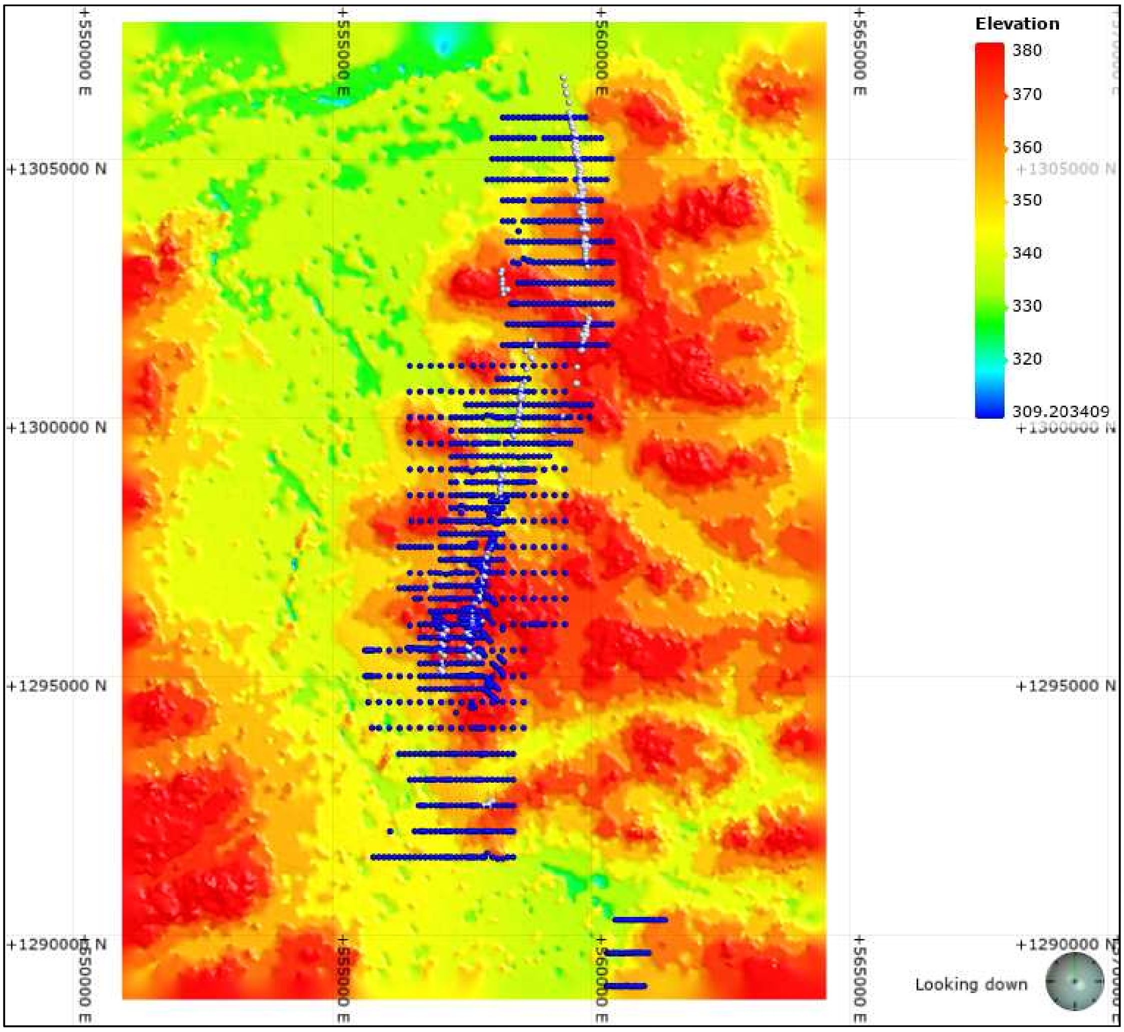


Figure 8-1: The Sanankoro final topography surface, coloured by elevation, shown alongside the historic (blue) and Cora Gold (white) drill hole collars.

8.2.2 Mineralisation Domains

SRK directly utilised the downhole assay data to model the mineralised domains by coding assay intervals into groups considered to form consistent mineralised corridors and subsequently turning these selections into discrete volumes. This was completed using the Leapfrog vein modelling tool, which works by selecting and coding specific intervals, which can be used to define discrete volumes. Footwall and hangingwall points are extracted from the manual selections and these are used to create automatically interpolated footwall and hangingwall surfaces which are used to define mineralisation shapes and volumes.

Before modelling, the raw gold assay data was composited to 3 m intervals to more readily evaluate the overall continuity in the mineralised zones, with vein interval selections completed directly on the raw un-composited assay data. Typically, the Sanankoro mineralisation is defined by a clear and significant increase in grade, relative to the surrounding host rock, which can be clearly identified through visual assessment of the downhole assay grades. Therefore, no specific modelling cut-off was applied; rather domain contacts / limits were defined based on the position of step changes in gold grade. That said, the mineralisation domains were generally restricted to a minimum modelling cut-off of 0.2 g/t Au, and limited to zones of > 0.2 g/t Au that could be correlated in the 3 m composites across at least 3 drillholes.

Visual comparison of the distribution of the highest grade downhole assay intervals with the induced polarization (IP) geophysical anomaly map, suggests a strong spatial correlation between the mineralised zones, and the location of sharp contrasts between high and low IP anomalies. It is considered that these IP anomaly contrasts are most likely associated with the deposit-scale tight folding and thrust faulting interpreted to act as a conduit for the mineralised quartz veins. The strike of the modelled veins was guided by the trend of the IP anomaly contrasts between drillhole sections, whilst the dip of the modelled veins was based on visual continuity in downhole assay grades, and the known steeply dipping to sub-vertical dip of the main mineralised vein set, as described in Section 4.3.2.

At this stage, given the relatively wide drillhole section spacing, modelling focussed on connecting mineralised intervals parallel to the main steeply dipping NNE-SSE striking vein set, this being the principal focus of artisanal exploration to date. It is possible that more detailed mapping and more close spaced drilling, with associated structural data, may allow for delineation of the less prominent steeply dipping E-W oriented and sub-horizontal vein sets in future updates.

In total, 29 mineralised bodies have been modelled across Zone A, Zone B, Zone B North and Selin. These are described in Table 8-1 and displayed in Figure 8-2 to Figure 8-6.

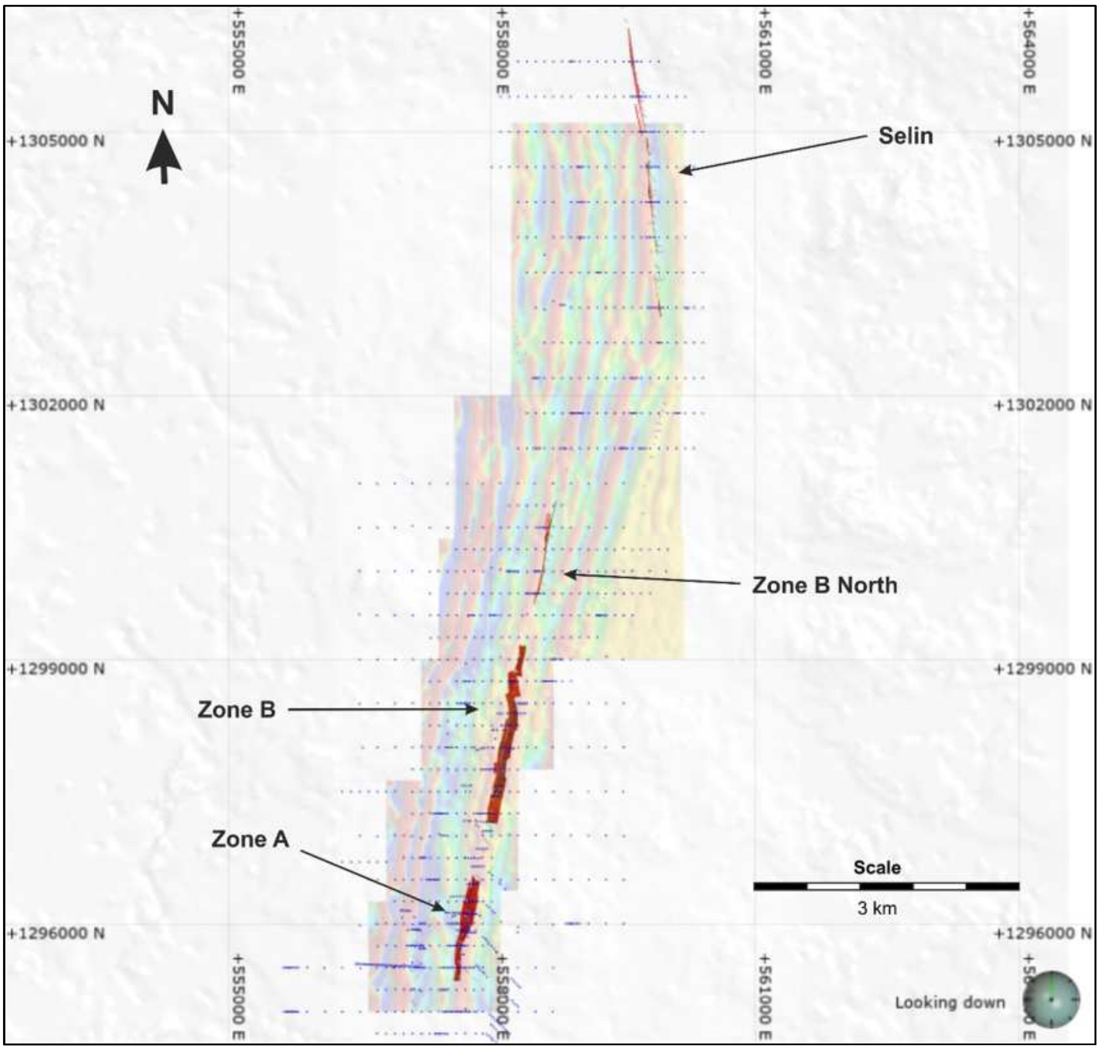


Figure 8-2: Map of the modelled mineralisation domains, shown relative to the IP survey map and drillhole collars.

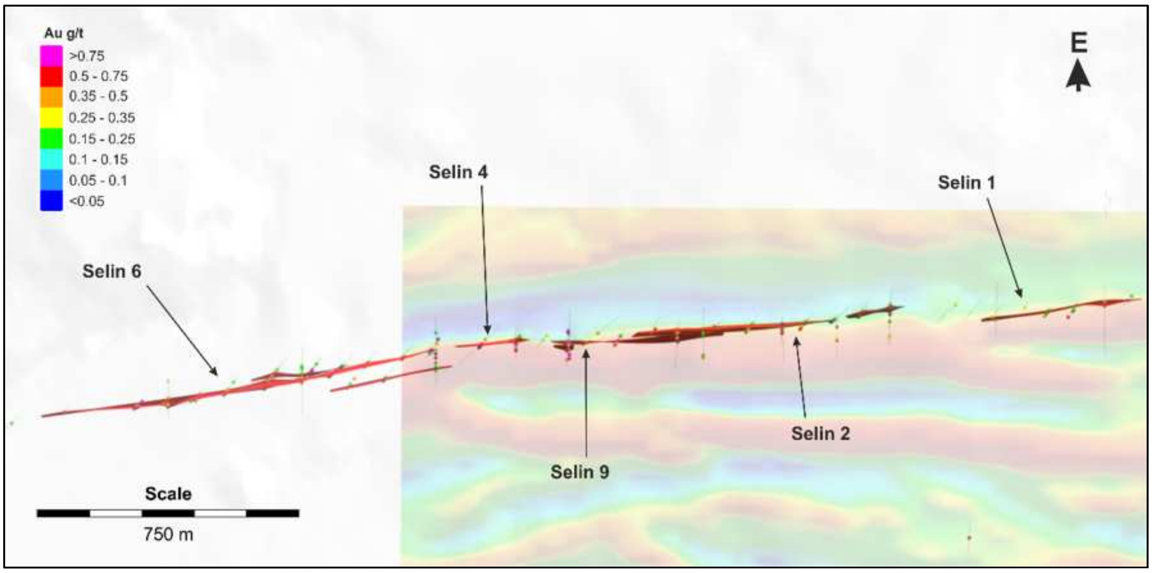


Figure 8-3: East-facing map of the Selin mineralisation domains, shown relative to the IP survey map and 3m composites >0.2 g/t Au. Key domains labelled.

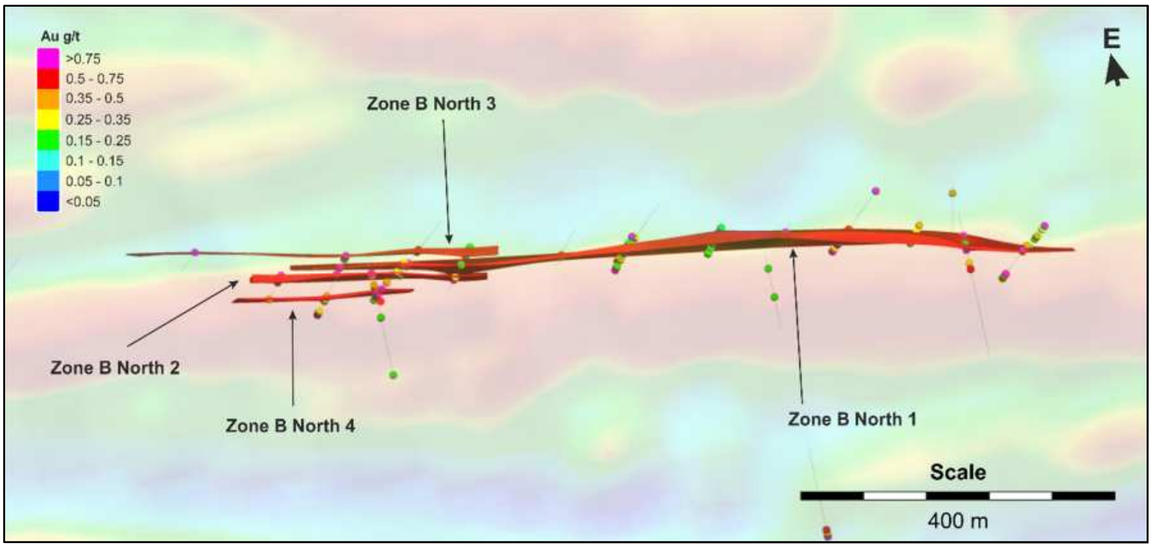


Figure 8-4: Map of the Zone B North mineralisation domains (facing towards 100°), shown relative to the IP survey map and 3m composites >0.2 g/t Au.

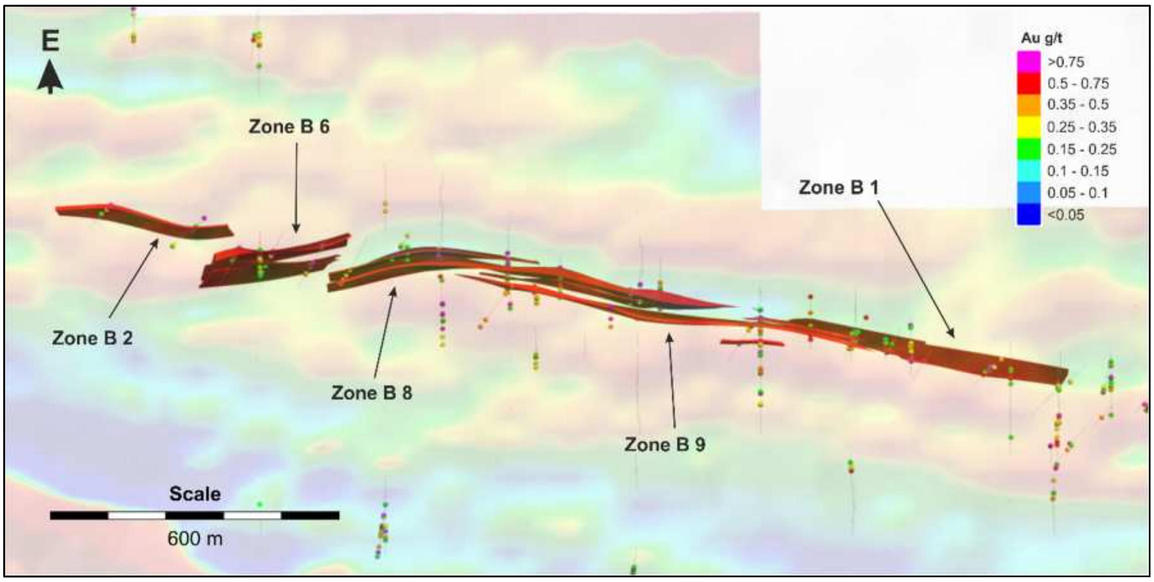


Figure 8-5: East-facing view (inclined at 75°) of the Zone B mineralisation domains, shown relative to the IP survey map and downhole 3m composites >0.2 g/t Au. Key domains labelled.

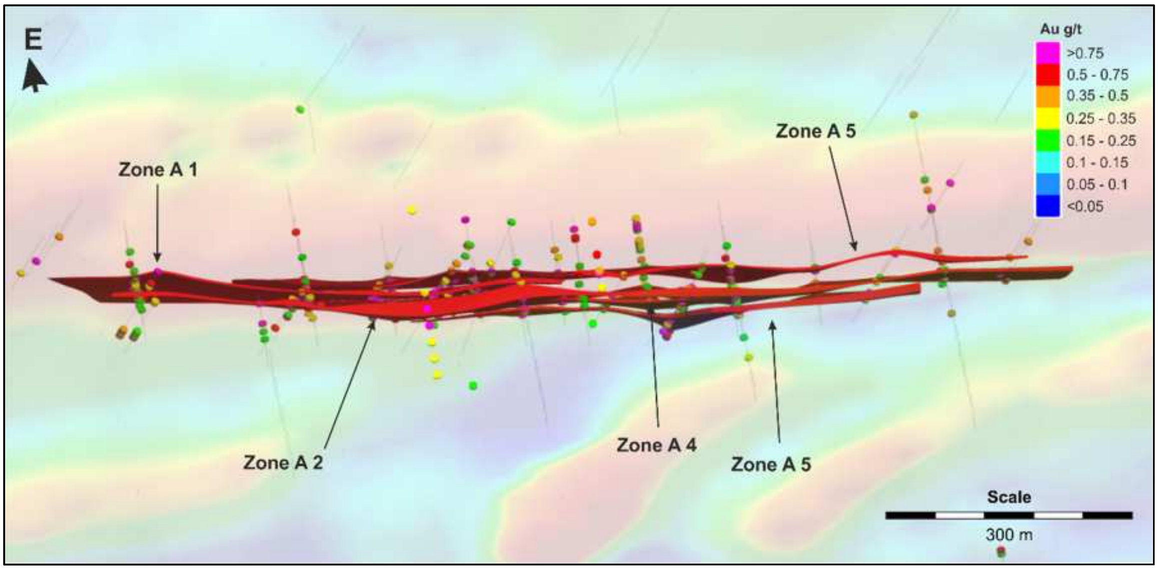


Figure 8-6: Inclined view (70° towards 100°) of the Zone A mineralisation domains (facing towards 100°), shown relative to the IP survey map and 3m composites >0.2 g/t Au.

Table 8-1: Orientation, strike extent and true thickness of the modelled mineralisation domains.

Domain	Zone	Dip (°)	Dip Direction (°)	Strike Length (m)	Average True Thickness (m)	Minimum True Thickness (m)	Maximum True Thickness (m)
Zone A	Zone A 1	75	100	600	4	1	8
	Zone A 2	72.5	97.5	1,150	9	1	19
	Zone A 3	72.5	97.5	950	4	1	10
	Zone A 4	72.5	100	200	4	1	10
	Zone A 5	75	97.5	600	4	1	9
Zone B	Zone B 1	70	100	400	5	1	10
	Zone B 2	80	97.5	350	5	3	8
	Zone B 3	72.5	102.5	350	5	1	8
	Zone B 4	80	82.5	250	5	2	7
	Zone B 5	75	92.5	100	3	1	4
	Zone B 6	77.5	77.5	250	9	4	17
	Zone B 7	80	82.5	250	3	2	4
	Zone B 8	77.5	92.5	800	11	1	26
	Zone B 9	75	100	800	5	1	12
	Zone B 11	77.5	102.5	350	3	1	4
	Zone B 12	77.5	87.5	400	2	1	4
Zone B North	Zone B N 1	90	100	950	7	1	18
	Zone B N 2	90	102.5	300	3	1	10
	Zone B N 3	87.5	102.5	450	5	1	11
	Zone B N 4	90	97.5	200	3	1	6
Selin	Selin 1	87.5	82.5	500	2	1	3
	Selin 2	90	85	600	5	2	11
	Selin 4	87.5	85	200	5	4	7
	Selin 5	87.5	77.5	350	4	2	7
	Selin 6	87.5	80	1,200	7	1	20
	Selin 7	87.5	80	300	6	2	12
	Selin 8	82.5	272.5	50	3	1	6
	Selin 9	90	87.5	450	2	1	6
	Selin 10	90	260	200	6	2	13

8.2.3 Weathering Model

The weathering profile observed at Sanankoro is transitional, from surface hardcap, through saprolite and saprock down to fresh un-weathered material at a depth of 30 – 125 m. The Cora Gold drillholes have been logged for regolith type, divided into “Cuirasse” (hardcap) “Saprolite”, “Saprock” and “Fresh”. SRK used these codes to define 4 weathering domains, namely hardcap, saprolite, saprock and fresh material. The weathering domains were modelled using an offset mesh function based on the topography surface, in order to honour the topographic control on the geometry of the weathering profile. A typical section through the weathering profile is displayed in Figure 8-7.

- **Base of Hardcap** – Minimum Depth = 0 m, Maximum Depth = 20 m, Average Depth = 5 m;
- **Base of Saprolite** - Minimum Depth = 10 m, Maximum Depth = 120 m, Average Depth = 55 m.
- **Base of Saprock** - Minimum Depth = 30 m, Maximum Depth = 125 m, Average Depth = 65 m.

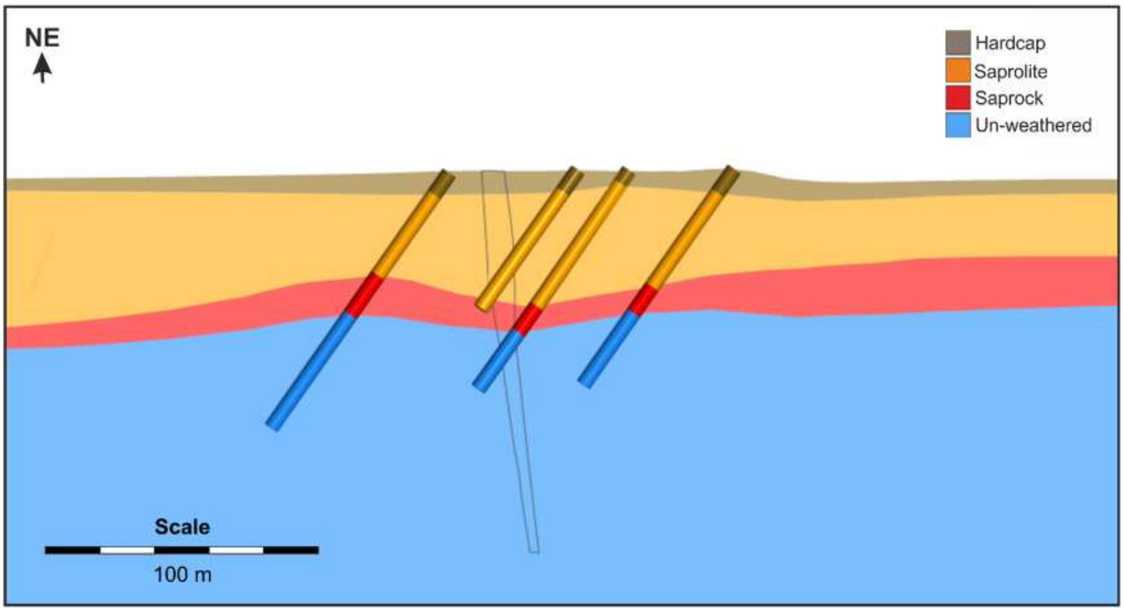


Figure 8-7: Northeast facing section through Zone B North showing the weathering model shown relative to the downhole regolith logging and the mineralisation outlines (in black).

8.3 Statistical and Geostatistical Analysis

8.3.1 Data Conditioning

Compositing

Data compositing is undertaken to reduce the inherent variability that exists within the raw assay grade population and to generate samples more appropriate to the scale of the mining operation envisaged. It is also necessary for the estimation process that all samples are assumed to be of equal weighting and should therefore be of equal length.

For all zones, the most common sample length is 1 m, although a significant number of 3 m samples are also included in the mineralisation wireframes in each zone. On average, the proportion of samples inside of the mineralisation wireframes that are 3 m in length is approximately 10%. For this reason, to avoid down-compositing of too many samples, it was decided to composite to 3 m. Composites of 3 m length were generated, using the mineralisation wireframes as compositing triggers. Given that the weathering surfaces represent gradational contacts, compositing was completed across the weathering surfaces without utilising these to trigger new composite samples.

The estimation process assumes that samples represent equal volumes. It may therefore be necessary to discard, or ignore, remnant composites smaller than the defined composite length (in this instance 3 m) generated in the downhole compositing process in the event that these may introduce bias into the estimation. Composite length analysis compares the length weighted mean grade of all samples within individual domains, with the mean grade of individual domains after the removal of remnant composites smaller than a defined length. This was conducted using the removal of remnant samples at intervals of 0.5 m up to the half the composite length of 3 m (Table 8-2.). The smallest difference between the length weighted mean of all samples, and the un-weighted mean grade after the removal of remnant composites at 0.5 m intervals determines the length of remnant composites to be removed before grade interpolation, whilst ensuring that not too many composite samples are removed from the dataset.

Table 8-2: Composite length analysis statistics for the Sanankoro mineralised zones; smallest mean differences for each domain are highlighted in yellow.

Zone	CLA Interval	Number of Composite Samples	Mean Au ppm	% Au ppm Difference	% of Samples Remaining
Zone A	Length Weighted	355	1.30	-	-
	>0.5 m	351	1.28	99.0	99
	>1.0 m	350	1.29	99.1	99
	>1.5 m	321	1.28	98.9	95
Zone B	Length Weighted	202	1.20	-	-
	>0.5 m	198	1.19	99.7	98
	>1.0 m	195	1.20	99.9	97
	>1.5 m	185	1.23	102.5	92
Zone B North	Length Weighted	107	1.27	-	-
	>0.5 m	103	1.28	100.2	96
	>1.0 m	103	1.28	100.2	96
	>1.5 m	95	1.33	104.2	89
Selin	Length Weighted	317	1.89	-	-
	>0.5 m	314	1.85	97.8	99
	>1.0 m	313	1.86	98.1	99
	>1.5 m	300	1.88	99.1	95

In the three of the four mineralised zones, the smallest difference between the length weighted mean of all samples, and the mean grade after the removal of remnant composites at 0.5 m intervals, is after the removal of composites <1.0 m in length. It is noted that the smallest difference in composite grade at Selin is after the removal of 1.5 m composites, however the total percentage of composite samples in <1.5 m in length across the deposit is quite large. Therefore, across all zones, it was decided to remove all remnant composites <1.0 m in length from the composite drill hole file used for grade estimation. The total number of samples removed from the composite file is 20, which represents 2.0% of all composite samples.

Treatment of High Grade Outliers

High grade capping is typically undertaken in order to reduce the impact on the interpolation of sample grades that are considered to be outside of the normal observed sample distribution and that can't be separately domained in order to be interpolated independently. Values above the cap value are reduced to the cap value.

SRK completed a capping analysis on the composite samples, based on the assessment of log probability plots, raw and log histograms, which were used to identify any sample grades outside of the main grade populations. At present, it is considered that the individual modelled mineralisation domains do not include sufficient sample points to appropriately assess capping values separately for each domain. The capping analysis was therefore completed by zone. In general, the gold grades associated with the oxidised mineralisation at Sanankoro are slightly elevated (in the order of 10-20%) in comparison to the sulphide mineralisation. For this reason, the capping analysis was completed separately for the oxide (all composite samples above the base of saprock surface described in Section 0) and sulphide samples (all composite samples below the base of saprock surface).

On the basis of the capping analysis, it was deemed necessary to cap the composited gold grades in the oxide mineralisation at Selin and Zone B North and in the sulphide mineralisation at Selin and Zone B. For all other zones SRK is satisfied that none of the composite samples fall outside of the main population and as such do not require capping. The capping limits applied and the impact on the mean gold grade of each domain are provided in Table 8-3. All composite sample grades above the capping limits applied were replaced with the value of the high grade cap.

Table 8-3: High grade caps applied to the Au composite samples, by zone and weathering state.

<table>
<tr><th>Zone</th><th>Weathering State</th><th>Selected High Grade Cap (Au ppm)</th><th>Number of Samples Capped</th><th>Mean Grade Pre-Capping (Au ppm)</th><th>Mean Grade Post-Capping (Au ppm)</th><th>% Reduction in Grade Post-Capping</th></tr>
<tr><td rowspan="2">Zone A</td><td>Oxide</td><td>-</td><td>-</td><td rowspan="2">1.25</td><td rowspan="2">1.25</td><td rowspan="2">-</td></tr>
<tr><td>Sulphide</td><td>-</td><td>-</td></tr>
<tr><td rowspan="2">Zone B</td><td>Oxide</td><td>-</td><td>-</td><td rowspan="2">1.20</td><td rowspan="2">1.17</td><td rowspan="2">-2.5%</td></tr>
<tr><td>Sulphide</td><td>5.0</td><td>1</td></tr>
<tr><td rowspan="2">Zone B N</td><td>Oxide</td><td>6.0</td><td>2</td><td rowspan="2">1.19</td><td rowspan="2">1.05</td><td rowspan="2">-11.6%</td></tr>
<tr><td>Sulphide</td><td>-</td><td>-</td></tr>
<tr><td rowspan="2">Selin</td><td>Oxide</td><td>12.0</td><td>7</td><td rowspan="2">1.82</td><td rowspan="2">1.66</td><td rowspan="2">-8.9%</td></tr>
<tr><td>Sulphide</td><td>9.0</td><td>1</td></tr>
</table>

8.3.2 Basic Statistics

The basic capped composite statistics for each domain are presented in Table 8-4. The distribution of gold grades are generally log-normal, with a very slight left skew, due to a small proportion of low grade composites included in the domain models, effectively internal waste, that cannot be practically removed at the current drillhole spacing. All domains are characterised by coefficients of variation ("CoV") close to or less than 1.0 after capping.

Table 8-4: The total length of samples, mean, minimum and maximum Au grades and coefficient of variation by estimation domain.

Zone	Domain	Total Length of Samples (m)	Mean (ppm)	Min (ppm)	Max (ppm)	CoV
Zone A	Zone A 1	127	1.10	0.09	4.45	0.89
	Zone A 2	640	1.45	0.01	9.87	1.07
	Zone A 3	174	0.84	0.04	8.03	1.39
	Zone A 4	51	1.18	0.23	4.88	0.98
	Zone A 5	84	0.93	0.07	5.00	1.34
ALL ZONE A		**1,076**	**1.25**	**0.01**	**9.87**	**1.13**
Zone B	Zone B 1	66	1.04	0.26	3.36	0.83
	Zone B 2	23	0.81	0.30	1.29	0.45
	Zone B 3	67	2.50	0.16	12.16	1.32
	Zone B 4	35	0.50	0.15	1.82	0.87
	Zone B 5	19	1.74	0.53	5.37	1.01
	Zone B 6	73	1.13	0.02	4.87	1.04
	Zone B 7	22	1.15	0.29	4.75	1.22
	Zone B 8	198	0.70	0.08	3.91	0.94
	Zone B 9	113	1.55	0.04	5.49	1.04
	Zone B 11	23	1.16	0.26	3.11	0.85
	Zone B 12	22	0.84	0.29	2.65	0.93
ALL ZONE B		**661**	**1.17**	**0.02**	**12.16**	**1.32**
Zone B North	Zone B N 1	241	1.05	0.02	6.00	1.09
	Zone B N 2	39	0.85	0.28	1.50	0.42
	Zone B N 3	38	1.49	0.26	6.00	1.16
	Zone B N 4	21	0.63	0.16	1.51	0.77
ALL ZONE B NORTH		**339**	**1.05**	**0.02**	**6.00**	**1.08**
Selin	Selin 1	23	0.90	0.01	2.14	0.66
	Selin 2	182	1.38	0.18	12.00	1.17
	Selin 4	58	1.80	0.34	10.93	1.28
	Selin 5	30	1.21	0.33	5.48	1.22
	Selin 6	536	1.81	0.01	12.00	1.36
	Selin 7	110	1.75	0.09	7.59	1.10
	Selin 8	35	2.61	0.32	9.00	1.06
	Selin 9	62	1.29	0.03	5.31	0.86
	Selin 10	58	1.30	0.11	5.78	1.06
ALL SELIN		**1,094**	**1.66**	**0.01**	**12.00**	**1.28**

8.3.3 Grade Continuity Analysis

Geostatistical analysis is the study of the spatial variability of an attribute, in this case composited gold grade. Gold grade continuity was assessed using experimental variograms, and also via isotropic radial basis function ("RBF") interpolants of the composite assays to assess trends in the distribution of Au grades.

Variography was only attempted on the largest and best informed domains in each zone, since all other domains do not contain a sufficient number of samples on which to undertake meaningful continuity analysis. Namely, the domains on which variography was attempted were the Zone A 2, Zone B 8, Zone B North 1 and Selin 6.

Prior to undertaking any variography, an RBF interpolation was completed on each of the domains in order to provide a broad indication as to the distribution of grade inside of each domain, and to identify any preferential mineralised trends. The RBF interpolants were completed using an isotropic search so as not to impart any pre-imposed trends on the resulting grade distribution plots.

Long-sections of the isotropic RBF interpolants for Zone A 2, Zone B 8, Zone B North 1 and Selin 6 are displayed in Figure 8-8 to Figure 8-11. The results of this exercise suggest that grade variations are more distinct along-strike than down-dip, indicating a structural control on the distribution of high grade zones, possibly relating to the intersection of the N-S trending mineralised structures with cross-cutting mineralised veins. This is most notable in Zone A 2, in which the distribution of higher and lower grade zones potentially suggests a shallow – moderately dipping plunge to the north. A clear plunge direction is less obvious for the other domains.

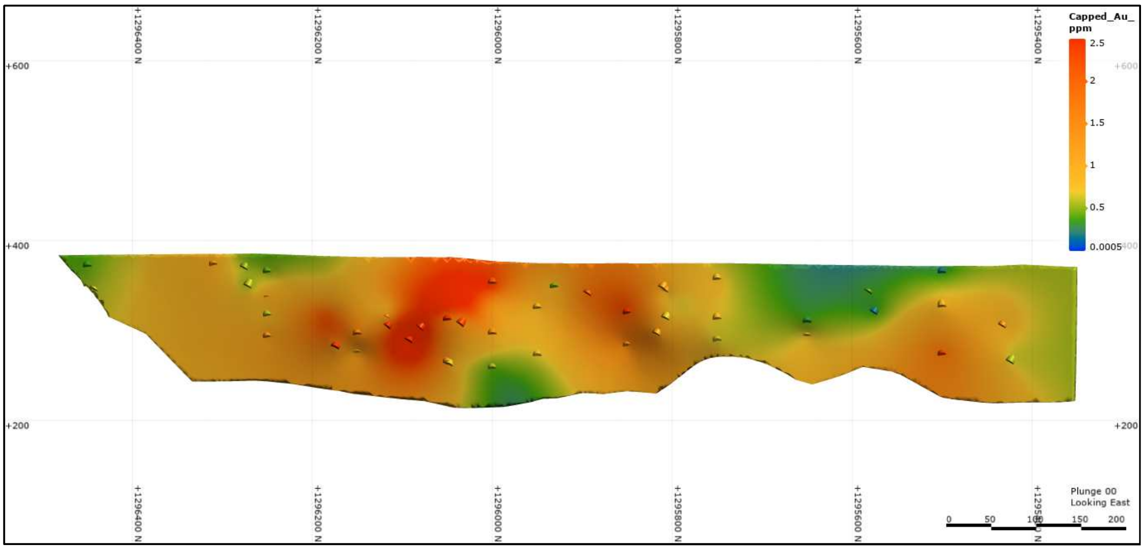


Figure 8-8: East facing long section of the Zone A 2 mineralisation wireframe, evaluated against an isotropic RBF interpolant of the assays inside the domain, displayed alongside the drillhole intersections.

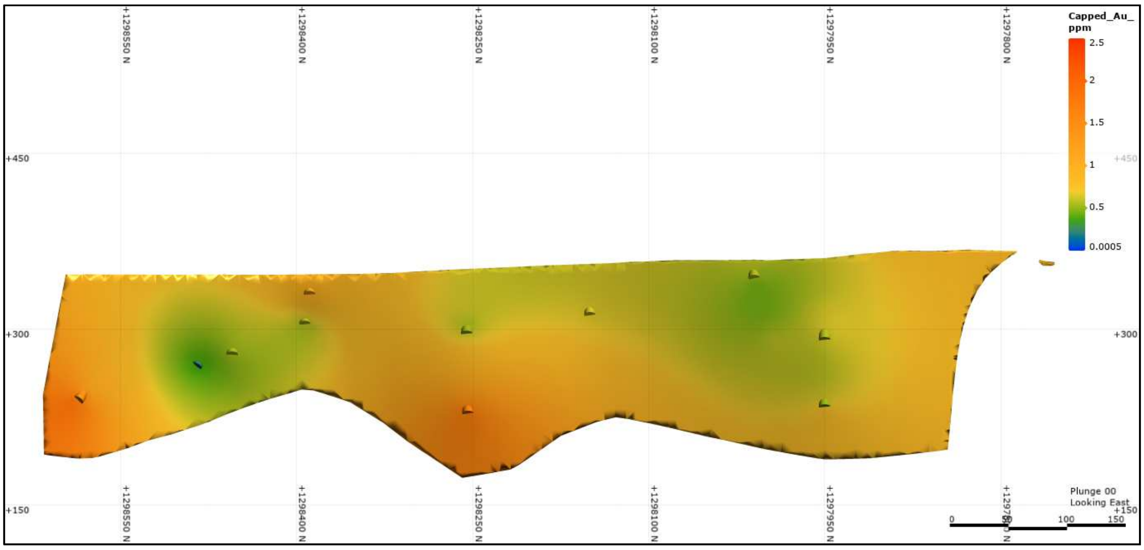


Figure 8-9: East facing long section of the Zone B 8 mineralisation wireframe, evaluated against an isotropic RBF interpolant of the assays inside the domain, displayed alongside the drillhole intersections.

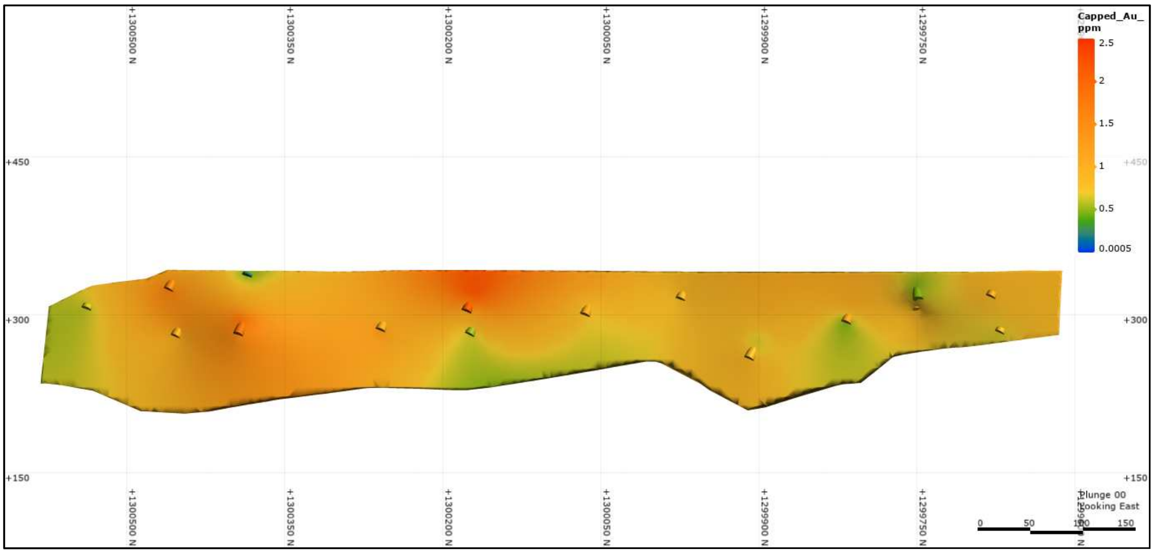


Figure 8-10: East facing long section of the Zone B North 1 mineralisation wireframe, evaluated against an isotropic RBF interpolant of the assays inside the domain, displayed alongside the drillhole intersections.

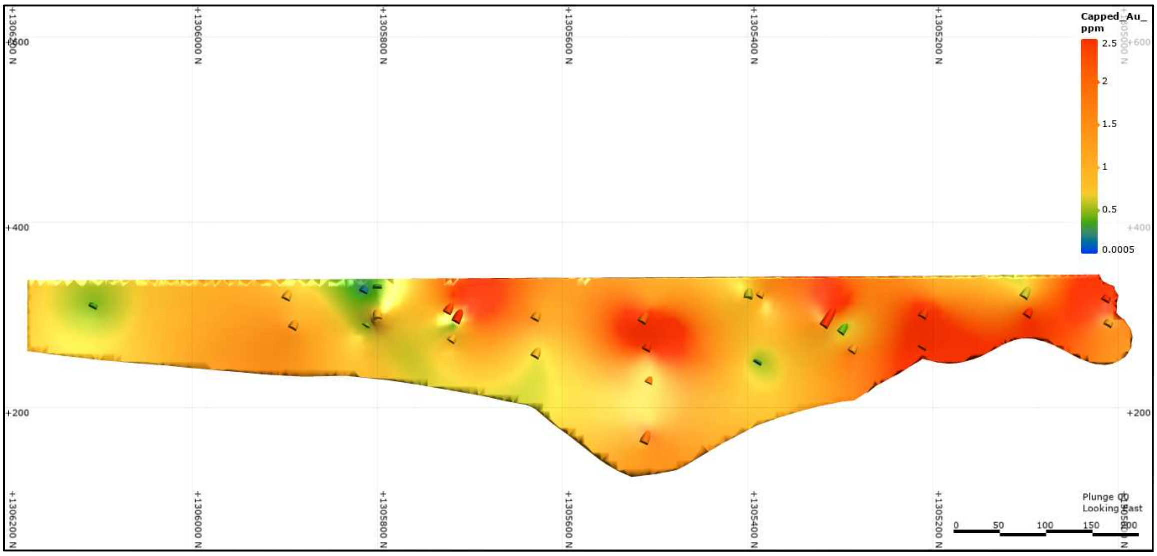


Figure 8-11: East facing long section of the Selin 6 mineralisation wireframe, evaluated against an isotropic RBF interpolant of the assays inside the domain, displayed alongside the drillhole intersections.

Downhole and directional variography was first attempted on Zone A 2, with search directions based on the moderate – shallow dipping plunge identified during RBF interpolant analysis. It was not possible to generate meaningful directional variograms, most likely a result of the small number of sample pairs in the direction of maximum grade continuity. Instead an omni-direction variogram was modelled. Both directional, and omni-directional variography was attempted on Zone B 8, Zone B North 1 and Selin 6, without success.

The downhole and omni-directional variograms for Zone A 2 are presented in Figure 8-12 and Figure 8-13, and the results summarised in Table 8-5. The omni-directional variogram presented for Zone A 2 is a relative variogram, as the standard semi-variogram for this domain was too noisy to model grade continuity. The purpose of a relative variogram is to reduce noise associated with high grade samples by dividing the gamma values from the sample pairs by the mean grade of the domain.

Table 8-5: Omni-directional variogram results for Zone A 2; all nugget and sill values are normalised as a ratio of the variance.

NUGGET	S1		S2		TOTAL SILL
	SILL	RANGE (m)	SILL	RANGE (m)	
0.4	0.3	50	0.25	150	0.95

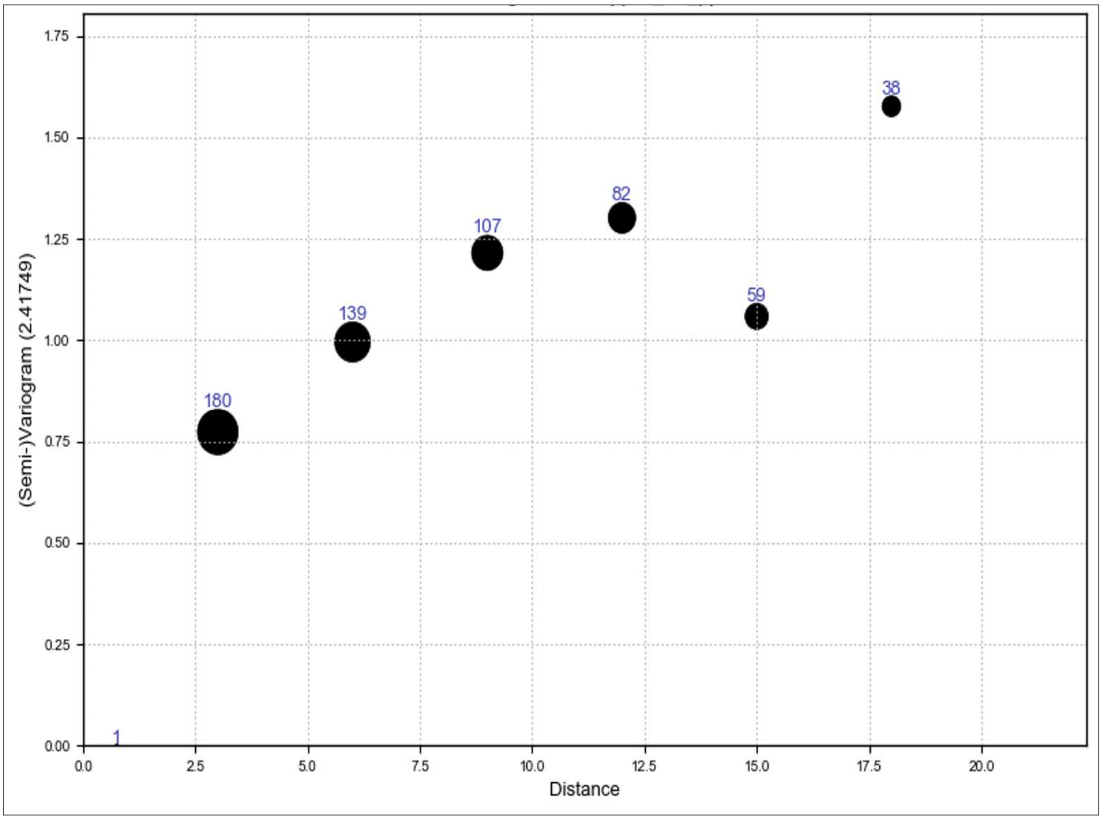


Figure 8-12: Downhole variogram for Zone A 2. Points scaled to number of pairs.

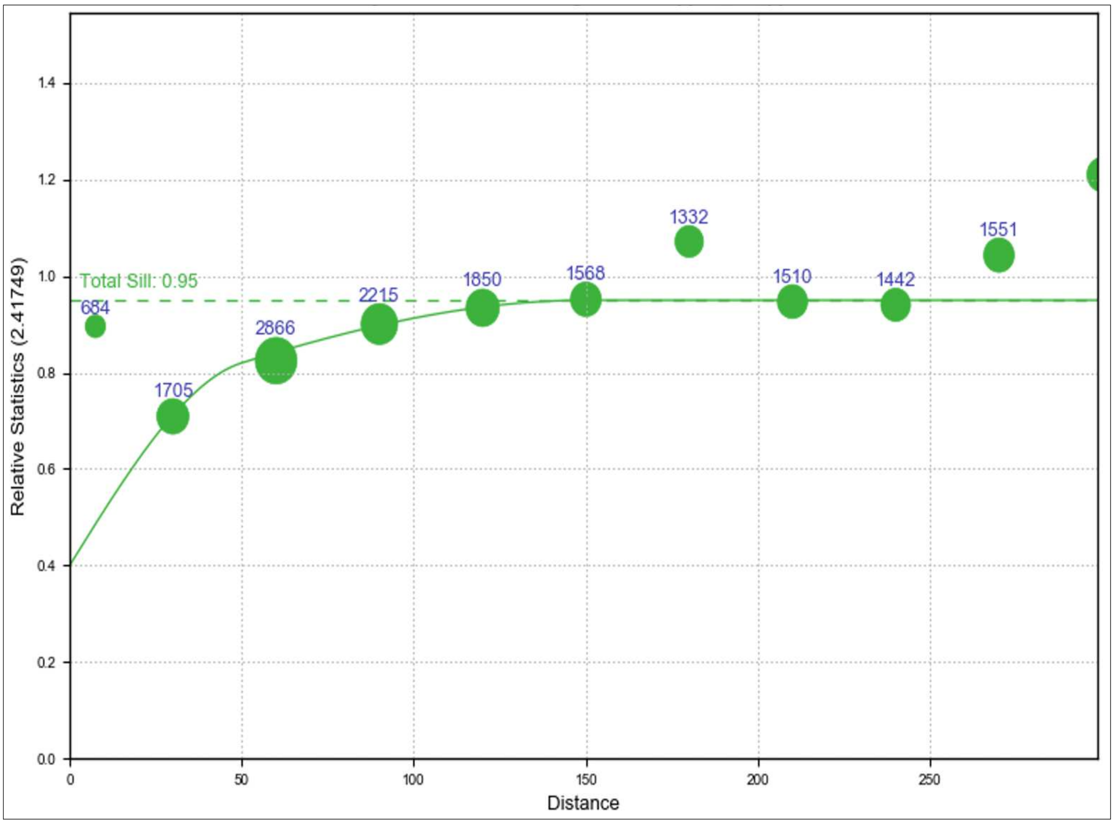


Figure 8-13: Omni-directional variogram for Zone A 2. Points scaled to number of pairs.

8.4 Block Model and Grade Estimation

8.4.1 Block Model Creation

Empty block models were generated within the solid wireframes of the mineralisation domains listed in Table 8-1. In addition, the block model was coded by the weathering model described in Section 0.

To reduce block model size, and to appropriately reflect the drillhole spacing and vein orientation in the parent block dimensions, separate block models were generated for each zone. Parent block sizes were selected based on the average drillhole spacing in each zone, being roughly half the on-section drillhole spacing and with approximately 2-3 columns of blocks between sections. To improve the geometric representation of the geological model, sub-blocking was allowed along the boundaries of the domains. Both the mineralisation wireframes and weathering profile wireframes were employed as sub-blocking triggers. The minimum sub block size was adjusted per area to appropriately reflect the geometry and volume of the mineralisation domains, whilst maintaining a practical block model file size. Both the mineralisation domain model and the weathering profile surfaces were employed as sub-blocking triggers. The parent block dimensions and minimum sub-block sizes for each area are provided in Table 8-6.

Table 8-6: Parent block and minimum sub-block dimensions.

Zone	Parent Block Dimensions (m)			Minimum Sub-Block Dimensions (m)		
	X	Y	Z	X	Y	Z
Zone A	10	25	15	0.5	1.25	1
Zone B	10	30	15	0.5	1.5	1
Zone B North	10	30	15	0.5	1.5	1
Selin	10	40	15	0.5	2	1

8.4.2 Grade Interpolation Parameters

Gold grades were interpolated into the block model, using the capped composite drillhole data from the corresponding mineralisation domain. The presently available assay data indicates that the difference in grade between weathering states is minor (an approximate 10-20% reduction in grade between oxide and sulphide zones) and gradational. As such, the weathering model was not used to sub-domain the grade interpolation.

Ordinary Kriging was used as the interpolation method for all domains. The mineralisation domains were all treated as hard boundaries in the estimation process. A discretization level of 3*3*3 was set for all estimates, and in all cases sub-block grades were assigned the grade of the parent block.

All mineralisation domains in Zone A were estimated using a moderately north-dipping search ellipse, with down-plunge range distances being set to approximately double the across-plunge range, in order to attempt to reflect the mineralisation plunge identified in the isotropic RBF interpolant described in Section 8.3.3. All other mineralisation domains were estimated using an isotropic ellipse, since no clear mineralisation plunge is evident for these domains at the current data spacing. The ellipsoid ranges were adjusted for each domain (whilst retaining a 2:1 ellipsoid ratio for Zone A and an isotropic ellipse for all other zones) to attempt to estimate each block using data from 2 – 3 drillhole sections, with the ellipsoid ranges limited to a maximum of 150 m, this being the modelled variogram range of Zone A 2.

As described in Section 8.3.3, the only domain in which it was possible to generate a meaningful variogram model was Zone A 2. Since the style of mineralisation is similar in all zones, the krigging variogram parameters for all domains were based on the results of the variography completed on this domain. This is based on the assumption that the grade continuity in Zone A, Zone B North and Selin will be comparable to Zone A. To attempt to better represent the shallow – moderately north plunging mineralisation trend interpreted in Zone A, the variogram parameters applied to the Zone A estimates were manually adjusted so that the down-plunge range was double that of the across-plunge range. For all other zones, the variogram ranges were un-changed from the omni-directional variogram ranges modelled for Zone A 2 (as outlined in Table 8-5).

Considering the average number of samples per drillhole and average number of drillholes per section in each domain, the minimum number of samples to be estimated into each block was adjusted for each domain to attempt to force the estimate to use samples from at least 2 sections in the estimation of each block. Additionally, in domain Selin 9, the maximum number of samples to be used from each drillhole for each block estimate was restricted to 4, to limit the impact of hole SC0012, which has a much larger intersection length than the other drillholes that intersect this domain.

Second, third and fourth searches, with progressively expanded ellipses and relaxed sample requirements were applied to fill any blocks not filled in the previous run. Specifically, in Search Volume 2 (“SV2”) the ellipse ranges were equal to 1.5 * the Search Volume 1 (“SV1”) ranges, in Search Volume 3 (“SV3”) the ellipse ranges were equal to 3 * the SV1 ranges, and in Search Volume 4 (“SV4”) an isotropic ellipse with a range of 500 m was employed. The minimum number of samples was relaxed in SV3 and SV4 and adjusted per domain to try to ensure that at samples from at least 2 drillholes are used for each block estimate (considering the average number of composites per drillhole intersection in each domain). For both SV3 and SV4 the maximum number of samples per block estimate was also restricted to try to avoid using samples from more than 3 sections in each block estimate, given the large ellipse size for these estimation runs.

The estimation sample selection parameters, including ellipsoid ranges, minimum and maximum number of samples to be estimated into each block, for all mineralisation domains, are provided in Table 8-7 to Table 8-10.

Table 8-7: Search Volume 1 estimation parameters.

Zone	Domain	Ellipsoid Ranges (m)			Ellipsoid Directions (°)			Min No. of Samples	Max No. of Samples	Max No. of Samples per Drillhole
		Max	Int	Min	Dip	Azimuth	Pitch			
Zone A	Zone A 1	125	125	65	45	10	90	10	40	-
	Zone A 2	150	150	75	45	10	90	20	40	-
	Zone A 3	115	115	60	45	10	90	10	40	-
	Zone A 4	85	85	45	45	10	90	10	40	-
	Zone A 5	150	150	75	45	10	90	5	40	-
Zone B	Zone B 1	75	75	75	0	0	0	5	40	-
	Zone B 2	150	150	150	0	0	0	5	40	-
	Zone B 3	120	120	120	0	0	0	5	40	-
	Zone B 4	120	120	120	0	0	0	5	40	-
	Zone B 5	70	70	70	0	0	0	5	40	-
	Zone B 6	75	75	75	0	0	0	10	40	-
	Zone B 7	120	120	120	0	0	0	5	40	-
	Zone B 8	150	150	150	0	0	0	15	40	-
	Zone B 9	130	130	130	0	0	0	10	40	-
	Zone B 11	150	150	150	0	0	0	5	40	-
	Zone B 12	150	150	150	0	0	0	5	40	-
Zone B N	Zone B N 1	135	135	135	0	0	0	10	40	-
	Zone B N 2	90	90	90	0	0	0	5	40	-
	Zone B N 3	135	135	135	0	0	0	5	40	-
	Zone B N 4	115	115	115	0	0	0	5	40	-
Selin	Selin 1	120	120	120	0	0	0	5	40	-
	Selin 2	135	135	135	0	0	0	15	40	-
	Selin 4	150	150	150	0	0	0	10	40	-
	Selin 5	150	150	150	0	0	0	5	40	-
	Selin 6	150	150	150	0	0	0	20	40	-
	Selin 7	100	100	100	0	0	0	15	40	-
	Selin 8	150	150	150	0	0	0	5	40	-
	Selin 9	135	135	135	0	0	0	5	40	4
	Selin 10	135	135	135	0	0	0	10	40	-

Table 8-8: Search Volume 2 estimation parameters.

Zone	Domain	Ellipsoid Ranges (m)			Ellipsoid Directions (°)			Min No. of Samples	Max No. of Samples	Max No. of Samples per Drillhole
		Max	Int	Min	Dip	Azimuth	Pitch			
Zone A	Zone A 1	190	190	95	45	10	90	10	30	-
	Zone A 2	225	225	115	45	10	90	20	30	-
	Zone A 3	175	175	90	45	10	90	10	30	-
	Zone A 4	130	130	65	45	10	90	10	30	-
	Zone A 5	225	225	115	45	10	90	5	30	-
Zone B	Zone B 1	115	115	115	0	0	0	5	30	-
	Zone B 2	225	225	225	0	0	0	5	30	-
	Zone B 3	180	180	180	0	0	0	5	30	-
	Zone B 4	180	180	180	0	0	0	5	30	-
	Zone B 5	105	105	105	0	0	0	5	30	-
	Zone B 6	115	115	115	0	0	0	10	30	-
	Zone B 7	180	180	180	0	0	0	5	30	-
	Zone B 8	225	225	225	0	0	0	15	30	-
	Zone B 9	195	195	195	0	0	0	10	30	-
	Zone B 11	225	225	225	0	0	0	5	30	-
	Zone B 12	225	225	225	0	0	0	5	30	-
Zone B N	Zone B N 1	205	205	205	0	0	0	10	30	-
	Zone B N 2	135	135	135	0	0	0	5	30	-
	Zone B N 3	205	205	205	0	0	0	5	30	-
	Zone B N 4	175	175	175	0	0	0	5	30	-
Selin	Selin 1	180	180	180	0	0	0	5	30	-
	Selin 2	205	205	205	0	0	0	15	30	-
	Selin 4	225	225	225	0	0	0	10	30	-
	Selin 5	225	225	225	0	0	0	5	30	-
	Selin 6	225	225	225	0	0	0	20	30	-
	Selin 7	150	150	150	0	0	0	15	30	-
	Selin 8	225	225	225	0	0	0	5	30	-
	Selin 9	205	205	205	0	0	0	5	30	4
	Selin 10	205	205	205	0	0	0	10	30	-

Table 8-9: Search Volume 3 estimation parameters.

Zone	Domain	Ellipsoid Ranges (m)			Ellipsoid Directions (°)			Min No. of Samples	Max No. of Samples	Max No. of Samples per Drillhole
		Max	Int	Min	Dip	Azimuth	Pitch			
Zone A	Zone A 1	375	375	195	45	10	90	3	25	-
	Zone A 2	450	450	225	45	10	90	5	25	-
	Zone A 3	345	345	175	45	10	90	3	15	-
	Zone A 4	255	255	135	45	10	90	3	20	-
	Zone A 5	450	450	225	45	10	90	4	15	-
Zone B	Zone B 1	225	225	225	0	0	0	4	15	-
	Zone B 2	450	450	450	0	0	0	5	10	-
	Zone B 3	360	360	360	0	0	0	4	15	-
	Zone B 4	360	360	360	0	0	0	3	15	-
	Zone B 5	210	210	210	0	0	0	3	20	-
	Zone B 6	225	225	225	0	0	0	5	25	-
	Zone B 7	360	360	360	0	0	0	3	10	-
	Zone B 8	450	450	450	0	0	0	5	25	-
	Zone B 9	390	390	390	0	0	0	4	20	-
	Zone B 11	450	450	450	0	0	0	3	10	-
	Zone B 12	450	450	450	0	0	0	2	10	-
Zone B N	Zone B N 1	405	405	405	0	0	0	5	25	-
	Zone B N 2	270	270	270	0	0	0	4	15	-
	Zone B N 3	405	405	405	0	0	0	5	10	-
	Zone B N 4	345	345	345	0	0	0	3	10	-
Selin	Selin 1	360	360	360	0	0	0	2	10	-
	Selin 2	405	405	405	0	0	0	5	25	-
	Selin 4	450	450	450	0	0	0	5	25	-
	Selin 5	450	450	450	0	0	0	5	10	-
	Selin 6	450	450	450	0	0	0	5	25	-
	Selin 7	300	300	300	0	0	0	5	25	-
	Selin 8	450	450	450	0	0	0	5	25	-
	Selin 9	405	405	405	0	0	0	3	15	-
	Selin 10	405	405	405	0	0	0	5	25	-

Table 8-10: Search Volume 4 estimation parameters.

Zone	Domain	Ellipsoid Ranges (m)			Ellipsoid Directions (°)			Min No. of Samples	Max No. of Samples	Max No. of Samples per Drillhole
		Max	Int	Min	Dip	Azimuth	Pitch			
Zone A	Zone A 1	500	500	500	0	0	0	3	25	-
	Zone A 2	500	500	500	0	0	0	5	25	-
	Zone A 3	500	500	500	0	0	0	3	15	-
	Zone A 4	500	500	500	0	0	0	3	20	-
	Zone A 5	500	500	500	0	0	0	4	15	-
Zone B	Zone B 1	500	500	500	0	0	0	4	15	-
	Zone B 2	500	500	500	0	0	0	5	10	-
	Zone B 3	500	500	500	0	0	0	4	15	-
	Zone B 4	500	500	500	0	0	0	3	15	-
	Zone B 5	500	500	500	0	0	0	3	20	-
	Zone B 6	500	500	500	0	0	0	5	25	-
	Zone B 7	500	500	500	0	0	0	3	10	-
	Zone B 8	500	500	500	0	0	0	5	25	-
	Zone B 9	500	500	500	0	0	0	4	20	-
	Zone B 11	500	500	500	0	0	0	3	10	-
	Zone B 12	500	500	500	0	0	0	2	10	-
Zone B N	Zone B N 1	500	500	500	0	0	0	5	25	-
	Zone B N 2	500	500	500	0	0	0	4	15	-
	Zone B N 3	500	500	500	0	0	0	5	10	-
	Zone B N 4	500	500	500	0	0	0	3	10	-
Selin	Selin 1	500	500	500	0	0	0	2	10	-
	Selin 2	500	500	500	0	0	0	5	25	-
	Selin 4	500	500	500	0	0	0	5	25	-
	Selin 5	500	500	500	0	0	0	5	10	-
	Selin 6	500	500	500	0	0	0	5	25	-
	Selin 7	500	500	500	0	0	0	5	25	-
	Selin 8	500	500	500	0	0	0	5	25	-
	Selin 9	500	500	500	0	0	0	3	15	-
	Selin 10	500	500	500	0	0	0	5	25	-

8.4.3 Block Model Validation

Visual Validation

Visual validation provides a comparison of the interpolated block model on a local scale. A thorough visual inspection has been undertaken in 3D, comparing the domain sample grades with the block grades in the corresponding modelled mineralisation domains. These visual checks generally demonstrate a strong comparison between local block estimates and nearby samples, without excessive smoothing in the block model. Figure 8-14 to Figure 8-17 show examples of the visual validation checks on the largest domains in each zone (namely Zone A 2, Zone B 8, Zone B North 1 and Selin 6) and highlight how the overall block grades correspond with composite sample grades. The interpreted shallow – moderately north dipping mineralisation plunge in Zone A 2 is well reflected in the block estimation.

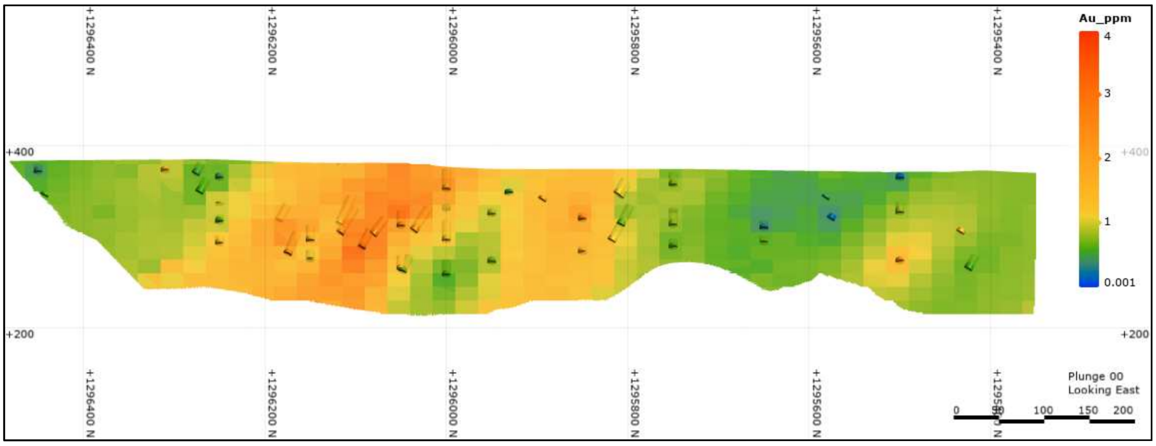


Figure 8-14: East facing long section of the estimated block model for Zone A 2, shown relative to the input drillhole data, composited to a single sample per intersection for visualisation purposes.

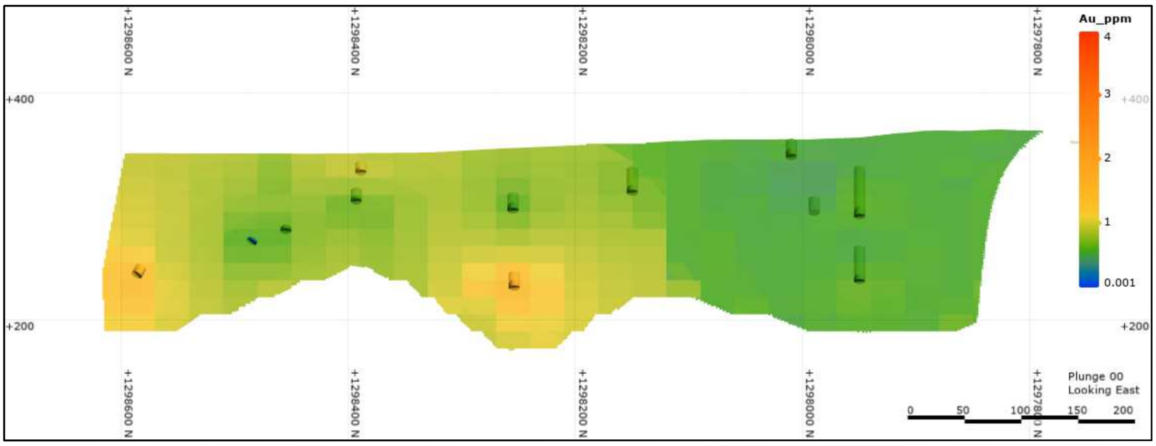


Figure 8-15: East facing long section of the estimated block model for Zone B 8, shown relative to the input drillhole data, composited to a single sample per intersection for visualisation purposes.

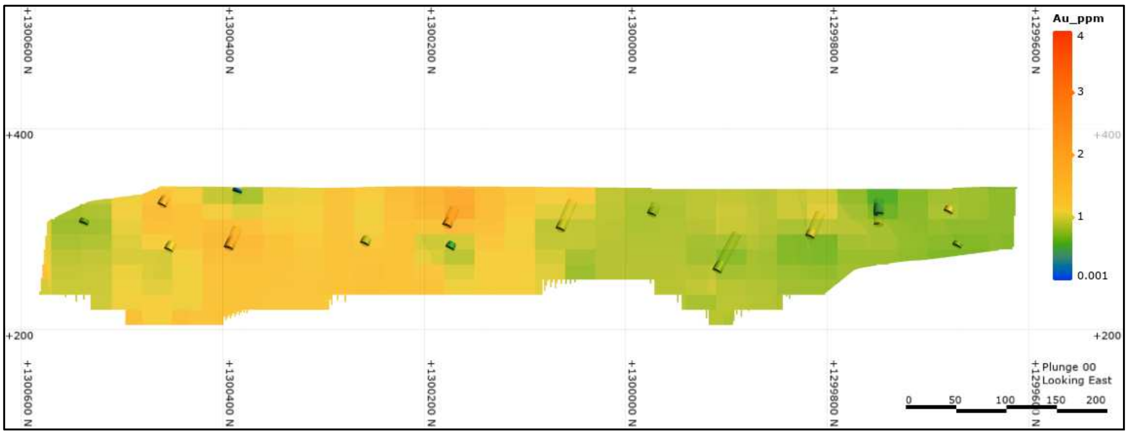


Figure 8-16: East facing long section of the estimated block model for Zone B North 1, shown relative to the input drillhole data, composited to a single sample per intersection for visualisation purposes.

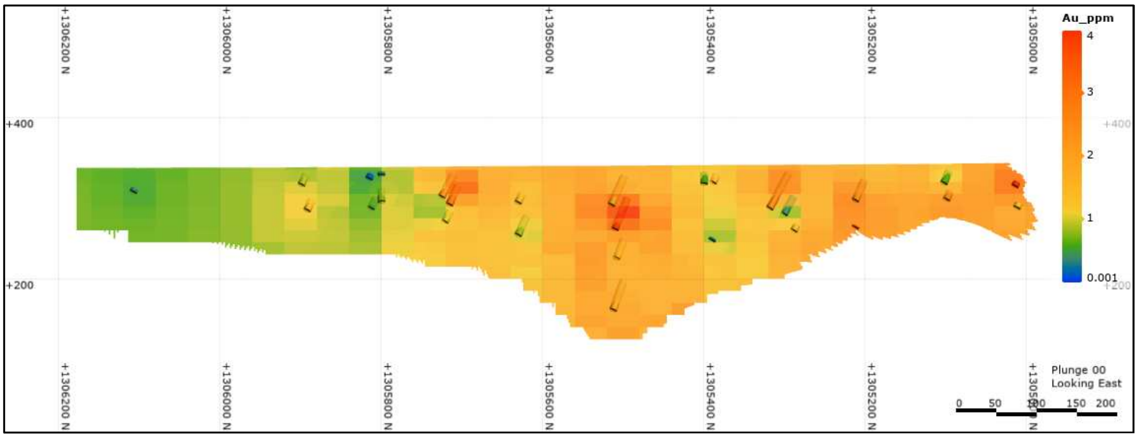


Figure 8-17: East facing long section of the estimated block model for Selin 6, shown relative to the input drillhole data, composited to a single sample per intersection for visualisation purposes.

Statistical Validation

The estimated block grades for the vein domains have been compared to the mean of the capped composite samples on which the estimate was based (Table 8-11). The results of this exercise indicate that in the significant majority of instances, the estimated block grades are within 10% of the mean capped composite grades. For a small number of mineralisation domains, the mean estimated block grades differ by more than 10% from the mean capped composite grades; these are all relatively poorly informed domains, with fewer than 50 composite samples available for estimation. SRK have investigated the cause of the discrepancies between the block model grade and input composite sample grades for these domains, which are primarily a function of either a single large intersection skewing the mean composite grade, or clustering of high or low grade drillhole intersections, as summarised below:

- **Zone A 5** *(mean composite grade 74% of block model grade)* – the highest drillhole grades are in more clustered areas of drilling, with the lowest grade drillhole intervals in areas of wider drillhole spacing.
- **Zone B 1** *(mean composite grade 85% of block model grade)* – the highest drillhole grades are in more clustered areas of drilling. Additionally, the lowest grade intersections typically relate to the deepest intersections in the model, which influence a greater number of blocks in the down-dip continuation of the model that has yet to be tested by drilling.
- **Zone B 3** *(mean composite grade 111% of block model grade)* – the highest grade intersections typically relate to the deepest intersections in the model, which influence a greater number of blocks in the down-dip continuation of the model that has yet to be tested by drilling.
- **Zone B 9** *(mean composite grade 89% of block model grade)* – the highest drillhole grades are in more clustered areas of drilling, with the lowest grade drillhole intervals in areas of wider drillhole spacing.
- **Selin 4** (mean composite grade 86% of block model grade) – the highest grade drillhole intersection in this domain is SC0214, which is also the longest intersection and therefore associated with the largest number of composite samples, which has a greater impact on the mean composite grade than the “de-clustered” block estimate.
- **Selin 8** *(mean composite grade 88% of block model grade)* – the highest grade drillhole intersection in this domain is SC0001, which is also the longest intersection and therefore associated with the largest number of composite samples, which has a greater impact on the mean composite grade than the “de-clustered” block estimate.

Table 8-11: Block estimate and capped composite mean grades by domain.

Zone	Domain	Mean Block Model Grade (Au ppm)	Mean Composite Grade (Au ppm)	Block Model Grade as % of Composite Grade
Zone A	Zone A 1	1.01	1.10	92%
	Zone A 2	1.38	1.45	95%
	Zone A 3	0.80	0.84	95%
	Zone A 4	1.21	1.18	102%
	Zone A 5	0.69	0.93	74%
ALL ZONE A		**1.16**	**1.25**	**93%**
Zone B	Zone B 1	0.88	1.04	85%
	Zone B 2	0.85	0.81	106%
	Zone B 3	2.78	2.50	111%
	Zone B 4	0.54	0.50	108%
	Zone B 5	1.87	1.74	107%
	Zone B 6	1.08	1.13	96%
	Zone B 7	1.15	1.15	101%
	Zone B 8	0.70	0.70	100%
	Zone B 9	1.38	1.55	89%
	Zone B 11	1.19	1.16	102%
	Zone B 12	0.77	0.84	92%
ALL ZONE B		**1.08**	**1.17**	**92%**
Zone B North	Zone B N 1	1.02	1.05	97%
	Zone B N 2	0.84	0.85	100%
	Zone B N 3	1.41	1.49	95%
	Zone B N 4	0.62	0.63	100%
ALL ZONE B NORTH		**1.01**	**1.05**	**96%**
Selin	Selin 1	0.92	0.90	102%
	Selin 2	1.31	1.38	95%
	Selin 4	1.55	1.80	86%
	Selin 5	1.30	1.21	107%
	Selin 6	1.64	1.81	90%
	Selin 7	1.63	1.75	93%
	Selin 8	2.29	2.61	88%
	Selin 9	1.30	1.29	101%
	Selin 10	1.35	1.30	104%
ALL SELIN		**1.52**	**1.66**	**92%**

In general, given the overall strong visual validation between the block and sample grades, and the rationale for the limited cases of significant differences between the block grade means and sample grade means, SRK is confident in the positive validation of the estimate of the Sanankoro mineralisation domains.

8.5 Density Assignment

Cora Gold provided SRK with density data from 5 alternative sources / methodologies, as outlined below:

- Drill core density determinations on the fresh rock, saprock and saprolite, using the water immersion method as described in Section 5.4.7.
- Drill core density determinations on the fresh rock saprock and saprolite, based on sample weight and volume measurements, as described in Section 5.4.7.
- Grab samples of saprolite material from the base of the artisanal workings, analysed for density using the waster immersion method, as described in Section 5.3.1.
- Grab samples of hardcap material, analysed for density using the waster immersion method, as described in Section 5.3.1.
- Field density determinations of the saprolite based on calculating the density of material removed from small excavations at the base on the artisanal workings, as described in Section 5.3.1.

The results of the density analyses are described in the sections below.

8.5.1 Field Density Determinations

In total, 7 density determinations were completed by excavating pits into mineralisation at the base of the artisanal workings. Of these, 4 were taken from Zone A, and 3 taken from Zone C, a lower grade mineralised zone on the Bokoro Structure, approximately 500 m west of Zone A. The results are provided in Table 8-12 below. The mean density of the saprolite samples taken from the small pit excavations is 2.13 g/cm^3.

Table 8-12: The results of the density determinations carried out on the saprolite mineralisation by the pit excavation method.

Zone	Sample Weight (kg)	Sample Volume (cm^3)	Density (g/cm^3)
Zone A	25.0	11,468	2.18
Zone A	25.0	11,074	2.26
Zone A	28.2	12,483	2.26
Zone A	25.9	12,000	2.16
Zone C	24.4	12,716	1.92
Zone C	20.2	10,268	1.96
Zone C	29.1	13,253	2.19

A total of 6 density determinations were completed on grab samples of the hardcap material, using the water immersion method. Of the 6 hardcap density determinations completed, 2 were taken from Zone A, 2 from Zone B and 2 from Zone B North. The results are provided in Table 8-13 below. The mean density of the hardcap samples is 2.54 g/cm^3.

Table 8-13: The results of the density determinations carried out on grab samples of the hardcap material by the water immersion method.

Zone	Density (g/cm^3)
Zone A	2.44
Zone A	2.58
Zone B	2.55
Zone B	2.56
Zone B North	2.48
Zone B North	2.64

Cora Gold completed a total of 31 density determinations on grab samples of saprolite mineralisation from the base of the artisanal workings, using the water immersion method. The densities derived from these samples are much lower than would normally be anticipated for material of this type, and notably significantly lower that the saprolite densities derived by both the pit excavation method (as outlined in Table 8-12) and from drill core analyses (as outlined in Section 8.5.2). Specifically, the mean density from the results of the saprolite grab samples is 1.62 g/cm^3, with minimum and maximum densities of 1.38 g/cm^3 and 1.83 g/cm^3. It is considered by SRK that these values are most likely to be erroneous; possibly as a result of trapping excessive air around the outside of the samples when weighing the samples in water (the samples were wrapped in cling film before weighing) due to the irregular shape of the grab samples. For this reason, the results of these analyses have been disregarded in the assignment of densities to the estimated block model.

8.5.2 Drill Core Density Determinations

As described in Section 5.4.7, Cora Gold completed density determinations on the diamond drill core at a downhole spacing of approximately 4 m. All samples were analysed by both the water immersion method and a volumetric method. The drill core density determinations are primarily taken from the un-weathered fresh rock, with a smaller number of samples analysed in the saprock and saprolite. No drill core density determinations were carried out of the hardcap material. The mean density values for both methodologies, by weathering state (as per the weathering model described in Section 0) are provided in Table 8-14.

Table 8-14: Mean drillcore density determinations by methodology and weathering state.

Density Analysis Type	Mean Fresh Density (g/cm^3)	Mean Saprock Density (g/cm^3)	Mean Saprolite Density (g/cm^3)
Water Immersion	2.71	2.06	1.90
Volumetric	2.73	2.18	2.09

A comparison of the water immersion and volumetric density values is provided in Figure 8-18. Here, the coloured by weathering state (blue = fresh, red = saprock, orange = saprolite). Both Figure 8-18 and Table 8-14 indicate that the water immersion method returns similar (albeit marginally lower) density values to the volumetric method for fresh rock samples, but markedly lower density values for saprock and saprolite samples. It is considered that this maybe a result of air being trapped around the outside of the drillcore when wrapping the saprolite and saprock samples in clingfilm prior to weighing in water. The consistency between the density of the saprolite measured by volumetric determinations on the drill core, and the pit excavation method (outlined in Table 8-12) supports this.

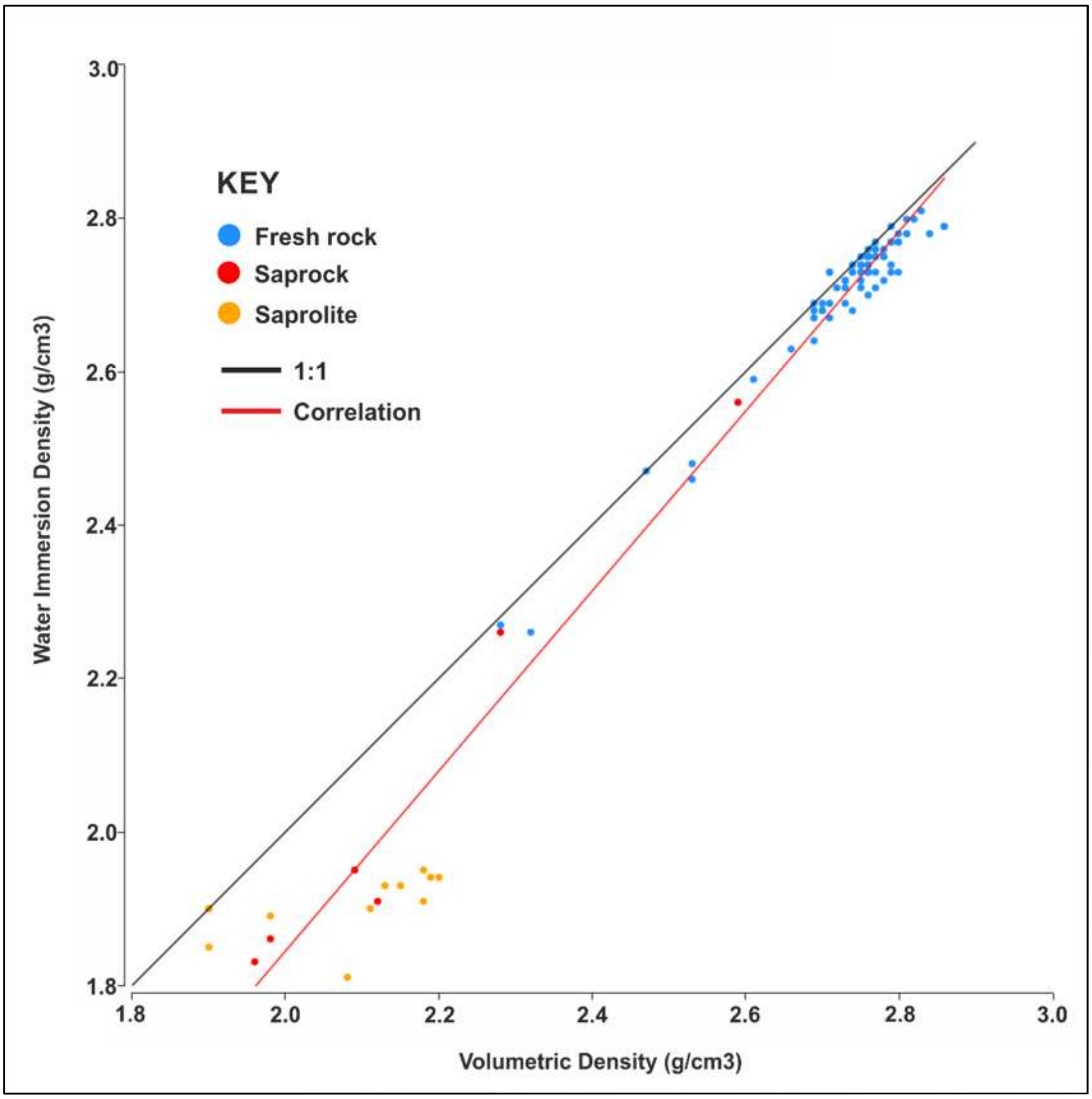


Figure 8-18: Scatterplot of volumetric drill core density analyses against water immersion drill core density analyses.

8.5.3 Block Model Density Values

Based on the results of the density analyses described in Sections 8.5.1 and 8.5.2, the estimated block model was assigned density values based on the following rationale:

- Fresh Rock – Assigned the mean density (rounded to the nearest 0.05 g/cm^3) of the fresh drill core samples analysed using the volumetric method.
- Saprolite and Saprock – Assigned the combined mean density of the saprolite and saprock drill core samples analysed using the volumetric method (rounded to the nearest 0.05 g/cm^3). This is also comparable to the mean density of saprolite density determinations taken from the pit excavation method (as outlined in Table 8-12). The mean density values determined for the saprock and saprolite are relatively similar, and at this stage it is considered that there is not a sufficient number of density determinations within the saprock layer to accurately apply a separate density value for this zone.
- Hardcap – Assigned the mean density value (rounded to the nearest 0.05 g/cm^3) of the

field grab samples taken from the hardcap, since at this stage this is the only data source available for the density of this material.

SRK completed an analysis of the difference in density between mineralised and un-mineralised rock (based on a 0.25 ppm Au cut-off) on the drill core density determinations. The difference in mean density between the mineralised and un-mineralised samples is minimal, and so, at this stage, it was decided to apply the same density values to the mineralisation and waste blocks, with weathering state being the only differentiator in the assignment of density values to the block model. This may change with the addition of more data as the project progresses.

The density values assigned to the block model are outlined in Table 8-15.

Table 8-15: Density values assigned to the Sanankoro weathering domains.

Weathering State	Assigned Density (g/cm^3)
Fresh	2.75
Saprock + Saprolite	2.15
Hardcap	2.55

8.6 Mineral Resource Classification

Block model quantities and grade estimates for the Sanankoro deposit were classified in accordance with the JORC Code (2012).

Mineral Resource classification is typically a subjective concept. Industry best practices suggest that resource classification should consider both the confidence in the geological continuity of the mineralised structures, the quality and quantity of exploration data supporting the estimates and the geostatistical confidence in the tonnage and grade estimates. SRK's approach to classification criteria aims to integrate both concepts to delineate regular areas at similar resource classification.

SRK have not completed a Competent Persons site visit to the Sanankoro Project. No independent checks on the logging, sampling or drill protocols put in place by Cora Gold have been completed by SRK. Dr Jonathan Forster, CEO and Head of Exploration for Cora Gold Ltd, acts as the Competent Person responsible for the geology, drilling and exploration protocols employed on site. That said, based on the information provided by Dr Jonathan Forster and Cora Gold on the drilling, sampling and sample analysis protocols employed during the Cora Gold drill campaigns, SRK considers that these are acceptable for the reporting of a Mineral Resource Estimate in line with the JORC Code (2012).

Cora Gold has put in place a logical system of QA/QC checks including the insertion of blanks, duplicates, standards and the use of an umpire lab. Validation checks of the blanks and duplicates are broadly within acceptable reporting limits. SRK and Cora Gold have identified that the standards used for the assessment of the bottle roll analyses return spurious and erratic results. This is discussed in detail in Section 0. Whilst it is recommended that Cora Gold attempt to urgently rectify this issue for future drill campaigns, SRK is sufficiently satisfied that the spurious results of the bottle roll standards are a function of the method used to create the standard samples, rather than indicating any serious fault in the accuracy of the analytical equipment at the SGS lab, or contamination of samples during sample preparation. This is substantiated by the satisfactory results of the blanks, duplicate and umpire lab analyses.

Validation of the historic Randgold and Gold Fields drillhole data against the Cora Gold drilling, indicates that the inclusion of this data in the mineralisation models and grade interpolation is unlikely to introduce any significant bias into the Mineral Resource Estimate, either in the grade of the estimated blocks or the volume of the mineralised envelopes. It is noted that approximately 25 % and 10 % of the total length of assay samples inside of the mineralisation wireframes are from AC and RAB holes respectively. SRK would recommend that Cora Gold focus on increasing the coverage of RC and DD drilling in future drill campaigns.

SRK consider that the predictability of the position and continuity of the main mineralised structures is high, given the current drillhole spacing. The orientation of the mineralised structures appears to be very consistent over large distances along-strike, and the trend of the mineralisation is well predicted by the induced polarization (IP) geophysical anomalies. For the most part, the location of mineralisation intersected by the most recent drill campaign either overlaps with, or is within a few meters of a previous iteration of the Sanankoro mineralisation wireframes (Figure 8-19), used to assist in deriving an Exploration Target for the Sanankoro Project in October 2018 (SRK UK, 2018). That said, there are relatively significant variations in thickness of the mineralised zones over quite short distances both down-dip and along-strike.

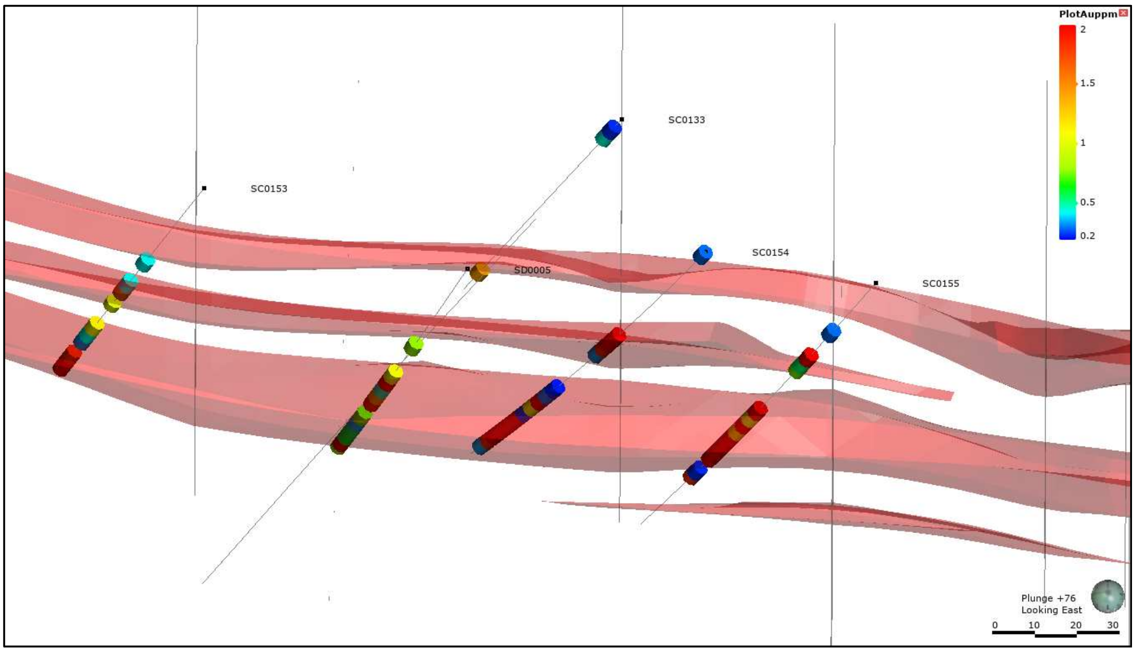


Figure 8-19: 3D view (76 degrees towards the east) of a portion of Zone A, showing downhole assays from the 2019 Cora Gold drilling, filtered above 0.2 g/t Au, alongside the mineralisation wireframes used to assist in deriving the October 2018 Exploration Target. Holes used in the derivation of the October 2018 model are displayed as grey traces.

Across Selin, Zone B, Zone B North and most notably Zone A there are identifiable higher and lower grade zones, that are based on multiple adjacent drillholes of reasonably consistent grade. Grade variations are more distinct along-strike than down-dip, indicating a structural control on the distribution of high grade zones, possibly relating to the intersection of the N-S trending mineralised structures, with cross-cutting mineralised veins. Notably, the distribution of higher and lower grade zones in the largest mineralisation domain at Zone A (Zone A 2) suggests a relatively consistent and predictable shallow dipping plunge to the north.

Given the above, SRK consider that, for the most part, the current drill coverage at Zone A, Zone B, Zone B North and Selin is appropriate for the reporting of an Inferred Mineral Resource. This excludes Selin 5 and Zone B 2, which, at this stage, are characterised by a low confidence in the geological interpretation and have too few drillhole intersections to justify the classification of Inferred Resources. Other than these two domains, all estimated blocks inside of the mineralisation domains across each of Zone A, Zone B, Zone B North and Selin, have been classified as Inferred to an approximate depth of 50 m down-dip beyond the deepest drillhole intersection on each drillhole section.

8.7 Mining Depletion

The estimated block model has been depleted to account for the artisanal mining activity described in Section 0. This was completed by delineating broad outlines of the extent of artisanal workings, as interpreted from Google Earth satellite imagery. Within these outlines, all blocks inside of the mineralisation domains to a depth of 15 m below surface were coded as depleted, based on an approximation by Cora Gold of the average maximum depth of the artisanal pits. Waste blocks were not depleted. This is considered a conservative approach, since the outlines used to deplete the block model cover all areas in which any amount of artisanal activity is evident. It is thought unlikely that all mineralisation within these outlines has been extracted to a depth of 15 m.

8.8 Pit Shell Optimisation

In order to determine the quantities of material offering “reasonable prospects for economic extraction”, SRK completed pit optimisation study based on reasonable, but optimistic, economic and mining assumptions to evaluate the proportions of the block model that could “reasonably expected” to be mined from an open pit.

Prior to optimisation, the block model was regularised to a block size of 2.5 m * 2.5 m * 5m, based on an assumption of reasonable minimum mining block dimensions, in order to account for mining recovery and dilution.

The optimisation was completed in NPV Scheduler (“NPVS”) software. NPVS uses the Lerchs-Grossmann algorithm for determining the shape of an optimal pit using a set of techno-economical input parameters. These mining assumptions and parameters are outlined in Table 8-16.

Table 8-16: Parameters applied in the generation of optimised pit shells for the Sanankoro resource.

Parameter	Units	Value
Production		
Production Rate	*tonnes per annum (tpa)*	1,000,000
Geotechnical		
Slope Angle - Saprolite and Hardcap	*Degrees*	34
Slope Angle - Saprock	*Degrees*	40
Slope Angle - Fresh rock	*Degrees*	42
Mining Factors		
Dilution	*Regularised block model (2.5 * 2.5 * 5 m) - no flat dilution rate*	
Recovery	*Regularised block model (2.5 * 2.5 * 5 m) - no flat dilution rate*	
Processing Recovery		
Hardcap - all zones	*%*	80.0
Saprolite + Saprock - Zone A and Zone B	*%*	95.7
Saprolite + Saprock - Zone B North and Selin	*%*	92.9
Fresh rock - all zones	*%*	80.0
Operating Costs		
Base Mining Cost		
Saprolite and Hardcap Ore	*USD / t*	3.5
Saprock and Fresh Ore	*USD / t*	4.0
Saprolite and Hardcap Waste	*USD / t*	3.0
Saprock and Fresh Waste	*USD / t*	3.5
Other Operating Costs		
Processing Cost - Oxide	*USD / t ore*	15.5
Processing Cost - Sulphide	*USD / t ore*	17.0
G&A	*USD / t ore*	2.0
Selling Cost	*%*	5
Metal Price		
Au	*USD / oz*	1,700*
Other		
Discount Rate	*%*	10

** Optimistic long-term gold price requested by Cora Gold*

The resulting pit shells for Zone A, Zone B, Zone B North and Selin are displayed in Figure 8-20 to Figure 8-23. For Zone A, the resource is constrained within a single pit, whilst the optimisation process for Zone B, Zone B North and Selin has produced multiple smaller pits that are disconnected along-strike. At surface, the total strike length of the optimised pit shells is approximately 0.9 Km at Zone A, 1.1 km at Zone B, 0.8 Km at Zone B North and 1.95 Km at Selin. The total 4.75 km surface strike length of the pit shells, represents 65% of the approximate 7.35 km surface strike extent of the modelled mineralised zones. The maximum depth of the pit shells is 130 m at Zone A, 130 m at Zone B, 50 m at Zone B North and 115 m at Selin.

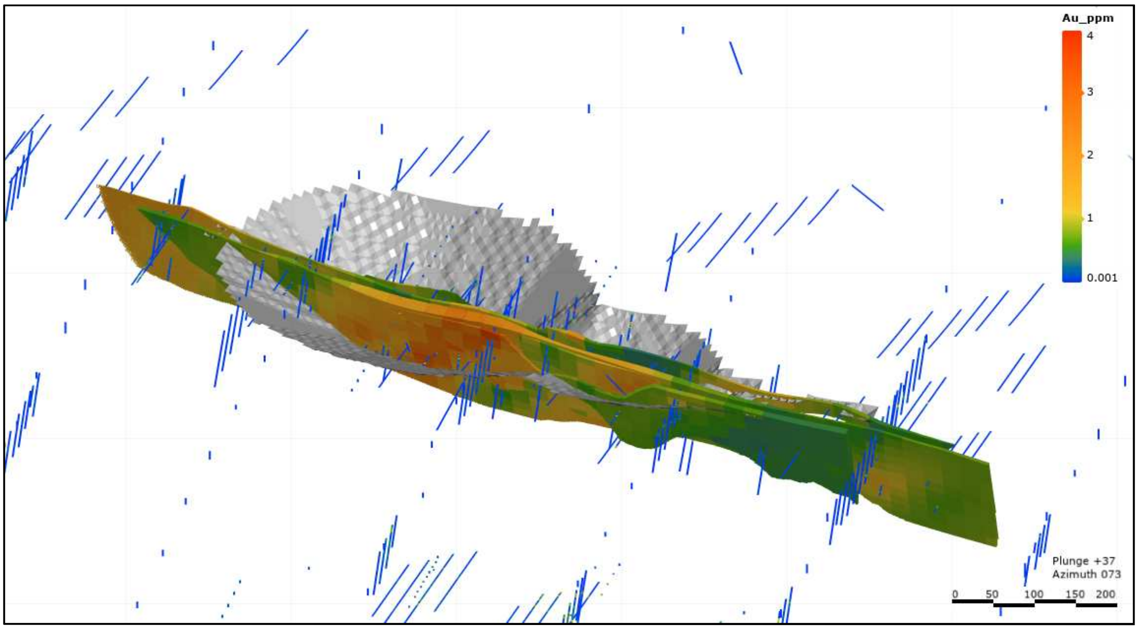


Figure 8-20: An inclined view (37° towards 073°) of the Inferred estimated Zone A block model shown alongside the 1,700 USD/oz optimised pit shell.

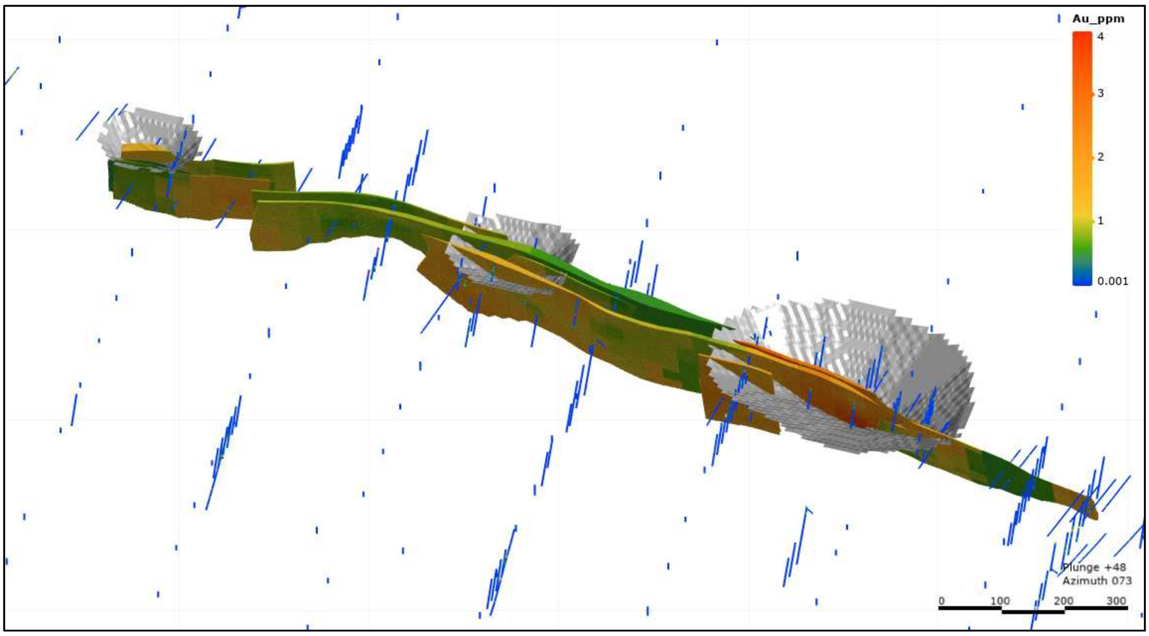


Figure 8-21: An inclined view (48° towards 073°) of the Inferred estimated Zone B block model shown alongside the 1,700 USD/oz optimised pit shell.

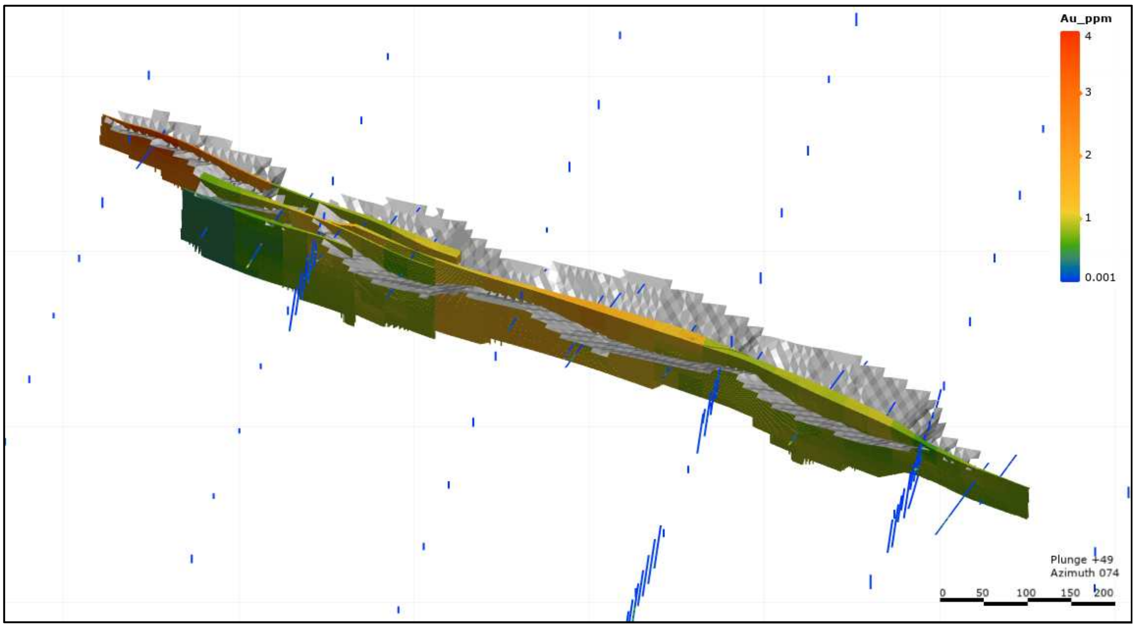


Figure 8-22: An inclined view (49° towards 074°) of the Inferred estimated Zone B North block model shown alongside the 1,700 USD/oz optimised pit shell.

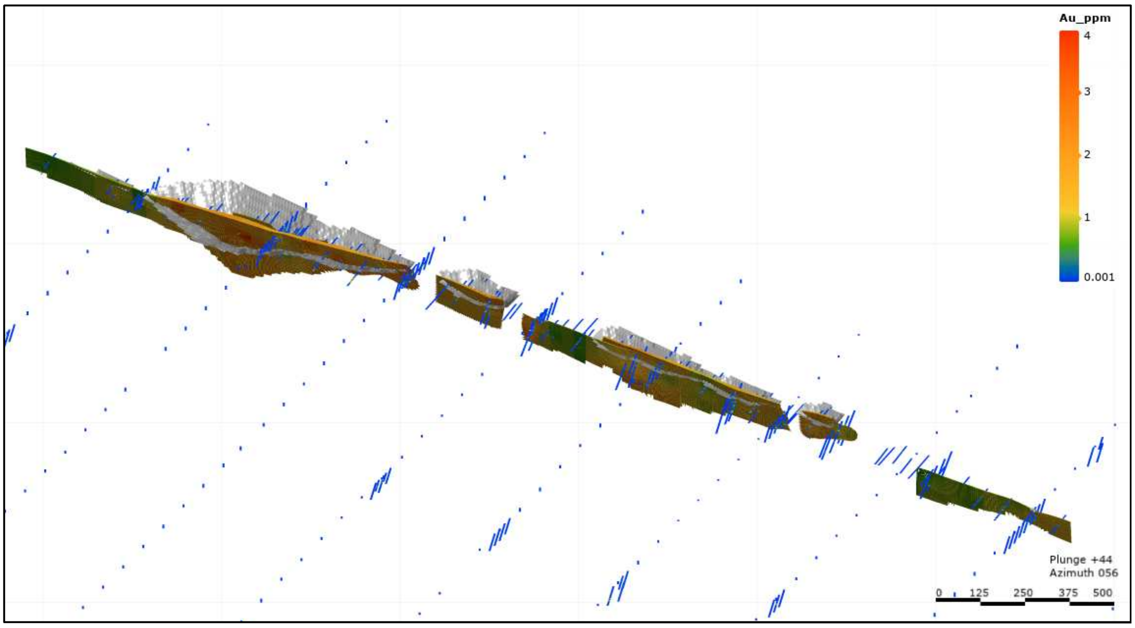


Figure 8-23: An inclined view (44° towards 056°) of the Inferred estimated Selin block model shown alongside the 1,700 USD/oz optimised pit shell.

8.9 Mineral Resource Statement

The Mineral Resource Statement generated by SRK has been restricted to all classified material falling within the optimised pit shells representing a metal price of 1,700 USD / oz and through the application of the parameters outlined in Section 8.8. Additionally, the Mineral Resource is reported above a marginal cut-off grade of 0.4 g/t for all oxide blocks (hardcap, saprolite and saprock) and 0.5 g/t for the sulphide blocks. This represents the material which SRK considers has reasonable prospects for eventual economic extraction.

Table 8-17 shows the resulting Mineral Resource Statement for the Sanankoro Project. In total, SRK has estimated an Inferred Mineral Resource of 5.0 Mt grading at 1.6 g/t Au. This includes 4.5 Mt of oxide mineralisation at 1.6 g/t Au and 0.5 Mt of sulphide mineralisation at 1.8 g/t Au.

The Mineral Resource Statement presented herein has been classified by Mr. Martin Pittuck, who is a Corporate Consultant (Mining Geology) of SRK UK, a Member of the Institute of Materials, Minerals and Mining (MIMM), a Fellow of the Geological Society of London (FGS) and a Chartered Engineer, UK (CEng). Mr Pittuck is responsible for the preparation of the Mineral Resource Estimate and takes overall responsibility for the resource estimation work and resulting Mineral Resource Statement.

SRK UK have not completed a Competent Persons site visit to the Sanankoro Project. Dr. Jonathan Forster, CEO and Head of Exploration for Cora Gold Ltd, acts as the Competent Person responsible for the geology, drilling and exploration protocols employed on site.

Both Mr Pittuck and Dr. Forster have sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration and to the activity which they are undertaking to qualify as a Competent Person as defined in the 2012 Edition of the 'Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves'. Both Mr Pittuck and Dr Forster consent to the inclusion in this announcement of the matters based on their information in the form and context in which it appears.

Mineral Resources that are not Mineral Reserves have no demonstrated economic viability. SRK are not aware of any factors (environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors) that have materially affected the Mineral Resource Estimate. It is uncertain is further exploration will convert Inferred Mineral Resources to higher confidence categories.

Table 8-17: Mineral Resource Statement for the Sanankoro Project, as of 5 December 2019.

Weathering State	Resource Classification	Tonnes (Mt)	Au g/t	Contained Au (Oz)
OXIDE	MEASURED	-	-	-
	INDICATED	-	-	--
	INFERRED	4.5	1.6	233,000
	TOTAL	**4.5**	**1.6**	**233,000**
SULPHIDE	MEASURED	-	-	-
	INDICATED	-	-	--
	INFERRED	0.5	1.8	32,000
	TOTAL	**0.5**	**1.8**	**32,000**
OXIDE + SULPHIDE	MEASURED	-	-	-
	INDICATED	-	-	--
	INFERRED	5.0	1.6	265,000
	TOTAL	**5.0**	**1.6**	**265,000**

Notes:

(1) The Inferred Mineral Resource Estimate is reported above a cut-off grade of 0.4 g/t for oxide material and 0.5 g/t for sulphide.
(2) The Mineral Resource Estimate for the Sanankoro deposit was constrained within grade based solids and within a Lerchs-Grossman optimised pit shell based on a gold price of 1,700 USD / oz and through the application of reasonable mining parameters.
(3) All figures are rounded to reflect the relative accuracy of the estimate.
(4) Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.
(5) It is uncertain is further exploration will convert Inferred Mineral Resources to higher confidence categories.

The Mineral Resource is delineated by zone, and by the weathering profile in Table 8-18 and Table 8-19 respectively.

Table 8-18: Mineral Resources by Zone.

Zone	Tonnes (Mt)	Au g/t	Contained Au (Oz)
Selin	1.9	1.8	108,000
Zone A	1.9	1.5	91,000
Zone B	0.7	2.0	47,000
Zone B North	0.5	1.1	19,000
TOTAL	**5.0**	**1.6**	**265,000**

Table 8-19: Mineral Resources by Weathering Profile Domain.

Zone	Tonnes (Mt)	Au g/t	Contained Au (Oz)
Hardcap	0.4	1.3	16,000
Saprolite	3.7	1.6	191,000
Saprock	0.4	1.9	27,000
Fresh	0.5	1.8	32,000
TOTAL	**5.0**	**1.6**	**265,000**

8.10 Grade Sensitivity Analysis

The Mineral Resource Estimate for the Sanankoro Project is sensitive to the selection of the reporting cut-off grade. To illustrate this sensitivity, the model quantities and grade estimates for the combined oxide and sulphide Inferred Resources are presented in Table 8-20 at cut-off grade increments of 0.1 g/t Au, up to a maximum cut-off of 3.4 g/t. Figure 8-24 presents the sensitivity of the estimate as a grade tonnage curve. The reader is cautioned that the figures presented in Table 8-20 and Figure 8-24 should not be misconstrued as a Mineral Resource Statement. The figures are only presented to show the sensitivity of the block model estimates to the selection of the Au cut-off value. For reference, the cut-off grade selected for the reporting of the Sanankoro Mineral Resource Statement outlined in Table 8-17 is 0.4 g/t for all oxide blocks (hardcap, saprolite and saprock) and 0.5 g/t for the sulphide blocks.

Table 8-20: Sanankoro Inferred block model tonnage and grades inside the optimised pit shell at various Au g/t cut-off grades.

Cut-Off Grade (Au g/t)	Tonnage (Mt)	Grade (Au g/t)
0	5.04	1.64
0.2	5.04	1.64
0.4	5.03	1.64
0.6	4.96	1.66
0.8	4.59	1.73
1	4.02	1.85
1.2	3.44	1.97
1.4	2.83	2.12
1.6	2.36	2.24
1.8	1.77	2.43
2	1.32	2.61
2.2	1.02	2.76
2.4	0.79	2.89
2.6	0.58	3.03
2.8	0.33	3.28
3	0.22	3.48
3.2	0.15	3.66
3.4	0.11	3.78

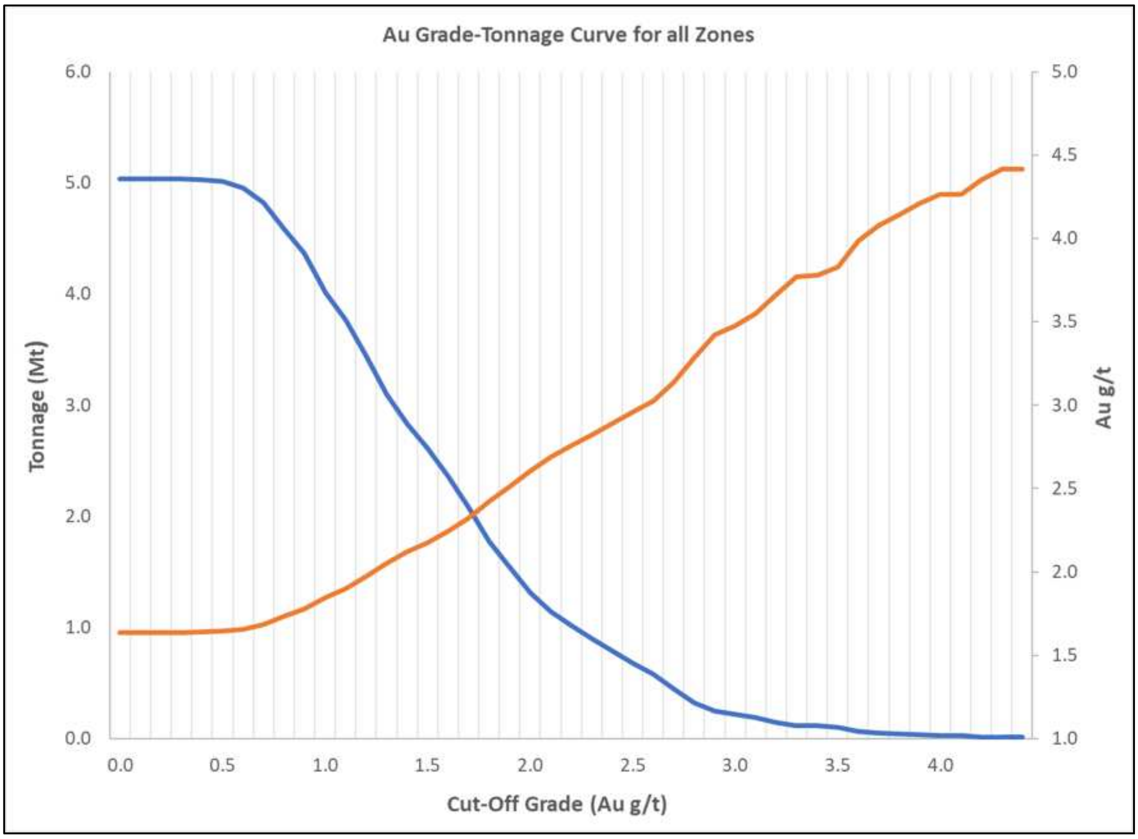


Figure 8-24: Sanankoro grade tonnage curve inside the optimised pit shell.

8.11 Comparison to Previous Resource Estimates

SRK is unaware of any previous Mineral Resource Estimates completed on the Sanankoro Project.

8.12 Exploration Target

In October 2018, SRK derived an Exploration Target for the Sanankoro Project, based on the following:

- Volumetric modelling and grade interpolation of mineralisation at Zone A, Zone B, Zone B North and Selin, in addition to two other zones, namely Zone C and Selin South, altogether representing a total strike length of ~11km. The volumetric modelling was limited to a depth of 100 m below surface.
- An additional strike length of ~33km of 2D map-lines representing positive exploration indications which may comprise mineralisation with thickness and grade similar to the modelled volumes.

A more detailed description of how the Exploration Target was derived can be found in the 2018 SRK UK Exploration Target report (SRK UK, 2018). The location and extent of the volumetric models and map lines used to derive the October 2018 Exploration Target, in relation to the optimised pit shells used to constrain the Inferred Mineral Resource presented in Table 8-17, are outlined in Figure 8-25 and Figure 8-26.

SRK is unaware of any new information which materially impacts on the assumptions upon which the Exploration Target is based. For this reason, an unchanged Exploration Target for the Sanankoro Project of **between 30 Mt and 50 Mt at a grade of between 1.0 and 1.3 g/t Au** is re-stated here.

For the avoidance of doubt, in respect to the Exploration Target, SRK notes:

- The potential quantity and grade as reported in respect of the Exploration Targets are conceptual in nature;
- There has been insufficient exploration to define a Mineral Resource; and
- It is uncertain if further exploration (as planned by the Company) will result in the determination of a Mineral Resource.

The <5 km total strike extent of the optimised pit shells used to constrain the Sanankoro Inferred Mineral Resource represents <15% of the total linear strike length of potential mineralised zones upon which the Exploration Target is based. It is noted that, of the approximate 1 – 2 million ounce Exploration Target range, approximately 700,000 ounces of gold are defined in the block model from which the 265,000 ounce Inferred Mineral Resource is derived (being inside the optimised pit and above cut-off grade).

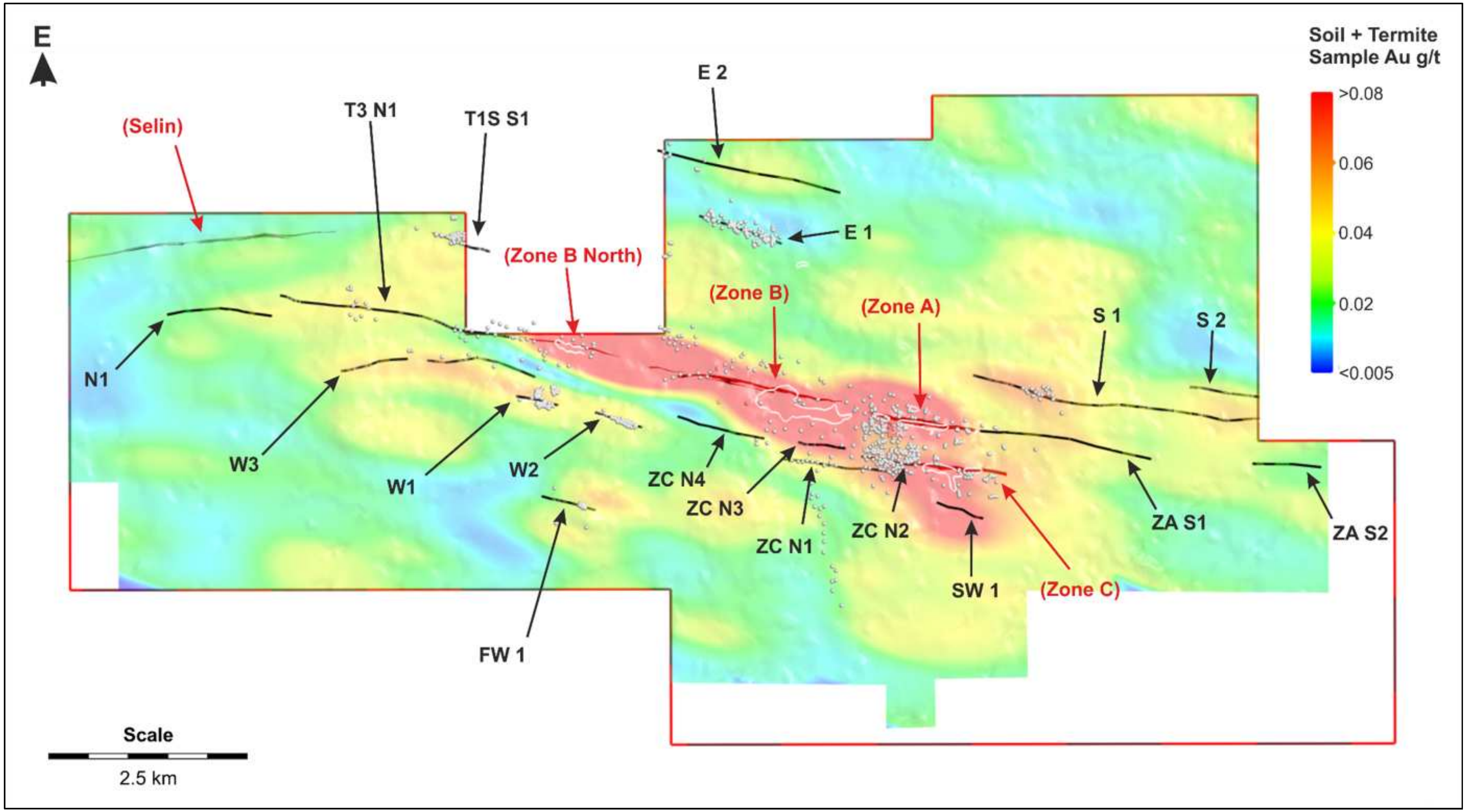


Figure 8-25: East-facing map of the mineralisation map-lines (in black) in the Sanankoro Permit and modelled mineralisation wireframes (in red), relative to the soil and termite sample grade trends and artisanal excavations (as white points and white outline strings).

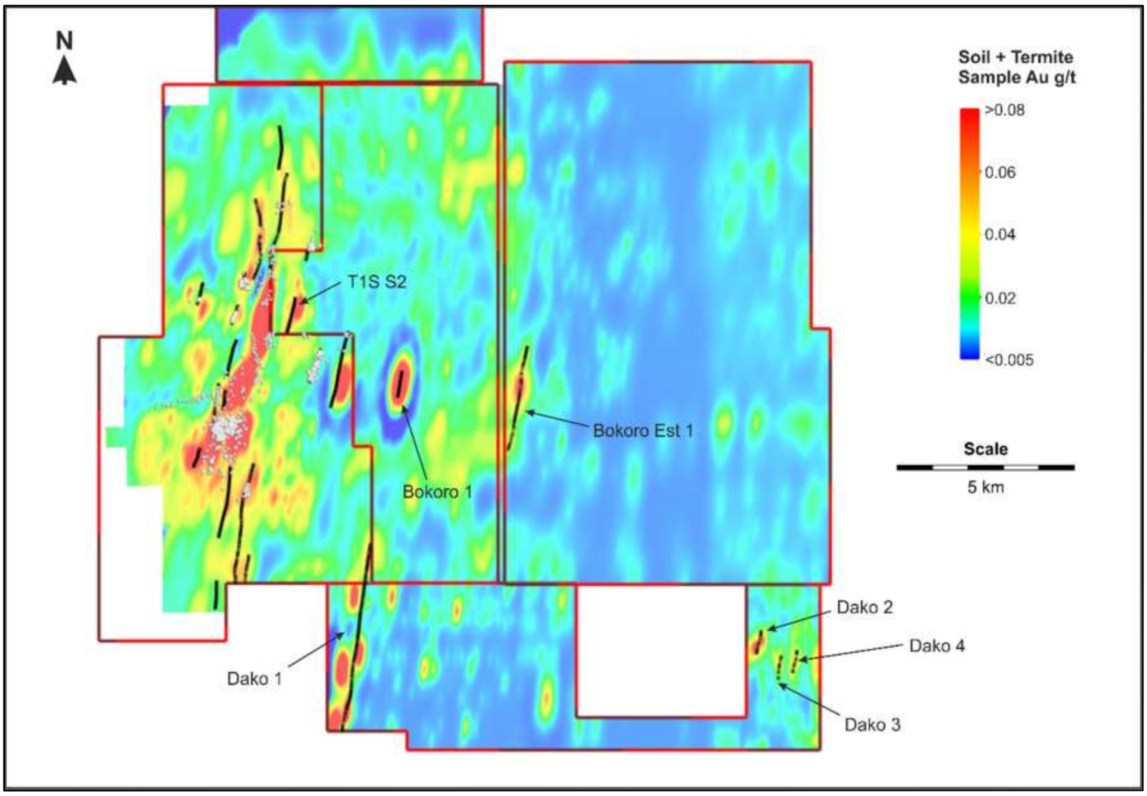


Figure 8-26: Sanankoro Project mineralisation map-lines (in black), relative to soil and termite sample grade trends and artisanal excavations (as white points and outline strings). Only prospect areas outside of the Sanankoro Licence are labelled.

9 CONCLUSIONS

The aim of the study presented here was to produce a Maiden Mineral Resource Estimate for the Sanankoro Project, based on the results of recent exploration and drilling by Cora Gold and historic drilling completed by Gold Fields and Randgold between the mid-2000’s and 2012.

In producing the updated Mineral Resource Estimate presented herein, SRK has:

- Reviewed the exploration database provided by Cora Gold, including both the recent drilling completed by Cora Gold, and the historic drilling completed by both Gold Fields and Randgold;
- Discussed the interpretation of the structural and geological controls on gold mineralisation at the Sanankoro Project with the Gora Gold geology team;
- Developed volumetric wireframe models for the Sanankoro mineralisation and weathering profile on the basis of the data provided by Cora Gold and the present interpretation of the controls on gold mineralisation;
- Undertaken statistical and geostatistical analyses of the assay and density data obtained during the various exploration campaigns;
- Interpolated the above data into 3D block models, tagged and sub-blocked by the mineralisation and weathering wireframes;
- Classified the block model in accordance with the JORC (2012) reporting code;
- Completed a pit optimisation exercise on the estimated block model, based on reasonable, but optimistic, economic and mining assumptions to evaluate the proportions of the block model that have “reasonable prospects for economic extraction” an open pit mine.
- Reported a Maiden Mineral Resource according to the guidelines for such set out in the JORC Code.

The Inferred Mineral Resource estimate derived by SRK as described above and as of 5 December 2019 is 5.0Mt with a mean grade of 1.6 g/t Au, which includes 4.5 Mt of oxide mineralisation at 1.6 g/t Au and 0.5 Mt of sulphide mineralisation at 1.8 g/t Au.

The Mineral Resource Statement has been classified by Mr. Martin Pittuck, who is a Corporate Consultant (Mining Geology) of SRK UK. Mr Pittuck is responsible for the preparation of the Mineral Resource Estimate and takes overall responsibility for the resource estimation work and resulting Mineral Resource Statement. SRK UK have not completed a Competent Persons site visit to the Sanankoro Project. Dr. Jonathan Forster, CEO and Head of Exploration for Cora Gold Ltd, acts as the Competent Person responsible for the geology, drilling and exploration protocols employed on site.

SRK has re-stated the existing Exploration Target for the Sanankoro Project of between 30 Mt and 50 Mt at a grade of between 1.0 and 1.3 g/t Au.

10 RECOMMENDATIONS

SRK provides the following general recommendations for future resource work at Sanankoro:

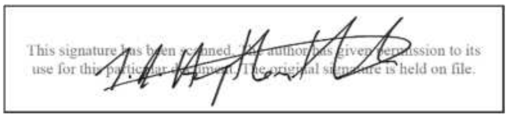
- As noted in Section 6.6.5, the bottle roll standard samples inserted into the sample stream by Cora Gold as part of the assay QAQC programme perform very poorly, with erratic results that have little correlation with the calculated expected sample grades. It is considered that this is most likely a function of the method used to create the standard samples, namely poor homogenisation of the gold pills with the blank material prior to assaying. SRK would recommend that all future bottle roll standards are created by mixing larger CRM samples with blank material, rather than 1 g gold pills. Specifically, it is recommended that multiple CRM samples of the same type are mixed with blank material for each bottle roll standard, in order to increase the proportion of Au bearing sample in the standard sample. The grade of the CRM’s used should be selected to produce standards with a grade range similar to that observed at Sanankoro.
- Cora Gold should start to conduct field duplicate analyses on diamond drill core, as a routine part of the QAQC programme, similar to the field duplicate analyses already carried out on the RC chips.
- It is recommended that Cora Gold continue with the approach of completing density determinations every 4 m on the diamond drill core. Density should continue to be calculated for each sample using both the water immersion and volumetric methods outlined in 5.4.7, in order to better understand and quantify the difference in results between the two methods and determine the most appropriate system moving forwards.
- There is currently very limited information relating to the density of the hardcap material (a total of 6 grab samples). It is recommended that a more detailed density programme is completed on the hardcap material, with care taken to ensure that the samples selected for density determinations are representative of the variation in material characteristics throughout the hardcap layer
- It is recommended that Cora Gold continue and add to the limited programme of pit excavation density calculations (as described in Section 5.3.1) to supplement and validate the density determinations completed on the saprolite and saprock drill core
- SRK consider that the main potential for the delineation of additional Mineral Resources is from the potential mineralised structures outside of the zones that have been the primary focus of drilling to date. SRK would recommend that a key focus of the next phase of exploration would be drilling of the best understood and most prospective exploration target zones.
- Additionally, it is recommended that Cora Gold conduct systematic trenching / channel sampling or grab sampling across the less informed 2D target map-lines used to define the Exploration Target to provide an un-biased indication of the grade of the veins at surface and to assist in determining priority drilling targets; if this might attract artisanal activity, alternatively consider RAB line coverage in these areas.
- Within Zone A, Zone B, Zone B North and Selin it is considered that the most prospective zones for the addition of ounces to the Mineral Resource are down-plunge of high grade zones identified in the block model grade estimates (such as the moderate north dipping plunge in Zone A), and following up on isolated mineralised intercepts across strike from the already delineated mineralised structures that may form additional mineralised

structures parallel to the main mineralised zones.

- SRK understand that, to date, Cora Gold have only completed assaying for Au. It is recommended that a wider suite of elements is analysed in future drill campaigns in order to understand the potential concentrations of any penalty or deleterious elements.
- Continue to orientate any diamond holes using a reliable orientation system such as the Reflex$_{TM}$ tool, to allow for the collection of structural data such as vein contacts. This is particularly important given that at least 3 sets of mineralised veins, or differing orientation, are recognised in the Sanankoro Project area.
- Consider twinning several of the existing AC and RAB holes with new RC or diamond holes; potentially allowing more reliance to be placed on historical data.
- Downhole surveying should be completed on all holes that exceed a depth of approximately 50 m. Initially SRK would recommend that downhole surveying is completed at increments of approximately 25 m, although this should be reviewed as future drill programmes progress, dependent on the degree of deviation observed. SRK would recommend that surveying is completed using a standard down-hole survey instrument such as a gyroscopic tool / Tropari.
- Commission a high-resolution topography survey to replace the ASTER digital elevation data employed in defining the topography surface used to limit the vertical extent of the 3D mineralisation models described in Section 8.2.2.
- Complete detailed surveying of the artisanal workings to allow for more accurate depletion of the resource.

For and on behalf of SRK Consulting (UK) Limited

Martin Pittuck,
Corporate Consultant (Mining Geology),
Project Director
SRK Consulting (UK) Limited



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TECHNICAL APPENDIX

A JORC TABLE 1

Section 1 – Sampling Techniques and Data

Criteria	JORC Code Explanation	Project Description
Sampling techniques	*Nature and quality of sampling (e.g. cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down-hole gamma sondes, or handheld XRF instruments, etc.). These examples should not be taken as limiting the broad meaning of sampling.* *Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used.* *Aspects of the determination of mineralisation that are Material to the Public Report. In cases where 'industry standard' work has been done this would be relatively simple (e.g. 'reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases, more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (e.g. submarine nodules) may warrant disclosure of detailed information*	The Sanankoro exploration database includes a combination of results from reverse circulation ("RC"), Air Core ("AC"), Diamond Core ("DC") and Rotary Air Blast Drilling ("RAB"), including historic drilling completed between the mid-, firstly by Randgold and subsequently by Gold Fields, and more recent drilling by Cora Gold between 2017 and 2019. Limited information is available on the sampling procedures put in place by Randgold and Gold Fields. For the Cora Gold drilling campaigns, RC and AC samples were collected each metre from the cyclone, into a 50kg plastic bag. Depending on the results of logging and/or panning, the samples are either composited to a 3 m composite sample or retained as a 1 m sample. The diamond drilling completed by Cora Gold was sampled at intervals determined by the project geologists, being broadly set to regular 1m intervals except were lithological or mineralised contacts were encountered. In addition to the drilling data available, other salient sampling data available for the project includes: - 200 m x 500 m regional termite mound sampling completed by Cora Hold; - 500 x 1000 m regional soil sampling; - 100 x 200 m detailed soil sampling over the central part of the Sanankoro Licence; - 50-100 x 400 m grid infill soil sampling, in the western portion of the Dako Licence; - Randgold 200 m x 500 m regional termite mound sampling; - Detailed 100 m x 200 m termite mound sampling completed in the east of the Dako Licence by Cora Gold; - Randgold 500 x 1000 m regional soil sampling;

Criteria	JORC Code Explanation	Project Description
		- Randgold 100 x 200 m detailed soil sampling over the central part of the Sanankoro Licence; - Gold Fields 50-100 x 400 m grid infill soil sampling, in the western portion of the Dako Licence; - Variably spaced (~10–500 m) termite mound panning data, detailing the number of gold grains observed at each location, over two separate areas of approximately 4 x 5 km in the east of the Sanankoro Licence and 2.5 x 5.5 km in the north of the Sanankoro Licence, completed by Cora Gold.
Drilling techniques	*Drill type (e.g. core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc.) and details (e.g. core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other type, whether core is oriented and if so, by what method, etc.).*	The Sanankoro drillhole database includes a combination of the following drill types: **Historic Randgold and Gold Fields Drilling:** - AC / RC – 1,007 holes for 32,840 m - RAB – 775 holes for 23,700 m - DC tails on AC / RC holes – 16 holes for 910 m **Cora Gold Drilling:** - AC / RC – 236 holes for 21,600 m - RAB – 17 holes for 340 m - DC tails on AC / RC holes – 10 holes for 550 m - DC – 1 hole for 70 m. Limited information is available on the drilling procedures put in place by Randgold and Gold Fields. For the Cora Gold drilling campaigns, Both AC/RC and Diamond drilling was completed by Target Drilling using a multi-purpose KL 900 truck mounted RC/core drill rig with a 350 psi / 1150 cfm compressor and 6m runs. A HQ3 drill core diameter was employed in unconsolidated ground, with HQ core collected in solid,

Criteria	JORC Code Explanation	Project Description
		fresh rock. The HQ3 core was drilled in 1.5 m runs and the HQ core drilled in 3 m runs. Downhole structural orientation has been completed on the diamond core tails of 5 of the Cora Gold holes. Using an ACT III H/H3 survey tool.
Drill sample recovery	*Method of recording and assessing core and chip sample recoveries and results assessed.* *Measures taken to maximise sample recovery and ensure representative nature of the samples* *Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.*	Both total core recovery ("TCR") and solid core recovery ("SCR") are recorded by for all Cora Gold diamond drillholes and 11 of the 16 historic diamond core holes. For the Cora Gold holes, Total core recovery is generally good, with an average recovery of approximately 92%. 100% core recovery is achieved for more than 90% of the geotechnical intervals with roughly 70% of geotechnical intervals having a recorded total core recovery of and greater than 90%. Only a small length of intervals (<4%) have a recovery of <50% and these are mostly smaller intervals with lengths of 1m or less. Core Recovery from the Gold Fields diamond holes is lower than the Cora Gold drilling, at an average of 83%. AC/RC samples are weighed after each run at the rig site. Recovery is generally good, comprising dry sample. In the event of wet samples that extend for greater than a run, the hole is stopped. No reconciliation between theoretical and actual recovery has yet been made.
Logging	*Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies.*	Geological and logging has been completed on all AC, RAB, RC and diamond drilling. Information recorded includes rock type, regolith type, the type and intensity of sulphide mineralisation, the type and intensity of veining, the style and intensity of alteration, colour, and a geological description. Geotechnical logging has been completed on both the historic and Cora Gold

Criteria	JORC Code Explanation	Project Description
		diamond tails. This includes the recording of core recovery data in addition to rock strength, the number of joints in 30° alpha angle buckets, joint roughness, infill mineralogy, the pervasiveness of alteration associated with jointing and the calculation of rock quality designation (“RQD”) from solid core recovery. Structural geotechnical logging has been completed on 3 of the structurally oriented diamond tails. This includes the recording of alpha and beta orientation data for individual joints, with associated micro roughness, infill type and infill thickness for each joint.
	Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc.) photography.	Both qualitative (geological logging codes) and quantitative (geotechnical parameters, quartz vein percentages, alteration mineral percentage estimations etc.) logging has been undertaken.
	The total length and percentage of the relevant intersections logged.	100% of the Cora Gold diamond holes and most of the historic drilling has been geologically logged.
Sub-sampling techniques and sample preparation	*If core, whether cut or sawn and whether quarter, half or all core taken.*	For the Cora Gold drilling, the core was split using a diamond core saw. In order to preserve the orientation lines for further structural measurements, the core is split vertically down the core axis normal to the foliation/bedding to produce two identically sized sections of half core. No information is available on diamond core sampling techniques for the historic drilling.
	If non-core, whether riffled, tube sampled, rotary split, etc. and whether sampled wet or dry.	For the Cora Gold RC and AC holes, a Rifle Splitter was used to homogenise and split the material collected at each metre (the material is passed through the splitter twice), with approximately 1/8 of the material taken for sampling. Depending on the results of logging and/or panning, the 1/8 samples are either composited to a 3 m composite sample or retained as a 1 m sample. To prepare samples for shipment to the analytical laboratory, the final 1/8 samples are

Criteria	JORC Code Explanation	Project Description
		homogenised further, by passing through a Gilson Porta Splitter (model SP2). For the 3 m composite samples, each 1 m sample is passed through the Gilson splitter and split into two, with one half stored as a field duplicate. The 3 m composite sample is then passed through the splitter again to homogenise it. For the 1 m samples, the sample is passed through the Gilson splitter, with one half taken for sampling and the other stored as a field duplicate. The final sample size was 1kg samples for 50g FA or 4Kg for Bottle Roll analysis. No information is available on chip sampling techniques for the historic drilling.
	For all sample types, the nature, quality and appropriateness of the sample preparation technique.	The preparation techniques are considered appropriate for the style of mineralisation.
	Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples. *Measures taken to ensure that the sampling is representative of the in-situ material collected, including for instance results for field duplicate/second-half sampling.*	For chip sampling, Cora Gold insert field duplicates into the sample stream at an insertion rate of 1 in 20 (5%), showing good correlation between duplicate assay analyses. No information is available on duplicate analyses for the historic drilling.
	Whether sample sizes are appropriate to the grain size of the material being sampled.	Samples are considered to be appropriate for the lithological contacts and mineralisation grain size. Cora Gold switched to bottle roll analyses for both chip and core drilling during the 2017-18 exploration campaign, to account for the nuggety nature of gold mineralisation in saprolitic material.

Criteria	JORC Code Explanation	Project Description
Quality of assay data and laboratory tests	*The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total.*	At the outset of Cora Gold's exploration programme in Q4 2017/Q1 2018 at Sanankoro, the oxide samples from the first round of drilling were sent to the SGS laboratory in Bamako, Mali for 50 g fire assay. This has remained the case for sulphide drill samples. During the first phase of drilling by Cora Gold, panning of the drill sample at the rig side confirmed that coarse gold (>100 micron) was a regular feature of the mineralisation style. It was therefore decided to preferentially use cyanide bottle roll as the analytical technique for determining the gold content for all subsequent oxide drill samples. For fire assay, the samples received at the analytical lab are crushed with 75% < 2mm followed by pulverisation with 85% < 75 micron. Once pulverised, the sample is split and a pulp sample is collected for Au 50g FA analysis and the reject stored. For bottle roll, a 4 kg (2 kg during earlier phases of drilling) dry sample is then crushed with 75% < 2 mm followed by pulverisation with 85% < 75 micron. Once pulverised, the 4 kg sample is split into a 2 kg sub sample and the reject stored. Bottle roll analysis is completed by AAS with a detection limit of 0.01 ppm. Where sample analysis is 0.5 ppm or better, the residue from the settled bottle roll is collected and analysed by 50 g fire assay, to enable a total gold assay of the sample to be calculated. All bottle roll analyses are completed at SGS Ouagadougou whilst all fire assay analyses are completed at SGS Bamako. No information is available on analytical techniques used for the historic Randgold and Gold Fields drilling.
	For geophysical tools, spectrometers, handheld XRF instruments, etc., the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their derivation, etc.	No downhole geophysical or XRF data collection has been completed.

Criteria	JORC Code Explanation	Project Description
	Nature of quality control procedures adopted (e.g. standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (i.e. lack of bias) and precision have been established.	For the Cora Gold drilling, the following QAQC procedure have been upheld: - Blanks were inserted at a frequency of 1 in 20 samples (5%). - Insertion of field duplicates, commencing part way through the Cora Gold drilling at a frequency of 1 in 20 (5%). Average frequency across all Cora Gold drill phases is equivalent to approximately 2.5%. - Repeat assays conducted on the pulverised RC rejects for bottle roll analyses, at an equivalent insertion rate of approximately 3.5%. - A combination of various standard samples analysed at a frequency of approximately 1 in 20 (5%). Standard samples include CRM's for fire assay QAQC, and larger bespoke standards for bottle roll QAQC. - A small number of duplicate check assays completed at ALS Shannon in Ireland. The results of the QAQC analyses undertaken by Cora Gold do not indicate any serious issues in the sample assays. Although the standards used for bottle roll analysis perform very poorly, this is considered to be most likely a result of the method used to prepare these samples, rather than indicating any fatal flaw in the analytical equipment. No information is available on QAQC analysis for the historic drilling.
Verification of sampling and assaying	*The verification of significant intersections by either independent or alternative company personnel.*	SRK UK has not visited the project site or completed any independent check sampling of material from the project.
	The use of twinned holes.	No twinned drilling has been undertaken.
	Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols.	Data is collected in the field on paper log sheets which are stored in files, with all data transferred into excel spreadsheets. This is reviewed by the geologist at the

Criteria	JORC Code Explanation	Project Description
		site prior to forwarding to the database manager based in the UK. The data is verified with queries returned to the field where necessary, prior to being saved into a project specific Access database in the UK where standard back up procedures are maintained. Drillhole data was provided to SRK in a series of Excel spreadsheets, with separate files for each drillhole data type (collar, survey, assay etc.) and separate files for both the Cora Gold and historic drill campaigns. SRK has not completed any verification of the data storage or input procedures.
	Discuss any adjustment to assay data.	To prevent the smoothing of higher grades in un-sampled intervals, SRK has replaced all absent or negative Au assay values with a low grade background value of 0.001 ppm.
Location of data points	*Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation.*	All Cora drillhole collars were surveyed with standard hand-held GPS equipment and later confirmed by a contractor, who re-surveyed the collars with Differential Global Positioning System “DGPS”. The method used to spatially survey the location of the historic drillhole collars is unclear. Downhole surveying has not been completed on most of the Cora drillholes. Specifically, downhole surveying is restricted to a total of 45 of the 264 Cora Gold holes, limited to the diamond core drilling and some of the deeper reverse circulation holes. For these holes downhole surveying is generally completed at 40 – 60 m increments, using a Reflex EZ-TRAC downhole survey tool. The reverse circulation, air core and rotary air blast hole that are not downhole surveyed range in length from 9 m to 140 m, with an average length of 80 m. For these holes, the hole dip and azimuth were derived from the measured hole dip and azimuth taken at the drillhole collars. For those holes on which downhole surveying has been completed, the downhole deviation in hole dip and azimuth is generally considered minimal, and that visual assessment of the 3D location of mineralised intercepts

Criteria	JORC Code Explanation	Project Description
		in un-surveyed holes typically indicates a reasonable consistency with mineralised intercepts in nearby holes on which downhole surveying has been completed.
	Specification of the grid system used.	Universal Transverse Mercator (UTM) projection Zone 29 North (29N) and the 1984 World Geodetic System (WGS84) datum.
	Quality and adequacy of topographic control.	In generating mineralised volumes to inform the Sanankoro Mineral Resource, a topography surface was generated from ASTER digital elevation data, which was locally adjusted to be consistent with the elevation of the Cora Gold drillhole collars. The topographic profile of the Sanankoro Project area is typically flat-lying. The resolution of the topography surface employed is considered appropriate for the use in deriving a suitably accurate model for use in Mineral Resource Estimation considering the surface relief of the project area.
Data spacing and distribution	*Data spacing for reporting of Exploration Results.*	Across the two Cora Gold drill campaigns, drilling has typically been completed on 60 – 120 m spaced sections, with between 1 and 5 holes per section. The spacing of the historic Randgold and Goldfields drilling is variable, and described in detail in Section 5.
	Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied	The spacing, in relation to the understanding of the controls on mineralisation, the continuity of the mineralised structures, and the Au grade distribution and continuity, is sufficient for the reporting of an Inferred Mineral Resource Estimate
	Whether sample compositing has been applied.	Depending on the results of logging and/or panning, the Cora Gold RC and AC chip samples were either composited to a 3 m composite sample or retained as a 1 m sample (generally the sampled considered to be low grade were composited), prior to assaying.

Criteria	JORC Code Explanation	Project Description
		All downhole assay intervals were composited inside of the modelled mineralisation wireframes during grade estimation, as described in Section 3 of this table.
Orientation of data in relation to geological structure	*Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type.* *If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.*	The majority of the historic RC and AC holes are drilled at a moderate dip to the east, roughly perpendicular to the main mineralisation trend. The Cora Gold drillholes were mostly drilled moderately dipping to the northeast, oblique to the main mineralised trend, in order to drill at an angle appropriate to both the main N-S mineralised trend and the subsidiary E-W trending vein set. As a result, the apparent thickness of the Cora Gold intersections are generally greater than the historic drillhole intersections, which are closer to true thicknesses.
Sample security	*The measures taken to ensure sample security.*	Since the RC and AC chips are sampled at the drill site, a tracking form is filled in and is signed by both the geologist on site and the driver of the vehicle transporting the samples. Once arrived at the camp, the samples are received by the camp manager, who will also sign and file the tracking form. Both the RC and RC chip samples, and the diamond core samples are sent from the field camp to Bamako, where they are directed onwards to either to SGS Ouagadougou or SGS Bamako. Transportation of the samples from the field camp to Bamako is by vehicle. Another a tracking form is filled in by the geologist on site and signed by both the geologist and the driver. A copy of the form is given to the driver (to be handed to the administrator in the Bamako office) and is also emailed to the office in Bamako on the day of departure. Once the samples have arrived at Bamako, they are inspected and (if relevant) air transport information is completed, before being sent to the lab.
Audits or reviews	*The results of any audits or reviews of sampling techniques and data.*	SRK UK has not visited the project site and is unaware of any audits or reviews of sampling techniques.

Section 2 Reporting of Exploration Results

Criteria	JORC Code explanation	Project Description
Mineral tenement and land tenure status	*Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings.*	The Sanankoro property consists of four contiguous permits (Sanankoro, Bokoro II, Bokoro Est and Dako) that encompass a total area of approximately 320 km^2. Details of the permits are provided in Section 3.2.1 and summarised in Table 3-1.
	The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.	The Sanankoro property consists of five contiguous permits (Sanankoro, Bokoro II, Bokoro Est, Dako and Kodiou) that encompass a total area of approximately 342 km2. Details of the permits are provided below: **Sanankoro Permit:** The current exploration permit held by Sankarani (a 95%-owned Malian subsidiary company of Cora Gold Ltd) was issued on the 2 July 2018 and represents the final 2 year exploration permit renewal period, being due to expire on 1 February 2020. **Bokoro II permit:** The current exploration permit held by Sankarani was renewed on 23 August 2019 and is due to expire on 25 August 2022. In accordance with the Malian Mining Code, the permit can be renewed once more for periods of two years, after the expiration of the current licence. **Bokoro Est permit:** The current exploration permit held by Sankarani was re-issued to Sankarani on 18 September 2019 and is due to expire on 18 September 2022. The permit can be renewed twice more for periods of 2 years, after the expiration of the current licence. **Dako permit:**

Criteria	JORC Code explanation	Project Description
		The current exploration permit held by Sankarani was re-issued to Sankarani on 31 December 2018 and is due to expire on the 31 December 2021. The permit can be renewed twice more for periods of 2 years, after the expiration of the current licence. **Kodiou permit:** The Kodiou Permit was granted as an exploration permit to a third party initially on 15 May 2015. The permit expires on 15 May 2022. Through a Joint Venture Agreement, Cora Gold have the option to earn up to 100% through payment of staged fees to the permit holder, subject to the 3rd party being paid a 1% NSR royalty of production from the permit area, with Cora given the right to buyout the 3rd party.
Exploration done by other parties	*Acknowledgment and appraisal of exploration by other parties.*	Much of the exploration data described herein and used to derive the Sanankoro exploration permit was captured by Randgold and subsequently Gold Fields between the mid-2000's and 2012. The exploration completed by Randgold and Gold Fields is described in detail in Section 5.1
Geology	*Deposit type, geological setting and style of mineralisation.*	Cora Gold have established a preliminary geological model that involves the rotation of the host Birimian sedimentary sequence (comprising interbedded volcanic tuffs, sandstones, siltstones and mudstone) into a N-S orientated sub vertical geometry. The package is repeated by regional-scale, steeply east-dipping reverse faults / thrusts, with associated tight to isoclinal folding. The faulting /shearing provided a focus for the development of extensive zones of quartz veining, iron carbonate and pyrite alteration in association with the gold mineralisation. The deep tropical weathering in the region has liberated and in parts re-mobilised the primary gold to depths of 40-100m or more.

Criteria	JORC Code explanation	Project Description
		A detailed description of the regional and local geology and mineralisation geometry and style is provided in Section 4.3.
Drill hole Information	*A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes:* • *easting and northing of the drill hole collar* • *elevation or RL (Reduced Level – elevation above sea level in meters) of the drill hole collar* • *dip and azimuth of the hole* • *down hole length and interception depth* • *hole length.* *If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.*	Listing this material would not add any further material understanding of the deposit and Mineral Resource. Furthermore, no Exploration Results are specifically reported.
Data aggregation methods	*In reporting Exploration Results, weighting averaging techniques, maximum and/or minimum grade truncations (e.g. cutting of high grades) and cut-off grades are usually Material and should be stated.* *Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail.* *The assumptions used for any reporting of metal equivalent values should be clearly stated*	Not applicable - No Exploration Results are specifically reported.
Relationship between mineralisation widths and	*These relationships are particularly important in the reporting of Exploration Results.* *If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported.* *If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (e.g. 'down hole length, true*	Not applicable - No Exploration Results are specifically reported.

Criteria	JORC Code explanation	Project Description
intercept lengths	*width not known').*	
Diagrams	*Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.*	Maps and sections are provided throughout the main body of the report.
Balanced reporting	*Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.*	Not applicable - No Exploration Results are specifically reported.
Other substantive exploration data	*Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical and rock characteristics; potential deleterious or contaminating substances.*	A comprehensive overview of all exploration completed on the property, including soil sampling, termite mound sampling, mapping of artisanal excavations, geophysical surveying etc. is provided in Section 5.
Further work	*The nature and scale of planned further work (e.g. tests for lateral extensions or depth extensions or large-scale step-out drilling).*	Recommendations for future exploration are provided in Section 0.
	Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive	Diagrams are provided in the main body of the report.

Section 3 Estimation and Reporting of Mineral Resources

Criteria	JORC Code explanation	Project Description
Database integrity	*Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes.*	SRK have not visited the Sanankoro site as part of this work and have not completed any review of the Cora Gold data input or database management procedures.
	Data validation procedures used.	SRK have completed a high level statistical and visual validation of the historic Randgold and Gold Fields drillhole data with the Cora Gold drilling. This includes Q-Q plot analysis and visual assessment of grade distribution in the Cora Gold and historic drillhole datasets, by visually comparing interpolants (based on the Leapfrog Radial Basis Function (“RBF”) in based on only the Cora Gold holes and only the historic holes, in specific zones. The results of the validation checks completed, which are described in 6.2, suggest that the historic Randgold and Gold Fields drillhole data is sufficiently robust for the use in the derivation of an Inferred Mineral Resource. The Sanankoro assay database comprises a combination of fire assays and bottle roll analyses. Scatterplot analysis of intervals analysed by both fire assay and bottle roll, as described in Section 6.5, shows an acceptable level of correlation between the fire assay and bottle roll analyses - it is considered appropriate to utilise both Au assay sources (fire assay and bottle roll) in deriving a Mineral Resource Estimate for the Project. SRK have completed a visual verification of the spatial location of the Cora Gold drillhole collars against the historic Gold Fields and Randgold collars. The result of this exercise highlighted some local discrepancies between the elevation of the Cora Gold collars (excluding those that were collared in the artisanal workings) and the historic collars. Specifically, when comparing close-spaced Cora Gold collars with historic Randgold and Goldfield collars, the historic collars often have an erratic distribution of elevation values and, on average are higher in elevation

Criteria	JORC Code explanation	Project Description
		that the Cora Gold collars. For this reason, for the purposes of mineralisation modelling, it was decided to generate a topography surface directly from the Cora Gold collars, (but retaining the trend of the ASTER digital terrane model between collar points), and to snap the historic collars to the elevation of this surface. The collar, survey and assay data were validated through import via the Seequent Leapfrog Geo ("Leapfrog") drillhole data validation routine, prior to completing any modelling. This checks for any overlapping intervals, from depths > to depths, duplicate locations, out of place non-numeric values, missing collar and survey data, any down-hole intervals that exceed the max collar depth etc.
Site visits	*Comment on any site visits undertaken by the Competent Person and the outcome of those visits.* *If no site visits have been undertaken indicate why this is the case.*	SRK have not completed a Competent Persons site visit to the Sanankoro Project. The geological interpretation of the deposit and controls on mineralisation have been developed by Cora Gold. All data upon which the Mineral Resource Estimate is based has been provided to SRK by Cora Gold, and SRK have not completed any independent checks on the logging, sampling or drill protocols put in place by Cora Gold. Dr. Jonathan Forster, CEO and Head of Exploration for Cora Gold Ltd, acts as the Competent Person responsible for the geology, drilling and exploration protocols employed on site.
Geological interpretation	*Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit.*	In SRK's opinion, given the stage of exploration, Cora Gold have developed a robust geological interpretation for the origin and nature of the Sanankoro gold mineralisation, which takes account of all available information. It is considered that the predictability of the position and continuity of the main mineralised structures is high, given the current drillhole spacing. The orientation of the mineralised structures appears to be very consistent over large distances along-strike, and the trend of the mineralisation is well predicted by induced polarization ("IP") geophysical anomalies. For the most part, the location of mineralisation intersected by the most recent drill campaign either overlaps with, or is within a few meters of, a previous iteration of the Sanankoro mineralisation

Criteria	JORC Code explanation	Project Description
		wireframes.
	Nature of the data used and of any assumptions made.	Data used to directly inform the wireframes and block model upon which the Mineral Resource Estimate is based include the following: Au assays from the Cora Gold AC, RC and DC drilling – used as a hard control on modelling of mineralisation wireframes, and for block model grade interpolation. Au assays from the Randgold and Gold Fields AC, RC, DC and RAB drilling – used as a hard control on modelling of mineralisation wireframes, and for block model grade interpolation. Regolith logging of the Cora Gold drillholes – directly used to model the weathering profile wireframes, which were subsequently employed as domains for density assignment. Bulk density determinations from Cora Gold diamond drilling – used to directly inform the density values applied to the block model Bulk density determinations from field grab samples – used to directly inform the density values applied to the block model
	The use of geology in guiding and controlling Mineral Resource estimation. *The effect, if any, of alternative interpretations on Mineral Resource estimation.*	Visual comparison of the distribution of the highest grade downhole assay intervals with the induced polarization (IP) geophysical anomaly map, suggests a strong spatial correlation between the mineralised zones, and the location of sharp contrasts between high and low IP anomalies. It is considered that these IP

Criteria	JORC Code explanation	Project Description
		anomaly contrasts are most likely associated with the deposit-scale thrust faulting interpreted to act as a conduit for the mineralised quartz veins. As such, the strike of the modelled veins was guided by the trend of the IP anomaly contrasts between drillhole sections, with the dip of the modelled veins being based on visual continuity in downhole assay grades, and the known steeply dipping to sub-vertical dip of the main mineralised vein set. Modelling was focussed on connecting mineralised intervals parallel to the main steeply dipping NNE-SSE striking vein set, this being the principal focus of artisanal exploration to date. It is possible that more close spaced drilling, with associated structural data, may allow for delineation of the less prominent steeply dipping E-W oriented and sub-horizontal vein sets in future updates. SRK used regolith logging completed on all Cora Gold drillholes to generate a weathering model, which was used to sub-domain the volumetric mineralisation model into “hardcap”, “saprolite”, “saprock” and “fresh” domains, for the application of density values.
	The factors affecting continuity both of grade and geology.	As presently defined, the modelled zones of mineralisation that inform the Mineral Resource Estimate are open down-dip. Drilling to date suggests that the individual mineralised structures have along-strike continuity (at Au grades sufficiently high to support the reporting of a Mineral Resource Estimate) of up to approximately 1 – 1.5 km. Across the mineralised structures at the Sanankoro Project exist identifiable higher and lower grade zones, that are based on multiple adjacent drillholes of reasonably consistent grade. Grade variations are more distinct along-strike than down-dip, indicating a structural control on the distribution of high-grade zones, possibly relating to the intersection of the N-S trending mineralised structures, with cross-cutting mineralised veins.

Criteria	JORC Code explanation	Project Description
		No cross-cutting structures are known to terminate the mineralised system.
Dimensions	*The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and lower limits of the Mineral Resource.*	The mineralisation volumes modelled by SRK cover a total linear strike length of approximately 13.5 km over 29 individual domains. The mineralised domains are restricted to four zones, namely Zone A, Zone B, Zone B North and Selin. Excluding across-strike overlap between individual domains, the total strike of the four mineralised zones is approximately 8 km. The mineralisation wireframes were limited to a vertical depth of 50 m below the deepest hole on each drill section. The maximum depth below surface of the mineralisation volumes is 215 m, whilst the average depth to the base of the mineralisation volumes is 125 m.
Estimation and modelling techniques	*The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used.*	The volumetric mineralisation model used to inform the Mineral Resource Estimate was constructed in Leapfrog Geo 4.5. Mineralisation wireframes were constructed based directly on the raw un-composited assay data. No specific modelling cut-off was applied; rather domain contacts / limits were defined based on the position of step changes in Au grade. That said, the mineralisation domains were restricted to a minimum modelling cut-off of 0.2 g/t, and limited to zones of > 0.2 g/t Au that could be correlated across at least 3 drillholes after compositing to 3 m. Weathering profile wireframes (namely hardcap, saprolite, saprock and fresh rock) were constructed based on regolith logging of the Cora Gold drillholes. The weathering domains were modelled using an offset mesh function based on the topography surface, in order to honour the topographic control on the geometry of the weathering profile. Assay composites of 3 m length were generated, using the mineralisation wireframes as compositing triggers. SRK completed a composite length analysis based on comparing the length weighted mean grade of all samples within individual domains, with the mean grade of individual domains after the removal of remnant composites smaller than a defined length. On this basis, it was decided to remove all remnant composites <1.0 m in length from the composite drill hole

Criteria	JORC Code explanation	Project Description
		file used for grade estimation. The total number of samples removed from the composite file is 20, which represents 2.0% of all composite samples. High grade caps were applied to the composite drillhole file, based on the capping analysis described in Section 8.3.1. Variography undertaken on the largest domain in Zone A (Zone A 2), indicates a nugget of approximately 40% and a total range in the order of 150 m (based on an omni-directional variogram. It was not possible to generate meaningful directional variograms for Zone A 2, most likely a result of the small number of sample pairs in the direction of maximum grade continuity. Both directional, and omni-directional variography was attempted on the other estimation domains, without success. Empty block models were generated within the solid wireframes of the mineralisation domains. Parent block sizes varied between zone, being between 10 mx * 25-40 my * 15 mz. The minimum sub block size was adjusted per area to appropriately reflect the geometry and volume of the mineralisation domains. Both the mineralisation wireframes and weathering profile wireframes were employed as sub-blocking triggers. Capped composite assay data was used to interpolate Au grade into the block model, independently for each mineralisation domain, according to the following criteria: - Au grades estimated into parent blocks, using only the composite assays in the corresponding mineralisation domain, by Ordinary Krigging (“OK”); - The krigging variogram parameters for all domains were based on the results of the variography completed on Zone A 2. This is based on the assumption that the grade continuity in Zone A, Zone B North and Selin will be comparable to Zone A; - To attempt to better represent the shallow – moderately north plunging mineralisation trend interpreted in Zone A, the variogram parameters applied to the Zone A estimates were manually adjusted so that the down-plunge range was

Criteria	JORC Code explanation	Project Description
		double that of the across-plunge range. For all other zones, the variogram ranges were un-changed from the omni-directional variogram ranges modelled for Zone A 2; - All mineralisation domains in Zone A were estimated using a moderately north-dipping search ellipse, with down-plunge range distances being set to approximately double the across-plunge range; - All other mineralisation domains were estimated using an isotropic ellipse; - Search ellipse size adjusted for each domain to estimate blocks using data from at least 2-3 drillhole fences; - The minimum number of samples to be estimated into each block was adjusted for each domain to attempt to force the estimate to use samples from at least 2 sections in the estimation of each block; - A discretization level of 3*3*3 was set for all estimates; - Sub-blocks assigned the grade of the parent block; - Second, third and fourth searches, with progressively expanded ellipses and relaxed sample requirements were applied to fill any blocks not filled in the previous run.
	The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data.	SRK are unaware of any existing Mineral Resource Estimates relating to the Sanankoro Project.
	The assumptions made regarding recovery of by-products.	No by-products are assumed to be economic at this stage.
	Estimation of deleterious elements or other non-grade variables of economic significance (e.g. sulphur for acid mine drainage characterisation).	No deleterious elements have been modelled at this stage.

Criteria	JORC Code explanation	Project Description
	In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed.	Parent block sizes for the interpolation of grades within the volumetric mineralisation model were selected based on the average drillhole spacing in each area, being roughly half the on-section drillhole spacing and with approximately 2-3 columns of blocks between sections. Search ellipse size adjusted for each domain, to estimate blocks using data from at least 2-3 drillhole fences.
	Any assumptions behind modelling of selective mining units.	No selective mining unit estimations have been made.
	Any assumptions about correlation between variables.	At this stage, Au is the only variable to have been modelled.
	Description of how the geological interpretation was used to control the resource estimates	Mineralisation domain wireframes used to code the model and drillholes to estimate grade into separate domains. The trend applied to the mineralisation wireframes was based on the assumption that the N-S vein set recognised in the Sanankoro Project area is the dominant mineralised orientation. The variogram parameters and search ellipse for Zone A estimates were manually adjusted to impart a moderately north dipping plunge, interpreted to be potentially related to the intersection of the main N-S trending mineralised vein set with cross-cutting mineralised veins, on the block model grade estimation.
	Discussion of basis for using or not using grade cutting or capping.	SRK completed a capping analysis based on the assessment of log probability plots, raw and log histograms, which were used to identify any sample grades outside of the main grade populations. The capping analysis was completed by zone, and separately for the oxide (hardcap, saprolite and saprock) and sulphide (fresh rock) samples. The following grade caps were applied: - Zone B Sulphide – 1 sample to 5 g/t resulting in a 2.5% reduction in mean sample grade for Zone B

Criteria	JORC Code explanation	Project Description
		- Zone B North Oxide – 2 samples to 6 g/t resulting in a 11.6% reduction in mean sample grade for Zone B North - Selin Oxide – 7 samples to 12 g/t and; Selin Sulphide – 1 sample to 9 g/t, in total resulting in an 8.9% reduction in mean sample grade for Selin. No capping was applied to any other domains.
	The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.	SRK completed a validation of the estimated blocks in the volumetric mineralisation using the following techniques: - Visual inspection of the block grades in 3D and section, comparing the input composite grades with the block grades in the corresponding domains; - Comparison of global mean block grades and sample grades within the mineralisation domains. Visual checks generally demonstrate a strong comparison between local block estimates and nearby samples, without excessive smoothing in the block model. In the significant majority of instances, the estimated block grades are within 10% of the mean capped composite grades. For a small number of mineralisation domains, the mean estimated block grades differ >10% from the mean capped composite grades. Given the overall strong visual validation between the block and sample grades, and clear reasons (as outlined in Section 8.4.3) for the limited cases of significant differences between the block grade means and sample grade means, SRK is confident in the positive validation of the estimate of the Sanankoro mineralisation domains.
Moisture	*Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.*	All tonnages are reported as dry tonnages.

Criteria	JORC Code explanation	Project Description
Cut-off parameters	*The basis of the adopted cut-off grade(s) or quality parameters applied.*	The Mineral Resource is reported above a calculated marginal cut-off grade of 0.4 g/t for all oxide blocks (hardcap, saprolite and saprock) and 0.5 g/t for the sulphide blocks.
Mining factors or assumptions	*Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.*	SRK completed pit optimisation study based on reasonable, but optimistic, economic and mining assumptions to evaluate the proportions of the block model that could "reasonably expected" to be mined from an open pit. The Mineral Resource Statement has been restricted to material falling within the resulting optimised pit shells. Optimisation parameters applied include the following: Gold Price: 1,700 USD/oz; - Pit slope angle - 34° in hardcap and saprolite, 40° in saprock, 42° in fresh rock; - Mining dilution – optimisation based on a regularised block model with a regularisation grid of 2.5 * 2.5 * 5 m; - Processing Recovery – 80% in hardcap, 95.7% in the saprolite and saprock at Zone A and Zone B, 92.9% in the saprolite and saprock at Zone B North and Selin; 80% in fresh rock; - Mining Cost – 3.5 USD/t in saprolite and hard cap ore, 4.0 USD/t in saprock and fresh ore, 3.0 USD/t in saprolite and hardcap waste, 3.5 USD/t in saprock and fresh waste. - Processing Cost – 15.5 USD/t ore in oxide, 17.0 USD/t ore in sulphide - G&A – 2.0 USD/t ore - Selling Cost – 5% - Discount Rate – 10%

Criteria	JORC Code explanation	Project Description
Metallurgical factors or assumptions	*The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.*	Processing recoveries and costs have been applied to the pit optimisation shell used to constrain the Mineral Resource Estimate. Processing recoveries have been assigned based on preliminary metallurgical testwork on the oxide and sulphide mineralisation, completed by Wardell Armstrong International ("WAI"), the results of which are summarised in Section 7.
Environmental factors or assumptions	*Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.*	SRK is unaware of any environmentally sensitive areas (for example, protected / conservation areas, forest reserves, national parks, etc.) or historical, archaeological, cultural or other heritage features that impact on the Sanankoro Project. SRK have not completed any environmental review in reporting the Mineral Resource Estimate.
Bulk density	*Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples. The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc.), moisture and differences between rock and alteration zones within the deposit. Discuss assumptions for bulk density estimates used in the evaluation process of the different materials.*	Average density values were assigned to the block model, based on the weathering profile domain coding, as described below. The difference in mean density between the mineralised and un-mineralised samples available at present is minimal, and so, at this stage, it was decided to apply the same density values to the mineralisation and waste blocks. Fresh Rock – Assigned the mean density (2.75 g/cm3) of unweathered drill core samples from the Cora Gold diamond drilling. Density was calculated using a simple volumetric method, where density = weight of sample / (length of sample (measured along the 3 different parts of the drill core perimeter to derive an average) * drill core diameter * □). Saprolite and Saprock – Assigned the mean density (2.15 g/cm3) of saprolite and

Criteria	JORC Code explanation	Project Description
		saprock drill core samples from the Cora Gold diamond drilling, as calculated using the volumetric method described above. Hardcap – Assigned the mean density (2.55 g/cm3) of field grab samples taken from the hardcap material. Density was calculated by Cora Gold by the water immersion method, with density calculated as "W1/(W1/(W2)", where W1 is the dry weight, and W2 is the weight in water.

Classification	*The basis for the classification of the Mineral Resources into varying confidence categories.* *Whether appropriate account has been taken of all relevant factor (i.e. relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data).* *Whether the result appropriately reflects the Competent Person's view of the deposit.*	SRK consider that the predictability of the position and continuity of the main mineralised structures is high, given the current drillhole spacing. The orientation of the mineralised structures appears to be very consistent over large distances along-strike, and the trend of the mineralisation is well predicted by the induced polarization (IP) geophysical anomalies The distribution of higher and lower grade zones in the largest mineralisation domain at Zone A (Zone A 2) suggests a relatively consistent and predictable shallow dipping plunge to the north. Trends in the distribution of Au grade in other domains are less clear at this stage. Validation checks on the historic Randgold and Gold Fields drilling indicates that the inclusion of this data in the mineralisation models and grade interpolation is unlikely to introduce any significant bias into the Mineral Resource Estimate. QAQC checks on the Cora Gold assays are mostly within acceptable reporting limits. That said, SRK and Cora Gold have identified that the standards used for the assessment of the bottle roll analyses return spurious and erratic results. This is discussed in detail in Section 0. SRK is sufficiently satisfied that the spurious results of the bottle roll standards are a function of the method used to create the standard samples, rather than indicating any serious fault in the accuracy of the analytical equipment at the SGS lab, or contamination of samples during sample preparation, and as such consider that the Cora Gold assay data is sufficiently robust for the use in a Mineral Resource Estimate. Given the above, SRK consider that, for the most part, the current drill coverage at Zone A, Zone B, Zone B North and Selin is appropriate for the reporting of an Inferred Mineral Resource. This excludes Selin 5 and Zone B 2, which, at this stage, are characterised by a low confidence in the geological interpretation and have too few drillhole intersections to justify the classification of Inferred Resources.

Criteria	JORC Code explanation	Project Description
Audits or reviews	*The results of any audits or reviews of Mineral Resource estimates.*	SRK is unaware of any pre-existing Mineral Resource Estimates relating to the Sanankoro Project.
Discussion of relative accuracy/ confidence	*Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate.* *The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used.* *These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.*	SRK has assigned portions of the deposit in the Inferred Mineral Resource category based on the drillhole spacing, quality of data and confidence in the continuity of mineralisation.

APPENDIX 2: A Report for the Mining Scoping Study on the Sanankoro Gold Project, Mali
(Completed by SRK Consulting)

A REPORT FOR THE MINING SCOPING STUDY ON THE SANANKORO GOLD PROJECT, MALI

Prepared For

Cora Gold

Report Prepared by

srk consulting

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UK30681

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EXECUTIVE SUMMARY
A REPORT FOR THE MINING SCOPING STUDY ON THE SANANKORO GOLD PROJECT, MALI

1 INTRODUCTION

SRK Consulting (UK) Limited ("SRK") is an associate company of the international group holding company, SRK Consulting (Global) Limited (the "SRK Group"). SRK has been requested by Cora Gold Limited ("Cora Gold", hereinafter also referred to as the "Company" or the "Client") to prepare a mining report section for the Scoping Study for their Sanankoro Project ("Sanankoro", or the "Project") located in Mali, West Africa. Hydrogeological and Geotechnical considerations are included.

2 MINERAL RESOURCE ESTIMATE

The Mineral Resource Estimate ("MRE") and supporting geological model used for this Scoping Study are described in a separate document, namely the "A Mineral Resource Estimate on the Sanankoro Gold Project, Mali, December 2019" by SRK referred hereby also as the "MRE".

3 HYDROGEOLOGY AND HYDROLOGY

SRK have completed a high-level scoping study review of the available hydrology and hydrogeological data for the Sanankoro project. These data are noted to be very limited, but have been used to inform recommendations for moving the project towards a Pre-Feasibility Study (PFS). The assumptions made are that all mining slopes will be within the saprolite formation and will need to be depressurised in order to achieve the pit slope angles defined by the geotechnical assessment.

The key hydrological risks identified relate to high intensity rainfall events resulting in either direct flooding of the pits or indirect recharging of the pit slope pore pressures; these risks should be quantified at PFS level following the installation of a site weather station and river flow gauges. The key hydrogeological risk for the project is the inability for the saprolite to remain depressurised; the hydrogeological system requires testing and conceptualisation in order to assess expected pore pressure responses to both climate and mining events. This assessment requires the establish of groundwater level monitoring and hydraulic testing within the key hydrogeological units.

4 GEOTECHNICAL

SRK have provided Scoping level geotechnical slope criteria for the Sanankoro project to feed into pit optimisation. The pits will be in the region of ~100m at the deepest sections and will primarily be formed within saprolite with minor saprock and fresh rock at the base of the slopes. Whilst limited geotechnical information exists for the fresh material, there is no geotechnical information for the saprolite. As such, SRK have relied on experience from developing pit slopes in other saprolite deposits to propose a range of saprolite slope angles for Sanankoro.

Several slope angles ranging from 26° to 38° were considered, with a slope angle of 34° chosen for input into pit optimisation. Within the deeper sections of the open pits, 34° can be considered steep and to achieve such an angle, high quality surface water management in addition to slope depressurisation drilling will be required to lower pore water pressure within the slope. Regardless of the success of the depressurisation programme, bench and possibly multi-bench failure may be expected as a result of remnant structure within the saprolite.

For the small sections of saprock and fresh material exposed at the toe of the slopes, SRK recommended 40° and 42° slope angles respectively. To verify the proposed Scoping level slope angles at the next project stage, geotechnical drilling, logging and sampling will be required in addition to hydrogeological testing to determine the susceptibility of the saprolite to slope depressurisation programmes.

5 MINING

5.1 Introduction

The Project comprises several distinct zones including Zone A, Zone B, Zone B North and Selin. The mining study has been completed for three production rates (0.5 Mtpa,1.0 Mtpa,1.5 Mtpa) recognized as Case 1, Case 2 and Case 3. The main objective of the Study was to understand how the different cases compare, their potential impact on mining costs for owner and contractor operated scenarios and to support any future exploration activities. The mining study is restricted to oxide material (hardcap, saprolite and saprock) and excludes sulphide (fresh) mineralisation.

5.2 Dilution and Recovery

In order to address mining modifying factors such as mining losses and dilution, the mineral resource model (in Datamine format) has been regularised to a block size of 2.5 x 2.5 x 5 m and used in pit optimisation and mine planning. A block size of 2.5 x 2.5 x 5 m is considered representative of the selective mining unit size estimated for small scale mining equipment (1.9 m^3 to 4 m^3 bucket excavators, 24 t to 40 t capacity haul trucks), and requires a relatively high level of selectivity. Above a marginal cut-off of 0.4 g/t Au, the dilution in all zones is estimated between 14% and 20% and recovery between 91% and 95%. The method of calculation is explained in the main text of the Report.

5.3 Pit Optimisation

The pit optimisation was completed for a selling price of USD1,500 /oz Au. Resulting pit shells were analysed to compare how the factored metal price (Revenue Factor or "RF") affects ore tonnage, grade and strip ratio. The pit optimisation parameters are shown in Table ES 1. The optimisation parameters outlined in Table ES 1 include recoveries, costs and slope angles for fresh rock (as an alternate pit optimisation was completed on both the oxide and fresh rock for the purposes of Mineral Resource reporting), however it should be stressed that the pit optimisation employed in the mining study considered only oxide material.

Table ES 1: Open Pit Parameters

Parameters	Units	Case 1	Case 2	Case 3	Comments
Production					
Production Rate - Ore	(tpa)	500,000	1,000,000	1,500,000	Cora Gold Assumption
Geotechnical					
Overall Slope Angle - Saprolite	(˚)	34	34	34	SRK Assumption
Overall Slope Angle - Saprock	(˚)	40	40	40	SRK Assumption
Overall Slope Angle - Fresh	(˚)	42	42	42	SRK Assumption
Mining Factors					
Dilution	(%)	Regularised Block Model 2.5x2.5x5m			See Section 5.2 for details
Recovery	(%)	Regularised Block Model 2.5x2.5x5m			See Section 5.2for details
Processing					
Hardcap - All Zones	(%)	80.0	80.0	80.0	WA Assumption
Zone A/B (sap/saprock)	(%)	95.7	95.7	95.7	WA Assumption
Selin + Zone B North (sap/saprock)	(%)	92.9	92.9	92.9	WA Assumption
Fresh - All Zones	(%)	80.0	80.0	80.0	WA Assumption
Operating Costs					
Mining Cost - Ore					
Saprolite	(US$/$t_{ore}$)	3.50	3.50	3.50	SRK Assumption
Sap Rock & Fresh	(US$/$t_{ore}$)	4.00	4.00	4.00	
Mining Cost - Waste					
Saprolite	(US$/$t_{waste}$)	3.0	3.0	3.0	SRK Assumption
Saprock & Fresh	(US$/$t_{waste}$)	3.50	3.50	3.50	
Processing - Saprolite, Saprock, Hardcap	(US$/$t_{ore}$)	16.2	15.5	14.7	WA Assumption
Processing - Fresh	(US$/$t_{ore}$)	17.0	17.0	17.0	WA Assumption
G&A	(US$m/Year)	1.0	2.0	3.0	
	(US$/$t_{ore}$)	2.0	2.0	2.0	WA Assumption
Selling Cost Au	(%)	5.0	5.0	5.0	SRK Assumption explained in Section 4.5.7 in more detail
	(US$/oz)	85.0	85.0	85.0	
	(US$/g)	2.5	2.5	2.5	
Metal Price					
Gold	(US$/oz)	1,500.0	1,500.0	1,500.0	Cora Gold Assumption
	(US$/g)	43.8	43.8	43.8	
Other					
Discount Rate	(%)	10.0	10.0	10.0	SRK Assumption
Cut-Off Grade					
Marginal - Saprolite, Saprock, Hardcap	(US$/$t_{ore}$)	18.2	17.5	16.7	
	(g/t Au)	0.4	0.4	0.4	
Marginal - Fresh	(US$/$t_{ore}$)	19	19	19	
	(g/t Au)	0.5	0.5	0.5	

The pit optimisation results show that the mining inventory grows with increasing RF on a broadly linear basis. That said, it should be noted that the total ore tonnage is relatively sensitive to the gold price selected for the pit optimisation. The total ore tonnage inside of the USD1,300/oz pit shell (which is a more similar price to the current long-term Au price forecast) is 2.8 Mt at 1.60 g/t Au, whilst the total ore tonnage inside of the USD1,500/oz pit shell is 4.1 Mt at 1.47 g/t Au. This represents a 46% increase in ore tonnage and 35% increase in contained ounces in the USD1,500/oz pit shell, compared to the USD1,300/oz pit shell. Total rock inside the USD1,500/oz pit shell is 28.4 Mt and total rock inside the USD1,300/oz pit shell is 17.0 Mt. The stripping ratio is 5.9 in the USD1,500/oz pit shell and 5.1 in the USD1,300/oz pit shell. After discussions between Cora Gold and SRK, Cora Gold requested that SRK use the USD1,500/oz Au pit shell (RF=100%) for the development of the strategic schedule. This is considered acceptable at a scoping level, however the sensitivity to Au price should be carefully considered as the Project develops.

5.4 Strategic Mine Schedule

SRK has developed a strategic level mining and processing schedule for the Zone A, Zone B, Zone B North and Selin using NPVS scheduling software. The mine schedule was completed for the three production cases and has been produced in annual periods. The mining schedule for Case 1, Case 2 and Case 3 are shown in Figure ES 1, Figure ES 2, Figure ES 3, respectively.

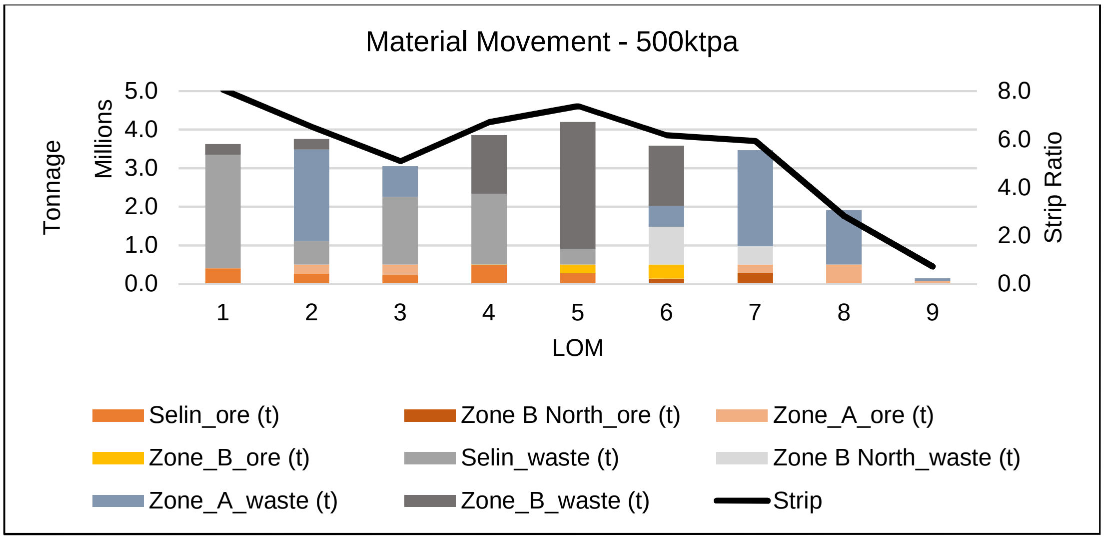

Figure ES 1: Case 1 – 0.5 Mtpa – Material Movement

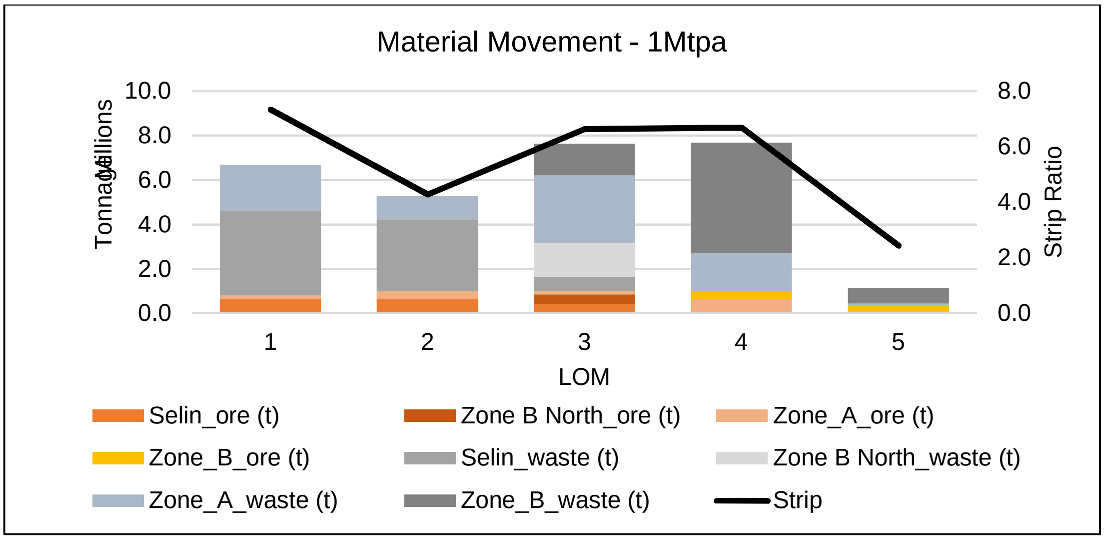

Figure ES 2: Case 2 – 1.0 Mtpa – Material Movement

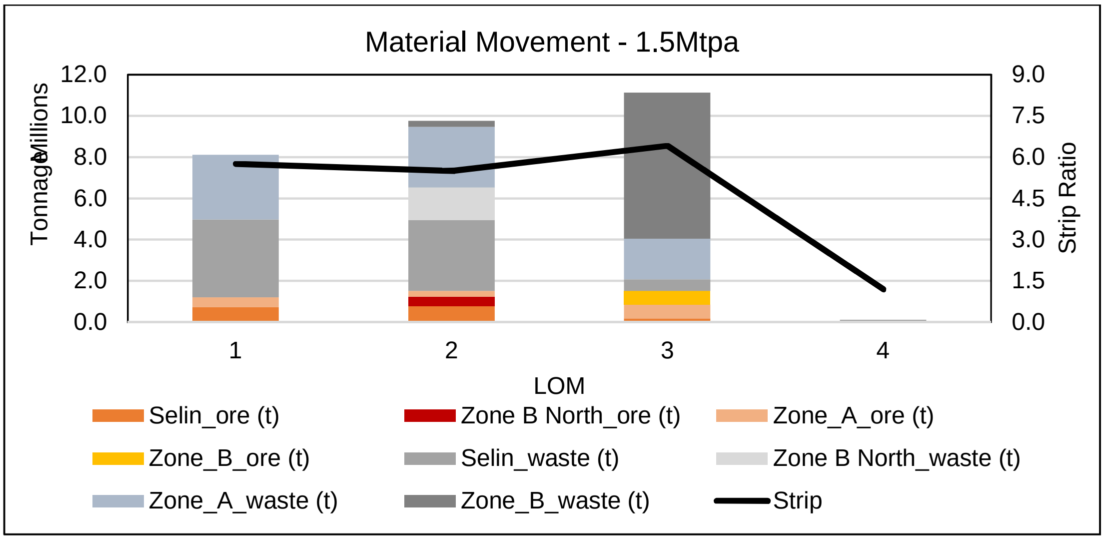

Figure ES 3: Case 3 – 1.5 Mtpa – Material Movement

5.5 Equipment Selection

Based on an estimation of equipment productivities, production profiles and the need for a selective mining method, the selected equipment sizes are shown in Table ES 2. The quantity of each piece of equipment is dependent on the strategic schedule for each Case.

Table ES 2: Equipment Selection

Equipment	Material Type	Unit	Equipment Size		
			0.5 Mtpa	1.0 Mtpa	1.5 Mtpa
Primary Backhoe	Ore	m^3 bucket width	1.9	1.9	1.9
Primary Wheel Loader	Waste	m^3 bucket width	4.2	7.1	8.4
Primary Truck	Ore	t capacity	24	24	24
Secondary Truck	Waste	t capacity	40	40	54
Primary Drill	Both	mm diameter	140	140	140

5.6 Operating Strategy

It is expected that the extraction method will be predominantly free digging, as the hardcap and saprock weathering domains do not require blasting. Drill and blast will be required in the saprock domain.

Ore and waste will be excavated by separate fleets in order to account for a relatively high level of mining selectivity.

Based on the pit locations and the distance between the zones, it is recommended to have three Waste Rock Dumps (“WRD”). The waste rock dump tonnage schedule is reflected by the yearly waste production, but no detailed scheduling has been done for the WRDs. A stockpiling strategy has not been considered in this study.

5.7 Capital and Operating Cost Estimation

A mining cost model has been developed to assess the mining capital and operating expenditures expected for the Sanankoro Project. This cost estimation is based on both contractor mining and owner operated options as requested by the Client. All capital and operating costs have been estimated from first principles but based on SRK’s experience of open pits in Mali, or benchmarked from the 2018 Infomine cost database.

The capital cost estimation includes equipment purchase, replacement and rebuild costs, as well as mobilisation/demobilisation and site establishment costs. The results of the capital cost estimation are shown in Table ES 3.The capital cost difference between an owner operated and a contractor option, is that contractor capital does not include equipment purchase and replacement costs.

In addition to the capital cost categories, a 15% capital cost contingency is applied to both the owner operated and contractor options.

Table ES 3: Capital Cost Estimation

Capital Cost	Units	Cost
Primary Backhoe	(US$m)	0.9
Primary Wheel Loader	(US$m)	0.8
Primary Truck	(US$m)	0.6
Secondary Truck	(US$m)	0.8
Primary Drill	(US$m)	1.4
Track Dozer	(US$m)	0.9
Grader	(US$m)	1.0
Water Truck	(US$m)	1.4
Service Truck	(US$m)	0.1
Fuel/Lube Truck	(US$m)	0.1
Small Crane	(US$m)	0.5
Lighting Plant	(US$m)	0.5
Light Vehicle	(US$m)	0.7
Site Establishment, Mobilisation, Demobilisation	(US$m)	1.7

The operating costs are broken down into four categories including labour, maintenance, consumables and grade control. The owner operated and contractor base unit cost for these categories are the same, therefore the varying factor is a contractor premium of 25% applied to the contractor option. Similar to the capital cost estimation, a 15% operating cost contingency is applied to both the owner operated and contractor options.

The labour and consumable unit costs are provided in Table ES 4 and Table ES 5, respectively.

Table ES 4: Consumable and Grade Control Costs

Consumables	Source	Units	Cost
Fuel	SRK estimate	(US$/l)	1.1
Power	SRK estimate	(US$/kWh)	0.049
Lube	SRK estimate	(US$/l)	4
AN	SRK estimate	(US$/t)	1500
Primer	SRK estimate	(US$/unit)	3.5
Detonator	SRK estimate	(US$/unit)	3
Surface Delay	SRK estimate	(US$/unit)	2
Blasting Accessories	SRK estimate	(%)	5
Sampling	SRK estimate	(US$/unit)	20
Stemming	SRK estimate	(US$/lcm)	10
Grade Control Drilling	SRK estimate	(US$M/yr)	1

Table ES 5: Labour Costs

Position	Salary (USD$pa)
Mine Operations	
Mine Manager	120,000
Superintendents	100,000
Supervisors	85,000
Trainer	75,000
Shovel Operator	40,000
Truck Operator	40,000
Loader Operator	45,000
Ancillary Operator	35,000
Driller Operator	35,000
Safety Manager	80,000
Human Resources Manager	80,000
Accountant	60,000
Security	15,000
Mine Maintenance	
Maintenance Superintendent	85,000
Maintenance Supervisor	75,000
Maintenance Planner	65,000
Maintenance Crew	25,000
Technical Services	
Chief Engineer	100,000
Chief Geologist	100,000
Senior Engineer	90,000
Senior Geologist	90,000
Planning Engineer	75,000
Mine Surveyor	40,000
Mine Geologist	60,000
Consultant (Hydro, Geotech, etc)	75,000
Administrative Assistant	25,000

The mining costs estimated in this Report are summarised in Table ES 6.

Table ES 6: Estimated Mining Costs

	Scenario	Unit	0.5 Mtpa	1.0 Mtpa	1.5 Mtpa
OPEX	Owner	(US$/t)	3.43	2.82	2.48
	Contractor	(US$/t)	4.17	3.43	3.02
CAPEX	Owner	(US$M)	19.6	32.6	31.3
	Contractor	(US$M)	3.2	5.9	6.6

5.8 Conclusions

The dilution in all zones is estimated between 14% and 20% and recovery between 91% and 95% based on a regularised block model to a selected SMU of 2.5 x 2.5 x 5 m and 0.4g/t Au cut-off grade. Based on the pit optimisation results, there are no visible or significant step changes in sensitivity in tonnes or grade around the considered prices however, fairly linear increase in the ore inventory is relatively steep. The Client and SRK selected the USD1,500/oz Au pit shell (RF=100%) for developing the mine design and strategic schedule.

The production rate of each schedule directly effects the length of the life of mine (LoM), material movement and equipment/personnel requirements. Three cases were considered. Case 1 at a production rate of 0.5 Mtpa can be achieved at total mining rate up to 4.3 Mtpa and contains the most balanced material movement. Case 2 and Case 3 at a production rate of 1 Mpta and 1.5 Mpta respectively, contain a total mining rate ranging between 5.0 Mpta and 11 Mpta. All cases require the same ore mining fleet that includes a minimum of 2 x 1.9 m^3

excavators, 2 x 24 t trucks and 1 x 140 mm production drill. The waste equipment size is dependent on the production rate of each case. Relatively long distances and resulting cycle times were observed for the ore haulage, due to the plant location and number of pits located around it. It is expected that the extraction method will be predominately free digging where drill and blast will only be required in the saprock weathering domain.

This Mining Section of the Scoping Study for Sanankoro has been completed by SRK while the main Report of the Scoping Study has been developed by Wardell Armstrong consultants. From this perspective, SRK has not identified any fatal flaws at the scoping level, however SRK is not able to comment on the overall project potential as it has not been involved in developing the Technical Economic Model ("TEM").

5.9 Recommendations

Based on the work undertaken as part of the Scoping Study, SRK makes the following mining study related recommendations:

- Mining operating costs should be confirmed through obtaining mining contractor costs, using the strategic schedule and equipment sizing and estimated requirements from the Scoping Study.
- Open pit optimisation and pit shell selection should be repeated when material changes to the geological model or operating / processing assumptions occur.
- Run trade-off studies with civil type of trucks hauling ore to the plant and consult tyre manufacturers regarding the tyres for high temperature environment;
- Verify potential for the fresh material;
- This Mining Section of the Scoping Study for Sanankoro has been completed by SRK, while the main Report of the Scoping Study has been developed by Wardell Armstrong consultants. SRK recommends using the annual cost estimate in the TEM for the entire Project.
- It is recommended by SRK that this scoping level mine planning should be updated when new geological information and an updated block model is available. Based on the results of the updated scoping level mine planning, if the Project is economically favourable, more detailed study (to Pre-Feasibility Level) should be undertaken to determine the mining parameters, costs and operating strategy at an increased level of detail.

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A SCOPING STUDY ON THE SANANKORO GOLD PROJECT, MALI

1 INTRODUCTION

1.1 Background

SRK Consulting (UK) Limited ("SRK") is an associate company of the international group holding company, SRK Consulting (Global) Limited (the "SRK Group"). SRK has been requested by Cora Gold Limited ("Cora Gold", hereinafter also referred to as the "Company" or the "Client") to prepare a mining report section for the Scoping Study ("Scoping Study") for the Sanankoro Project ("Sanankoro", or the "Project") located in Mali, West Africa. Hydrogeological and Geotechnical considerations are included.

1.2 Property Description, Location and History

1.2.1 Location

The Sanankoro property lies approximately 110 km south west of Bamako, predominantly within the Kangaba Cercle, Koulikoro Region in southwest Mali, although the southern-most part of the licence area extends into the Yanfolila Cercle of the Sikasso Region.

The geographical location of the Sanankoro property permits is shown in Figure 1-1.

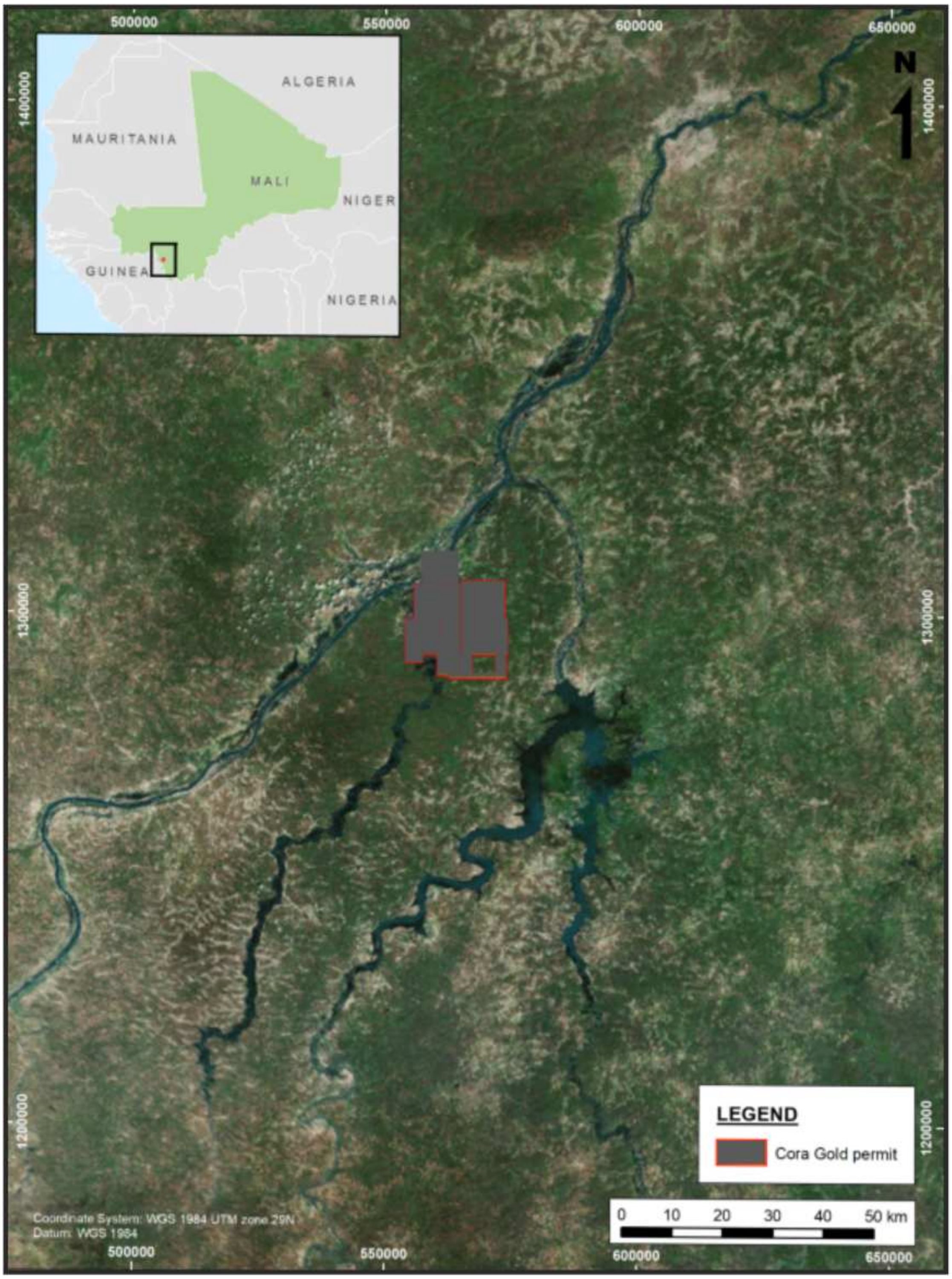


Figure 1-1: The Sanankoro property permit outlines, shown relative to Google Earth Satellite imagery and, inset, within the West Africa region map.

1.3 Physiography, Climate and Environment

Mali has a varied landscape and three distinct climatic and vegetation zones: the Saharan zone in the north; the semi-arid Sahelian Zone in the centre; and the raised savannah, or 'Sudanese' zone in the south. Northern Mali is covered by the southern extension of the Sahara Desert, and as such is arid with a hot almost rainless climate. The Sahelian zone is concentrated around the River Niger and marks the transition from desert into raised savannah.

In the raised savannah zone in the south, where the Sanankoro Project is located, there are two distinct seasons: a dry season lasting from mid-October to late-April, when virtually no rain falls and a rainy season from late-April to mid-October. Total annual rainfall for this region is around 1,200 mm per year, which is concentrated within these months and can impact infrastructure during this time. Temperatures are high year round (20-35°C), and peak at the end of the dry season where temperatures often exceed 40°C, particularly in the Saharan north.

The physiography of the property is typically flat-lying with shallow topography although does include several hills with elevations of up to 410 m, around 40-50 m above the surrounding plains. Drainage is moderately well developed and typically flows to the west into the Niger River. Vegetation within the property typically consists of sparse trees and bushes.

1.4 Artisanal Mining

The Sanankoro property is associated with extensive artisanal gold mining activity, mainly within the Sanankoro permit. Discontinuous open-pit workings extend over a distance of just over 10 km, with individual workings up to 3 km in length and 500 m in width. The open-pit workings are typically not very deep (< 15 m) which appears to be due to the instability of the regolith. However, vertical shafts are common in the base of the open-pits, locally extending the depth of the workings by up to a further 5-10 m. Further details can be found in the MRE Report.

1.5 Scope of Work

1.5.1 Mining

SRK has been requested to provide a preliminary technical assessment, with the main objective to describe a preferable mining operation. The resource block model, completed by SRK, forms the basis of the information for the Study. The mining study is focussed on only the oxide mineralisation.

Work completed includes:

- A pit optimisation study for each zone, based on optimistic conceptual parameters, with sensitivity testing for processing recovery;
- High-level analysis of potential mining dilution and mining losses for each area;
- An outline of potential contractor mining costs;
- A high-level Mine Schedule indicating potential mine production tonnes and grades outputs, presented in annual increments for the proposed Life of Mine ("LoM"). Mine scheduling has been tested at 3 rates (0.5 Mtpa, 1 Mtpa and 1.5 Mtpa) as requested by Cora Gold.

- Identification of areas around the pits that could accommodate volumes of waste and the completion of a highly simplified waste dump design;
- Generation of a conceptual mining study report.

1.5.2 Geotechnics

The following tasks were envisaged for the mining geotechnics to support a scoping level mining assessment:

- Review of the logging database and accompanying core photographs to understand the quantity and quality of the data being collected. The review has been focussed on the type of data being recorded to understand if it can be used to characterise the rock and generate rock mass ratings;
- Development of initial scoping level oxide and fresh rock pit slope angles based on SRK's understanding of the rock mass conditions, the available geotechnical data, and benchmarked against similar operations;
- Preparation of a short report section detailing conclusions on the quality and quantity of the data with recommendations on the geotechnical data required to be collected for a Pre-Feasibility Study.

1.5.3 Hydrogeology

SRK's has completed a high-level review of the available hydrology and hydrogeological data and provided recommendations for work required to move the project towards a Pre-Feasibility Study.

1.6 Declaration, Limitations and Cautionary Statements

SRK's estimates presented in this Report are based on information provided by the Company throughout the course of SRK's investigations, 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.

This report includes technical information, which 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.

1.7 Qualifications of Consultants

SRK is an associate company of the international group holding company SRK Consulting (Global) Limited. The SRK Group comprises over 1,400 staff, offering expertise in a wide range of resource engineering disciplines with 45 offices located on six continents. The SRK Group's independence is ensured by the fact that it holds no equity in any project. This permits the SRK Group to provide its clients with conflict-free and objective recommendations on crucial judgement issues.

The SRK Group has a demonstrated track record in undertaking independent assessments of resources and reserves, project evaluations and audits, Mineral Experts' Reports, Competent Persons' Reports, Mineral Resource and Ore Reserve Compliance Audits, Independent

Valuation Reports and independent feasibility evaluations to bankable standards on behalf of exploration and mining companies and financial institutions worldwide. The SRK Group has also worked with a large number of major international mining companies and their projects, providing mining industry consultancy service inputs.

2 HYDROGEOLOGY AND HYDROLOGY

2.1 Introduction

SRK have been requested to complete a high-level review of the available hydrology and hydrogeological data for the Sanankoro Project (the "Project"). This review should provide recommendations for work required to move the Project towards a Pre-Feasibility Study.

The recommendations made herein are based on the following assumptions:

- All mining slopes will be developed within the saprolite material (with a small section of saprock at the base);
- It is assumed that the saprolite pit slopes must be depressurised (in terms of pore water pressures) in order to achieve the currently planned pit slope angles.

2.2 Available Information

2.2.1 Site Data

SRK have received the following hydrogeological and hydrological data from Cora Gold relating to the Project:

- RC drilling programme groundwater strike database:

  "Plot_ML16_DH_WaterTable_31Aug19_1254 (2).xls"
- Site water supply well installation report:

  "RAPPORT DE FORAGE - SANANKORO - Mars 2018.pdf"
- Cora Gold responses to SRK hydrogeological and hydrological questions, submitted via email on 26/09/2019.

SRK have also reviewed and incorporated relevant information from previous reports prepared for Cora Gold:

- SRK 2018 Report: "30220 Sanankoro Exploration Target Report Final"
- SRK 2019 Report: "30487_Sanankoro_Model_Update_Memo"

2.2.2 Public Domain Data

Meteorological data. The National Oceanic and Atmospheric Administration database[1] ("NOAA") was accessed to review the publicly available meteorological data surrounding the project site. The closest meteorological station found was Siguri at 80 km, follow by Senou, at

[1] Accessed via https://www.noaa.gov/

100 km from the site. The Senou station contains the most complete data record in a radius of 200 km from the site.

River flow data. The Global Runoff Database[2] ("GRDC") was accessed to review the publicly available river discharge data surrounding the project site. A total of 20 regional river gauges were noted to surround the project site, but these all represent much larger watershed areas of up to 120,000 km^2. They are, therefore, not appropriate to utilise in an assessment of the project catchment characteristics. It was also noted that less than 60% of the data record was available, with a significant component of the daily and monthly information missing.

Hydrogeological data. The British Geological Survey ("BGS") Africa Groundwater Atlas[3] hydrogeological data was obtained from the public domain. This dataset includes regional scale characterisation of the major hydrogeological units and estimated borehole water yields (based on aggregated datasets). No other public domain hydrogeological data was obtained.

2.3 Preliminary Conceptual Model

2.4 Hydrology

In the south of Mali, where the Project is located, there are two distinct climatic seasons: a dry season lasting from mid-October to late-April, when virtually no rain falls and a rainy season from late-April to mid-October. Total annual rainfall for this region is around 1,200 mm per year, which is concentrated within the rainy season. Temperatures are high year-round (20-35°C), and peak at the end of the dry season where temperatures often exceed 40°C.

The physiography of the property is typically flat-lying with shallow topography although the area does include several hills with elevations of up to 410 m, around 40-50 m above the surrounding plains. Drainage is moderately well developed and typically flows to the west into the Niger River, generally as ephemeral streams that are dry for much of the year. Vegetation within the property typically consists of sparse trees and bushes.

A number of the proposed mining targets are within a close proximity to significant perennial river systems that are understood to over-top their banks during the wet season, namely the River Niger and the River Fie (see Figure 2-1). The proposed pit at Selin (in the north of the deposit) is located 1.5km southeast of the River Niger, and the proposed mining target at Zone A is located 2.0km northeast of the River Fie.

A high-level review of elevation differences at the project site relative to these river systems (utilising SRTM topography) suggests that direct flooding of the proposed pits is not expected. This should be confirmed with further work, however, utilising more detailed topography and characterisation of the rainfall-runoff relationship and estimated maximum river flood levels.

[2] Accessed via https://www.bafg.de/GRDC/EN/01_GRDC/13_dtbse/database_node.html
[3] Accessed via https://www.bgs.ac.uk/africagroundwateratlas/downloadGIS.html

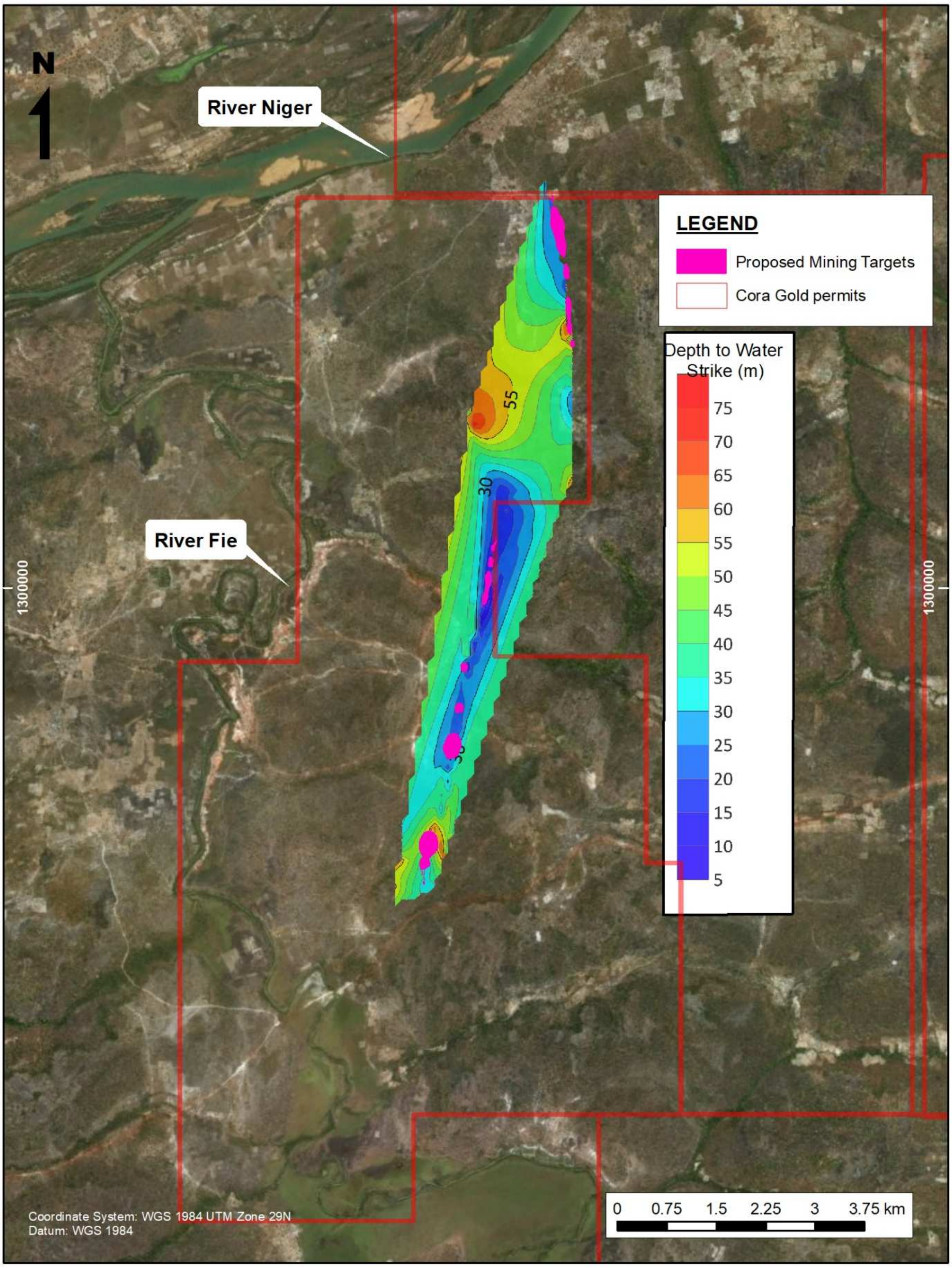


Figure 2-1: Sanankoro groundwater levels and hydrology map overview

Note: groundwater levels have been obtained from RC drilling water strike data only and may, therefore, represent the first intersection of mobile water in the saprock rather than the static piezometric (pore pressure) levels within the saprolite. Data points are limited to the proposed mining targets, with interpolation away from these targets no considered to be robust.

2.5 Hydrogeology

2.5.1 Groundwater Recharge

The process of rainfall and surface water infiltrating down through the soil and unsaturated units to the groundwater table is referred to as recharge. Groundwater recharge can dominate trends in groundwater regimes where the vertical permeabilities are high enough, with quick responses of groundwater levels to rainfall or large regional gradients. This is not expected to be the case at Sanankoro due to the low vertical permeability of the hard cap and clay-rich saprolite units.

Groundwater recharge is expected to occur predominantly along relic structures in the saprolite, which may offer a higher permeability pathway. The extent of recharge responses in the groundwater is not currently understood, as it requires a transient record of water level responses.

2.5.2 Key Geological Units

The mining concept considered in this scoping study involves excavation into the weathered oxide ores only, to include the saprolite and saprock. No mining is considered to advance into the fresh basement rock.

Overburden

The project site is overlain in part by alluvium and colluvium deposits, but these deposits are often absent (where the hard cap or saprolite outcrops). Where present, these deposits are between 1 to 2m thick and are expected to represent a higher permeability formation that should allow a faster recharge and storage of rainfall and surface waters into a perched system above the saprolite (as interflow). Where absent, rapid run-off and ponding of surface water is expected on top of the hard cap and saprolitic material, forming ephemeral drainage systems.

The potential for lateral hydraulic connection between the surface water systems and the proposed pits, via the overburden units, is considered to be limited. This is due to the thin nature of these deposits and their incomplete distribution.

Weathered bedrock

The weathered bedrock materials at Sanankoro are characteristic of a deep residual soil profile (or Saprolite). Saprolitic weathering typically results in an increasing grainsize profile with depth. The finer sized fraction is located near surface. This grainsize relationship is directly related to the increased mechanical and chemical weathering nearer the surface, with the chemical processes dominant due to infiltration and shallow groundwater flow systems.

The deep tropical weathering at Sanankoro has occurred to an average depth of 65m and a maximum depth of 125m. The weathering profile has been divided into the following units:

- Hard Cap (uppermost weathered unit)
- Saprolite
- Saprock (bottom weathered unit)

The upper unit of Hard Cap is observed to occur to an average depth of 5m and a maximum depth of 20m (based on RC drilling data). This unit is not noted to occur at all drilled locations at the site. Hydraulically, the hard cap is expected to exhibit a low permeability and storage capacity, and act as a barrier to vertical flows of water (i.e. limiting vertical recharge from rainfall events).

The Sanankoro deposit is characterized by a significant thickness of saprolite that is locally depressed along steeply dipping shear zones and is the dominant lithology in the proposed pits. The saprolite thickness ranges from 5m to 120m. The geotechnical variability of the saprolite units will likely result in challenging mining, stability and depressurization conditions. Saprolites, where unstructured and clay-rich, are typically low permeability and challenging to depressurise (when saturated).

Saprock units, which underly the saprolite, contain relict structures and foliations from the metasedimentary bedrock which may act as permeable flow pathways. The saprock is also found to be more competent and therefore brittle, with fracture zones likely to result in elevated permeabilities. This unit is expected to be the highest permeability unit intercepted by the mine and may provide an opportunity to underdrain the overlying clay-rich saprolite to reduce the pore-pressures.

Fresh bedrock

The Sanankoro property is underlain by several different Paleoproterozoic volcano-sedimentary formations (refer to A Mineral Resource Estimate on the Sanankoro Gold Project, Mali, December 2019 by SRK for detailed geology). These metasedimentary units have undergone significant deformation and are generally steeply dipping and folded. The permeability of the intact bedrock is expected to very low, with the bedrock considered as a fractured aquifer hydrogeological system, where groundwater flows are limited to the availability of interconnected fractures. Shears, fractures and faulting are understood to be common at the project site, and there is, therefore, a chance for higher permeability zones to be present within the bedrock units.

Structural Features

Structurally, the property includes mapped and inferred linear and curvilinear N-S and NE-SW orientated faults. The dominant form of structural development is shear / thrust fronts with secondary internal shear zones and local folding, most of which are now steeply dipping.

The deposit broadly occurs within 3 parallel planar zones that dip steeply to the east at approximately 70^{o}. Broadly speaking, these structures are characterised by steeply dipping sandstone, siltstone and phyllite sequences

The hydrogeological significance of these large structural features needs to be determined, but there is a potential for the shear zones to act as either a barrier to flow (if fractures are healed by mineralisation) or as a conduit to flow (if the fractures remain open). As the mine plan does not intercept the bedrock, the characterisation of the shear zones within the saprock is considered to be critical.

2.5.3 Groundwater Levels

Groundwater level information for the Sanankoro property is available from RC drilling water strike data, collected during the period December 2017 to June 2019. A total of 190 boreholes have been included in the database. These water strike data are likely to represent the first intersection of mobile water in the saprock rather than the top of the saturated formations in the saprolite, due to the low permeability expected in the clay-rich saprolite unit.

The observed depth to water is highly variable, and ranges from 7 to 78m below the ground surface (Figure 2-1). The shallowest water strikes appear to occur in the centre of the property, at Zone B and Zone B North, but the variation in observed depths within these locations is also noted to be high.

The only direct static water level measurement obtained at the site is available from the water supply well, which was recorded as 16m below ground level (on 05/04/2018). This borehole was drilled to a final depth of 105m and had the screened casing sections in the fissured and fractured argillite bedrock.

A comparison between the depth to water and the combined thickness of the saprolite and saprock at each location demonstrates a weak trend of deeper water strikes with a greater thickness of the weathered units (Figure 2-2). Whilst the correlation is not strong, this is assumed to indicate that where a thicker weathered zone is present, the saprolite unit is proportionally thicker and the saprock (where permeability is expected to be higher) is therefore intercepted at a greater depth.

The intensity of water returns following the water-strike during RC drilling has also been recorded, as a qualitative observation ranging from a low intensity (category “0”) to a higher intensity (category “4”). No correlation between the depth of water-strike or mining target and the water strike intensity has been noted, but it is assumed to be dependent on the level of fracturing within the saprock unit intercepted.

The spatial distribution and intensity of water level strikes is illustrated as a series of cross-sections in Figure 2-3.

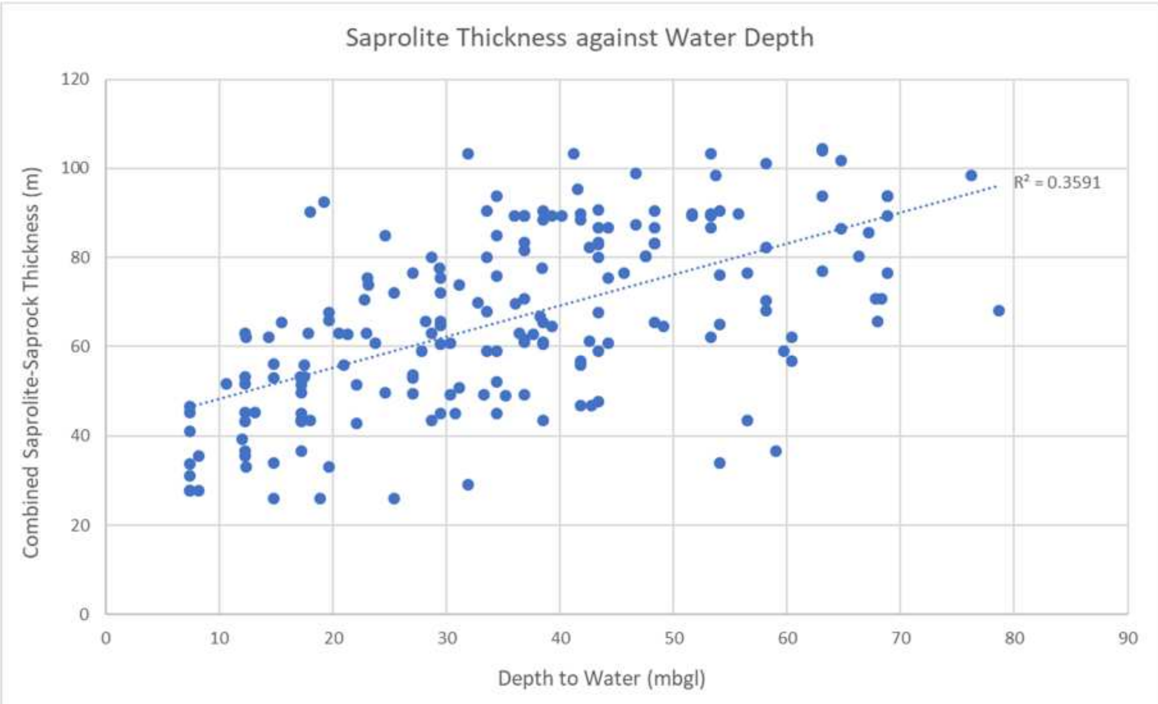


Figure 2-2: Sanankoro groundwater depth against weathered unit thickness

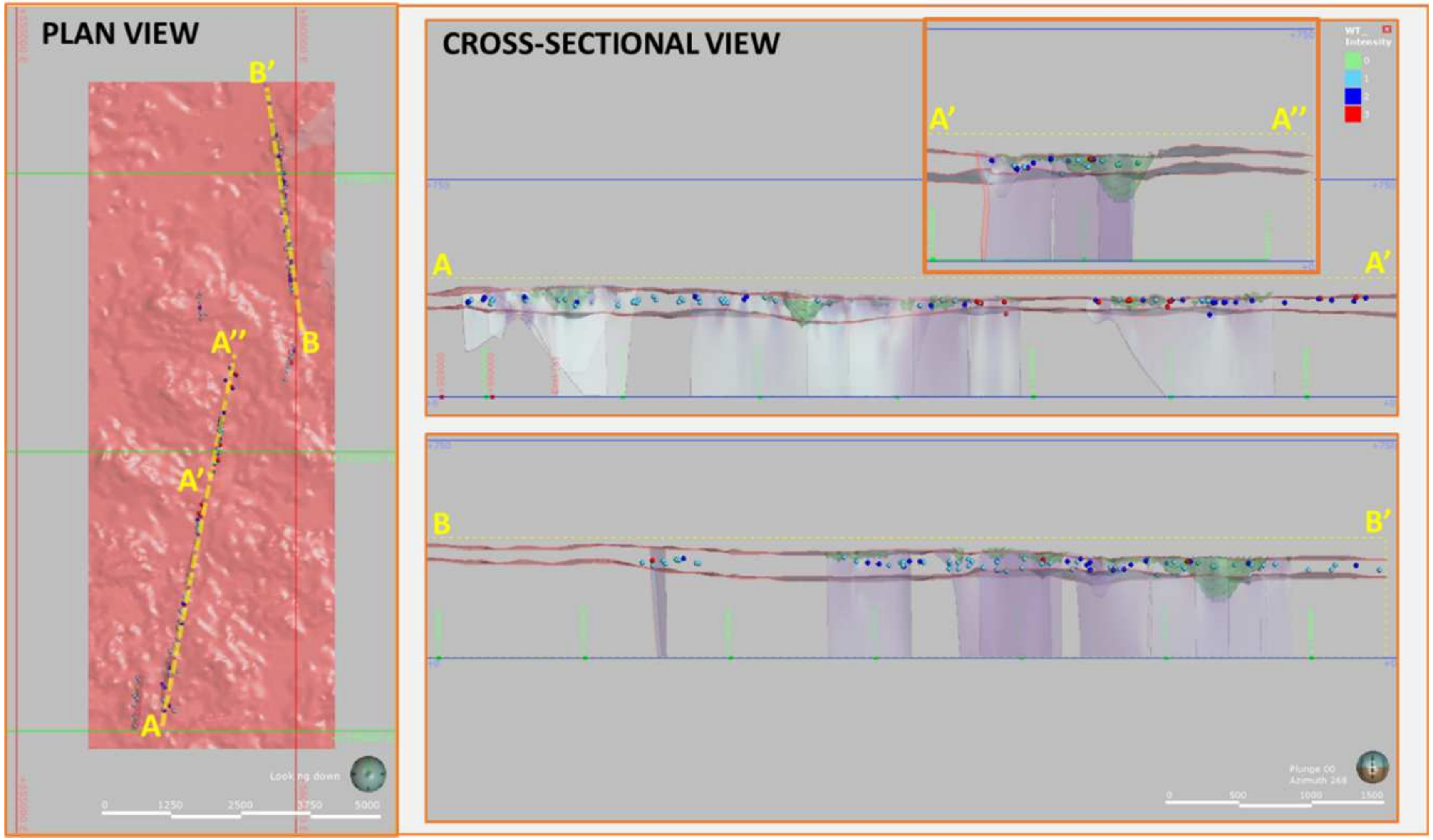


Figure 2-3: Observed groundwater strike depths and intensity within the weathered units

Note: Water level points are coloured by the observed flow rate intensity during RC drilling. Green represents a low intensity and red a high intensity (these observations are not quantified by flow rates)

2.5.4 Hydraulic Properties

The only hydraulic test work that has been completed at the project site was during the installation of a water supply borehole. This borehole was drilled to a depth of 105m and was screened between 54 and 104m (to allow inflows between these depths) and is, therefore, assumed to represent the hydraulic properties of the bedrock formations at this location.

A step-drawdown test was conducted by the contractor following the installation of this pumping well, to estimate the optimum long-term yield of the borehole. A maximum pumping rate of 3.6 m^3/h was used during this test, which resulted in a drawdown of the water level of 45m. It is not known what sustained yield has been achieved by this production well whilst it has been in operation, but this is considered to represent a low-yielding borehole.

Analysis of the pumping test data by the same contractor concluded an aquifer transmissivity of $1.3e^{-5}$ m^2/s, which equates to a hydraulic conductivity of $3e^{-7}$ m/s (based on a saturated thickness of 50m). This is considered a median value for a fractured igneous or metamorphic rock.

The BGS Africa Groundwater Atlas dataset defines the regional hydrogeology surrounding the project site as comprising crystalline basement aquifers with typical weathered/fractured aquifer properties. This dataset also suggests the aquifer productivity category for the site as moderate, with estimates of potential borehole yields to be 7 to 18 m^3/hr. This is noted to be slightly higher than the flow rates observed from the exploration site production well.

2.6 Water Management Design Considerations

2.6.1 Hydrology

The key hydrological risks that are considered likely to have a potential for impact on the water management design are:

- Rivers Niger and Fie flood events, and over land connection with the mining operation; and
- Rapid run-off of surface water during high intensity rain events, impacting the mining operation via direct flooding of pits and indirect recharging of pit slope pore pressures (where paleo-structures are present in the sap, resulting in a reduced FoS).

2.6.2 Hydrogeology

The key hydrogeological risks that are considered likely to have a potential for impact on the water management design are:

- Hydraulic connection between unconsolidated overburden units and the adjacent river systems resulting high pit inflows, especially after rainfall events;
- Hydraulic connection between saprock units and the adjacent river systems;
- Elevated pore water pressures in the saprolite and saprock pit slopes leading to an inability to meet slope design angles; and
- Hydraulic properties of the saprolite, saprock and underlying fresh bedrock, are not conducive to depressurisation.

3 GEOTECHNICAL

3.1 Introduction

SRK have been requested to provide slope angles for the open pits associated with the Sanankoro project to allow inputs for scoping study level pit optimisation. The slope angles provided have been based on the following assumptions:

- All mine slopes will be formed within saprolite (with a small section of saprock at the base)
- Maximum slope height will be in the region of 100 – 125 m (based on weathering surfaces developed by SRK of the drillhole regolith logging completed by Cora Gold geologists)

3.2 Available Information

SRK have not been provided with any geotechnical information for the saprolite material. However, in relation to the underlying rock, the following information has been received.

- Core photos – There are limited photos of the saprolite although saprock, transitional rock and fresh rock photos have been provided.
- Rock mass logging of fresh rock for twenty-two boreholes
- Structural logs (with alpha/beta measurements) of three boreholes

Given that oxide pits are the primary focus and no geotechnical information is available for the saprolite, SRK do not consider it appropriate to undertake detailed analytical work at this stage of the project.

3.3 Saprolite Slope Design Considerations

As a result of the lack of geotechnical information with regards the conditions of the saprolite, SRK make the following considerations:

- The saprolite slope angles presented herein are based on experience of designing slopes in (likely) similar materials and engineering judgement.
- The weathering profile at Sanankoro is transitional, from surface hardcap, through saprolite and saprock down to the unweathered fresh material;
- The Cora Gold regolith logging indicates that total depth of the weathering profile at Sanankoro ranges from 30 m to 125 m, with an average depth of 60 – 70 m. The primary component of the weathered package is saprolitic material which, on average, extends to 55 m below surface.
- Given the thickness of saprolite, SRK would recommend that the sensitivity to geotechnical slope angle is tested at a range of pit slope angles between 26° and 38°. After discussions with the Cora Gold technical team, **it has been decided to complete the final pit shell optimisation on the Sanankoro block model at a slope angle of 34° in the saprolite material**. SRK consider this angle to be optimistic, but appropriate for the level of study;
- It is likely that developing overall saprolite slope angles in excess of 30° for ~100m high slopes will lead to single and possibly multi-bench instability. Such instabilities may have a negative effect on production and will need to be managed by implementation of appropriate Ground Control Management Plans.

- The key drivers for the design of the saprolite slopes will be:
  - The residual strength of the saprolite.
  - The presence of relict structure within the saprolite. Whilst residual strength will control 'mass' instability, relict structure within the saprolite can also give rise to structural instability along faults, joints or fabric. Given the highly laminated nature of the protolith, it is possible there will be relict structure present within the saprolite.
  - The presence of pore pressure within the slope can reduce saprolite strength and result in instability.
- The upper limit of the saprolite slope angles presented above will only be achievable if the following are implemented:
  - Pit slope depressurisation, which will likely be in the form of horizontal dewatering wells, in-pit sumps and possible undraining by mining into the underlying fresh material as early as possible.
  - The success of the depressurisation will be controlled by the mineralogy/clay content of the saprolite.
  - Implementation of effective surface water management plans to minimise the effects of transient pore pressure changes as a result of seasonal weather conditions.
- Given the extremely long strike length of the deposits, it may be possible to develop a mining method that reduces the length of exposed saprolite slope and allows for progressive backfill.
- Regardless of the overall slope angle, bench scale failure will always be likely.
- It may be possible to 'manage' slope failures if a robust Ground Control Management Plan is developed and implemented. If so, it may be acceptable for the client to approve a slightly higher risk profile for the slope design.
- It may be possible to have a slightly steeper slope design in the saprock. It is understood that the saprock zone is, on average, approximately 15 m thick, but can be up 50m thick in places. Whilst there is no geotechnical data for the saprock, it is proposed that **an overall saprock slope angle of 40° is considered reasonable** (although this could vary depending on the thickness of the saprock)

Figure 3-1 and Figure 3-2 present typical intersections of saprolite and saprock.

Figure 3-1: Typical saprolite intersection

Figure 3-2: Typical saprock intersection

3.4 Rock Slope Design Considerations

While the focus of the geotechnical analysis completed by SRK has been providing optimisation angles for the saprolite and saprock slopes, SRK have also provided rock slope angles for use in pit optimisation of the fresh rock. Whilst at this stage of the project, no analytical work has been undertaken, given the information available, SRK present the following recommendations:

- The rock below the saprock boundary appears to be highly variable. Whilst some core photographs show a competent rock mass which could be considered to have high RMR values, other boreholes show a highly altered rock mass which would have very low to low RMR values. In addition to the highly altered rock, there are sections of 'rock' material that are highly fractured and would represent low RMR material.
- The spatial distribution of such material needs to be understood as the project develops.
- For the fresh rock slope SRK recommend that pit optimisation is undertaken on a slope

angle of 42°

Figure 3-3 and Figure 3-4 show the variability in the rock mess below the oxide zone with material; varying from very competent strong rock to highly altered, very weak rock.

Figure 3-3: Typical fresh rock intersection (laminated siltstone)

Figure 3-4: Poor quality fresh rock transitioning into highly altered rock

4 MINING

4.1 Objectives

The objectives of the mining study have been to describe a preferable mining method for the operation and assess future mining costs for the Project, based on the current level of geological understanding. The mining study considers a multi-pit project, containing four zones:, Zone A, Zone B, Zone B North and Selin. The results of the mine planning and mining cost estimations have been used for providing recommendations to guide future exploration and further mine planning activities for the Project.

4.2 Dilution and Recovery

The mineral resource model for all zones have been regularised to a block size of 2.5 x 2.5 x 5 m to incorporate mining dilution for use in pit optimisation and mine planning. This regular block size is known as the minimum Selective Mining Unit ("SMU") and the regularised model as the Mining Model. The estimation of dilution on a local level is achieved through the regularisation process and is an important step in the mine planning process to estimate the likely diluted tonnes and grade.

A block size of 2.5 x 2.5 x 5 m is considered representative of the SMU size estimated for small scale mining equipment (1.9 to 4 m^3 bucket excavators, 24 to 40 t capacity haul trucks), and requires a relatively high level of selectivity.

The orebody width (above a 0.4 g/t Au cut-off) ranges from 1.5m to 26 m. Therefore, in order to avoid excessive levels of dilution, a relatively selective mining method will need to be implemented.

4.2.1 Regularisation Results

Above a marginal cut-off of 0.4 g/t Au, the mining recovery in all zones is between 91% and 95% and the dilution is estimated between 14% and 20%. The regularized recovery and dilution at a marginal cut-off grade are calculated using the following formulas:

$$Recovery\ \% = \frac{\text{Regularized Metal Content}}{\text{Resource Metal Content}}$$

$$Dilution\ \% = \frac{\text{Regularized Cumulative Tonnes}}{(\text{Resource Cumulative Tonnes} * \text{Regularized Recovery \%})} * 100$$

A summary of the diluted model tonnage and grade above 0.4 g/t Au can be found in

Table 4-1. These factors are regarded as reasonable given thickness (1.5 m to 26 m) and geometry of the orebody. The comparison includes all mineralisation classifications (inferred and unclassified).

Histograms comparing the mineral resource models and diluted model tonnage above a range of cut-offs are shown for each zone from Figure 4-1 to Figure 4-4 (not constrained within any pit shell).

Table 4-1: Summary Global Mineral Resource and Mining Block Model Inventories

	Unit	Selin	Zone B North	Zone A	Zone B
Cut-Off	(g/t Au)	0.40	0.40	0.40	0.40
In Situ Tonnage	(Mt)	8.4	3.6	6.2	15.0
	(g/t Au)	1.4	1.0	1.2	1.1
	(koz Au)	373.4	119.2	231.0	510.2
Diluted Tonnage	(Mt)	9.5	3.9	6.4	16.0
	(g/t Au)	1.2	0.9	1.0	0.9
	(koz Au)	354.2	111.6	209.9	470.9
Dilution	(%)	120	115	114	116
Recovery	(%)	95	94	91	92

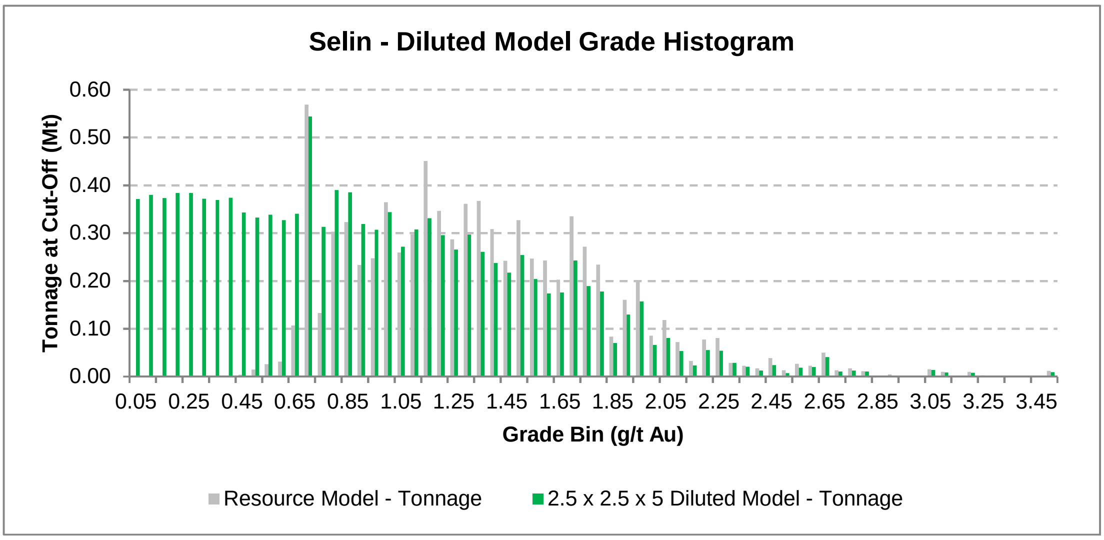


Figure 4-1: Selin - Diluted Model Grade-Tonnage Histogram

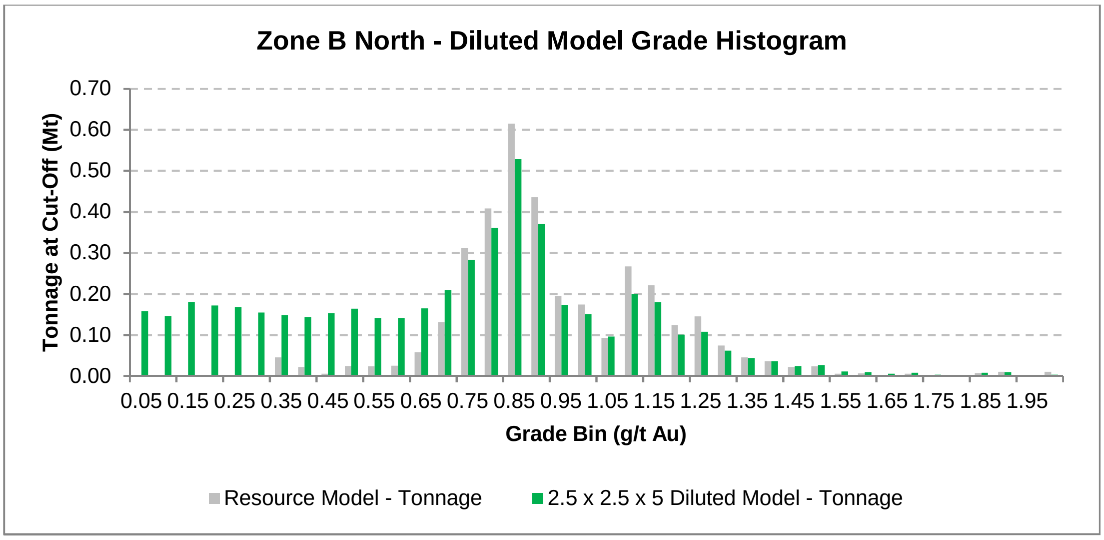


Figure 4-2: Zone B North - Diluted Model Grade-Tonnage Histogram

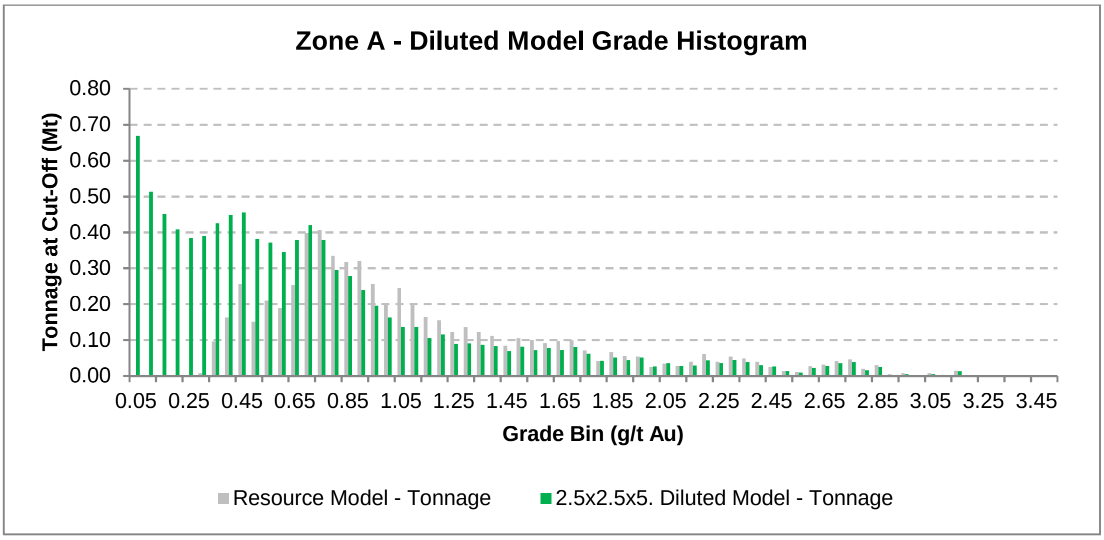


Figure 4-3: Zone A - Diluted Model Grade-Tonnage Histogram

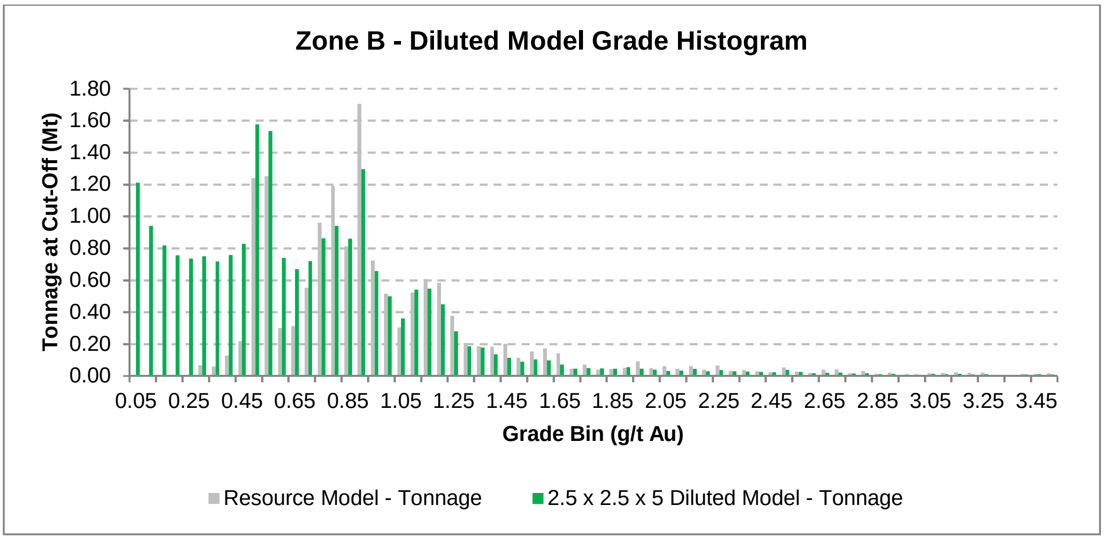


Figure 4-4: Zone B - Diluted Model Grade-Tonnage Histogram

4.3 Open Pit Optimisation

4.4 Approach

In order to assess the open pit potential for the Sanankoro deposit, SRK has undertaken open pit optimisation using NPVS software. NPVS uses the Lerchs-Grossmann algorithm for determining the shape of an optimal pit using a set of techno-economical input parameters.

The optimisation process produces a series of “nested” pit shells, where each shell provides the mining inventory which generates the maximum undiscounted cashflow (excluding capital costs) at a given metal price. The nested pit shells provide an indication of the sensitivity of the deposit at various metal prices given the same input costs and modifying factors. Revenue Factor (“RF”) is the factor by which the revenue for each block is scaled in order to produce one of the nested pit shells.

The nested pit shells were evaluated at metal prices of USD1,300/oz Au and USD1,500/oz Au, with USD1,500/oz Au selected as the base case metal price.

The objective of the open pit optimisation has been to assess the potential economic pit extents, understand the physical characteristics of the deposit, and to select a pit shell to use as the basis for the strategic scheduling.

The optimisation process has included Inferred classified material only, as the resource model does not contain Indicated or Measured material. The pit optimisation used in the mining study only considered oxide material (hardcap, saprolite and saprock). An alternate pit optimisation was completed to consider both oxide and sulphide (fresh) mineralisation for the purposes of producing an optimised pit shell for the reporting of a Mineral Resource Estimate. However, the results presented and discussed in the mining study only relate to the oxide material.

The results of the pit optimisation are presented in the following sections.

4.5 Pit Optimisation Parameters

The pit optimisation parameters for the base case pit optimisation (USD1,500/oz) are shown in Table 4-2 and are discussed below. The optimisation parameters outlined in this section include recoveries, costs and slope angles for fresh rock (as an alternate pit optimisation was completed on both the oxide and fresh rock for the purposes of Mineral Resource reporting), however it should be stressed that the pit optimisation employed in the mining study considered only oxide material.

4.5.1 Mining Model

The 2.5 m x 2.5 m x 5 m diluted model has been used for the pit optimisation and no additional mining dilution or losses have been applied. This is regarded as an appropriate approach for incorporating dilution on a local level based on the selectivity level of the current and planned equipment size.

The pit optimisation cases were completed on the inferred classified material and the top three weathering domains only (hardcap, saprolite and saprock). The fresh weathering state is not included within the mining study.

4.5.2 Geotechnical

Geotechnical slope angles have been applied separately to the weathering domains (hardcap, saprolite, saprock and fresh), and are based on preliminary slope angles assumed by SRK as part of the Scoping Study. An example west-east cross-sectional view of the geotechnical domains is shown in Figure 4-5, along with the Selin resource pit shell. It can be noted that, within Selin, the hardcap domain extends up to 10 m depth in the west (to 330 m RL) and 5 m in the east, comprising 7% of the resource. The saprolite and saprock domains extend to 70 m depth (260 m RL) and account for 82% of Selin’s resource pit shell. The remaining 11% consists of the fresh weathering domain. The same geotechnical domaining was used across Zone A, Zone B, Zone B North and Selin.

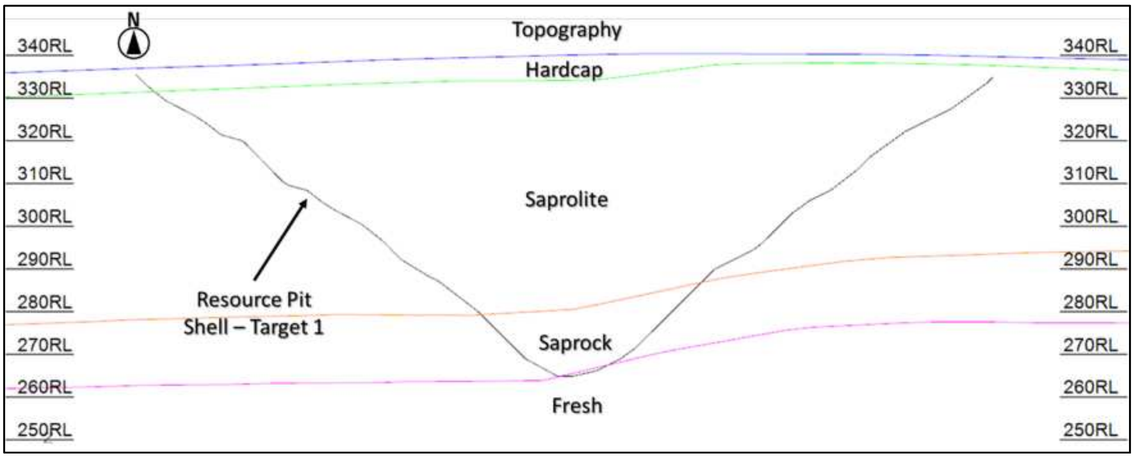


Figure 4-5: Geotechnical Domains

4.5.3 Processing Recoveries

The processing recoveries provided to SRK by Wardell Armstrong vary depending on the weathering domain and the mining zone. Processing recoveries applied for the hardcap and fresh material are 80%, whilst the processing recoveries applied to the saprolite and saprock are 92.9% for Zone A and Zone B and 95.7% for Zone B North and Selin.

4.5.4 Mining Cost

The mining operating cost estimations were assigned per rock type (ore, waste) and weathering domains (hardcap, saprolite, saprock, fresh). All mining cost assumptions are based on SRK's experience of mining costs achieved at open pit operations in Mali. The mining costs used for ore and waste in the top weathered domains (hardcap and saprolite) were USD3.50/t and USD3.00/t, respectively. These mining costs were increased by USD0.50/t, to USD4.00/t in ore and USD3.50/t in waste, to account for the change in extraction method from free digging to drill and blast in the less weathered domains (saprock and fresh).

It is recommended that future pit optimisations use the latest estimates of mining costs estimated in this Scoping Study or any new quotes that the Client may obtain for the Project following conclusions of this Study.

4.5.5 Processing Costs

The pit optimisation was tested on three production rates including 0.5 Mtpa, 1.0 Mtpa, and 1.5 Mtpa recognized in this study as Case 1, Case 2 and Case 3, respectively. The three cases contain unique processing rates resulting in slightly varying processing costs and related marginal cut-off grades. The base-case is the Case 2 pit optimisation (1.0Mtpa). The processing costs provided by Wardell Armstrong vary depending on the weathering state and the production rate. The processing cost ranged between USD14.70/t and USD17.00/t, increasing with depth and decreasing at higher production rates.

4.5.6 General and Administration Costs

A fixed cost assumption for general and administration (G&A) of USD2.00/t was applied to ore tonnage and used along with processing cost to calculate the marginal cut-off USD/$t_{ore.}$

4.5.7 Selling Costs

The fixed cost assumption for selling cost of 5% accounts for 3% royalties stated in the 2012 Mining code in Mali plus an additional 2% to account for overhead costs.

4.5.8 Cut-Off Grade

Based on the input parameters used for the pit optimisation, the marginal cut-off grade is estimated at 0.4 g/t Au for saprolite, saprock and hardcap and 0.5 g/t Au for fresh rock. For strategic scheduling purposes, all mineralisation (classified as Inferred) above a marginal cut-off of 0.4 g/t Au has been treated as diluted ore ton. The marginal cut-off grades have been calculated with the following formula:

$$\boldsymbol{Marginal\ Cut-off\ Grade} = \frac{(\mathbf{c})}{(\mathbf{y}(\boldsymbol{s}-\mathbf{r}))}$$

Where,

- r = Selling unit cost
- c = cost to process ore
- y = metal recovery
- s = unit metal sale price

Table 4-2: Pit Optimisation Parameters

Parameters	Units	Case 1	Case 2	Case 3	Comments
Production					
Production Rate - Ore	(tpa)	500,000	1,000,000	1,500,000	Cora Gold Assumption
Geotechnical					
Overall Slope Angle - Saprolite	(˚)	34	34	34	SRK Assumption
Overall Slope Angle - Saprock	(˚)	40	40	40	SRK Assumption
Overall Slope Angle - Fresh	(˚)	42	42	42	SRK Assumption
Mining Factors					
Dilution	(%)	Regularised Block Model 2.5x2.5x5m			See Section 4.2 for details
Recovery	(%)	Regularised Block Model 2.5x2.5x5m			See Section 4.2 for details
Processing					
Hardcap - All Zones	(%)	80.0	80.0	80.0	WA Assumption
Zone A/B (sap/saprock)	(%)	95.7	95.7	95.7	WA Assumption
Selin + Zone B North (sap/saprock)	(%)	92.9	92.9	92.9	WA Assumption
Fresh - All Zones	(%)	80.0	80.0	80.0	WA Assumption
Operating Costs					
Mining Cost - Ore					
Saprolite	(US$/$t_{ore}$)	3.50	3.50	3.50	SRK Assumption
Sap Rock & Fresh	(US$/$t_{ore}$)	4.00	4.00	4.00	
Mining Cost - Waste					
Saprolite	(US$/$t_{waste}$)	3.0	3.0	3.0	SRK Assumption
Saprock & Fresh	(US$/$t_{waste}$)	3.50	3.50	3.50	
Processing - Saprolite, Saprock, Hardcap	(US$/$t_{ore}$)	16.2	15.5	14.7	WA Assumption
Processing - Fresh	(US$/$t_{ore}$)	17.0	17.0	17.0	WA Assumption
G&A	(US$m/Year)	1.0	2.0	3.0	
	(US$/$t_{ore}$)	2.0	2.0	2.0	WA Assumption
Selling Cost Au	(%)	5.0	5.0	5.0	

	(US$/oz) (US$/g)	85.0 2.5	85.0 2.5	85.0 2.5	SRK Assumption, explained in Section 4.5.7 in more detail
Metal Price					
Gold	(US$/oz) (US$/g)	1,500.0 43.8	1,500.0 43.8	1,500.0 43.8	Cora Gold Assumption
Other					
Discount Rate	(%)	10.0	10.0	10.0	SRK Assumption
Cut-Off Grade					
Marginal - Saprolite, Saprock, Hardcap	(US$/$t_{ore}$) (g/t Au)	18.2 0.4	17.5 0.4	16.7 0.4	
Marginal - Fresh	(US$/$t_{ore}$) (g/t Au)	19 0.5	19 0.5	19 0.5	

4.6 Pit Optimisation Results

The pit optimisation was performed on all cases, however only the results of Case 2 (1.0Mtpa) will be discussed in this Section. Summary tables for the pit optimisation results for each zone and case can be found in Appendix A.

The optimisation results are presented in a metal price sensitivity graph, which shows how the physical characteristics of the deposit are sensitive to various metal prices.**Error! Reference source not found.** Figure 4-6 to Figure 4-9 **Error! Reference source not found.**show the metal price sensitivity results for the base case metal price (USD1,500/oz), comparing how diluted tonnes, grade and strip ratio are affected within each of the zones. The metal price sensitivity graphs for Case 1 and Case 3 can be found in Appendix B.

4.6.1 Diluted Ore Tonnage

The diluted ore tonnage for all zones increases gradually as the metal price increases, and no significant step increases are seen at metal prices above USD1,500/oz Au. However, a notable increase is observed at Selin, where the diluted ore tonnage increases rapidly by 400kt between USD600/oz Au and USD650/oz Au. This increase indicates that the ore inventory is less sensitive to metal price over USD700/oz although remains quite sensitive in Selin (the curves are linear but also relatively steep).

4.6.2 Strip Ratio

Strip ratio for all zones increases gradually as the metal price increases, with the exception of Zone A and Zone B. The strip ratio in Zone A is variable and sensitive to metal price between USD800/oz Au and USD1,200/oz Au, however it later stabilises and gradually increases with metal price. The strip ratio in Zone B increases from 9.2 to 11.4 (t:t) between USD1,300/oz Au and USD1,500/oz Au. The strip ratio decreases and stabilizes at metal prices above USD1,500/oz Au, indicating that the strip ratio is more sensitive to metal prices below USD1,500/oz Au in Zone B. These metal price sensitivities are not observed in the other zones.

4.6.3 Diluted Grade

The diluted grade decreases gradually as metal price increases for each zone. No notable step changes are observed, which implyies that diluted grade is not sensitive to metal price variability.

4.6.4 Pit Shell Selection

Table 4-3 summarises the inventories for the USD1,300/oz Au and USD1,500/oz Au pit shells at RF=100%. The pit optimisation results show that the mining inventory grows with increasing

RF on a broadly linear basis. That said, it is noted that the total ore tonnage is relatively sensitive to the gold price selected for the pit optimisation. The total ore tonnage inside of the USD1,300/oz pit shell is 2.8 Mt at 1.60 g/t Au, whilst the total ore tonnage inside of the USD1,500/oz pit shell is 4.1 Mt at 1.47 g/t Au. This represents a 46% increase in ore tonnage and 35% increase in contained ounces in the USD1,500/oz pit shell, compared to the USD1,300/oz pit shell. Additionally, it is noted that the USD1,500/oz pit shell includes a 10.1 Mt increase in waste tonnes compared to the USD1,300/oz pit shell, resulting in an increase of strip ratio from 5.1 to 5.9 (t:t). After discussions between Cora Gold and SRK, Cora Gold requested that SRK use the USD1,500/oz Au pit shell (RF=100%) for the development of the strategic schedule. This is considered acceptable at a scoping level, however the sensitivity to Au price should be carefully considered as the Project develops.

Table 4-3: Pit Optimisation Results – USD1,300/oz Au vs. USD1,500/oz Au

Gold Price	Zone	Rock (Mt)	SR (t:t)	Waste (Mt)	Ore (t)	Grade (g/t Au)
1500	Zone A	9.33	6.1	8.00	1.4	1.43
	Zone B	7.72	11.4	7.10	0.6	1.64
	Zone B North	1.95	3.6	1.50	0.4	1.00
	Selin	9.39	4.7	7.70	1.7	1.55
	Total	**28.4**	**5.9**	**24.3**	**4.1**	**1.47**
1300	Zone A	6.07	5.7	5.17	0.9	1.62
	Zone B	3.09	8.9	2.78	0.3	1.80
	Zone B North	0.88	2.8	0.65	0.2	1.05
	Selin	6.99	4.1	5.63	1.4	1.64
	Total	**17.04**	**5.1**	**14.2**	**2.8**	**1.60**

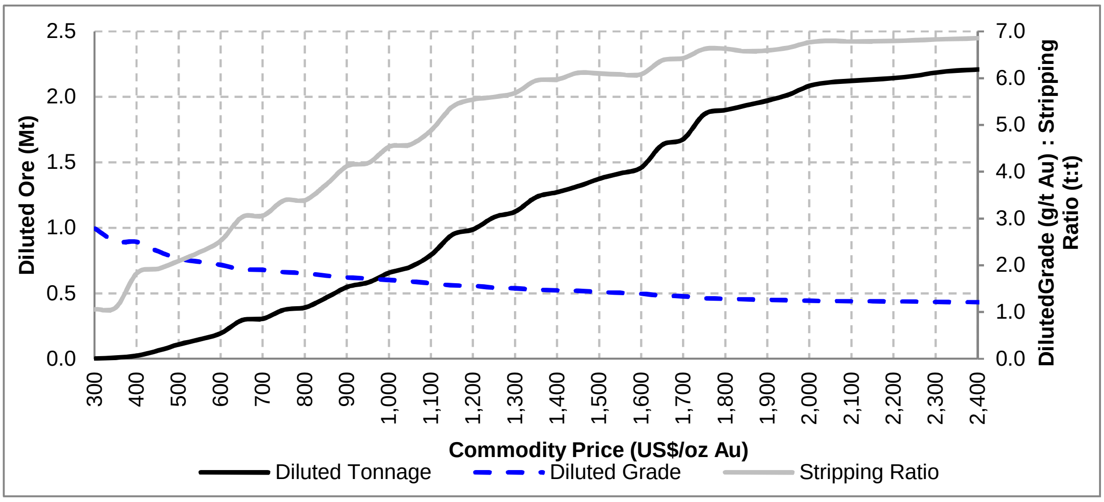


Figure 4-6: Zone A - Metal Price Sensitivity

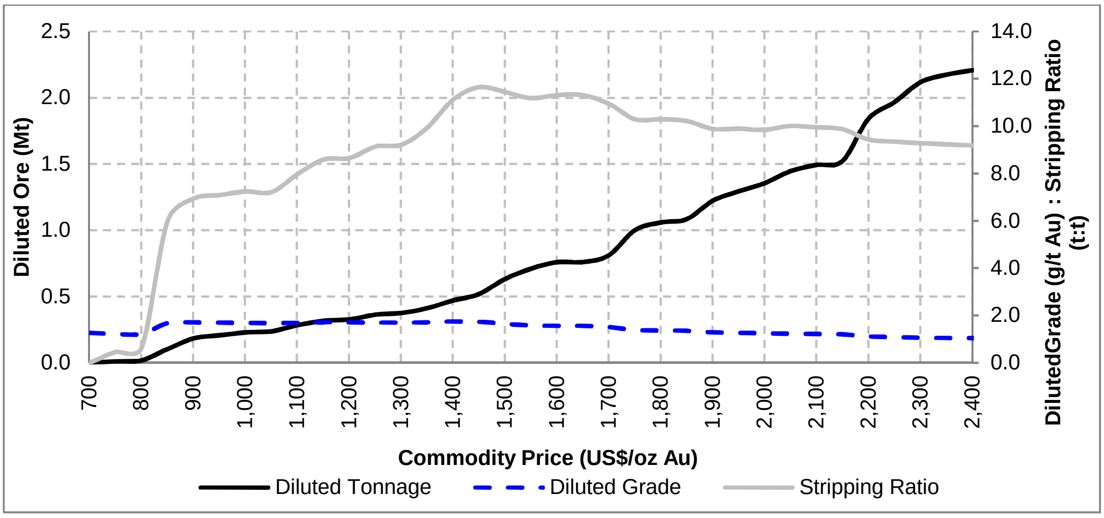


Figure 4-7: Zone B - Metal Price Sensitivity

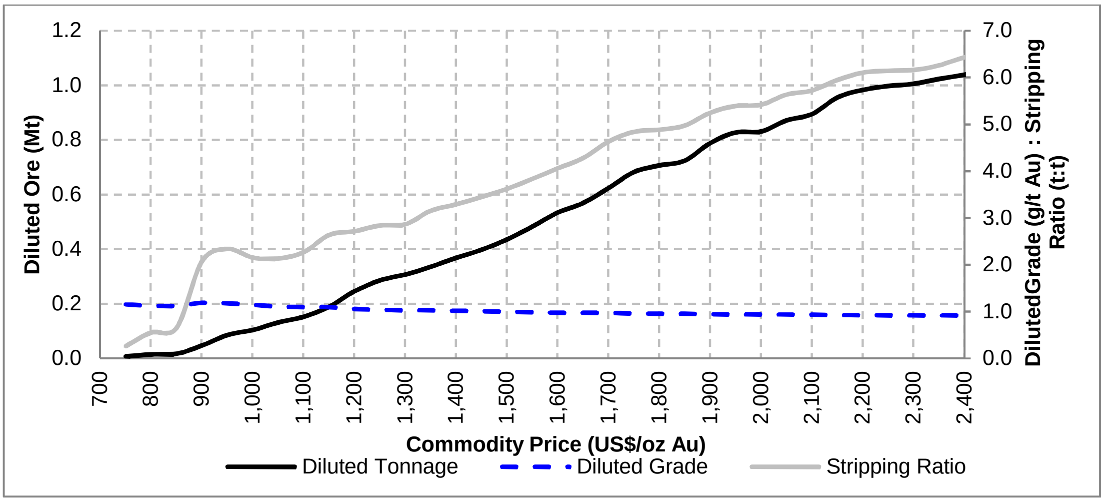


Figure 4-8: Zone B North - Metal Price Sensitivity

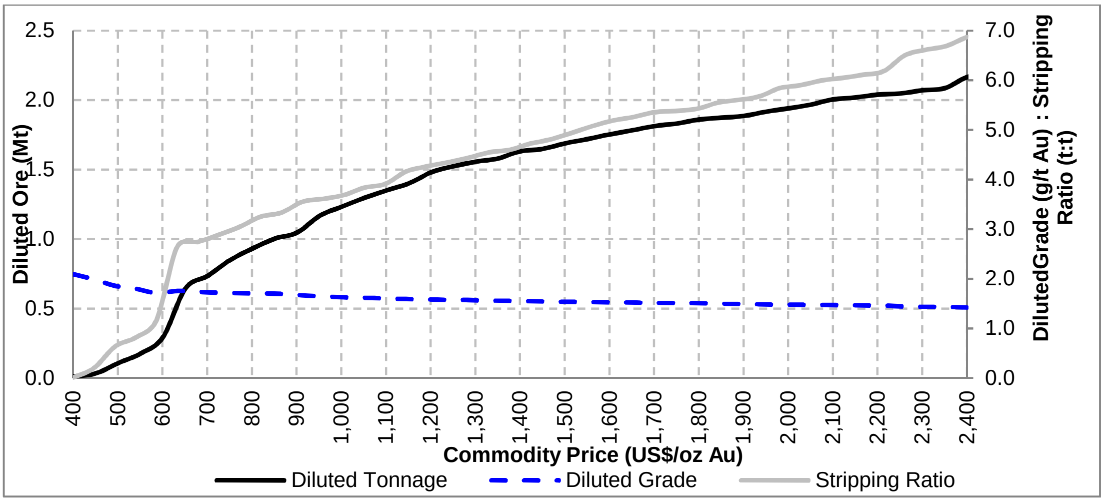


Figure 4-9: Selin - Metal Price Sensitivity

4.6.5 Sensitivity Analysis

Pit optimisation has been run at a range of slope angles to determine the economic and practical implications of a range of the geotechnical parameters. The results of the slope angle sensitivity are summarised for the USD1,500/oz Au pit shell in Table 4-4.

The slope angle sensitivity analysis was completed on each zone for Case 2 (1 Mtpa). The pit optimisation results are reasonably sensitive to slope angle due to the relatively high strip ratio of the pits. It is considered that slope angle is likely to have a significant impact on the quantity of waste mined and overall mining costs. The case chosen for strategic scheduling was "Case 2C" agreed between SRK and the Client based on the geotechnical considerations provided by SRK. The selected case is highlighted in Table 4-4.

Table 4-4: Pit Optimisation Sensitivity Results

Scenario No.	Zone	OSA			Strip Ratio	Ore Insitu	Grade
		Saprolite	Saprock	Fresh	t:t	Mt	g/t Au
Case 2A	Zone A	30	40	42	7	1.5	1.38
	Zone B	30	40	42	13	0.7	1.58
	Zone B North	30	40	42	4	0.5	0.97
	Selin	30	40	42	5	1.7	1.54
Case 2B	Zone A	26	40	42	8	1.3	1.41
	Zone B	26	40	42	12	0.3	1.67
	Zone B North	26	40	42	4	0.4	0.99
	Selin	26	40	42	6	1.6	1.56
Case 2C	Zone A	34	40	42	6	1.6	1.35
	Zone B	34	40	42	11	0.8	1.52
	Zone B North	34	40	42	4	0.6	0.97
	Selin	34	40	42	5	1.8	1.53

4.7 Pushbacks

The USD1,500/oz Au pit shell has been divided into a series of phases called pushbacks. Several pushbacks have been defined for each zone, depending on their size and geometrical characteristics. Table 4-5 and Figure 4-10 to Figure 4-12 provide the results of the pushback generation.

Table 4-5: USD1500/oz Pit Shell Pushbacks

Zone	Pushbacks	Rock	Total Ore	Total Waste	Strip
		Mt	Mt	Mt	t:t
Zone A	1	3.61	0.52	3.09	5.9
	2	4.07	0.52	3.55	6.8
	3	1.65	0.34	1.32	3.9
Total Zone A		9.33	1.38	7.96	5.8
Zone B	1	7.72	0.63	7.08	11.2
Zone B North	1	1.95	0.44	1.51	3.5
Selin	1	4.26	0.69	3.57	5.2
	2	2.88	0.48	2.40	5.0
	3	1.38	0.30	1.08	3.7
	4	0.87	0.22	0.65	2.9
Total Selin		9.39	1.69	7.7	4.6

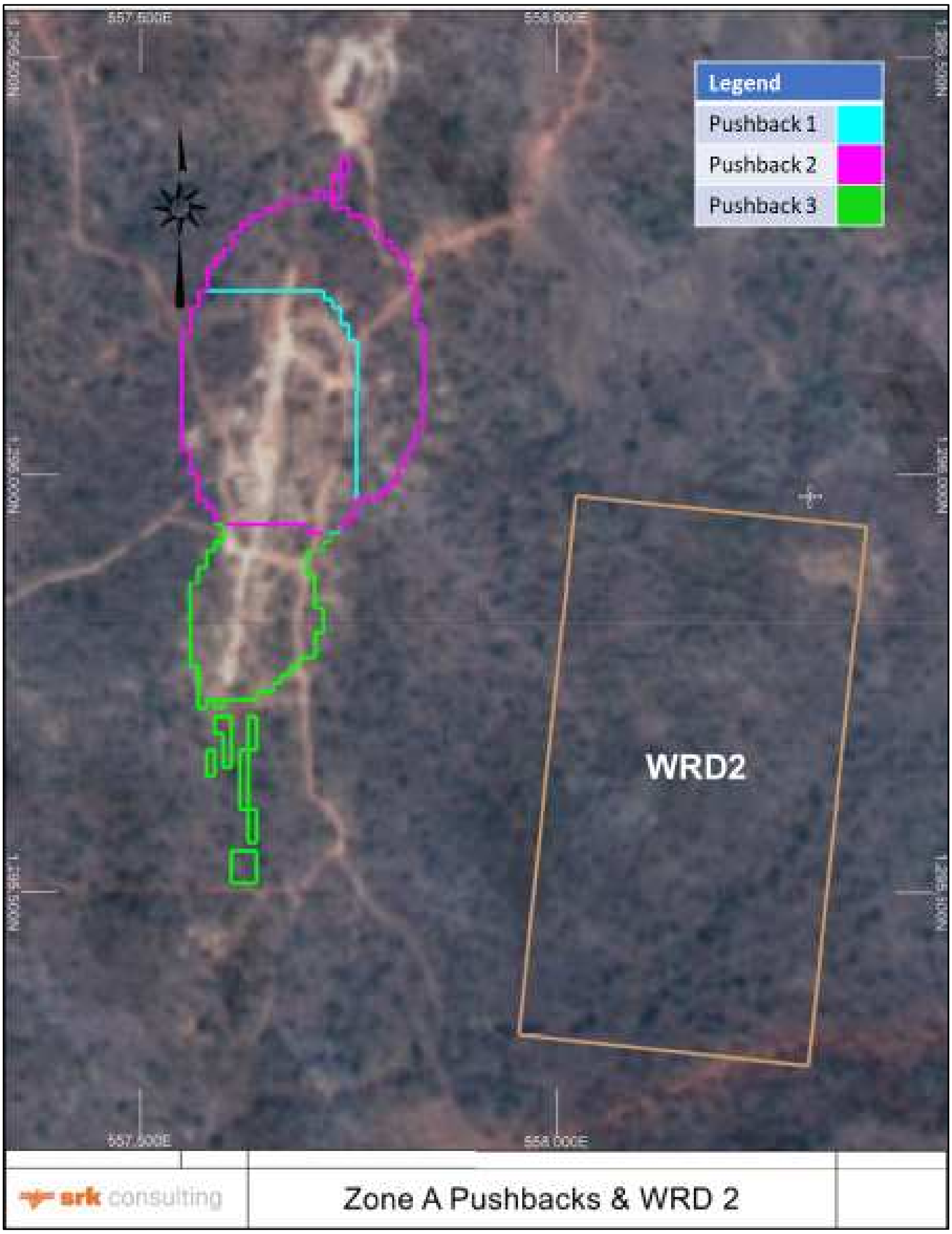


Figure 4-10: Zone A Pushback & WRD 2

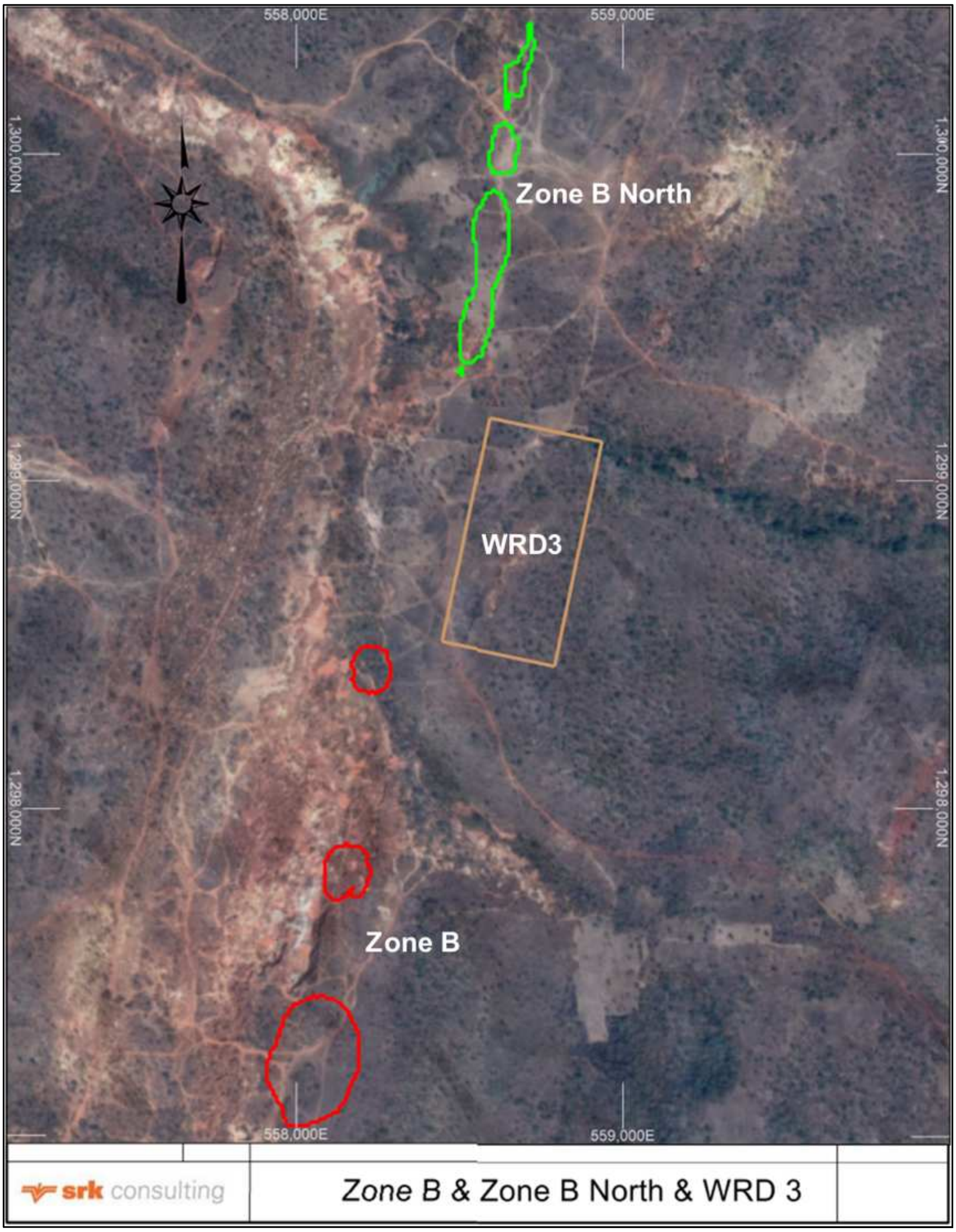


Figure 4-11: Zone B and Zone B North & WRD 3

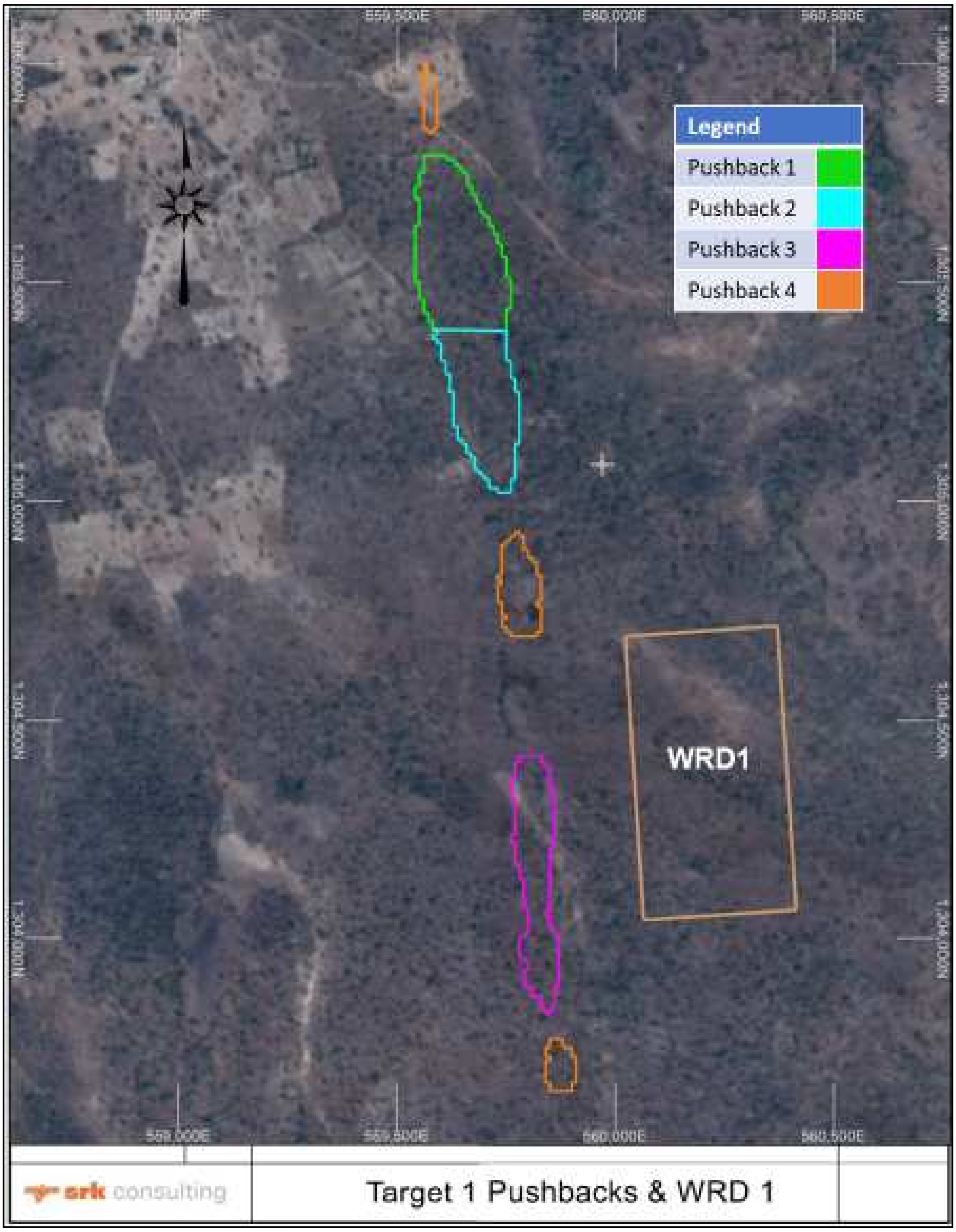


Figure 4-12: Target 1 Pushbacks & WRD 1

4.8 Strategic Schedule

4.8.1 Approach

SRK has developed a strategic level mining schedule for the Sanankoro Project using NPVS scheduling software. The strategic schedule uses the diluted mining model to estimate the scheduling quantities. The processing rate is based on the considered production targets. In this process, SRK tried to smooth the mining mass movements on an annual basis where possible. The mine plan presented in this report is for Case 1, Case 2 and Case 3 at a USD1,500/oz Au pit shell.

4.8.2 Scheduling Parameters

The key scheduling parameters used are listed below:

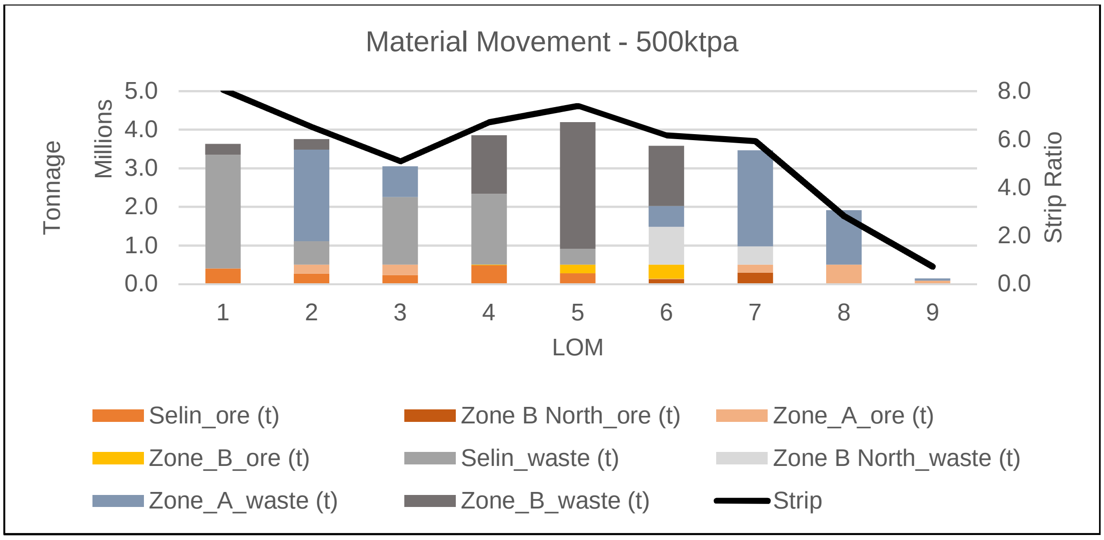
- Scheduling on an annual basis;
- Mining rate was suited to achieve the targeted production rates (Cases 1, 2 and 3 were 0.5 Mtpa,1 Mtpa and 1.5 Mtpa respectively);
- Max sink rate of 50 m per year per pushback (10 x 5 m benches);
- Mill feed is at 80% capacity in year 1 ramping up to 100% at the start of year 2;
- Primary target was to maximize NPV by mining the higher Au grades and low strip ratio areas first;
- Secondary target was to balance the material movement with an average strip ratio for each of the zones

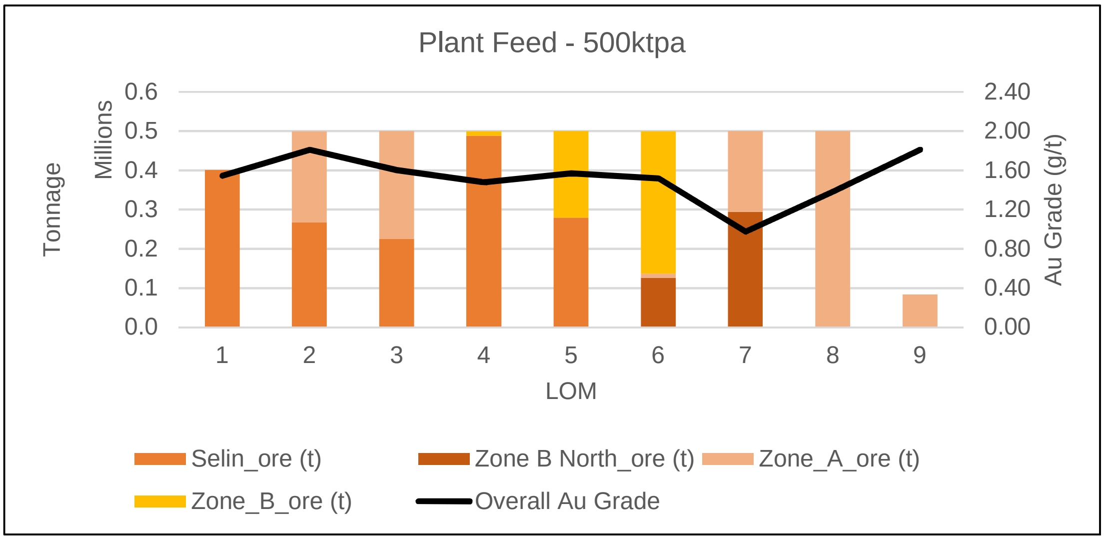
4.8.3 Strategic Schedule Results

The strategic mining and processing schedules are summarised in the following tables and figures listed below:

- Case 1 - 0.5 Mtpa in Table 4-6 and Figure 4-13
- Case 2 – 1.0 Mtpa in Table 4-7 and Figure 4-14
- Case 3 - 1.5 Mtpa in Table 4-8 and Figure 4-15

4.8.4 Case 1 - 0.5Mtpa Schedule





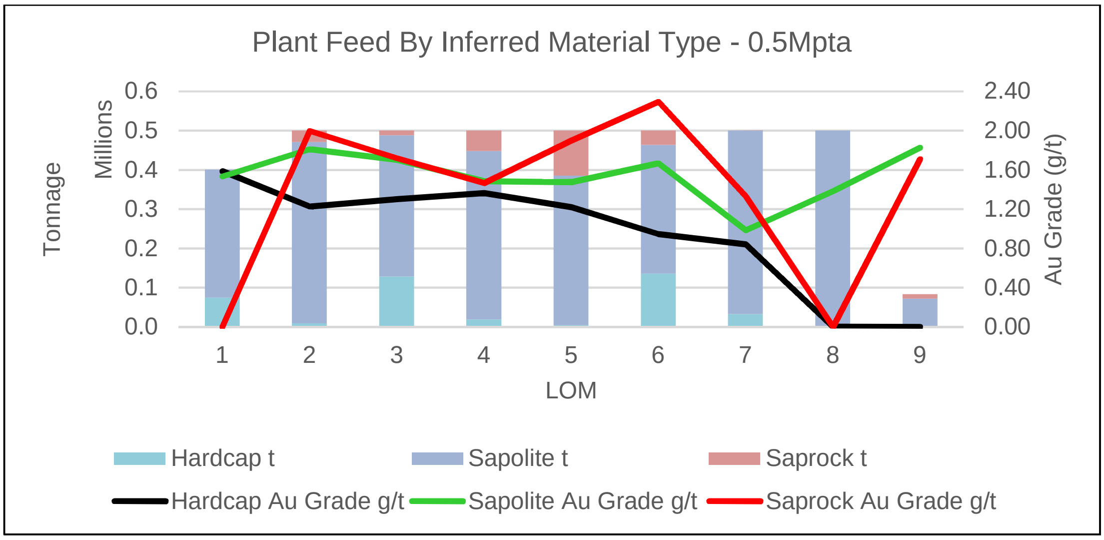
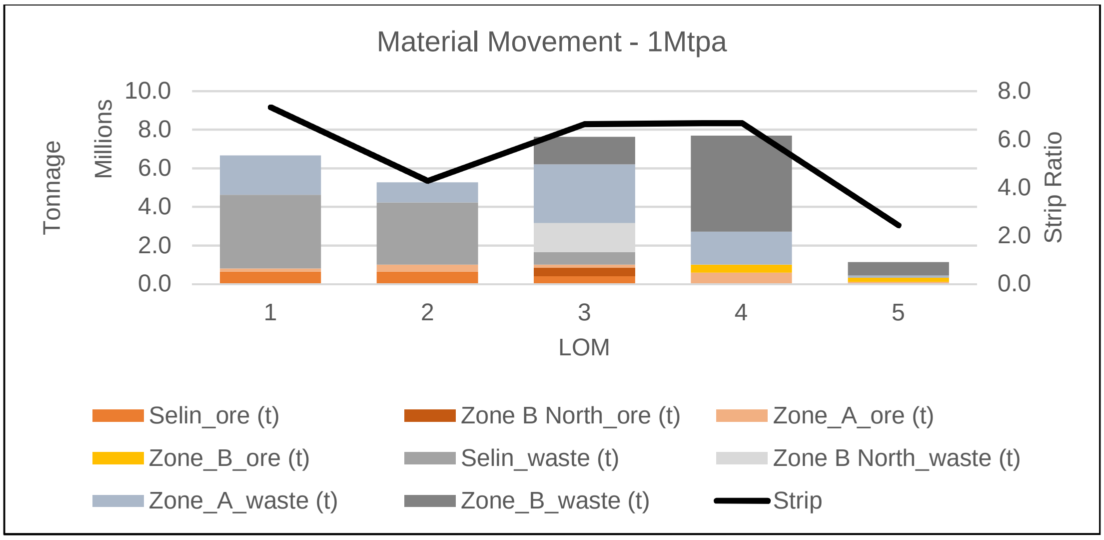


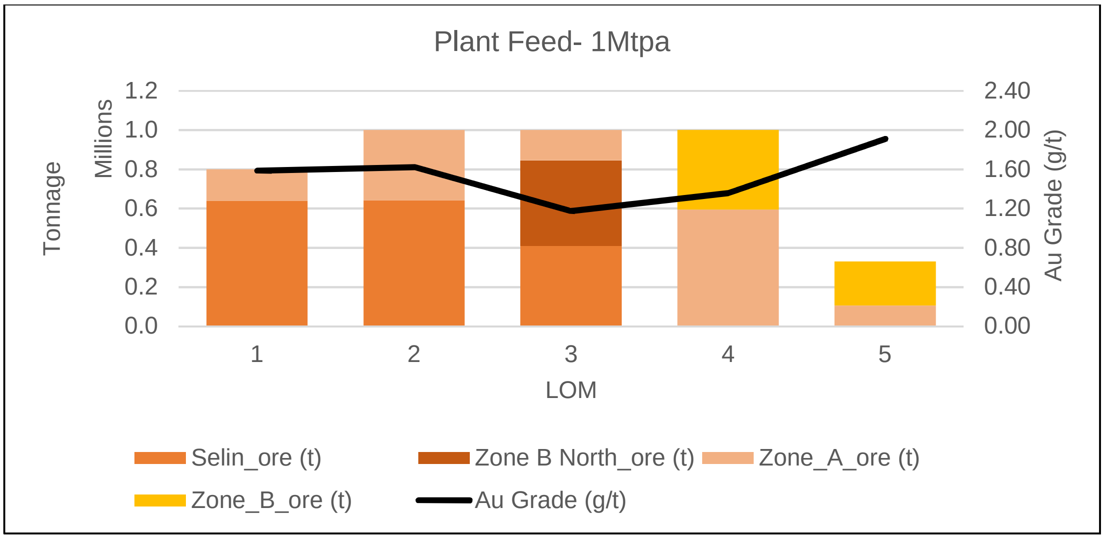
Figure 4-13: Case 1 - 0.5Mtpa - Strategic Mining and Processing Schedule

Table 4-6: Case 1 - 0.5 Mtpa - Strategic Mining and Processing Schedule

Case	Units	Project Year Total	1	2	3	4	5	6	7	8	9
Physicals											
Rock	**(kt)**	27,574	3,626	3,755	3,049	3,852	4,192	3,582	3,466	1,910	143
Ore	**(kt)**	3,984	400	500	500	500	500	500	500	500	83
Waste	**(kt)**	23,591	3,226	3,255	2,549	3,352	3,692	3,082	2,966	1,410	60
Strip Ratio	**(t: t)**	6	8	7	5	7	7	6	6	3	1
Grade	**(g/t)**	1.49	1.54	1.81	1.60	1.48	1.57	1.52	0.98	1.38	1.81
Metal Content	**(koz Au)**	191	20	29	26	24	25	24	16	22	5
Processing											
Mill Feed	**(kt)**	6	8	7	5	7	7	6	6	3	1
	(g/t Au)	1.49	1.54	1.81	1.60	1.48	1.57	1.52	0.98	1.38	1.81
Hardcap	**(kt)**	401	74	9	128	18	4	135	33	0	0
	(g/t Au)	1.20	1.59	1.23	1.30	1.36	1.22	0.95	0.84	0.00	0.00
Saprolite	**(kt)**	3326	326	462	360	430	381	328	467	500	72
	(g/t Au)	1.50	1.53	1.81	1.70	1.48	1.48	1.67	0.98	1.38	1.83
Saprock	**(kt)**	256	0	29	13	52	115	36	0	0	11
	(g/t Au)	1.86	0.00	2.00	1.72	1.47	1.90	2.30	1.33	0.00	1.71

4.8.5 Case 2 – 1 Mtpa Schedule





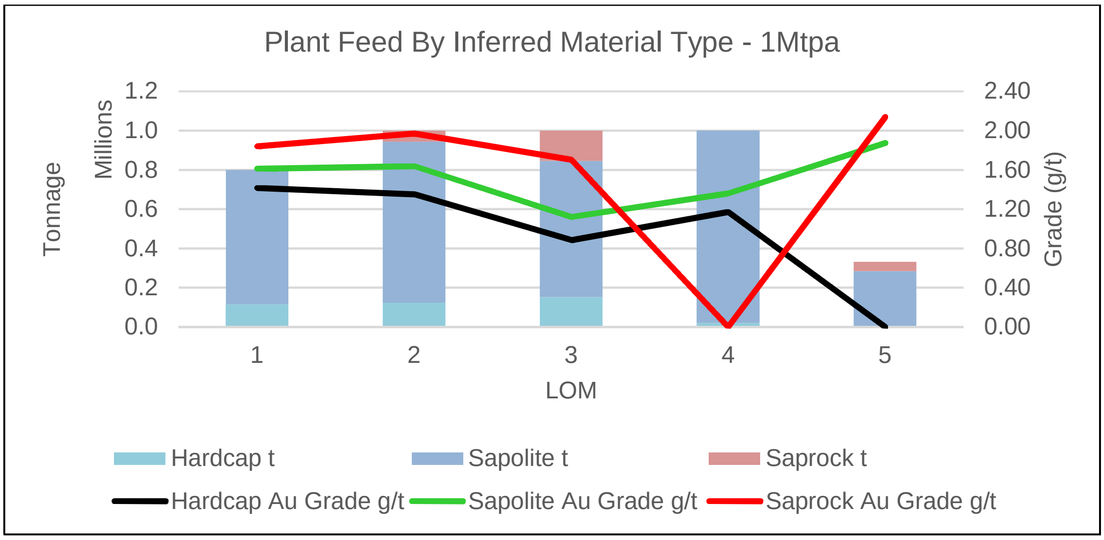
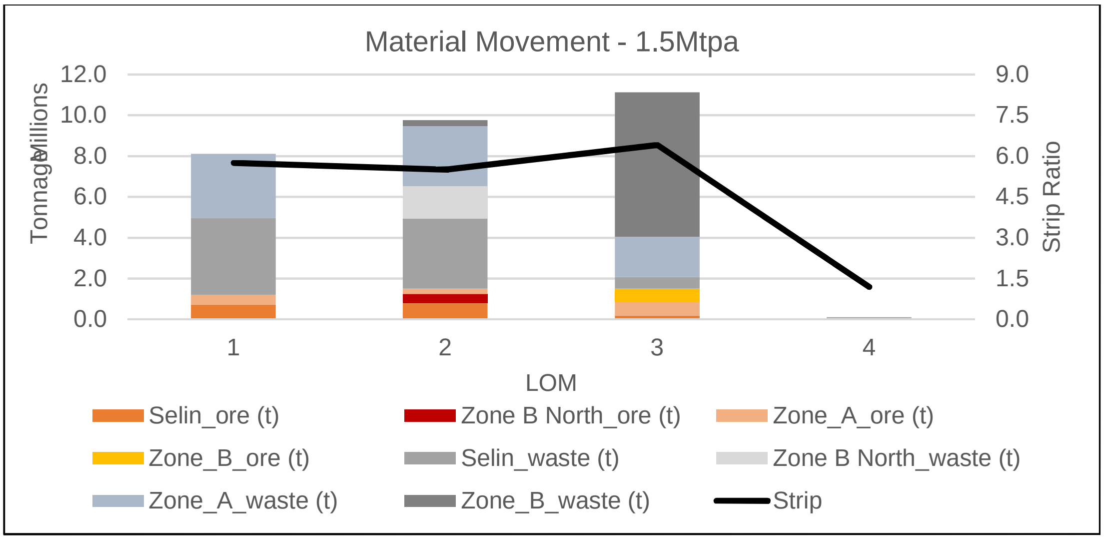


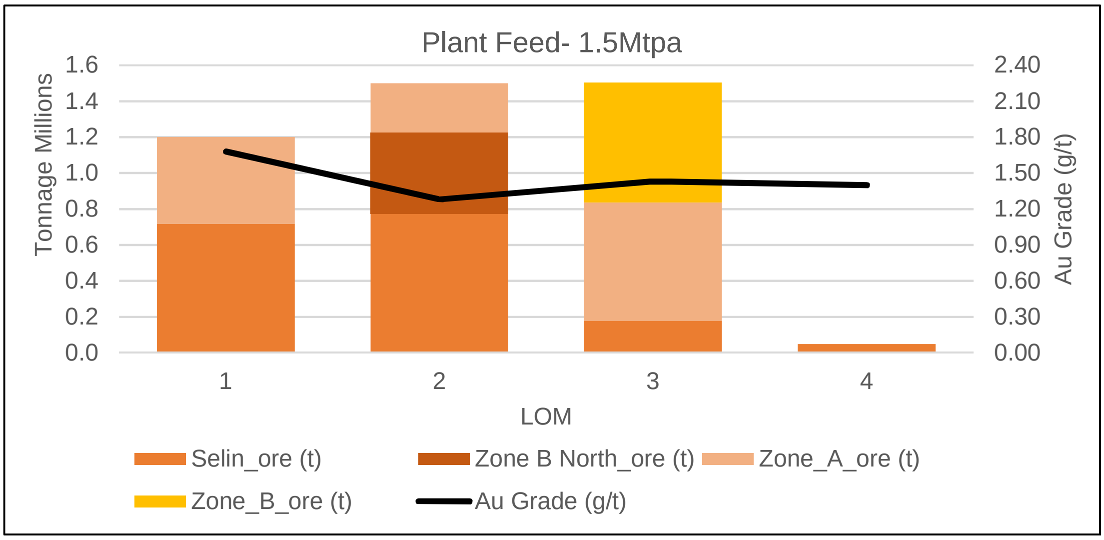
Figure 4-14: Case 2 – 1 Mtpa Strategic Mining and Processing Schedule

Table 4-7: Case 2 – 1 Mtpa - Strategic Mining and Processing Schedule

Case	Units	Project Year Total	1	2	3	4	5
Physicals							
Rock	**(kt)**	28,391	6,669	5,276	7,626	7,684	1,136
Ore	**(kt)**	4,131	800	1,000	1,000	1,000	330
Waste	**(kt)**	24,260	5,869	4,276	6,626	6,684	805
Strip Ratio	**(t: t)**	6	7	4	7	7	2
Grade	**(g/t)**	1.47	1.59	1.62	1.17	1.36	1.91
Metal Content	**(koz Au)**	195	41	52	38	44	20
Processing							
Mill Feed	**(kt)**	4,131	800	1,000	1,000	1,000	330
	(g/t Au)	1.47	1.59	1.62	1.17	1.36	1.91
Hardcap	**(kt)**	408	114	122	152	19	0
	(g/t Au)	1.19	1.42	1.35	0.88	1.17	0.00
Saprolite	**(kt)**	3462	685	820	693	981	283
	(g/t Au)	1.47	1.61	1.64	1.12	1.36	1.87
Saprock	**(kt)**	260.6	0.8	57.9	155.0	0.0	47.0
	(g/t Au)	1.84	1.84	1.97	1.70	0.00	2.14

4.8.6 Case 3 - 1.5 Mtpa Schedule





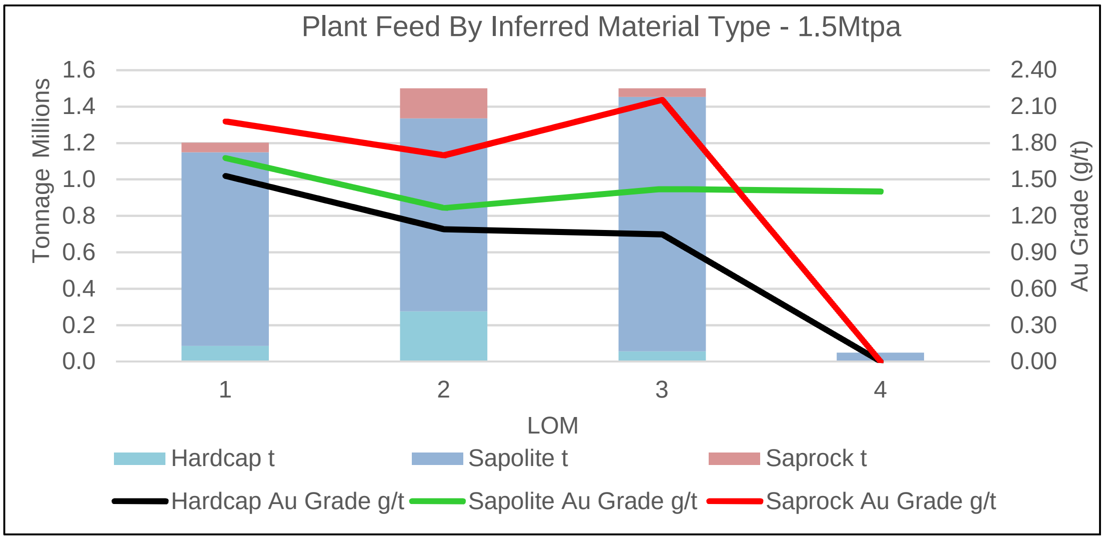


Figure 4-15: Case 3 - 1.5 Mtpa Strategic Mining and Processing Schedule

Table 4-8: Case 3 - 1.5 Mtpa - Strategic Mining and Processing Schedule

Case	Units	Project Year Total	1	2	3	4
Physicals						
Rock	**(kt)**	29,078	8,106	9,748	11,118	106
Ore	**(kt)**	4,249	1,200	1,500	1,500	49
Waste	**(kt)**	24,829	6,905	8,248	9,618	58
Strip Ratio	**(t: t)**	6	6	5	6	1
Grade	**(g/t)**	1.45	1.68	1.28	1.43	1.40
Metal Content	**(koz Au)**	198	65	62	69	2
Processing						
Mill Feed	**(kt)**	4,249	1,200	1,500	1,500	49
	(g/t Au)	1.45	1.68	1.28	1.43	1.40
Hardcap	**(kt)**	416	85	275	55	0
	(g/t Au)	1.17	1.53	1.09	1.05	0.00
Saprolite	**(kt)**	3569	1063	1060	1397	49
	(g/t Au)	1.45	1.68	1.27	1.42	1.40
Saprock	**(kt)**	264.1	52.0	164.6	47.4	0.0
	(g/t Au)	1.84	1.98	1.70	2.16	0.00

4.9 Mining Equipment

4.10 Equipment Selection

SRK has undertaken a high-level estimate of the likely number of excavators, trucks and production drills, to provide an indication to the Client of the potential size and quantity of equipment required for the operation. Selective mining is required to achieve the smallest mining unit (SMU) of 2.5 m x 2.5 m x 5 m, therefore the mining fleets for ore and waste are separate. The load and haul equipment selected for waste is larger than for ore.

This equipment estimate has been completed for the three Cases with the equipment size and the maximum equipment requirements provided in Table 4-9 and Table 4-10, respectively. As the mining schedule outlines that multiple zones are to be operated concurrently, SRK suggest that there are at least two ore and waste loading and hauling units are available throughout the Life of Mine (“LoM”).

Table 4-9: Equipment Selection - All Cases

Equipment	Material Type	Unit	Equipment Size 0.5Mtpa	1Mtpa	1.5Mtpa
Primary Backhoe	Ore	m^3 bucket width	1.9	1.9	1.9
Primary Wheel Loader	Waste	m^3 bucket width	4.2	7.1	8.4
Primary Truck	Ore	t capacity	24	24	24
Secondary Truck	Waste	t capacity	40	40	54
Primary Drill	Both	mm diameter	140	140	140

Table 4-10: Equipment LOM Max Requirement

Equipment	Material Type	Max Requirement 0.5Mtpa	1Mtpa	1.5Mtpa
Primary Backhoe	Ore	2	2	2
Primary Wheel Loader	Waste	2	3	3
Primary Truck	Ore	3	5	6
Secondary Truck	Waste	6	9	8
Primary Drill	Both	1	1	1

4.10.1 Loading Equipment Productivity

Table 4-11 shows the estimated loading productivity for the ore and waste loading fleets for each zone and production rate case. In the event that the loading productivity of a primary backhoe exceeds the annual ore tonnage requirements, the primary backhoe is also used for waste. This increases the utilization of the ore loading fleet

Table 4-12 shows a detailed breakdown of the loading productivity estimation for ore and waste for Case 2 (1.0 Mtpa). The detailed loading productivity breakdown tables for the other production rate cases can be found in Appendix C.

Table 4-11: Loading Productivity – All Cases

Equipment	Material Type	Unit	Loading Productivity		
			0.5Mtpa	1Mtpa	1.5Mtpa
Primary Backhoe	Ore	Mtpa	1.1	1	1
Primary Wheel Loader	Waste		1.8	3.5	3.7

Table 4-12: Case 2 – 1.0 Mtpa Loader Productivity Estimate

Material Type	Units	Selin - Ore	Selin - Waste	Zone B North - Ore	Zone B North - Waste	Zone A - Ore	Zone A - Waste	Zone B - Ore	Zone B - Waste
Loading Unit		Primary Backhoe	Primary Wheel Loader	Primary Backhoe	Primary Wheel Loader	Primary Backhoe	Primary Wheel Loader	Primary Backhoe	Primary Wheel Loader
Bucket Size	(m^3)	1.9	7.1	1.9	7.1	1.9	7.1	1.9	7.1
Loading Spot Time	(min.)	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50
Loading Cycle Time	(min.)	0.67	0.67	0.67	0.67	0.67	0.67	0.67	0.67
First Bucket Dump	(min.)	0.33	0.33	0.33	0.33	0.33	0.33	0.33	0.33
Haulage Unit		Primary Truck	Secondary Truck	Primary Truck	Secondary Truck	Primary Truck	Secondary Truck	Primary Truck	Secondary Truck
Capacity	(t)	25.0	40.0	25.0	40.0	25.0	40.0	25.0	40.00
Capacity	(m^3)	11.4	18.2	11.4	18.2	11.4	18.2	11.4	18.18
Truck Fill Factor	(%)	100	100	100	100	100	100	100	100
Capacity	(t)	25.0	40.0	25.0	40.0	25.0	40.0	25.0	40.0
Capacity	(m^3)	11.4	18.2	11.4	18.2	11.4	18.2	11.4	18.2
Dump & Spot Time	(min.)	5.88	5.88	5.88	5.88	5.88	5.88	5.88	5.88
Loading Parameters									
Bucket Fill Factor	(%)	90	90	90	90	90	90	90	95
In-Situ Density	(t/bcm)	2.20	2.25	2.20	2.30	2.20	2.20	2.20	2.20
Swell Factor	(lcm/bcm)	1.30	1.30	1.30	1.30	1.30	1.30	1.30	1.30
Loose Dry Density	(dt/lcm)	1.69	1.73	1.69	1.77	1.69	1.69	1.69	1.69
Moisture Factor	(%)	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00
Passes	(#)	8.2	3.4	8.2	3.4	8.2	3.5	8.2	3.5
Passes (Rounded)	(#)	8	3	8	3	8	4	8	4
Loaded Quantity	(t)	19.2	31.5	19.2	32.2	19.2	30.8	19.2	30.8
Loaded Volume	(m^3)	11.4	18.2	11.4	18.2	11.4	18.2	11.4	18.2
Loading Productivity									
Theoretical Loading Cycle Time	(min.)	5.50	2.17	5.50	2.17	5.50	2.83	5.50	2.83
Job Efficiency Factor	(%)	85	85	85	85	85	85	85	85
Adjusted Loading Cycle Time	(min.)	6.47	2.55	6.47	2.55	6.47	3.33	6.47	3.33
Loader Productivity	**(dt/doh)**	**170**	**705**	**170**	**721**	**170**	**527**	**170**	**527**
Operating Efficiency	(%)	85.0	83.0	85.0	83.0	85.0	83.0	85.0	83.0
Loader Productivity	(t/op. hr)	144	586	144	599	144	438	144	438
Loading Unit Utilisation	(%)	65.1	56.1	65.1	56.1	65.1	56.1	65.1	56.1
Loading Productivity	**(Mtpa)**	**1.0**	**3.5**	**1.0**	**3.5**	**1.0**	**2.6**	**1.0**	**2.6**

4.10.2 Haulage Estimation

Haulage cycle times have been estimated based on the bench schedule from the strategic mine schedule, the location of the plant and the waste rock dumps. The key haulage parameter assumptions are shown in Table 4-13 and the resulting estimated cycle times for each production case are shown in Figure 4-16 to Figure 4-21 .

A 90% operational efficiency adjustment has been applied to the estimated cycle times, to account for operational delays.

Table 4-13: Ore Haulage Parameter Assumptions

	Unit	Value
Haul Speed		
Average Haul Speed	(kmph)	18
Haul Distances		
Selin	(m)	Variable - Based on bench schedule
Input Flat Haul Distance One Way	(m)	450
Expit Flat Haul Distance One Way (Ore)	(m)	4875
Expit Flat Haul Distance One Way (Waste)	(m)	525
Zone A	(m)	
Input Flat Haul Distance One Way	(m)	250
Expit Flat Haul Distance One Way (Ore)	(m)	4500
Expit Flat Haul Distance One Way (Waste)	(m)	200
Zone B North	(m)	
Input Flat Haul Distance One Way	(m)	450
Expit Flat Haul Distance One Way (Ore)	(m)	550
Expit Flat Haul Distance One Way (Waste)	(m)	700
Zone B	(m)	
Input Flat Haul Distance One Way	(m)	200
Expit Flat Haul Distance One Way (Ore)	(m)	2400
Expit Flat Haul Distance One Way (Waste)	(m)	875

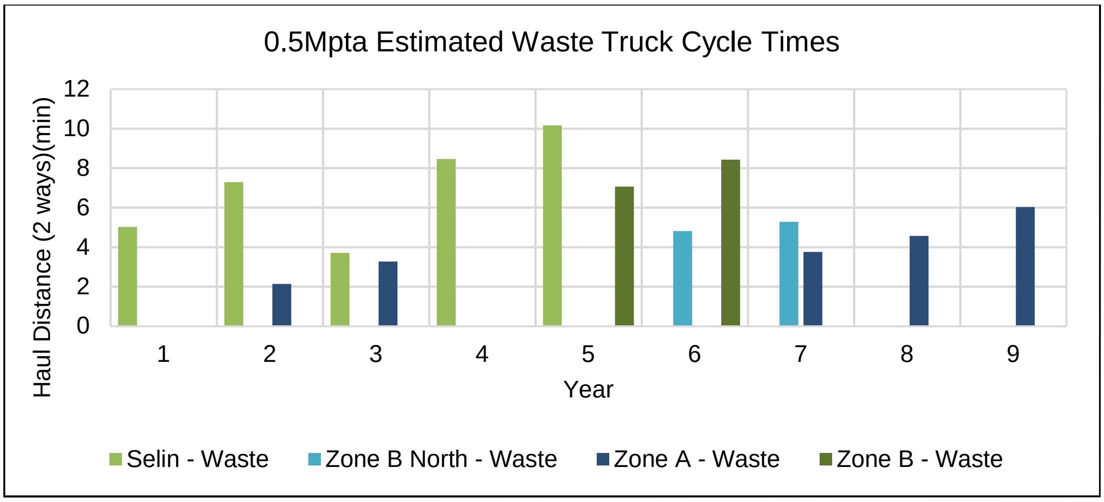


Figure 4-16: Case 1 - 0.5 Mtpa Estimated Ore Haulage Cycle Times

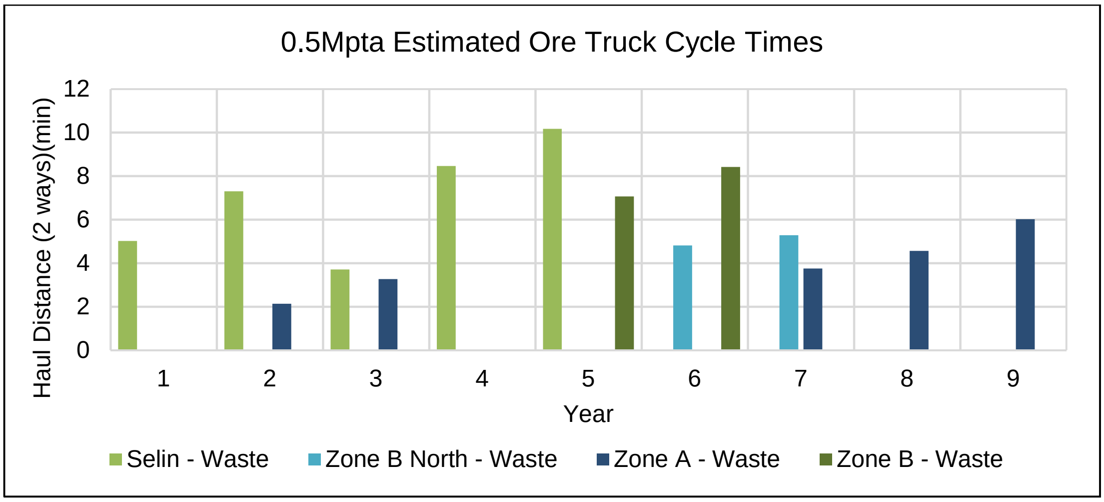


Figure 4-17: Case 1 - 0.5 Mtpa Estimated Waste Haulage Cycle Times

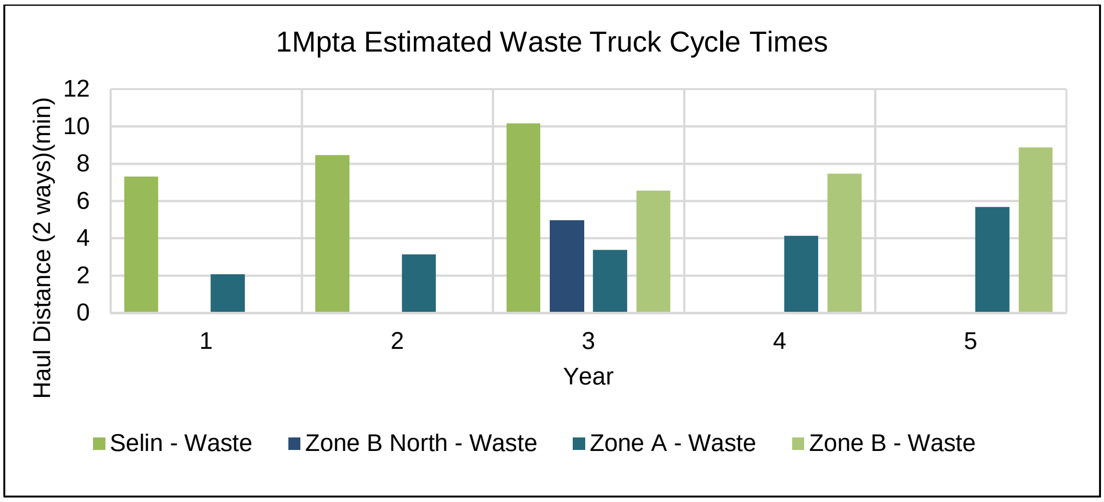


Figure 4-18: Case 2 – 1.0 Mtpa Estimated Ore Haulage Cycle Time

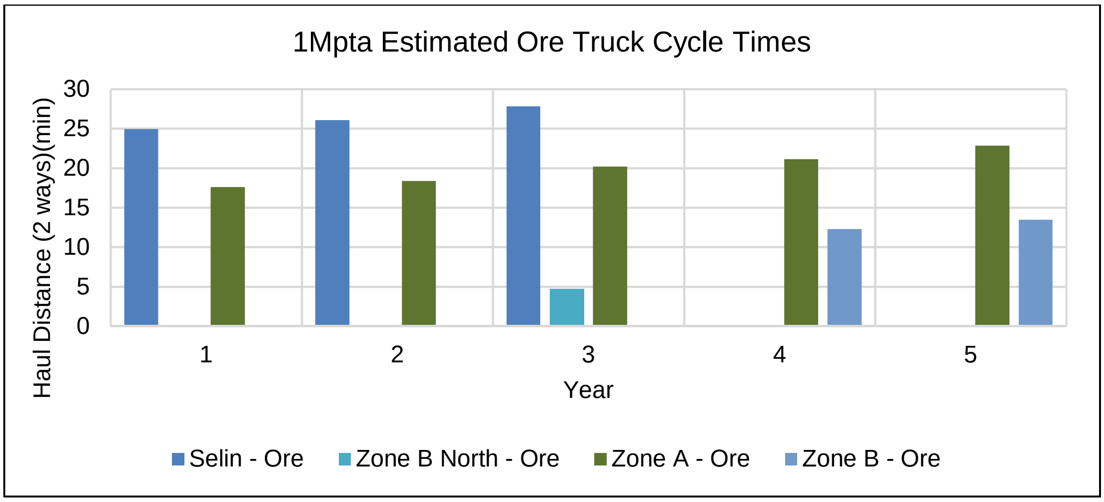


Figure 4-19: Case 2 – 1.0 Mtpa Estimated Waste Haulage Cycle Time

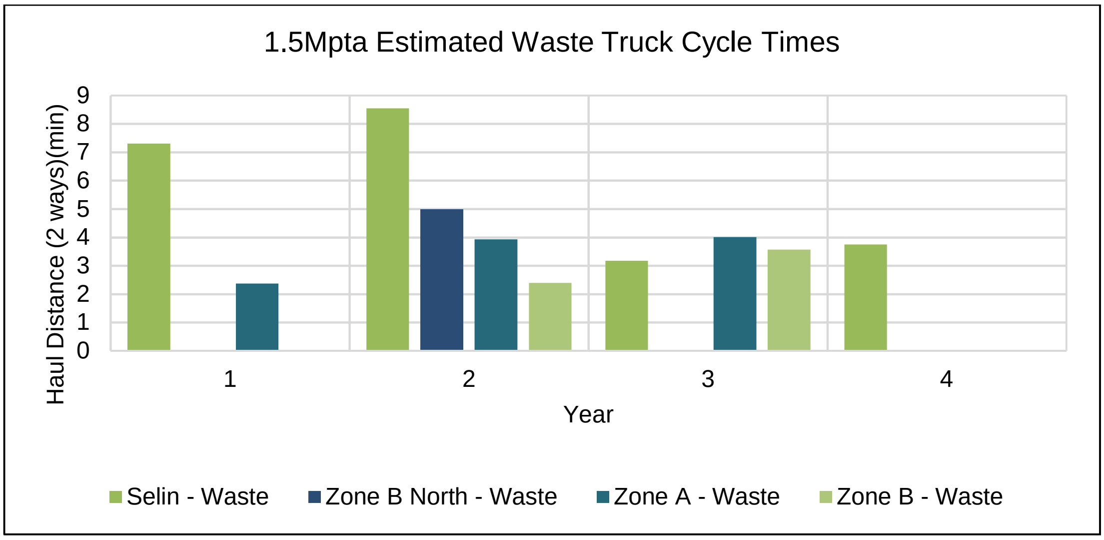


Figure 4-20: Case 3 - 1.5 Mtpa Estimated Ore Haulage Cycle Times

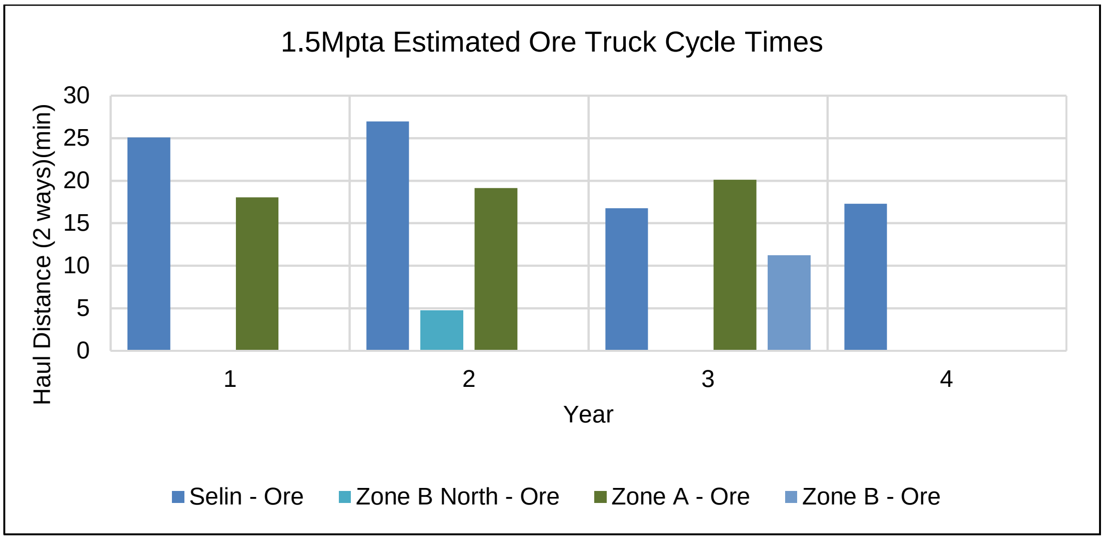


Figure 4-21: Case 3 - 1.5 Mtpa Estimated Waste Haulage Cycle Times

4.10.3 Drill and Blast

A high-level drill and blast estimate have been completed to determine the potential drill productivity for estimating the number of required drills for each production rate case. Table 4-14 shows the blasting requirement for each zone.

The blasting requirement is low, as saprock is the only weathering domain that requires drill and blast. The drill and blast pattern assumptions are shown in Table 4-15. The assumptions are based on a typical production blast pattern and results in an estimated drilling productivity of 2.9 Mtpa.

Table 4-14: Drill and Blast Requirement by Rock Type

Material Type	Blasting (%)
Selin - Ore	15.0
Selin - Waste	5.0
Zone B North - Ore	0.0
Zone B North - Waste	0.0
Zone A - Ore	5.0
Zone A - Waste	0.0
Zone B - Ore	10.0
Zone B - Waste	0.0

Table 4-15: Drill and Blast Pattern Assumptions

	Units	Ore / Waste
Material Properties		
Density	(t/m^3)	2.20
Production Patterns		
Input Parameters		
Drill	-	**Production Drill**
Bench Height	(m)	5.0
Hole Diameter	(mm)	140.0
Subdrill	(m)	1.00
Spacing	(m)	4.00
Burden	(m)	3.50
Stemming Height	(m)	3.20
Re-drill/Drilling Overlap Factor	(%)	10.00
Rod Length	(m)	12.00
Hoisting Rate	(m/min)	30.00
Cleaning, retract, tramming, etc	(min.)	3.50
Add/remove rods	(min.)	4.00
Primers per Hole	(#)	1
Explosive Product		1
Explosive Density	(t/m^3)	2
Checks		ANFO
Bench Height : Hole Diameter	(m:m)	0.81
Subdrill to Hole Diameter	(m:m)	
Stemming to Burden	(m:m)	36
Drilling		7.1
Hole Depth	(m)	0.91
Volume Rock per Hole	(m^3)	
Quantity Rock per Hole	(t)	6.0
Yield of Rock	(m^3 rock/m drilled)	70.0
Yield of Rock	(t rock/m drilled)	154.0
Penetration Rate	(m/hr)	11.7
Drill time per Hole	(min.)	25.7
Productivity per meter	(m/doh)	25.0
Productivity per tonne	(t/doh)	18.1
Operating Hours per Year	(hrs)	19.9
	(Mtpa)	510.5
Blasting		2.9
Stemming Volume	(m^3)	
Volume of Charge	(m^3)	0.05
Charge Height	(m)	0.04
Charge per Hole	(kg)	2.8
Powder Factor	**(kg/m^3)**	34.9
Powder Factor	**(kg/t)**	0.50

4.10.4 Equipment Requirements

Based on the estimated equipment productivities, cycle times and strategic schedules, the estimated equipment requirements for each case are shown in Figure 4-22 to Figure 4-24.

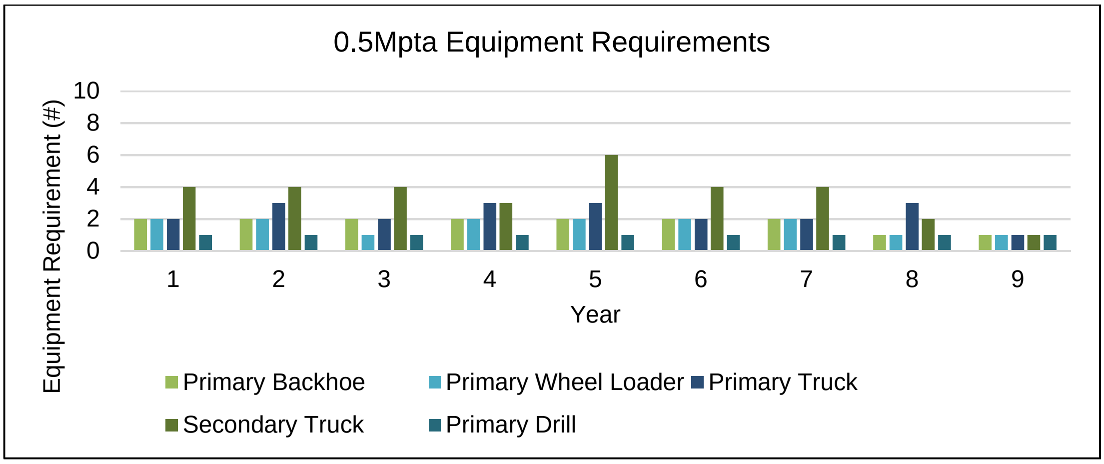


Figure 4-22: Case 1 Estimated Equipment Requirements

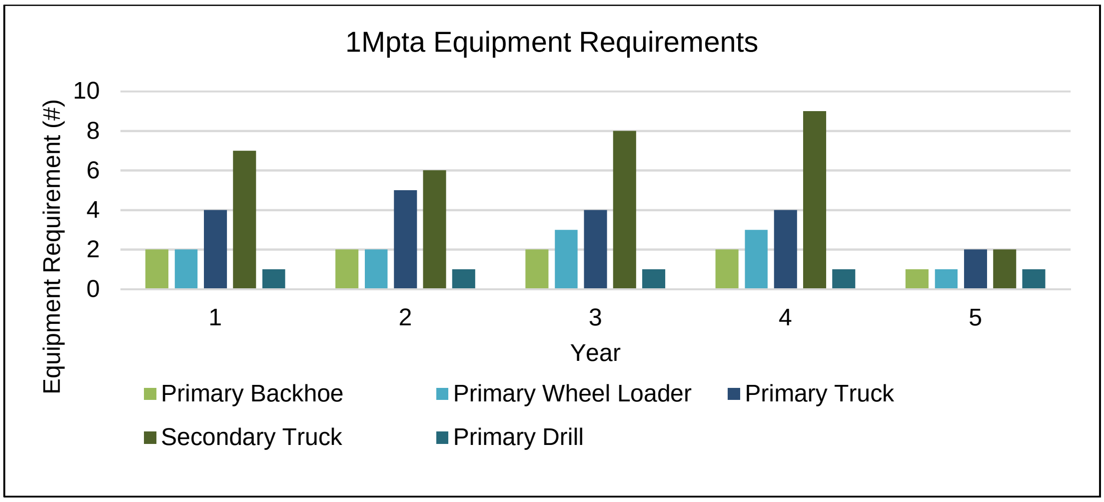


Figure 4-23: Case 2 Estimated Equipment Requirements

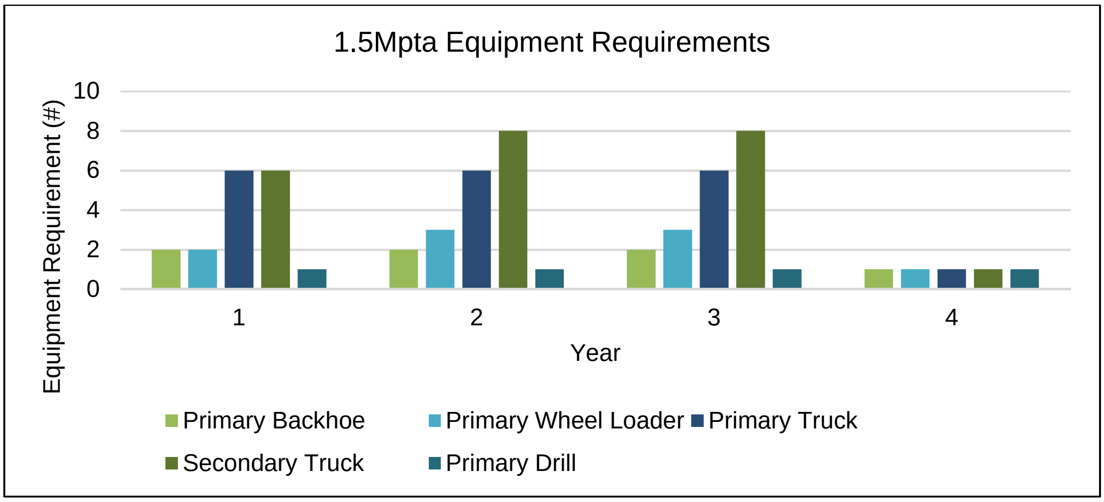


Figure 4-24: Case 3 Estimated Equipment Requirements

4.11 Operating Strategy

4.12 Approach

An operating strategy has been developed for Sanankoro based on the operating parameters specific to the deposit, selected mining equipment, and the mine layout.

4.13 Mine Layout

Table 4-25 shows the general site layout and includes the ultimate pit design crest, waste dump boundaries and processing plant.

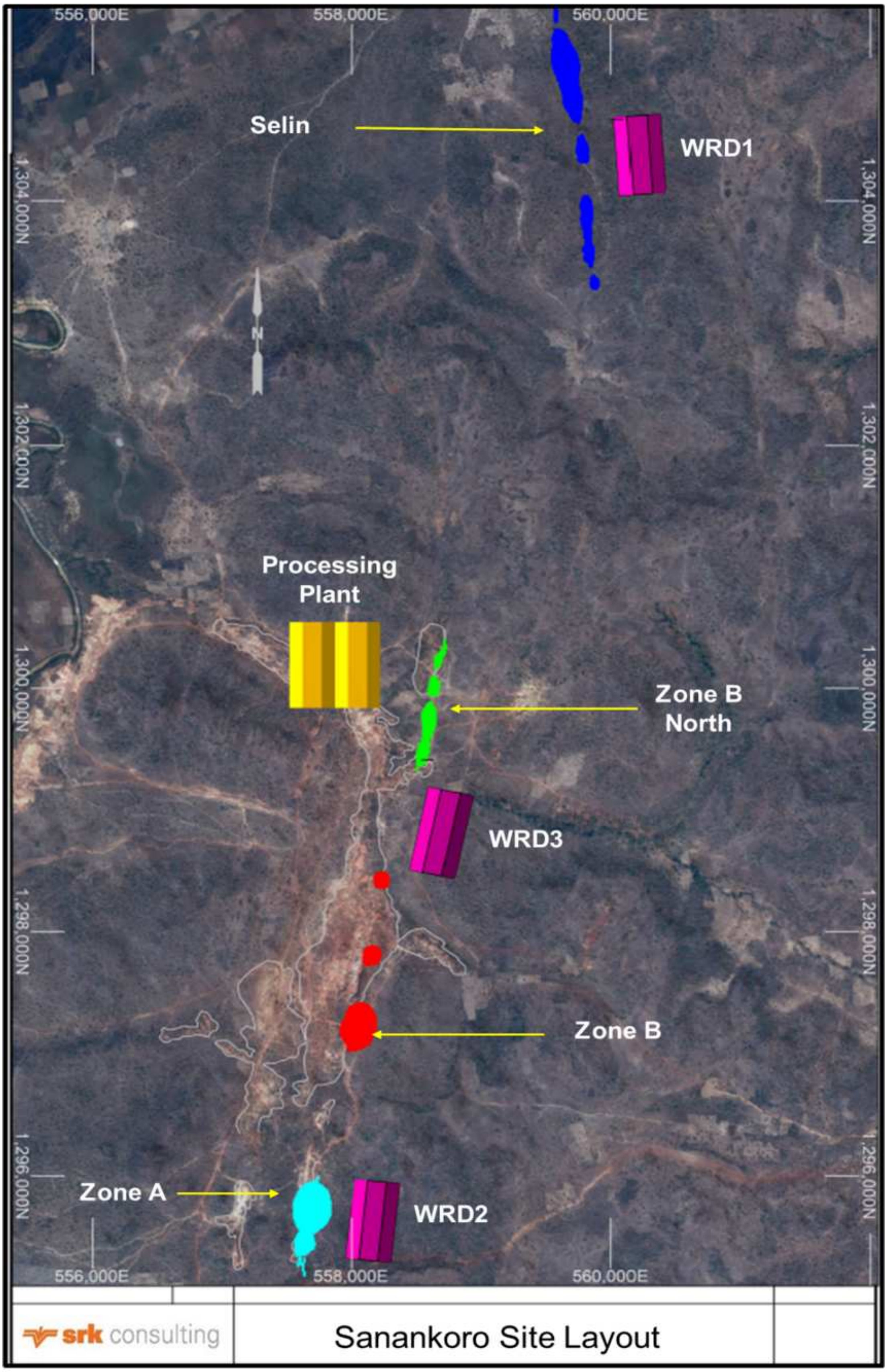


Figure 4-25: Site Layout

4.14 Mining Method

The following mining method has been proposed for the Sanankoro project and is the same across all production rate cases:

- It is expected that the extraction method will be predominately free digging, as the hardcap and saprock weathering domains do not need to be subject to drill and blasting;
- It is expected that saprock will require drill and blast;
- Ore will be excavated by a 1.9 m^3 primary backhoe accompanied with 24 t articulated haul trucks;
- Waste will be excavated by a primary wheel loader with haul truck, where the size of the equipment is dependent on the production rate case

4.15 Drilling and Blasting

Two types of drilling activities are expected at the Sanankoro operation, including production and infill. SRK expects that a production drill and a RC drill rig will be required to undertake all drilling activities.

The blasting activities at Sanankoro will only be required for material in the saprock weathering domain. Based on the dry conditions in Mali it is expected that ammonium nitrate fuel oil ("ANFO") product will be used. It is expected that one primer per blasthole will be used for ore and waste.

4.16 Grade Control

Grade control is the process in which the mill feed grade is verified and managed. Optimising this process and maintaining enough selectivity in mining will be essential to achieving the mine plan at the Sanankoro deposit.

Sampling will provide detailed grade distributions of the mining area in order to selectively excavate the mineralisation. SRK recommends that in-fill (or grade control) drilling with an RC drill rig should be conducted over 5 m benches to facilitate short term mine planning. The blastholes from the production patterns will also be sampled. Once the results of the samples are carried out, bench plans will be created by the geologists that outline mineralised and non-mineralised areas. These outlines will be used to mark out the different zones for excavation.

4.16.1 Waste Rock Dump Material Profile

SRK has developed a high-level waste rock dump schedule for all three production cases on an annual basis.

Table 4-16 shows the geotechnical waste rock dump design parameters. These design parameters are based on SRK's experience with the type of waste rock material present at Sanankoro, however it is recommended that the design parameters are confirmed by more detailed analysis in subsequent studies.

Table 4-16: Geotechnical Waste Rock Dump Design Parameters

Parameter	Unit	Waste Dump
Bench Angle	(º)	22
Bench Height	(m)	10
Berm Width	(m)	5
Overall Slope Angle	(º)	18
Max Height	(m)	50

This waste rock dump parameters shown in Table 4-16 were used to verify the capacity of the waste rock dump facilities for the three cases (0.5 Mtpa,1.0 Mtpa,1.5 Mtpa). The pit shells in the Sanankoro license span 10 km across the property. As a result of the large distance between the zones, it is recommended by SRK that three waste rock dumps facilities should be built.

Table 4-17 to Table 4-19 shows the results of the available volume for each waste rock dump and for each case.

Table 4-17: 0.5 Mtpa - Final Waste Rock Dump Design Capacity

WRD	Zone	Contingency	Waste Requirements (Mm^3)	Swell Factor (lcm:bcm)	Design Volume (Mm^3)
WRD1	Selin	20%	3.4	1.30	5.2
WRD2	Zone A	20%	3.4	1.30	5.4
WRD3	Zone B/ Zone B N	20%	3.7	1.30	5.7

Table 4-18: 1.0 Mtpa - Final Waste Rock Dump Design Capacity

WRD	Zone	Contingency	Waste Requirements (Mm^3)	Swell Factor (lcm:bcm)	Design Volume (Mm^3)
WRD1	Selin	20%	3.4	1.30	5.3
WRD2	Zone A	20%	3.6	1.30	5.6
WRD3	Zone B/ Zone B N	20%	3.8	1.30	5.8

Table 4-19: 1.5 Mtpa - Final Waste Rock Dump Design Capacity

WRD	Zone	Contingency	Waste Requirements (Mm^3)	Swell Factor (lcm:bcm)	Design Volume (Mm^3)
WRD1	Selin	20%	3.5	1.30	5.4
WRD2	Zone A	20%	3.6	1.30	5.7
WRD3	Zone B/ Zone B N	20%	4.0	1.30	6.1

4.17 Capital and Operating Cost Estimation

4.17.1 Approach

A mining cost model has been developed to assess the mining capital and operating expenditures expected for the Sanankoro operation, based on both a contractor mining and owner operated option as requested by the Client. The mine schedules for each case have been used as a basis for the cost estimation. Generally, the capital and operating unit cost assumptions are the same for all cases for the same equipment, but it should be reminded to the reader that some of the equipment varies depending on the production rate.

The cost estimate is based primarily on SRK's experience in open pits in Mali and the 2018 Infomine cost database.

The capital and operational cost estimates for each production rate case schedule can be found in Table 4-23 to Table 4-25.

4.17.2 Capital Cost Estimate

The equipment capital cost and equipment replacement and rebuild cost for the loading, hauling, drilling and ancillary equipment are shown in Table 4-20. SRK has used the 2018 Infomine cost database as a benchmark.

The mobilisation/demobilisation and site establishment costs are based on an SRK estimate that these costs are approximately 10% of the total equipment purchase, replacement and rebuilding costs in the first and last year of LoM. These cost estimates were compared to benchmarks from SRK's experience at open pits in Mali. The mobilisation/demobilisation and site establishment costs are shown in Table 4-20.

The capital cost difference between an owner operated and a contractor option is that the contractor capital does not include the equipment purchase and replacement costs. This equipment capital will be built into the contractor's operating cost (contractor rate). In addition to the capital cost categories, a 15% capital cost contingency is applied to both the owner operated and contractor option to account for additional costs.

Table 4-20: Capital Cost Estimates

Capital Cost	Units	Cost
Primary Backhoe	(US$m)	0.9
Primary Wheel Loader	(US$m)	0.8
Primary Truck	(US$m)	0.6
Secondary Truck	(US$m)	0.8
Primary Drill	(US$m)	1.4
Track Dozer	(US$m)	0.9
Grader	(US$m)	1.0
Water Truck	(US$m)	1.4
Service Truck	(US$m)	0.1
Fuel/Lube Truck	(US$m)	0.1
Small Crane	(US$m)	0.5
Lighting Plant	(US$m)	0.5
Light Vehicle	(US$m)	0.7
Site Establishment, Mobilisation, Demobilisation	(US$m)	1.7

4.17.3 Operating Cost Estimate

The operating cost is broken down into categories – primarily labour, maintenance, consumables and grade control. The owner operated and contractor base unit cost for these categories are the same, therefore the varying factor is a contractor premium of 25% applied to the contractor option. In addition to the cost categories, a 15% operating cost contingency is applied to both the owner operated and contractor option to account for additional costs.

The labour and consumable unit costs are provided in Table 4-21 and Table 4-22, respectively. Detailed operating cost estimates are presented in Table 4-23 to Table 4-25.

Table 4-21: Consumable and Grade Control Costs

Consumables	Source	Units	Cost
Fuel	SRK estimate	(US$/l)	1.1
Power	SRK estimate	(US$/kWh)	0.049
Lube	SRK estimate	(US$/l)	4
AN	SRK estimate	(US$/t)	1500
Primer	SRK estimate	(US$/unit)	3.5
Detonator	SRK estimate	(US$/unit)	3
Surface Delay	SRK estimate	(US$/unit)	2
Blasting Accessories	SRK estimate	(%)	5
Sampling	SRK estimate	(US$/unit)	20
Stemming	SRK estimate	(US$/lcm)	10
Grade Control Drilling	SRK estimate	(US$M/yr)	1

Table 4-22: Labour Costs

Position	Salary (US$pa)
Mine Operations	
Mine Manager	120,000
Superintendents	100,000
Supervisors	85,000
Trainer	75,000
Shovel Operator	40,000
Truck Operator	40,000
Loader Operator	45,000
Ancillary Operator	35,000
Driller Operator	35,000
Safety Manager	80,000
Human Resources Manager	80,000
Accountant	60,000
Security	15,000
Mine Maintenance	
Maintenance Superintendent	85,000
Maintenance Supervisor	75,000
Maintenance Planner	65,000
Maintenance Crew	25,000
Technical Services	
Chief Engineer	100,000
Chief Geologist	100,000
Senior Engineer	90,000
Senior Geologist	90,000
Planning Engineer	75,000
Mine Surveyor	40,000
Mine Geologist	60,000
Consultant (Hydro, Geotech, etc)	75,000
Administrative Assistant	25,000

Table 4-23: Case 1 – 0.5 Mtpa Capital and Operating Costs

Period	Unit	Total	1	2	3	4	5	6	7	8	9
Capital Costs	**(US$m)**	**3.4**	**2.2**	**0.0**	**0.0**	**0.0**	**0.0**	**0.0**	**0.1**	**0.0**	**1.1**
Equipment Mobilization & Establish Site Facilities	(US$m)	**1.0**	1.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Equipment Demobilization & Disestablish Site Facilities	(US$m)	**1.0**	0.0	0.0	0.0	0.0		0.0	0.0	0.0	1.0
Miscellaneous	(US$m)	**1.0**	0.9	0.0	0.0	0.0	0.0	0.0	0.1	0.0	0.0
Contingency	(US$m)	**0.4**	0.3	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.1
Operating Costs - Category	**(US$/t)**	**4.24**	**3.81**	**3.72**	**4.25**	**3.52**	**3.78**	**3.77**	**3.88**	**5.94**	**8.4**
Labour	(US$/t)	**1.57**	1.36	1.35	1.56	1.28	1.31	1.38	1.42	2.32	3.8
Maintenance	(US$/t)	**0.18**	0.16	0.16	0.18	0.15	0.17	0.16	0.16	0.24	0.3
Fuel	(US$/t)	**0.88**	0.83	0.80	0.89	0.75	0.86	0.81	0.83	1.15	1.1
Lubricants	(US$/t)	**0.10**	0.09	0.09	0.10	0.08	0.10	0.09	0.09	0.13	0.1
Tires	(US$/t)	**0.10**	0.09	0.09	0.10	0.08	0.11	0.09	0.09	0.11	0.1
Wear Parts	(US$/t)	**0.02**	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.04	0.1
Explosives	(US$/t)	**0.01**	0.02	0.01	0.02	0.02	0.01	0.00	0.00	0.01	0.0
Grade Control	(US$/t)	**0.16**	0.14	0.13	0.16	0.13	0.12	0.14	0.14	0.26	0.5
Miscellaneous	(US$/t)	**0.00**	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.0
Contingency	(US$/t)	**0.45**	0.41	0.40	0.46	0.38	0.40	0.40	0.42	0.64	0.9
Contractor Premium	(US$/t)	**0.76**	0.68	0.66	0.76	0.63	0.67	0.67	0.69	1.06	1.5
Operating Costs - Activity	**(US$/t)**	**4.24**	**3.81**	**3.72**	**4.25**	**3.52**	**3.78**	**3.10**	**3.19**	**4.88**	**6.9**
Management	(US$/t)	**0.78**	0.66	0.64	0.78	0.62	0.57	0.67	0.69	1.25	2.4
Loading	(US$/t)	**0.47**	0.45	0.46	0.44	0.45	0.44	0.46	0.47	0.47	0.4
Hauling	(US$/t)	**0.87**	0.82	0.82	0.89	0.71	1.02	0.80	0.82	1.08	0.4
Drilling	(US$/t)	**0.06**	0.06	0.04	0.06	0.05	0.04	0.04	0.04	0.08	0.1
Ancillary	(US$/t)	**0.68**	0.57	0.55	0.68	0.54	0.50	0.58	0.60	1.10	2.1
Blasting	(US$/t)	**0.01**	0.02	0.01	0.02	0.02	0.01	0.00	0.00	0.01	0.0
Grade Control	(US$/t)	**0.16**	0.14	0.13	0.16	0.13	0.12	0.14	0.14	0.26	0.5
Miscellaneous	(US$/t)	**0.00**	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.01	0.0
Contingency	(US$/t)	**0.45**	0.41	0.40	0.46	0.38	0.40	0.40	0.42	0.64	0.9
Contractor Premium	(US$/t)	**0.76**	0.68	0.66	0.76	0.63	0.67	0.67	0.69	1.06	1.5

Table 4-24: Case 2 – 1.0 Mtpa Capital and Operating Costs

Period	Unit	Total	1	2	3	4	5
Capital Costs	**(US$m)**	**7.6**	**2.9**	**0.0**	**0.0**	**1.8**	**2.9**
Equipment Mobilization & Establish Site Facilities	(US$m)	**1.7**	1.7	0.0	0.0	0.0	0.0
Equipment Demobilization & Disestablish Site Facilities	(US$m)	**1.7**	0.0	0.0	0.0	0.0	1.7
Miscellaneous	(US$m)	**0.9**	0.9	0.0	0.0	0.0	0.0
Contingency	(US$m)	**0.6**	0.4	0.0	0.0	0.0	0.3
Operating Costs - Category	**(US$/t)**	**3.43**	**3.09**	**3.74**	**2.86**	**2.96**	**10.97**
Labour	(US$/t)	**1.08**	0.94	1.19	0.87	0.88	4.08
Maintenance	(US$/t)	**0.16**	0.15	0.18	0.14	0.15	0.45
Fuel	(US$/t)	**0.79**	0.73	0.85	0.69	0.73	1.94
Lubricants	(US$/t)	**0.09**	0.09	0.10	0.08	0.09	0.21
Tires	(US$/t)	**0.11**	0.11	0.11	0.11	0.12	0.17
Wear Parts	(US$/t)	**0.02**	0.02	0.03	0.02	0.02	0.10
Explosives	(US$/t)	**0.01**	0.02	0.02	0.01	0.00	0.00
Grade Control	(US$/t)	**0.18**	0.15	0.19	0.13	0.13	0.88
Miscellaneous	(US$/t)	**0.00**	0.00	0.00	0.00	0.00	0.01
Contingency	(US$/t)	**0.37**	0.33	0.40	0.31	0.32	1.18
Contractor Premium	(US$/t)	**0.61**	0.55	0.67	0.51	0.53	1.96
Operating Costs - Activity	**(US$/t)**	**3.43**	**3.09**	**3.74**	**2.86**	**2.96**	**10.97**
Management	(US$/t)	**0.43**	0.37	0.47	0.32	0.32	2.18
Loading	(US$/t)	**0.36**	0.33	0.36	0.35	0.37	0.58
Hauling	(US$/t)	**0.87**	0.82	0.98	0.79	0.86	1.25
Drilling	(US$/t)	**0.03**	0.04	0.05	0.02	0.02	0.12
Ancillary	(US$/t)	**0.56**	0.48	0.61	0.42	0.41	2.81
Blasting	(US$/t)	**0.01**	0.02	0.02	0.01	0.00	0.00
Grade Control	(US$/t)	**0.18**	0.15	0.19	0.13	0.13	0.88
Miscellaneous	(US$/t)	**0.00**	0.00	0.00	0.00	0.00	0.01
Contingency	(US$/t)	**0.37**	0.33	0.40	0.31	0.32	1.18
Contractor Premium	(US$/t)	**0.61**	0.55	0.67	0.51	0.53	1.96

Table 4-25: Case 3 – 1.5 Mtpa Capital and Operating Costs

Period	Units	Total	1	2	3	4
Capital Costs	**(US$m)**	**6.6**	**3.9**	**0.0**	**0.0**	**2.7**
Equipment Mobilization & Establish Site Facilities	(US$m)	**1.9**	1.9	0.0	0.0	0.0
Equipment Demobilization & Disestablish Site Facilities	(US$m)	**1.9**	0.0	0.0	0.0	1.9
Miscellaneous	(US$m)	**0.9**	0.9	0.0	0.0	0.0
Contractor Premium	(US$m)	**1.2**	0.7	0.0	0.0	0.5
Contingency	(US$m)	**0.7**	0.4	0.0	0.0	0.3
Operating Costs - Category	**(US$/t)**	**3.02**	**2.95**	**2.87**	**2.52**	**72.43**
Labour	(US$/t)	**0.86**	0.80	0.75	0.66	36.93
Maintenance	(US$/t)	**0.15**	0.14	0.15	0.13	2.29
Fuel	(US$/t)	**0.76**	0.75	0.77	0.68	10.29
Lubricants	(US$/t)	**0.09**	0.09	0.09	0.08	1.09
Tires	(US$/t)	**0.11**	0.11	0.12	0.11	0.47
Wear Parts	(US$/t)	**0.02**	0.02	0.02	0.01	0.50
Explosives	(US$/t)	**0.01**	0.02	0.01	0.00	0.04
Grade Control	(US$/t)	**0.15**	0.19	0.15	0.13	0.00
Miscellaneous	(US$/t)	**0.00**	0.00	0.00	0.00	0.12
Contingency	(US$/t)	**0.32**	0.32	0.31	0.27	7.76
Contractor Premium	(US$/t)	**0.54**	0.53	0.51	0.45	12.93
Operating Costs - Activity	**(US$/t)**	**3.02**	**2.95**	**2.87**	**2.52**	**72.43**
Management	(US$/t)	**0.34**	0.30	0.25	0.22	23.21
Loading	(US$/t)	**0.37**	0.36	0.37	0.36	3.52
Hauling	(US$/t)	**0.79**	0.82	0.82	0.70	3.68
Drilling	(US$/t)	**0.03**	0.03	0.03	0.02	1.35
Ancillary	(US$/t)	**0.46**	0.39	0.41	0.36	19.81
Blasting	(US$/t)	**0.01**	0.02	0.01	0.00	0.04
Grade Control	(US$/t)	**0.15**	0.19	0.15	0.13	0.00
Miscellaneous	(US$/t)	**0.00**	0.00	0.00	0.00	0.12
Contingency	(US$/t)	**0.32**	0.32	0.31	0.27	7.76
Contractor Premium	(US$/t)	**0.54**	0.53	0.51	0.45	12.93

5 CONCLUSIONS

5.1 Mining

5.1.1 Dilution

The dilution in all zones is estimated between 14% and 20% and recovery between 91% and 95% based on a regularised block model to a selected mining unit of 2.5 x 2.5 x 5 m and a 0.4 g/t Au cut-off grade. This is regarded as reasonable for the width of the orebody (1.5 - 26 m) and would require relatively small-scale, higher selectivity mining methods (1.9 m^3 bucket capacity excavators, 24 - 40 t capacity trucks).

5.1.2 Pit Optimisation

The pit optimisation results show that the mining inventory grows with increasing RF on a broadly linear basis. That said, it should be noted that the total ore tonnage is relatively sensitive to the gold price selected for the pit optimisation. The total ore tonnage inside of the USD1,300/oz pit shell (which is a more similar price to the current long-term Au price forecast) is 2.8 Mt at 1.60 g/t Au, whilst the total ore tonnage inside of the USD1,500/oz pit shell is 4.1 Mt at 1.47 g/t Au. This represents a 46% increase in ore tonnage and 35% increase in contained ounces in the USD1,500/oz pit shell, compared to the USD1,300/oz pit shell. Total rock inside the USD1,500/oz pit shell is 28.4 Mt and total rock inside the USD1,300/oz pit shell is 17.0 Mt. The stripping ratio is 5.9 in the USD1,500/oz pit shell and 5.1 in the USD1,300/oz pit shell. After discussions between Cora Gold and SRK, Cora Gold requested that SRK use the USD1,500/oz Au pit shell (RF=100%) for the development of the strategic schedule. This is considered acceptable at a scoping level, however the sensitivity to Au price should be carefully considered as the Project develops.

5.1.3 Pushback

Pushback selection has been undertaken and indicates that Selin and Zone A are the only zones that require multiple pushbacks. The pushbacks make it possible to initially target a lower strip ratio in all scheduled cases.

5.1.4 Strategic Mine Schedule

The results of the mine schedule are summarised below:

- All cases contain a sink rate of 50 m per cutback per year;
- Case 1 at a production rate of 0.5 Mtpa can be achieved at a total mining rate up to 4.3 Mtpa;
- Case 2 at a production rate of 1.0 Mtpa can be achieved at a total mining rate ranging between 5.0 Mtpa and 8.0 Mtpa;
- Case 3 at a production rate of 1.5 Mtpa can be achieved at a total mining rate ranging between 8.0 Mtpa and 11 Mtpa.
- Where possible, the higher grades were scheduled to be mined at the beginning of the LoM, however SRK notes that the grade distribution wasn't particularly large and the differences are not significant, meaning the grade is relatively constant across the LoM, usually fluctuating more towards the LoM, where the smaller zones are mined;

- The three production rate cases differ in duration of the LoM between 9 for Case 1 and 4 years for Case 3, therefore production rate will have significant impact on the LoM duration;
- Mining the Sanankoro zones will provide a fairly stable strip ratio profile across the LoM with some local variations in certain years.

5.1.5 Mining Equipment

The results of the equipment schedule are summarised below:

- All cases require the same ore mining fleet that includes a minimum of 2 x 1.9 m^3 excavators, 2 x 24 t trucks and 1 x 140 mm production drill;
- Future studies may analyse civil type of trucks hauling ore to the plant, due to some long distances from the pits;
- It has been assumed that the ore trucks will be equipped with tyres suitable for longer distances in high temperatures;
- The waste equipment size is dependent on the production rate of each case;
- Sanankoro is relatively small operation with several pits spread over the area at some distance. The Project may therefore suffer from the “economy of scale” effect, especially for the cases assuming lower production rates;
- Primary mining equipment and support ancillary equipment (dozers, graders, water trucks, service vehicles) will be required.

5.1.6 Operating Strategy

The results of the operating strategy are summarised below:

- It is expected that the extraction method will be predominantly free digging as the hardcap and saprock weathering domains do not need to be drill and blasted. Drill and blast will be required in the saprock domain;
- Ore and waste will be excavated by separate fleets in order to account for a relatively high level of selective mining required;
- Based on the mine plans and the distance between the zones, it is recommended to have at least three waste rock dumps. SRK have not noted any significant limitations in available surface space at this stage;
- A stockpile strategy is not considered in this study.

5.1.7 Capital and Operating Costs

The results of the capital and operating cost estimates are summarised below:

- As expected for all cases, capital costs for the owner operated scenario are higher than the contracted option, with included equipment purchase and replacement costs;
- It is noted that the operating costs for contractor operated scenario would be higher as a result of the 25% contractor premium;
- The operating cost for owner operated and contractor options decreases as the production rate increases because certain cost items related to for example G&A or part of the

ancillary fleet do not change with the production rate. Therefore, there is a higher unit cost per tonne of rock for lower production rates, which is also related to the economy of scale effect. On the other hand, the equipment size for higher production rates have been adjusted to account for higher mass movement rates within certain years. This reduces the operating cost as less equipment is required for higher production rates.

- Labour rates may seem to be relatively high but are expected to be so, due to the geographical location of the Project;
- The capital costs for owner operated and contractor scenarios decrease with low production rates as a result of lower requirement for equipment throughout the mine life.

The mining costs estimated in this Report are summarised in Table 5-1.

Table 5-1: Estimated Mining Costs

	Scenario	Unit	0.5 Mtpa	1.0 Mtpa	1.5 Mtpa
OPEX	Owner	(US$/t)	3.43	2.82	2.48
	Contractor	(US$/t)	4.17	3.43	3.02
CAPEX	Owner	(US$M)	19.6	32.6	31.3
	Contractor	(US$M)	3.2	5.9	6.6

6 RECOMMENDATIONS

Based on the work undertaken as part of the scoping study, SRK makes the following mining study related recommendations:

- Mining operating costs should be confirmed through obtaining mining contractor costs, using the strategic schedule and equipment sizing and estimated requirements from the Scoping Study;
- Open pit optimisation and pit shell selection should be repeated when material changes to the geological model or operating / processing assumptions occur;
- Run trade-off studies with civil type of trucks hauling ore to the plant and consult tyre manufacturers regarding the tires for high temperature environment;
- Verify potential for the fresh material;
- This Mining Section of the Scoping Study for Sanankoro has been completed by SRK while the main Report of the Scoping Study has been developed by Wardell Armstrong consultants. SRK recommends using the annual cost estimate in the TEM for the entire Project;
- It is recommended by SRK that this scoping level mine planning should be updated when new geological information and an updated block model is available. Based on the results of the updated scoping level mine planning, if the Project is economically favourable, more detailed study (to Pre-Feasibility Level) should be undertaken to determine the mining parameters, costs and operating strategy at an increased level of detail.

For and on behalf of SRK Consulting (UK) Limited

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APPENDIX

A PIT OPTIMISATION SUMMARY TABLES

Case 1

Table A 1: Case 1 – 0.5 Mtpa – Selin

Optimisation Results	Units	1300 US$/oz	1500 US$/oz	1600 US$/oz	1700 US$/oz	1800 US$/oz
Insitu						
Inventory	(Mt)	1.5	1.7	1.7	1.8	1.8
	(g/t Au)	1.59	1.56	1.54	1.54	1.52
	(koz Au)	79	83	86	88	90
Modifying Factors						
Mining Dilution	(%)	0.0	0.0	0.0	0.0	0.0
Dilutant	(g/t Au)	0.0	0.0	0.0	0.0	0.0
Mining Recovery	(%)	100.0	100.0	100.0	100.0	100.0
Au Metallurgical Recovery	(%)	91.2	91.3	91.3	91.4	91.4
Diluted						
Inventory	(Mt)	1.5	1.7	1.7	1.8	1.8
	(g/t Au)	1.59	1.56	1.54	1.54	1.52
	(koz Au)	79	83	86	88	90
Quantities						
Total Rock	(Mt)	8.3	9.5	10.2	11.0	11.8
Mineral Inventory	(Mt)	1.5	1.7	1.7	1.8	1.8
Waste + OM	(Mt)	6.7	7.8	8.5	9.2	10.0
Waste	(Mt)	6.5	7.5	8.2	8.9	9.6
Inventory (Below Cut-off)	(Mt)	0.2	0.3	0.3	0.3	0.3
Stripping Ratio	(t:t)	4.4	4.7	4.9	5.2	5.4
Product						
Recovered Metal	(koz Au)	72	76	78	81	83
Project Life	(years)	3.1	3.3	3.5	3.6	3.7

Table A 2: Case 1 – 0.5 Mtpa – Zone A

Optimisation Results	Units	1300 US$/oz	1400 US$/oz	1500 US$/oz	1600 US$/oz	1700 US$/oz
Insitu						
Inventory	(Mt)	1.1	1.3	1.3	1.4	1.6
	(g/t Au)	1.52	1.48	1.46	1.42	1.36
	(koz Au)	54	60	62	65	71
Modifying Factors						
Mining Dilution	(%)	0.0	0.0	0.0	0.0	0.0
Dilutant	(g/t Au)	0.0	0.0	0.0	0.0	0.0
Mining Recovery	(%)	100.0	100.0	100.0	100.0	100.0
Au Metallurgical Recovery	(%)	95.6	95.6	95.6	95.6	95.6
Diluted						
Inventory	(Mt)	1.1	1.3	1.3	1.4	1.6
	(g/t Au)	1.52	1.48	1.46	1.42	1.36
	(koz Au)	54	60	62	65	71
Quantities						
Total Rock	(Mt)	7.5	8.9	9.4	10.2	12.1
Mineral Inventory	(Mt)	1.1	1.3	1.3	1.4	1.6
Waste + OM	(Mt)	6.4	7.6	8.1	8.7	10.5
Waste	(Mt)	6.1	7.2	7.7	8.3	9.9
Inventory (Below Cut-off)	(Mt)	0.3	0.4	0.4	0.5	0.6
Stripping Ratio	(t:t)	5.8	6.0	6.2	6.2	6.5
Product						
Recovered Metal	(koz Au)	52	57	59	62	68
Project Life	(years)	2.2	2.5	2.6	2.8	3.2

Table A 3: Case 1 – 0.5 Mtpa – Zone B North

Optimisation Results	Units	1500 US$/oz	1600 US$/oz	1700 US$/oz	1800 US$/oz	1900 US$/oz
Insitu						
Inventory	(Mt)	0.4	0.5	0.6	0.7	0.8
	(g/t Au)	1.01	0.99	0.98	0.96	0.95
	(koz Au)	14	15	19	21	23
Modifying Factors						
Mining Dilution	(%)	0.0	0.0	0.0	0.0	0.0
Dilutant	(g/t Au)	0.0	0.0	0.0	0.0	0.0
Mining Recovery	(%)	100.0	100.0	100.0	100.0	100.0
Au Metallurgical Recovery	(%)	89.3	89.6	90.2	90.4	90.6
Diluted						
Inventory	(Mt)	0.4	0.5	0.6	0.7	0.8
	(g/t Au)	1.01	0.99	0.98	0.96	0.95
	(koz Au)	14	15	19	21	23
Quantities						
Total Rock	(Mt)	1.9	2.3	3.3	4.0	4.7
Mineral Inventory	(Mt)	0.4	0.5	0.6	0.7	0.8
Waste + OM	(Mt)	1.5	1.9	2.7	3.4	4.0
Waste	(Mt)	1.5	1.8	2.6	3.2	3.8
Inventory (Below Cut-off)	(Mt)	0.1	0.1	0.1	0.1	0.2
Stripping Ratio	(t:t)	3.6	3.9	4.6	4.9	5.2
Product						
Recovered Metal	(koz Au)	12	14	17	19	21
Project Life	(years)	0.8	1.0	1.2	1.4	1.5

Table A 4: Case 1 – 0.5 Mtpa – Zone B

Optimisation Results	Units	1300 US$/oz	1400 US$/oz	1500 US$/oz	1600 US$/oz	1700 US$/oz
Insitu						
Inventory	(Mt)	0.4	0.4	0.6	0.7	0.8
	(g/t Au)	1.70	1.73	1.69	1.59	1.53
	(koz Au)	20	25	32	38	39
Modifying Factors						
Mining Dilution	(%)	0.0	0.0	0.0	0.0	0.0
Dilutant	(g/t Au)	0.0	0.0	0.0	0.0	0.0
Mining Recovery	(%)	100.0	100.0	100.0	100.0	100.0
Au Metallurgical Recovery	(%)	95.1	95.2	95.3	95.4	95.4
Diluted						
Inventory	(Mt)	0.4	0.4	0.6	0.7	0.8
	(g/t Au)	1.70	1.73	1.69	1.59	1.53
	(koz Au)	20	25	32	38	39
Quantities						
Total Rock	(Mt)	3.8	5.2	7.6	9.2	9.6
Mineral Inventory	(Mt)	0.4	0.4	0.6	0.7	0.8
Waste + OM	(Mt)	3.5	4.8	7.1	8.5	8.8
Waste	(Mt)	3.4	4.7	6.9	8.3	8.6
Inventory (Below Cut-off)	(Mt)	0.1	0.1	0.1	0.2	0.2
Stripping Ratio	(t:t)	9.3	10.7	11.8	11.5	11.1
Product						
Recovered Metal	(koz Au)	19	24	31	36	37
Project Life	(years)	0.7	0.9	1.2	1.5	1.6

Case 2

Table A 5: Case 2 – 1.0 Mtpa – Selin

Optimisation Results	Units	1300 US$/oz	1400 US$/oz	1500 US$/oz	1600 US$/oz	1700 US$/oz
Insitu						
Inventory	(Mt)	1.6	1.6	1.7	1.8	1.8
	(g/t Au)	1.58	1.56	1.55	1.54	1.53
	(koz Au)	79	82	84	87	89
Modifying Factors						
Mining Dilution	(%)	0.0	0.0	0.0	0.0	0.0
Dilutant	(g/t Au)	0.0	0.0	0.0	0.0	0.0
Mining Recovery	(%)	100.0	100.0	100.0	100.0	100.0
Au Metallurgical Recovery	(%)	91.2	91.3	91.3	91.3	91.4
Diluted						
Inventory	(Mt)	1.6	1.6	1.7	1.8	1.8
	(g/t Au)	1.58	1.56	1.55	1.54	1.53
	(koz Au)	79	82	84	87	89
Quantities						
Total Rock	(Mt)	8.3	9.1	9.7	10.4	11.2
Mineral Inventory	(Mt)	1.6	1.6	1.7	1.8	1.8
Waste + OM	(Mt)	6.8	7.4	8.0	8.7	9.4
Waste	(Mt)	6.5	7.2	7.7	8.4	9.1
Inventory (Below Cut-off)	(Mt)	0.2	0.3	0.3	0.3	0.3
Stripping Ratio	(t:t)	4.3	4.5	4.7	4.9	5.2
Product						
Recovered Metal	(koz Au)	72	75	77	79	81
Project Life	(years)	3.1	3.3	3.4	3.5	3.6

Table A 6: Case 2 – 1.0 Mtpa –Zone A

Optimisation Results	Units	1300 US$/oz	1400 US$/oz	1500 US$/oz	1600 US$/oz	1700 US$/oz
Insitu						
Inventory	(Mt)	1.1	1.3	1.4	1.5	1.7
	(g/t Au)	1.51	1.46	1.43	1.39	1.34
	(koz Au)	55	60	63	66	72
Modifying Factors						
Mining Dilution	(%)	0.0	0.0	0.0	0.0	0.0
Dilutant	(g/t Au)	0.0	0.0	0.0	0.0	0.0
Mining Recovery	(%)	100.0	100.0	100.0	100.0	100.0
Au Metallurgical Recovery	(%)	95.6	95.6	95.6	95.6	95.6
Diluted						
Inventory	(Mt)	1.1	1.3	1.4	1.5	1.7
	(g/t Au)	1.51	1.46	1.43	1.39	1.34
	(koz Au)	55	60	63	66	72
Quantities						
Total Rock	(Mt)	7.5	8.9	9.8	10.3	12.5
Mineral Inventory	(Mt)	1.1	1.3	1.4	1.5	1.7
Waste + OM	(Mt)	6.4	7.6	8.4	8.9	10.8
Waste	(Mt)	6.1	7.2	8.0	8.4	10.2
Inventory (Below Cut-off)	(Mt)	0.3	0.4	0.4	0.5	0.6
Stripping Ratio	(t:t)	5.7	6.0	6.1	6.1	6.4
Product						
Recovered Metal	(koz Au)	52	57	60	63	69
Project Life	(years)	2.3	2.5	2.7	2.9	3.4

Table A 7: Case 2 – 1.0 Mtpa – Zone B North

Optimisation Results	Units	1300 US$/oz	1400 US$/oz	1500 US$/oz	1600 US$/oz	1700 US$/oz
Insitu						
Inventory	(Mt)	0.3	0.4	0.4	0.5	0.6
	(g/t Au)	1.03	1.02	1.00	0.97	0.97
	(koz Au)	10	12	14	17	19
Modifying Factors						
Mining Dilution	(%)	0.0	0.0	0.0	0.0	0.0
Dilutant	(g/t Au)	0.0	0.0	0.0	0.0	0.0
Mining Recovery	(%)	100.0	100.0	100.0	100.0	100.0
Au Metallurgical Recovery	(%)	88.3	88.9	89.3	89.9	90.2
Diluted						
Inventory	(Mt)	0.3	0.4	0.4	0.5	0.6
	(g/t Au)	1.03	1.02	1.00	0.97	0.97
	(koz Au)	10	12	14	17	19
Quantities						
Total Rock	(Mt)	1.2	1.6	2.0	2.7	3.5
Mineral Inventory	(Mt)	0.3	0.4	0.4	0.5	0.6
Waste + OM	(Mt)	0.9	1.2	1.6	2.2	2.9
Waste	(Mt)	0.8	1.2	1.5	2.1	2.8
Inventory (Below Cut-off)	(Mt)	0.0	0.0	0.1	0.1	0.1
Stripping Ratio	(t:t)	2.9	3.3	3.6	4.1	4.6
Product						
Recovered Metal	(koz Au)	9	11	12	15	18
Project Life	(years)	0.6	0.7	0.9	1.1	1.2

Table A 8: Case 2 – 1.0 Mtpa – Zone B

Optimisation Results	Units	1300 US$/oz	1400 US$/oz	1500 US$/oz	1600 US$/oz	1700 US$/oz
Insitu						
Inventory	(Mt)	0.4	0.5	0.6	0.8	0.8
	(g/t Au)	1.69	1.74	1.64	1.56	1.51
	(koz Au)	20	26	33	38	39
Modifying Factors						
Mining Dilution	(%)	0.0	0.0	0.0	0.0	0.0
Dilutant	(g/t Au)	0.0	0.0	0.0	0.0	0.0
Mining Recovery	(%)	100.0	100.0	100.0	100.0	100.0
Au Metallurgical Recovery	(%)	95.1	95.3	95.4	95.4	95.4
Diluted						
Inventory	(Mt)	0.4	0.5	0.6	0.8	0.8
	(g/t Au)	1.69	1.74	1.64	1.56	1.51
	(koz Au)	20	26	33	38	39
Quantities						
Total Rock	(Mt)	3.8	5.7	7.9	9.3	9.7
Mineral Inventory	(Mt)	0.4	0.5	0.6	0.8	0.8
Waste + OM	(Mt)	3.5	5.2	7.2	8.6	8.9
Waste	(Mt)	3.4	5.1	7.1	8.4	8.7
Inventory (Below Cut-off)	(Mt)	0.1	0.1	0.1	0.2	0.2
Stripping Ratio	(t:t)	9.2	11.1	11.4	11.3	10.9
Product						
Recovered Metal	(koz Au)	19	25	32	36	38
Project Life	(years)	0.8	0.9	1.3	1.5	1.6

Case 3

Table A 9: Case 3 – 1.5 Mtpa – Selin

Optimisation Results	Units	1300 US$/oz	1400 US$/oz	1500 US$/oz	1600 US$/oz	1700 US$/oz
Insitu						
Inventory	(Mt)	1.6	1.6	1.7	1.8	1.8
	(g/t Au)	1.57	1.55	1.54	1.53	1.52
	(koz Au)	79	82	85	87	89
Modifying Factors						
Mining Dilution	(%)	0.0	0.0	0.0	0.0	0.0
Dilutant	(g/t Au)	0.0	0.0	0.0	0.0	0.0
Mining Recovery	(%)	100.0	100.0	100.0	100.0	100.0
Au Metallurgical Recovery	(%)	91.2	91.3	91.3	91.3	91.4
Diluted						
Inventory	(Mt)	1.6	1.6	1.7	1.8	1.8
	(g/t Au)	1.57	1.55	1.54	1.53	1.52
	(koz Au)	79	82	85	87	89
Quantities						
Total Rock	(Mt)	8.3	9.1	9.8	10.5	11.3
Mineral Inventory	(Mt)	1.6	1.6	1.7	1.8	1.8
Waste + OM	(Mt)	6.8	7.5	8.1	8.7	9.4
Waste	(Mt)	6.5	7.2	7.8	8.4	9.1
Inventory (Below Cut-off)	(Mt)	0.2	0.2	0.3	0.3	0.3
Stripping Ratio	(t:t)	4.3	4.5	4.7	4.9	5.2
Product						
Recovered Metal	(koz Au)	72	75	77	79	82
Project Life	(years)	1.0	1.1	1.1	1.1	1.2

Table A 10: Case 3 – 1.5 Mtpa –Zone A

Optimisation Results	Units	1300 US$/oz	1400 US$/oz	1500 US$/oz	1600 US$/oz	1700 US$/oz
Insitu						
Inventory	(Mt)	1.2	1.3	1.4	1.5	1.8
	(g/t Au)	1.49	1.45	1.41	1.37	1.30
	(koz Au)	57	60	64	68	76
Modifying Factors						
Mining Dilution	(%)	0.0	0.0	0.0	0.0	0.0
Dilutant	(g/t Au)	0.0	0.0	0.0	0.0	0.0
Mining Recovery	(%)	100.0	100.0	100.0	100.0	100.0
Au Metallurgical Recovery	(%)	95.6	95.6	95.6	95.6	95.6
Diluted						
Inventory	(Mt)	1.2	1.3	1.4	1.5	1.8
	(g/t Au)	1.49	1.45	1.41	1.37	1.30
	(koz Au)	57	60	64	68	76
Quantities						
Total Rock	(Mt)	8.0	8.9	9.9	11.0	13.6
Mineral Inventory	(Mt)	1.2	1.3	1.4	1.5	1.8
Waste + OM	(Mt)	6.9	7.6	8.5	9.5	11.8
Waste	(Mt)	6.5	7.2	8.0	9.0	11.2
Inventory (Below Cut-off)	(Mt)	0.3	0.4	0.4	0.5	0.6
Stripping Ratio	(t:t)	5.8	5.9	6.0	6.1	6.5
Product						
Recovered Metal	(koz Au)	54	58	61	65	73
Project Life	(years)	2.4	2.6	2.8	3.1	3.6

Table A 11: Case 3 – 1.5 Mtpa – Zone B North

Optimisation Results	Units	1300 US$/oz	1400 US$/oz	1500 US$/oz	1600 US$/oz	1700 US$/oz
Insitu						
Inventory	(Mt)	1.6	1.6	1.7	1.8	1.8
	(g/t Au)	1.57	1.55	1.54	1.53	1.52
	(koz Au)	79	82	85	87	89
Modifying Factors						
Mining Dilution	(%)	0.0	0.0	0.0	0.0	0.0
Dilutant	(g/t Au)	0.0	0.0	0.0	0.0	0.0
Mining Recovery	(%)	100.0	100.0	100.0	100.0	100.0
Au Metallurgical Recovery	(%)	91.2	91.3	91.3	91.3	91.4
Diluted						
Inventory	(Mt)	1.6	1.6	1.7	1.8	1.8
	(g/t Au)	1.57	1.55	1.54	1.53	1.52
	(koz Au)	79	82	85	87	89
Quantities						
Total Rock	(Mt)	8.3	9.1	9.8	10.5	11.3
Mineral Inventory	(Mt)	1.6	1.6	1.7	1.8	1.8
Waste + OM	(Mt)	6.8	7.5	8.1	8.7	9.4
Waste	(Mt)	6.5	7.2	7.8	8.4	9.1
Inventory (Below Cut-off)	(Mt)	0.2	0.2	0.3	0.3	0.3
Stripping Ratio	(t:t)	4.3	4.5	4.7	4.9	5.2
Product						
Recovered Metal	(koz Au)	72	75	77	79	82
Project Life	(years)	1.0	1.1	1.1	1.1	1.2

Table A 12: Case 3 – 1.5 Mtpa – Zone B

Optimisation Results	Units	1301 US$/oz	1400 US$/oz	1500 US$/oz	1600 US$/oz	1700 US$/oz
Insitu						
Inventory	(Mt)	0.4	0.5	0.7	0.8	0.9
	(g/t Au)	1.69	1.74	1.61	1.55	1.44
	(koz Au)	20	27	35	38	42
Modifying Factors						
Mining Dilution	(%)	0.0	0.0	0.0	0.0	0.0
Dilutant	(g/t Au)	0.0	0.0	0.0	0.0	0.0
Mining Recovery	(%)	100.0	100.0	100.0	100.0	100.0
Au Metallurgical Recovery	(%)	95.1	95.3	95.4	95.4	95.4
Diluted						
Inventory	(Mt)	0.4	0.5	0.7	0.8	0.9
	(g/t Au)	1.69	1.74	1.61	1.55	1.44
	(koz Au)	20	27	35	38	42
Quantities						
Total Rock	(Mt)	3.8	5.7	8.2	9.3	10.5
Mineral Inventory	(Mt)	0.4	0.5	0.7	0.8	0.9
Waste + OM	(Mt)	3.5	5.3	7.5	8.6	9.6
Waste	(Mt)	3.4	5.2	7.4	8.4	9.4
Inventory (Below Cut-off)	(Mt)	0.1	0.1	0.2	0.2	0.2
Stripping Ratio	(t:t)	9.2	11.1	11.3	11.2	10.6
Product						
Recovered Metal	(koz Au)	19	25	33	36	40
Project Life	(years)	0.8	1.0	1.3	1.5	1.8

APPENDIX

B METAL PRICE SENSITIVITY

Case 1

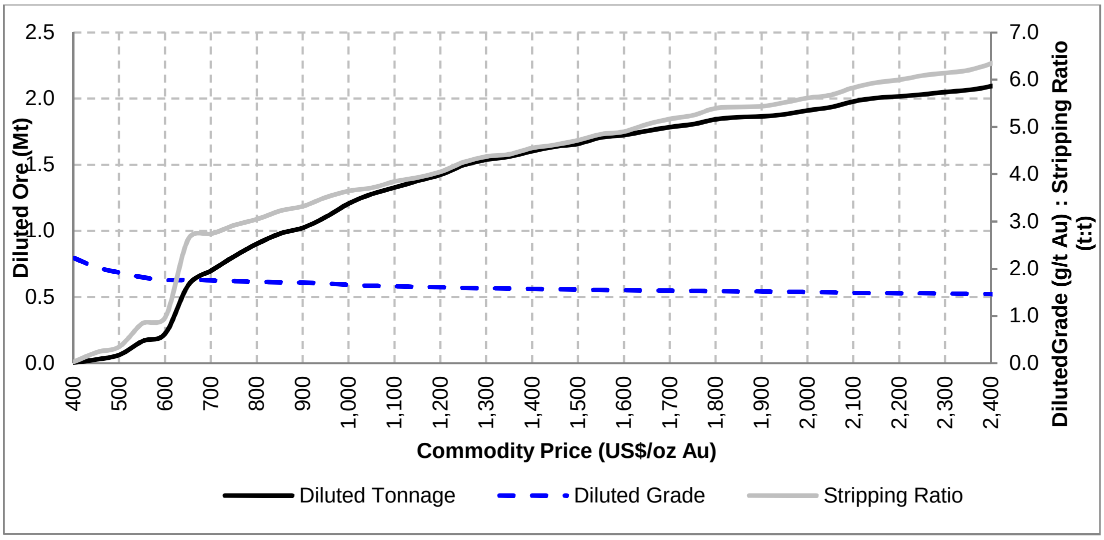

Figure B 1: Selin - Metal Price Sensitivity

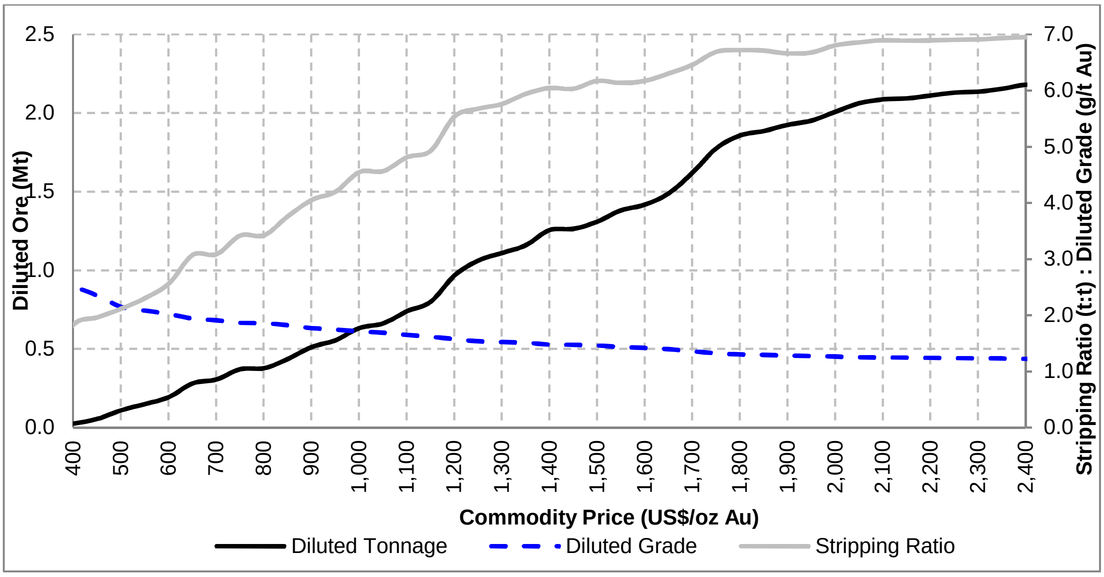

Figure B 2: Zone A - Metal Price Sensitivity

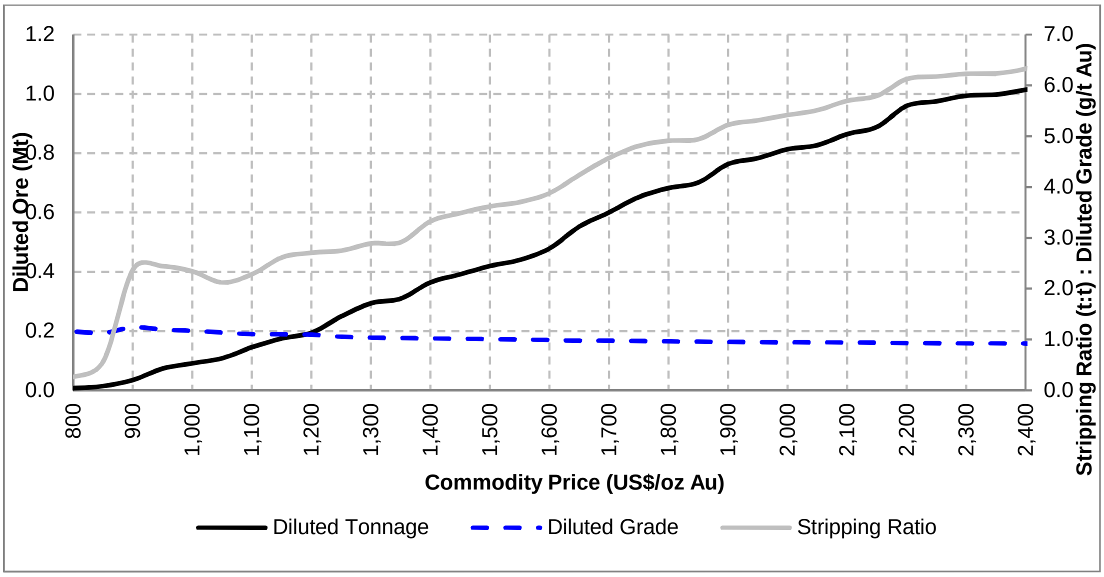

Figure B 3: Zone B North - Metal Price Sensitivity

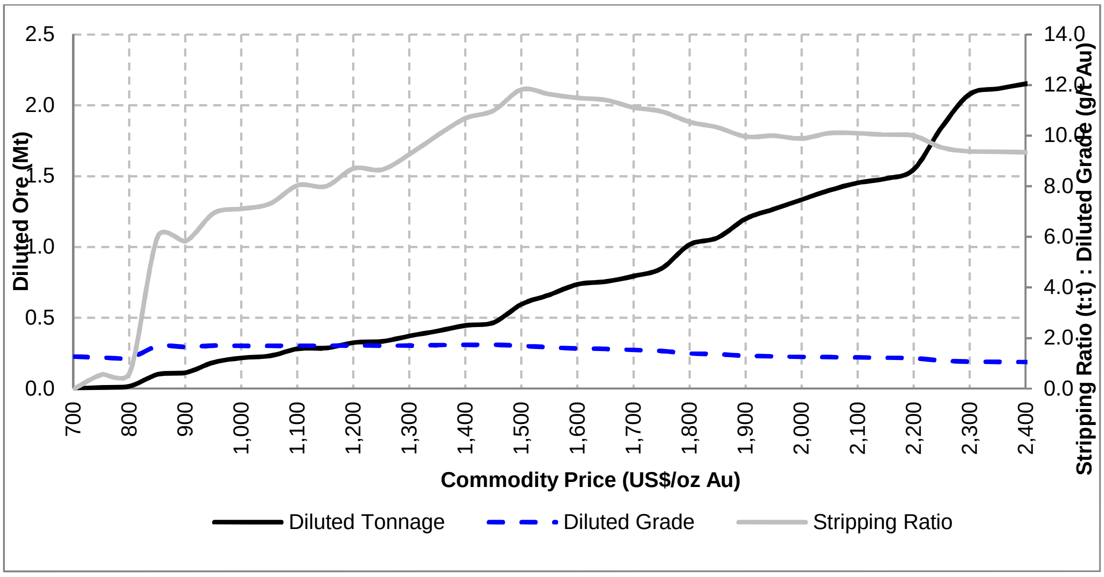

Figure B 4: Zone B - Metal Price Sensitivity

Case 3

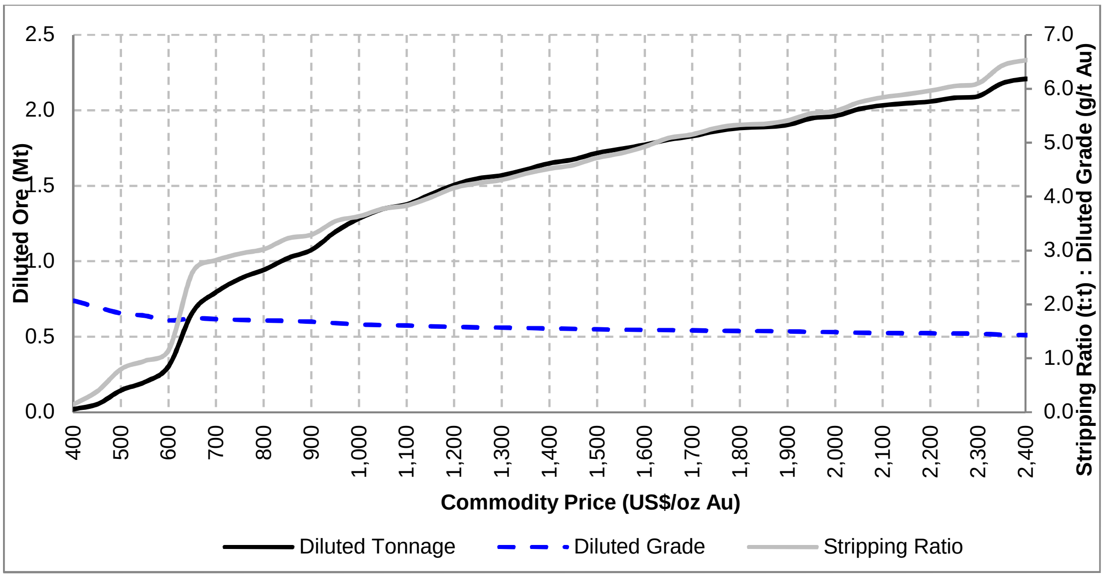

Figure B 5: Selin - Metal Price Sensitivity

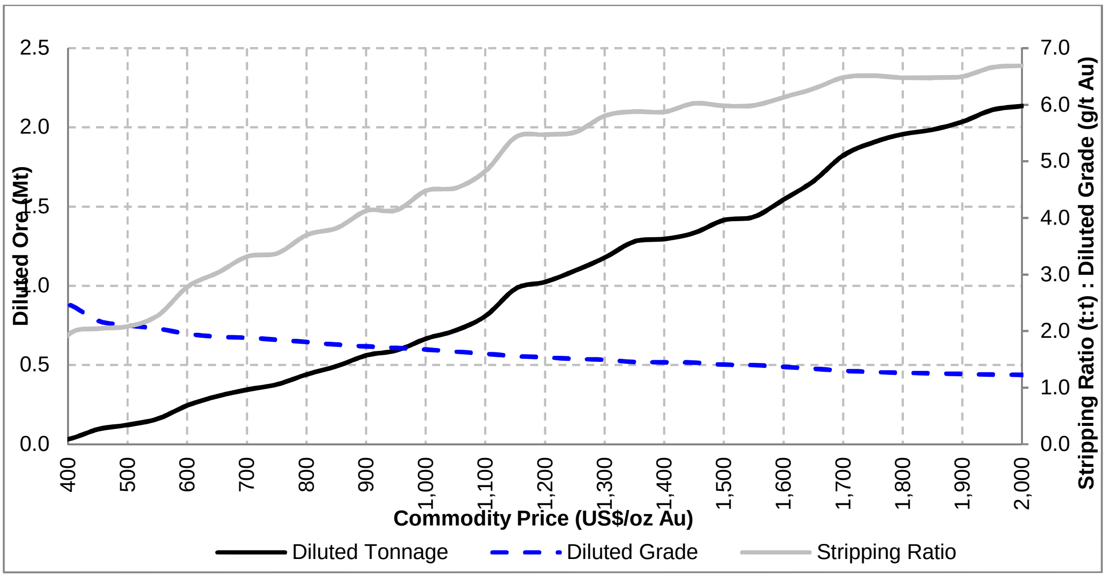

Figure B 6: Zone A - Metal Price Sensitivity

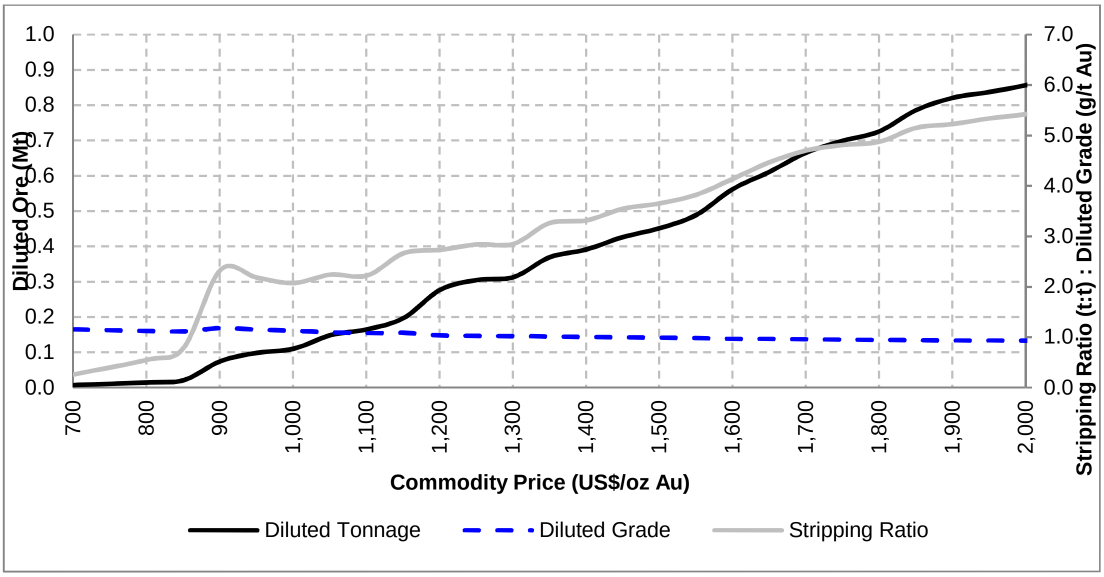

Figure B 7: Zone B North - Metal Price Sensitivity

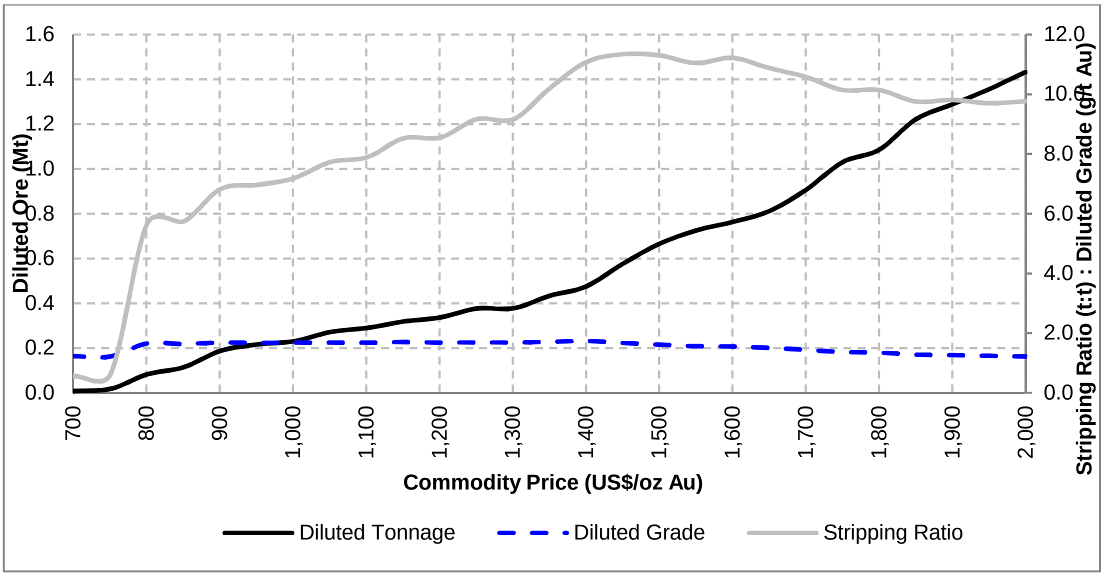

Figure B 8: Zone B - Metal Price Sensitivity

APPENDIX

C LOADER PRODUCTIVITY ESTIMATIONS

Table C 1: Case 1 – 0.5 Mtpa Loader Productivity Estimate

Material Type	Units	Selin - Ore	Selin - Waste	Zone B North - Ore	Zone B North - Waste	Zone A - Ore	Zone A - Waste	Zone B - Ore	Zone B - Waste
Loading Unit		Primary Backhoe	Primary Wheel Loader	Primary Backhoe	Primary Wheel Loader	Primary Backhoe	Primary Wheel Loader	Primary Backhoe	Primary Wheel Loader
Bucket Size	(m3)	1.9	7.1	1.9	7.1	1.9	7.1	1.9	7.1
Loading Spot Time	(min.)	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50
Loading Cycle Time	(min.)	0.67	0.67	0.67	0.67	0.67	0.67	0.67	0.67
First Bucket Dump	(min.)	0.33	0.33	0.33	0.33	0.33	0.33	0.33	0.33
Haulage Unit		Primary Truck	Secondary Truck	Primary Truck	Secondary Truck	Primary Truck	Secondary Truck	Primary Truck	Secondary Truck
Capacity	(t)	24.0	41.0	24.0	41.0	24.0	41.0	24.0	41.00
Capacity	(m3)	10.9	18.6	10.9	18.6	10.9	18.6	10.9	18.64
Truck Fill Factor	(%)	100	100	100	100	100	100	100	100
Capacity	(t)	24.0	41.0	24.0	41.0	24.0	41.0	24.0	41.0
Capacity	(m3)	10.9	18.6	10.9	18.6	10.9	18.6	10.9	18.6
Dump & Spot Time	(min.)	5.88	5.88	5.88	5.88	5.88	5.88	5.88	5.88
Loading Parameters									
Bucket Fill Factor	(%)	2.20	2.25	2.20	2.30	2.20	2.20	2.20	2.20
In-Situ Density	(t/bcm)	1.30	1.30	1.30	1.30	1.30	1.30	1.30	1.30
Swell Factor	(lcm/bcm)	1.69	1.73	1.69	1.77	1.69	1.69	1.69	1.69
Loose Dry Density	(dt/lcm)	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00
Moisture Factor	(%)	7.5	5.7	7.5	5.5	7.5	5.8	7.5	6.1
Passes	(#)	7	6	7	6	7	6	7	6
Passes (Rounded)	(#)								
Loaded Quantity	(t)	18.5	32.3	18.5	33.0	18.5	31.5	18.5	31.5
Loaded Volume	(m3)	10.9	18.6	10.9	18.6	10.9	18.6	10.9	18.6
Loading Productivity									
Theoretical Loading Cycle Time	(min.)	4.83	4.17	4.83	4.17	4.83	4.17	4.83	4.17
Job Efficiency Factor	(%)	85	85	85	85	85	85	85	85
Adjusted Loading Cycle Time	(min.)	5.69	4.90	5.69	4.90	5.69	4.90	5.69	4.90
Loader Productivity	**(dt/doh)**	186	376	186	384	186	368	186	368
Operating Efficiency	(%)	85.0	83.0	85.0	83.0	85.0	83.0	85.0	83.0
Loader Productivity	(t/op. hr)	158	312	158	319	158	305	158	305
Loading Unit Utilisation	(%)	65.1	56.1	65.1	56.1	65.1	56.1	65.1	56.1
Loading Productivity	**(Mtpa)**	2.20	2.25	2.20	2.30	2.20	2.20	2.20	2.20

Table C 2: Case 3 – 1.5 Mtpa Loader Productivity Estimate

Material Type	Units	Selin - Ore	Selin - Waste	Zone B North - Ore	Zone B North - Waste	Zone A - Ore	Zone A - Waste	Zone B - Ore	Zone B - Waste
Loading Unit		Primary Backhoe	Primary Wheel Loader	Primary Backhoe	Primary Wheel Loader	Primary Backhoe	Primary Wheel Loader	Primary Backhoe	Primary Wheel Loader
Bucket Size	(m3)	1.9	7.1	1.9	7.1	1.9	7.1	1.9	7.1
Loading Spot Time	(min.)	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50
Loading Cycle Time	(min.)	0.67	0.67	0.67	0.67	0.67	0.67	0.67	0.67
First Bucket Dump	(min.)	0.33	0.33	0.33	0.33	0.33	0.33	0.33	0.33
Haulage Unit		Primary Truck	Secondary Truck	Primary Truck	Secondary Truck	Primary Truck	Secondary Truck	Primary Truck	Secondary Truck
Capacity	(t)	24.0	54.0	24.0	54.0	24.0	54.0	24.0	54.00
Capacity	(m3)	10.9	24.5	10.9	24.5	10.9	24.5	10.9	24.55
Truck Fill Factor	(%)	100	100	100	100	100	100	100	100
Capacity	(t)	24.0	54.0	24.0	54.0	24.0	54.0	24.0	54.0
Capacity	(m3)	10.9	24.5	10.9	24.5	10.9	24.5	10.9	24.5
Dump & Spot Time	(min.)	5.88	5.88	5.88	5.88	5.88	5.88	5.88	5.88
Loading Parameters									
Bucket Fill Factor	(%)	2.20	2.25	2.20	2.30	2.20	2.20	2.20	2.20
In-Situ Density	(t/bcm)	1.30	1.30	1.30	1.30	1.30	1.30	1.30	1.30
Swell Factor	(lcm/bcm)	1.69	1.73	1.69	1.77	1.69	1.69	1.69	1.69
Loose Dry Density	(dt/lcm)	5.00	5.00	5.00	5.00	5.00	5.00	5.00	5.00
Moisture Factor	(%)	7.9	3.9	7.9	3.8	7.9	4.0	7.9	4.0
Passes	(#)	8	4	8	4	8	4	8	4
Passes (Rounded)	(#)								
Loaded Quantity	(t)	18.5	42.5	18.5	43.4	18.5	41.5	18.5	41.5
Loaded Volume	(m3)	10.9	24.5	10.9	24.5	10.9	24.5	10.9	24.5
Loading Productivity									
Theoretical Loading Cycle Time	(min.)	5.50	2.83	5.50	2.83	5.50	2.83	5.50	2.83
Job Efficiency Factor	(%)	85	85	85	85	85	85	85	85
Adjusted Loading Cycle Time	(min.)	6.47	3.33	6.47	3.33	6.47	3.33	6.47	3.33
Loader Productivity	**(dt/doh)**	163	728	163	744	163	712	163	712
Operating Efficiency	(%)	85.0	83.0	85.0	83.0	85.0	83.0	85.0	83.0
Loader Productivity	(t/op. hr)	139	605	139	618	139	591	139	591
Loading Unit Utilisation	(%)	65.1	56.1	65.1	56.1	65.1	56.1	65.1	56.1
Loading Productivity	**(Mtpa)**	0.9	3.6	0.9	3.7	0.9	3.5	0.9	3.5

APPENDIX 3: Metallurgical Testing on Samples of Oxide Mineralisation
(Completed by WAI)

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CORA GOLD

SANANKORO GOLD PROJECT, MALI

METALLURGICAL TESTING ON SAMPLES OF OXIDE MINERALISATION

July 2019

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CORA GOLD

SANANKORO GOLD PROJECT, MALI

METALLURGICAL TESTING ON SAMPLES OF OXIDE MINERALISATION

July 2019

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EXECUTIVE SUMMARY

Wardell Armstrong International (WAI) was commissioned by Cora Gold (the Client) to undertake a programme of metallurgical testing on samples of oxide gold mineralisation from the Sanankoro deposit, Mali.

Initially, testing was undertaken using two samples of exploration drill core representing different areas of the Sanankoro deposit and consisted of a programme of testing including; head assay, coarse ore bottle roll testing and whole ore leach testing.

Subsequently, a Master Composite sample was prepared by blending the two samples. This sample was then subjected to a further programme of testing consisting of; head assay, gravity-leach testing, whole ore leach testing, agglomeration & percolation testing and column leach testing.

Phase 1

Head Assay

Detailed chemical head assay was performed on each of the samples submitted for testing. A summary of the results is given in the Table below.

Phase 1 Head Assay Results			
Element	Units	Assay	
		SD0005	SD0006
$Au_{(FA)}$	ppm	0.61	3.35
$Au_{(AR)}$	ppm	0.64	2.70
Ag	ppm	<0.5	1.2
Cu	%	0.005	0.008
Pb	%	0.003	0.002
Zn	%	0.004	0.010
Fe	%	2.31	4.35
As	%	0.046	0.028
$S_{(TOT)}$	%	0.022	0.046
$C_{(TOT)}$	%	0.027	0.13

Coarse Ore Bottle Roll Testing

Coarse ore bottle roll testing was performed to provide an indication of the maximum gold and silver recoveries achievable at coarse particle sizes, typical of those used in heap leach operations.

Testing was conducted to investigate the effect of crush size on leach response with each sample tested at three crush sizes; -20.0mm, -12.5mm and -6.3mm, in duplicate.

A summary of the average gold and silver recoveries achieved for each sample/crush size is given in the Table below.

Summary of Coarse Ore Bottle Roll Leach Test Results					
Sample	Crush Size	Reagent Consumption (kg/t)		Recovery (%)	
		Lime	Cyanide	Au	Ag
SD0005	-20.0mm	1.20	0.45	78.4	14.8
	-12.5mm	1.08	0.71	84.2	21.9
	-6.3mm	1.00	0.53	97.6	17.9
SD0006	-20.0mm	1.98	0.57	66.8	35.1
	-12.5mm	2.08	0.62	81.6	45.4
	-6.3mm	1.80	0.76	93.1	47.9

Whole Ore Leach Testing

A single agitated leach test was performed on each of the samples to determine metal recoveries achievable at fine grind sizes typical of CIL type operations.

Results are summarised in the Table below.

Phase 1, Whole Ore Cyanide Leach Test Results				
Sample	Reagent Consumption (kg/t)		Recovery (%)	
	Lime	Cyanide	Au	Ag
SD0005	0.72	1.35	97.4	32.7
SD0006	1.49	1.50	96.7	67.3

Phase 2

Head Assay

Following preparation of the Master Composite, a representative sub-sample was submitted for head assay for gold and silver. Results are summarised in the Table below.

Master Composite Head Assay Result		
Element	Units	Assay
$Au_{(AR)}$	ppm	2.74
Ag	ppm	0.8

Gravity-Leach Testing

Gravity-leach testing was undertaken to investigate the total amount of gold recoverable through the combination of gravity preconcentration followed by cyanide leaching of the gravity tailings.

Results of the gravity preconcentration stage are summarised in the Table below.

Master Composite Gravity Concentration Test Results					
Product	**Stage**	**Grind Size (µm)**	**Mass (%)**	**Assay, Au (ppm)**	**Recovery, Au (%)**
Concentrate	1	212	0.34	406	50.65
	2	75	0.36	167	22.37
	Total	**-**	***0.70***	***282***	***73.02***
Tailings	-	-	99.30	0.74	26.98
Feed	-	-	100.00	2.71	-

Following completion of the gravity testing, the gravity tailings were subjected to kinetic cyanide leach testing at two cyanide concentrations. Results of these tests are summarised in the Table below,

Master Composite Gravity Tailings Cyanide Leach Test Results				
Cyanide Concentration (g/L)	**Reagent Consumption (kg/t)**		**Recovery (%)**	
	Lime	**Cyanide**	**Au**	**Ag**
1.0	0.49	1.21	94.1	34.1
0.5	0.87	0.53	92.1	36.2

Whole Ore Leach Testing

A series of whole ore leach tests were conducted to investigate the amount of gold and silver that could be recovered from the Master Composite through direct cyanide leaching at fine particle sizes.

Testing investigated two key variables; grind size and cyanide concentration. A summary of the results is given in the Table below.

Master Composite Whole Ore Cyanide Leach Test Results					
Grind Size (µm)	**Cyanide Concentration (g/L)**	**Reagent Consumption (kg/t)**		**Recovery (%)**	
		Lime	**Cyanide**	**Au**	**Ag**
150µm	1.0	0.88	1.21	95.0	58.3
125µm	1.0	1.10	1.09	98.0	62.4
106µm	1.0	1.02	1.17	95.3	53.6
75µm	1.0	1.00	0.98	98.0	59.2
75µm	0.5	1.22	0.49	97.3	56.6
75µm	0.25	1.53	0.08	92.9	62.4

Agglomeration & Percolation Testing

Agglomeration and percolation testing was performed to investigate the need to agglomerate the Master Composite with cement prior to column leaching. The sample was subjected to five percolation tests, four of which were performed on material which had been agglomerated with cement.

A summary of the results is given below.

Master Composite Percolation Test Results			
Cement Addition (kg/t)	Drainage Flowrate ($l/m^2/hr$)		
	Minimum	Maximum	Average
0	14	45	**25**
5	587	965	**780**
10	2,542	3,940	**3,136**
15	4,724	11,630	**7,208**
22.5	10,002	18,773	**12,794**

Column Leach Test

A single column leach test was conducted on the Master Composite to provide an indication of the gold and silver recoveries and leach kinetics achievable under heap leach conditions.

A 40kg sample of the Master Composite was subjected to testing for a total of 105 days (95 days under direct irrigation) using a 1.0g/L cyanide solution at a target application rate of $10l/m^2/hr$. Regular samples of the pregnant leach solution (PLS) were taken in order to measure levels of gold and silver extraction after which, the solution was passed through a column containing activated carbon in order to also determine metal recoveries onto the loaded carbon.

Results of the column leach test are summarised in the Table below.

Master Composite Column Leach Test Results					
Reagent Consumption (kg/t)		Recovery to PLS (%)		Recovery to Carbon (%)	
Lime	Cyanide	Au	Ag	Au	Ag
0.12	0.62	56.0	37.0	56.3	42.9

1 INTRODUCTION

Wardell Armstrong International (WAI) was commissioned by Cora Gold (the Client) to undertake a programme of metallurgical testing on samples of oxide gold mineralisation from the Sanankoro deposit, Mali.

The objective of the test programme was to evaluate the amount of gold that could be recovered from the material primarily by means of cyanide leaching with testing undertaken both at coarse particle sizes, indicative of heap leaching, and fine sizes, indicative of agitated tank leaching. Testing was also undertaken to investigate whether the material was amenable to processing by means of gravity concentration.

Testing was conducted by Wardell Armstrong International between January and July 2019. A summary of the testwork findings are detailed within this report.

2 SAMPLE RECEIPT AND PREPARATION

2.1 Sample Receipt

A total of 107 intervals of exploration drill core representing two different areas of the Sanankoro deposit and weighing a total of 207.6kg was submitted to Wardell Armstrong in January 2019 for testing.

A summary of the samples received is shown below in Table 2.1 with full details given in Appendix 1.

Table 2.1: Summary of Samples Received		
Drill Hole	**# Intervals**	**Mass (kg)**
SD0005	46	85.50
SD0006	61	122.13

Upon receipt, each interval was weighed and logged into the laboratory sample tracking system. Selected intervals were also photographed as shown below in Photo 2.1 through Photo 2.4.

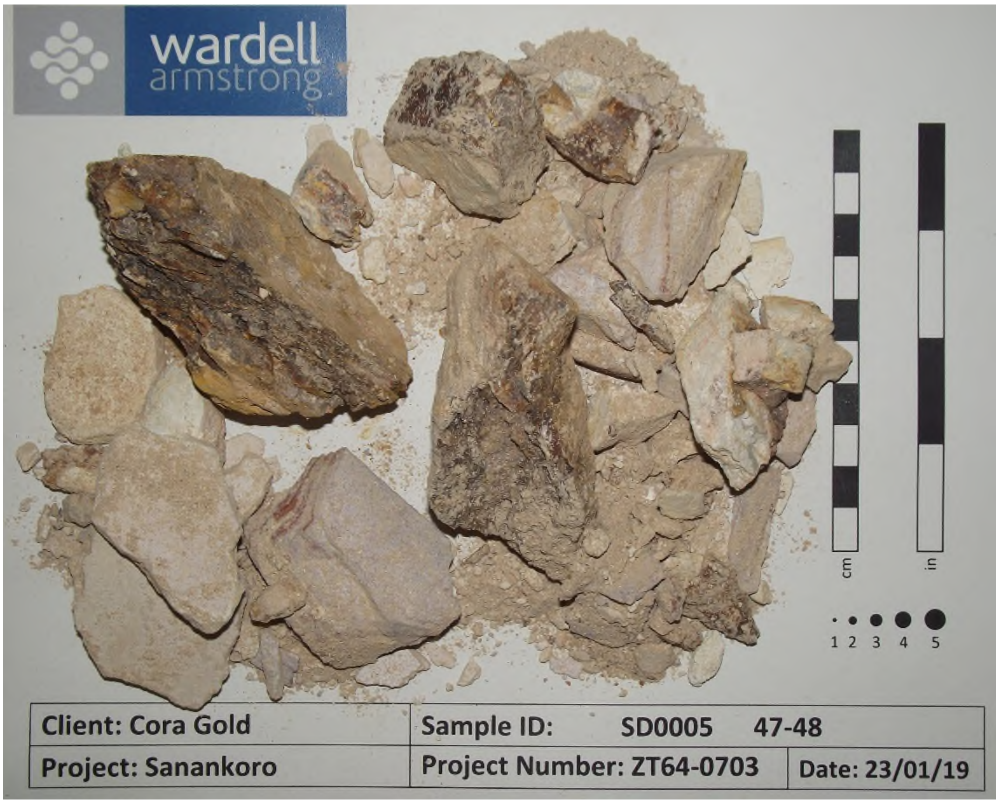


Photo 2.1: Photograph of SD0005, 47-48m Sample

Photo 2.2: Photograph of SD0005, 73-74m Sample

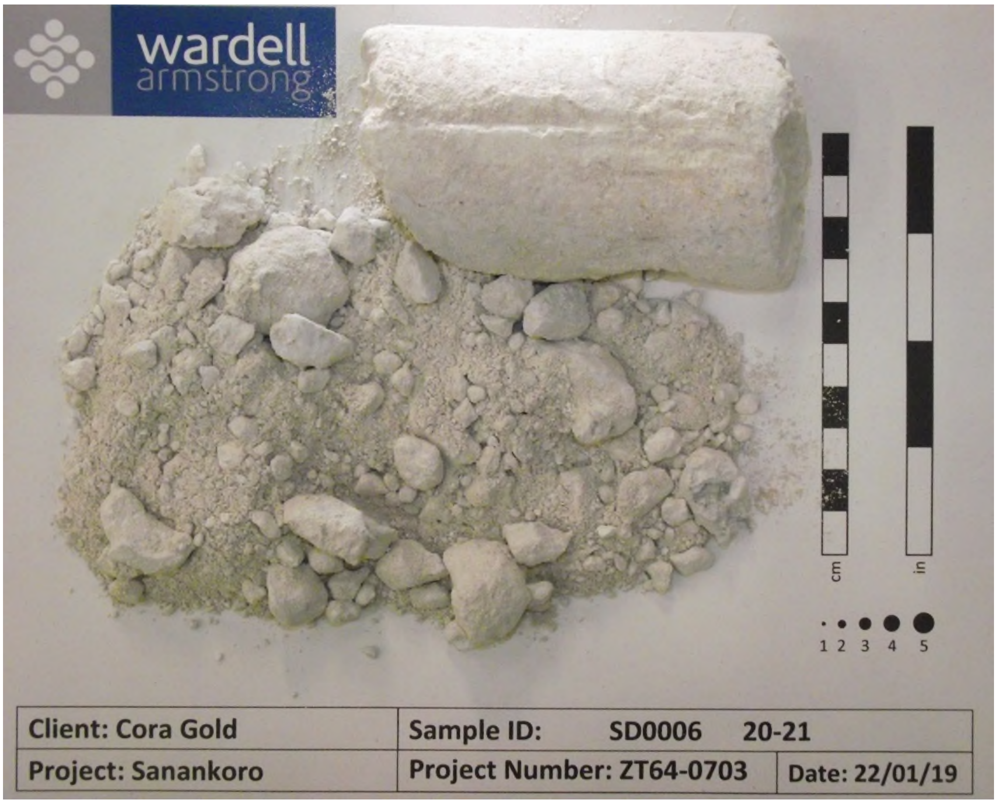


Photo 2.3: Photograph of SD0006, 20-21m Sample

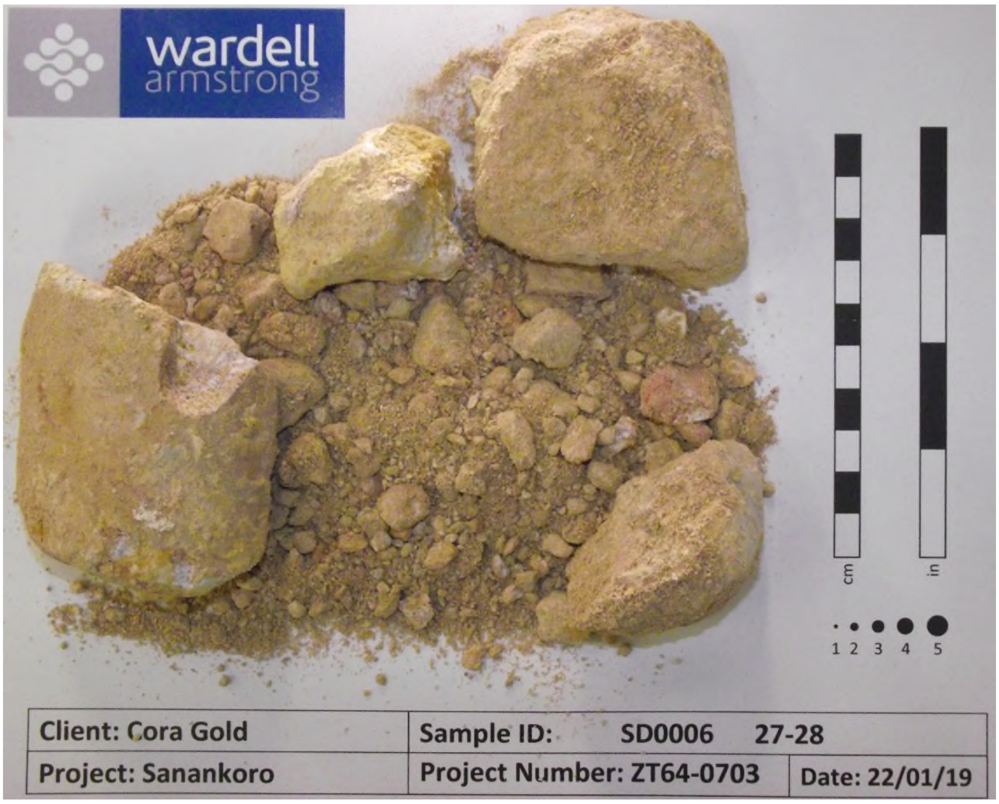


Photo 2.4: Photograph of SD0006, 27-28m Sample

2.2 Sample Preparation

2.2.1 Phase 1

Initially, sample preparation was undertaken on all of the intervals of the SD0005 sample and 45 of the 61 SD0006 intervals submitted as selected by the client. Full details of the intervals selected for testing are again given in Appendix 1.

Each sample was prepared by first stage crushing to 100% passing 20.0mm. The sample was then riffled to remove 20kg for head assay and leach testing, 6kg for coarse ore bottle roll testing and 12kg for further preparation. All remaining material was placed into storage pending further testing.

Following riffling, the 12kg sample was stage crushed to 100% passing 12.5mm and split into two equal 6kg samples. One of the 6kg samples was retained for coarse ore bottle roll testing whilst the second sample was crushed to 100% passing 6.3mm, again for coarse ore bottle roll testing at the finer particle size.

A summary of the sample preparation methodology is given overleaf in Figure 2.1.

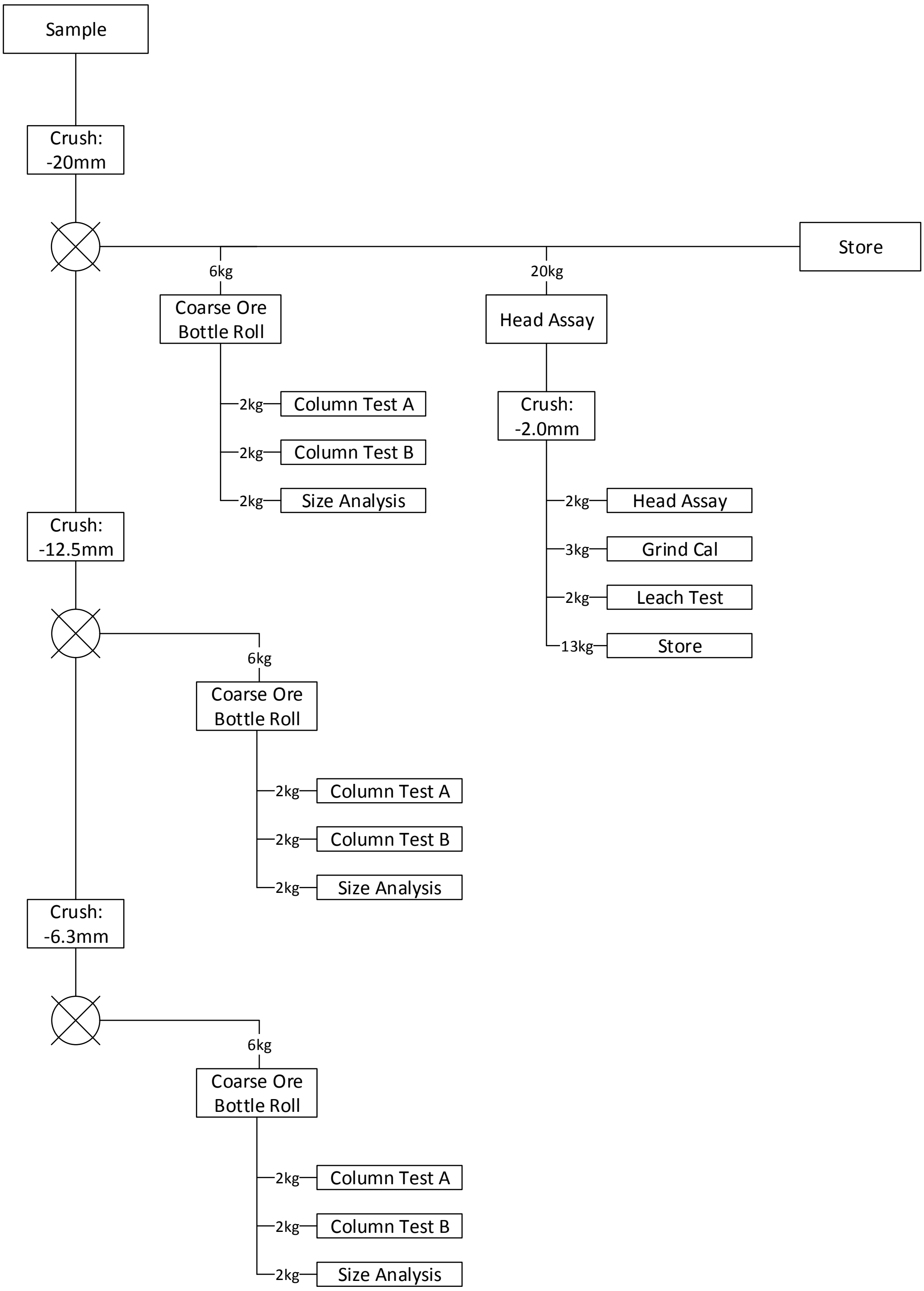


Figure 2.1: Phase 1 Sample Preparation Methodology

2.2.2 Phase 2

Following completion of the first phase of testing, a composite of the two samples was prepared for further testing using the residual -20.0mm material which had been placed into storage.

The composite sample was prepared using 47kg of SD0005 and 50kg of SD0006 which was mixed thoroughly prior to undergoing further preparation.

Once blended, a 53kg sub-sample was removed for agglomeration & percolation and column simulated heap leach testing with all remaining sample crushed to 100% passing 2.0mm. This material was then riffled into a number of 1kg sub-samples for gravity and leach testing.

A summary of the Phase 2 sample preparation methodology is given in Figure 2.2.

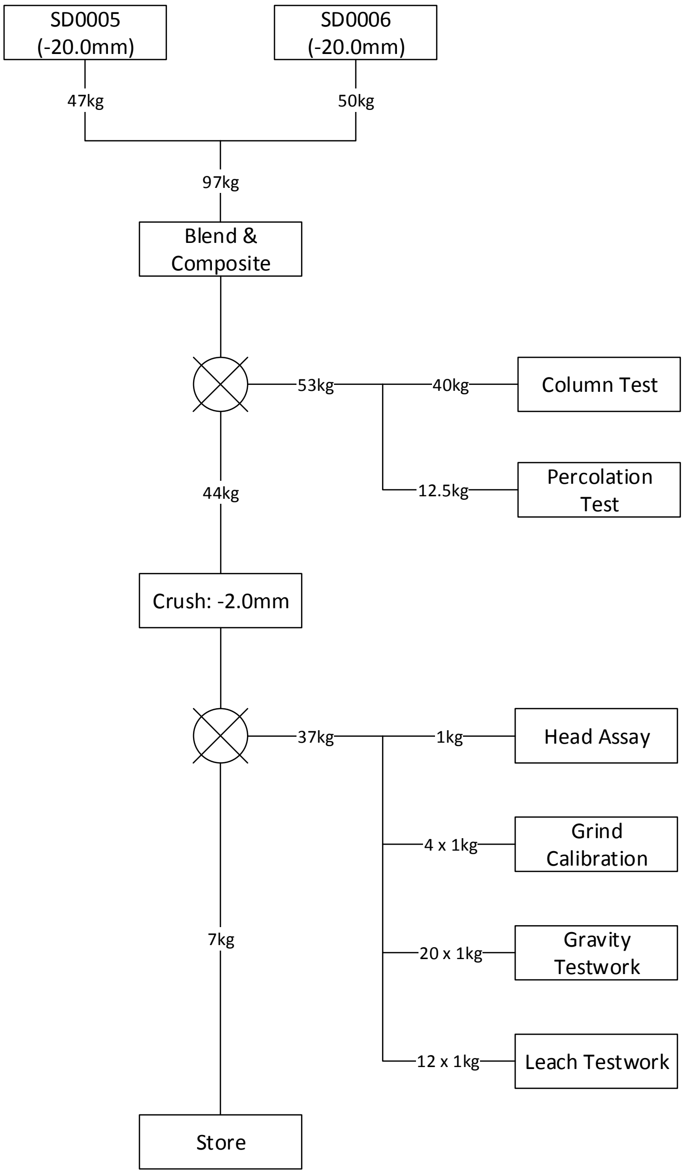


Figure 2.2: Phase 2 Sample Preparation Methodology

3 PHASE 1 TESTWORK

An initial programme of testing was conducted on each of the samples to investigate the gold recoveries achievable at both coarse and fine particle sizes. The programme of testing consisted of:

- Head Assay;
- Grind Calibration;
- Coarse Ore Bottle Roll Testing;
- Particle Size Analysis & Size-by-Size analysis for Gold; and
- Whole Ore Leach Testing.

Details of the test programme are given below.

3.1 Head Assay

Detailed chemical head assay was performed on each of the samples to determine the level of a range of elements of interest. The analyses were performed on a representative sub-sample of -2.0mm material which had been further crushed and pulverised to 100% passing 75µm.

Samples were analysed for gold by both Fire Assay (FA) and Aqua Regia (AR) methods along with separate assay for a range of base metals, sulphur and carbon. Results of the head assay are given in Table 3.1.

Table 3.1: Phase 1 Head Assay Results

Element	Units	Assay	
		SD0005	SD0006
$Au_{(FA)}$	ppm	0.61	3.35
$Au_{(AR)}$	ppm	0.64	2.70
Ag	ppm	<0.5	1.2
Cu	%	0.005	0.008
Pb	%	0.003	0.002
Zn	%	0.004	0.010
Fe	%	2.31	4.35
As	%	0.046	0.028
$S_{(TOT)}$	%	0.022	0.046
$C_{(TOT)}$	%	0.027	0.13

The results showed gold grades of 0.61g/t Au for the SD0005 sample and 3.35g/t Au for the SD0006 sample based on Fire Assay. Values based on the Aqua Regia analysis were 0.64g/t and 2.70g/t Au respectively.

Base metal levels in both samples were low as were the concentration of both total sulphur and total carbon.

3.2 Grind Calibration

A multi-point grind calibration was conducted on each of the samples to determine the length of time required to grind the samples to a product particle size of 80% passing 75µm.

The grind calibrations were performed by grinding two samples of -2.0mm material for 5 and 10 minutes respectively after which, the resulting products were sized into a number of discrete fractions and the particle size D_{80} calculated.

The resulting product sizes were then plotted against their respective grind times to generate a calibration curve which was used to interpolate the grinding time necessary to achieve the target particle size distribution.

The grind calibration curve for each of the samples is shown below in Figure 3.1. Full details are given in Appendix 2.

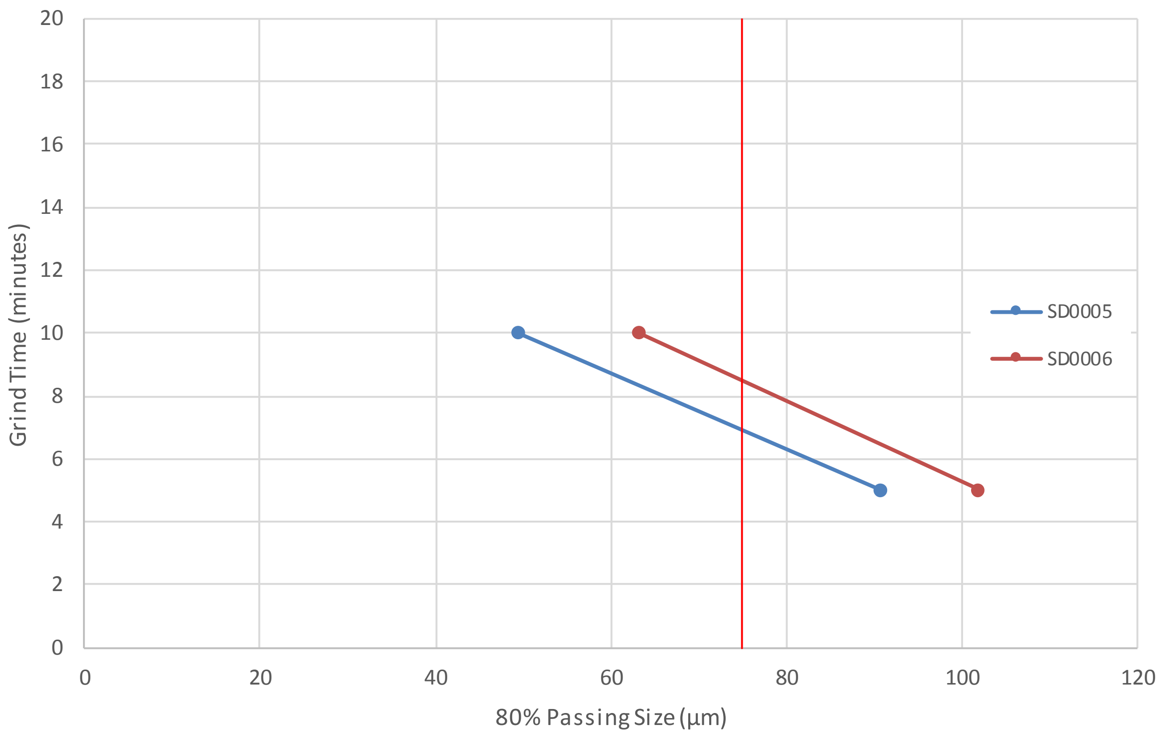


Figure 3.1: Phase 1 Grind Calibration Curves

3.3 Coarse Ore Bottle Roll Testing

Coarse ore bottle roll testing was conducted to identify the maximum gold and silver recoveries achievable from each of the samples at crush sizes typical of conventional heap leach operations.

The testing undertaken investigated three crush sizes; -20.0, -12.5 and -6.3mm with all testing conducted at a fixed cyanide concentration of 2.0g/L. All coarse ore bottle roll tests were performed in duplicate to minimise the effect of particle size related sampling variance on the leach results.

A summary of the coarse ore bottle roll test conditions is given below:

- Sample Size: 2kg (in duplicate),
- Method: Bottle Roll,
- Percent Solids: 40% w/w,
- Crush Size
  - i) -20.0mm,
  - ii) -12.5mm,
  - iii) -6.3mm,
- Leach Duration: 21 Days,
- Cyanide Concentration: 2.0g/L (maintained),
- pH: 10.5 – 11.0 (maintained),
- Solution Assays: Au & Ag,
- Solution Intervals: 1, 2, 4, 7, 10, 14, 17 & 21 Days,
- Tailings Assay: Au & Ag.

The results of the coarse ore bottle roll tests are given below in Section 3.3.1 and 3.3.2. Full results can be found in Appendix 3.

3.3.1 SD0005

The results of the coarse ore bottle roll tests conducted on the SD0005 sample are summarised overleaf in Table 3.2 and Figure 3.2.

Table 3.2: SD0005 Coarse Ore Bottle Roll Leach Test Results

Crush Size	Test	Reagent Consumption (kg/t)		Recovery (%)	
		Lime	Cyanide	Au	Ag
-20.0mm	A	1.25	0.23	79.9	8.7
	B	1.15	0.42	76.8	20.9
	Average	**1.20**	**0.45**	**78.4**	**14.8**
-12.5mm	A	1.05	0.90	74.8	21.9
	B	1.10	0.53	93.5	21.9
	Average	**1.08**	**0.71**	**84.2**	**21.9**
-6.3mm	A	1.00	0.45	98.3	19.9
	B	1.00	0.61	97.0	15.9
	Average	**1.00**	**0.53**	**97.6**	**17.9**

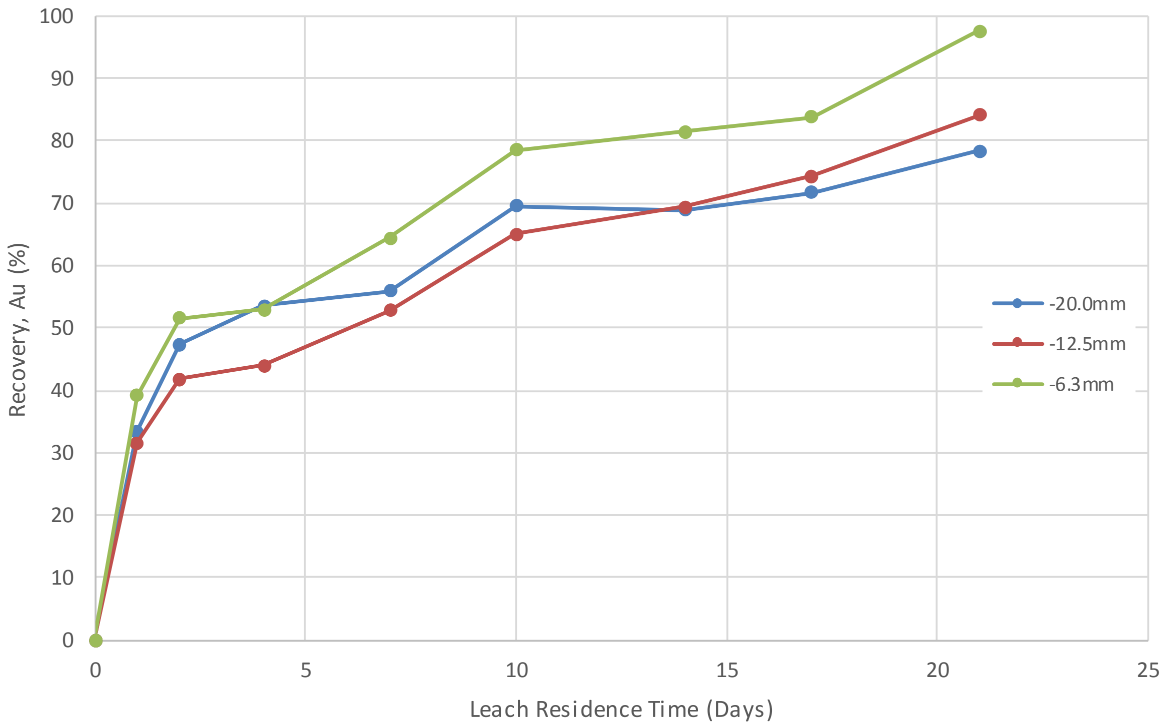


Figure 3.2: Gold Recovery vs. Residence Time, SD0005 Coarse Ore Bottle Roll Tests

The results showed average final gold recoveries after 21 days of leaching to range from 78.4% at the -20.0mm crush size to 97.6% at the -6.3mm crush size, a difference of 19.2%. Silver recoveries ranged from 14.8% to 21.9%, a difference of 7.1%.

For all three of the crush sizes tested, the leach kinetics indicated that the extraction of gold had not reached equilibrium at the end of the test (Day 21) and that, if the leach residence time had been extended, higher gold recoveries may have been achieved.

With respect to reagent consumption, the results showed a reduction in lime consumption, reducing from 1.20kg/t for the -20.0mm material to 1.00kg/t for the -6.3mm material. Cyanide consumption ranged from 0.45kg/t (-20.0mm) to 0.71kg/t (-12.5mm).

3.3.2 SD0006

The results of the coarse ore bottle roll testing performed on the SD0006 are summarised below in Table 3.3 and Figure 3.3.

Table 3.3: SD0006 Coarse Ore Bottle Roll Leach Test Results

Crush Size	Test	Reagent Consumption (kg/t)		Recovery (%)	
		Lime	Cyanide	Au	Ag
-20.0mm	A	2.25	0.42	63.5	45.0
	B	1.70	0.72	70.0	25.3
	Average	**1.98**	**0.57**	**66.8**	**35.1**
-12.5mm	A	2.30	0.64	90.6	45.2
	B	1.85	0.60	72.6	45.6
	Average	**2.08**	**0.62**	**81.6**	**45.4**
-6.3mm	A	1.70	0.75	95.0	49.6
	B	1.90	0.76	91.3	46.2
	Average	**1.80**	**0.76**	**93.1**	**47.9**

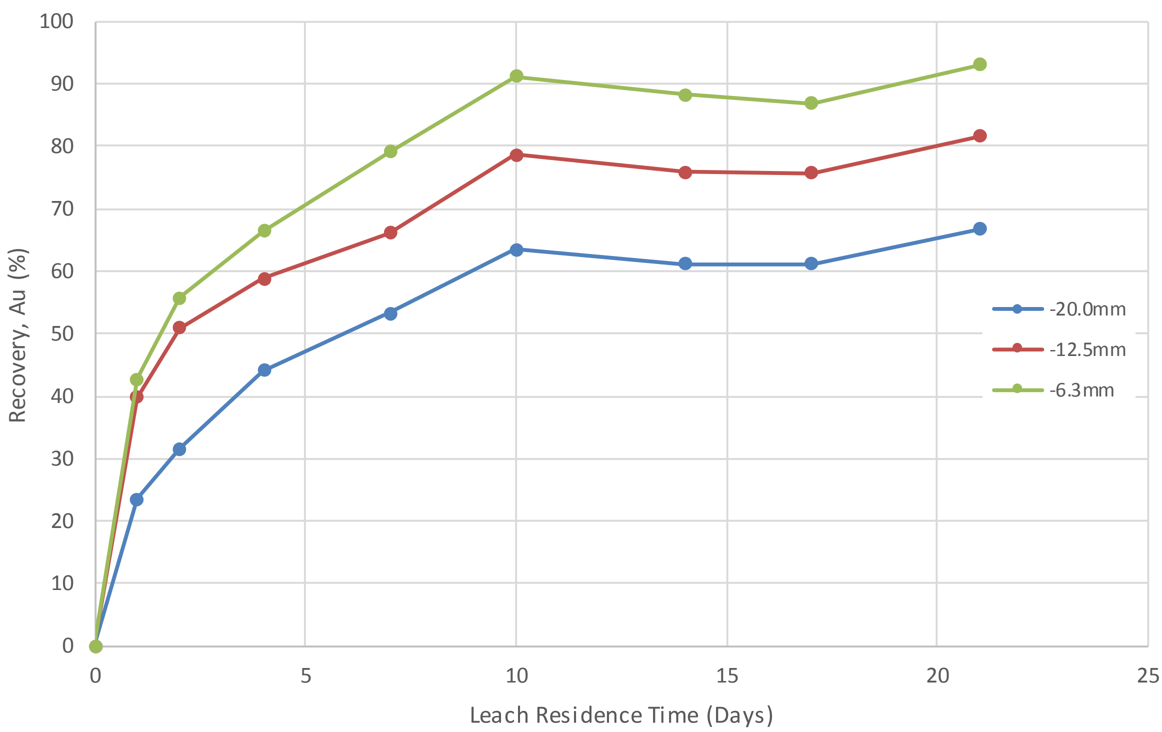


Figure 3.3: Gold Recovery vs. Residence Time, SD0006 Coarse Ore Bottle Roll Tests

The results of the testing showed gold recoveries to range from 66.8% at a crush size of -20.0mm to 93.1% at a crush size of -6.3mm, an increase of 26.3%, whilst silver recoveries ranged from 35.1% (-20.0mm) to 47.9% (-6.3mm), an increase of 12.8%.

Unlike the results for the SD0005 sample, analysis of the leach kinetics indicated that maximum gold recovery had been achieved around Day 10 of the test for all three of the crush sizes tested with some reduction in gold levels between this point and the end of the test (Day 21).

Although a trend such as this is symptomatic of preg-robbing, given the low carbon content of the sample, it is felt that the trend is most likely due to the presence of a high amount of fines in the sample as determined from the particle size analysis.

When compared with the results for the SD0005 sample, levels of both lime and cyanide consumption were higher than those observed previously with lime consumptions ranging from 1.80kg/t (-6.3mm) to 2.08kg/t (-12.5mm) whilst cyanide consumption ranged from 0.57kg/t (-20.0mm) to 0.76kg/t (-6.3mm).

The slightly higher cyanide consumptions when compared with the SD0005 sample is most likely due to the higher quantity of gold present in the sample.

3.4 Particle Size Analysis

Particle size analysis, including assay by size, was undertaken on samples of the bottle roll feed and leach residue to allow calculation of both the mass and gold distribution within each sample and gold recovery by particle size.

Results are summarised below with full data given in Appendix 4.

3.4.1 Bottle Roll Feed & Leach Residue Particle Size Analysis

3.4.1.1 SD0005

-20.0mm Crush Size

The distribution of mass and gold within the SD0005 -20.0mm bottle roll feed and leach residue is summarised overleaf in Table 3.4 and Figure 3.4.

Table 3.4: Particle & Gold Size Analysis, SD0005 -20.0mm, Bottle Roll Feed & Leach Residue								
Particle Size (mm)	Bottle Roll Feed				Bottle Roll Leach Residue			
	Mass		Au		Mass		Au	
	%	∑ % Passing	%	∑ % Passing	%	∑ % Passing	%	∑ % Passing
-20+12.5	3.29	96.71	6.92	93.08	3.10	96.90	0.79	99.21
-12.5+9.5	2.47	94.25	9.92	83.16	1.00	95.91	0.39	98.82
-9.5+6.3	3.81	90.44	34.42	48.74	2.59	93.31	5.66	93.16
-6.3+3.35	4.61	85.83	5.86	42.88	3.09	90.22	3.18	89.97
-3.35+1.18	6.08	79.74	9.41	33.47	3.57	86.66	76.49	13.48
-1.18+0.50	6.23	73.51	1.89	31.58	3.86	82.80	4.68	8.80
-0.50+0.15	8.83	64.68	1.51	30.07	7.42	75.37	1.03	7.77
-0.15	64.68	0.00	30.07	0.00	75.37	0.00	7.77	0.00
D_{80}		**1.28**		**9.21**		**0.31**		**3.07**

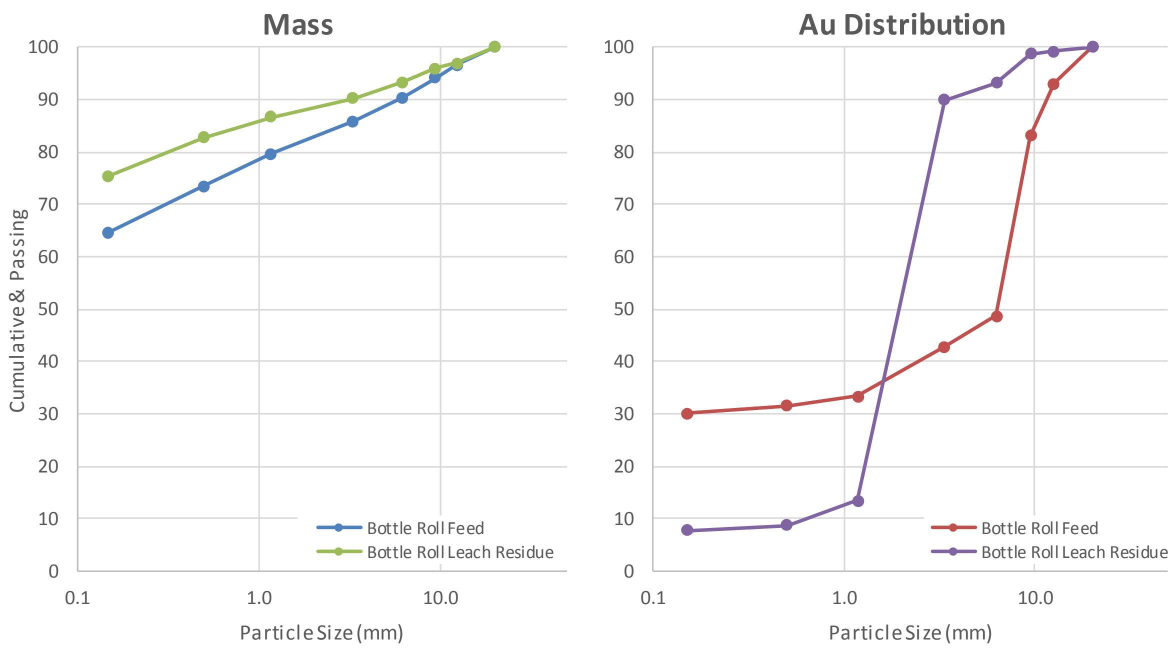


Figure 3.4: Mass & Gold Distribution, SD0005 -20.0mm Bottle Roll Feed & Leach Residue

The results of the size analysis of the bottle roll feed showed the material to have a D_{80} of 1.28mm which was considerably finer than the top size of -20.0mm. The discrepancy is most likely due to a combination of the natural softness of the material, being almost saprolitic in nature, combined potentially with some over crushing that may have occurred during preparation of the samples.

Despite the fine particle size distribution, the chemical analysis showed the gold to have maintained a relatively coarse distribution with a D_{80} value of 9.21mm.

Analysis of the bottle roll leach residue showed it to have a finer particle size distribution than that of the feed with a D_{80} of 0.31mm, indicating some break-up of material during testing which would again support the observation that the material is relatively soft and naturally breakable. The distribution of gold within the leach residue was also observed to have reduced in size with the D_{80} value falling to 3.07mm.

-12.5mm Crush Size

The distribution of mass and gold within the SD0005 -12.5mm bottle roll feed and leach residue is summarised below in Table 3.5 and Figure 3.5.

Table 3.5: Particle & Gold Size Analysis, SD0005 -12.5mm, Bottle Roll Feed & Leach Residue

Particle Size (mm)	Bottle Roll Feed				Bottle Roll Leach Residue			
	Mass		Au Distribution		Mass		Au Distribution	
	%	∑ % Passing	%	∑ % Passing	%	∑ % Passing	%	∑ % Passing
-12.5+9.5	2.27	97.73	1.98	98.02	2.76	97.24	19.22	80.78
-9.5+6.3	5.70	92.02	55.92	42.10	3.61	93.62	2.03	78.75
-6.3+3.35	5.38	86.64	2.62	39.48	3.55	90.07	9.44	69.31
-3.35+1.18	6.47	80.17	4.22	35.27	3.98	86.09	29.52	39.79
-1.18+0.50	6.39	73.78	25.80	9.47	3.43	82.66	1.19	38.60
-0.50+0.15	9.58	64.20	3.37	6.10	6.72	75.94	34.72	3.88
-0.15	64.20	0.00	6.10	0.00	75.94	0.00	3.88	0.00
D_{80}		**1.15**		**8.47**		**0.31**		**8.01**

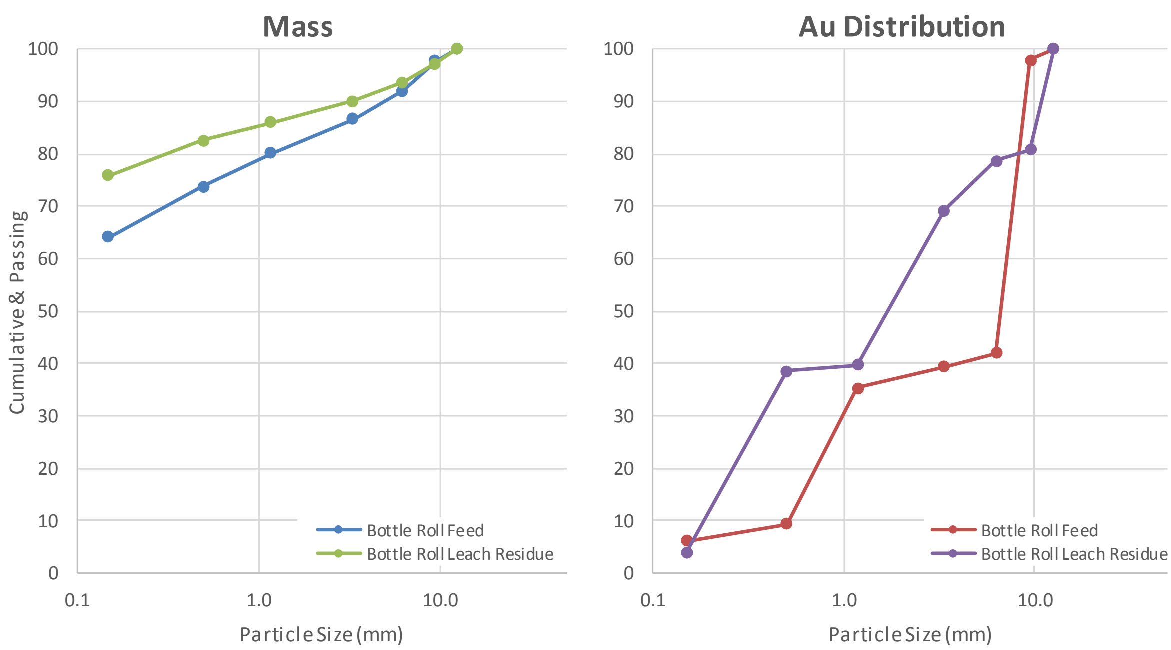


Figure 3.5: Mass & Gold Distribution, SD0005 -12.5mm Bottle Roll Feed & Leach Residue

Data showed the D_{80} of the bottle roll feed to be 1.15mm, again considerably below that of the particle top size, which reduced to 0.31mm following leaching.

Analysis of the distribution of gold within the samples showed the distribution to have remained broadly consistent with a D_{80} of 8.47mm in the feed compared with 8.01mm in the leach residue.

-6.3mm Crush Size

The distribution of mass and gold within the SD0005 -6.3mm bottle roll feed and leach residue is summarised below in Table 3.6 and Figure 3.6.

Table 3.6: Particle & Gold Size Analysis, SD0005 -6.3mm, Bottle Roll Feed & Leach Residue

Particle Size (mm)	Bottle Roll Feed				Bottle Roll Leach Residue			
	Mass		Au Distribution		Mass		Au Distribution	
	%	∑ % Passing	%	∑ % Passing	%	∑ % Passing	%	∑ % Passing
-6.3+3.35	5.17	94.83	25.70	74.30	4.63	95.37	37.80	62.20
-3.35+1.18	7.71	87.13	34.93	39.36	6.70	88.68	50.13	12.07
-1.18+0.50	6.49	80.64	10.80	28.56	5.25	83.43	1.55	10.53
-0.50+0.15	9.67	70.97	4.36	24.20	7.80	75.63	1.95	8.58
-0.15	70.97	0.00	24.20	0.00	75.63	0.00	8.58	0.00
D_{80}		**0.46**		**4.00**		**0.29**		**4.74**

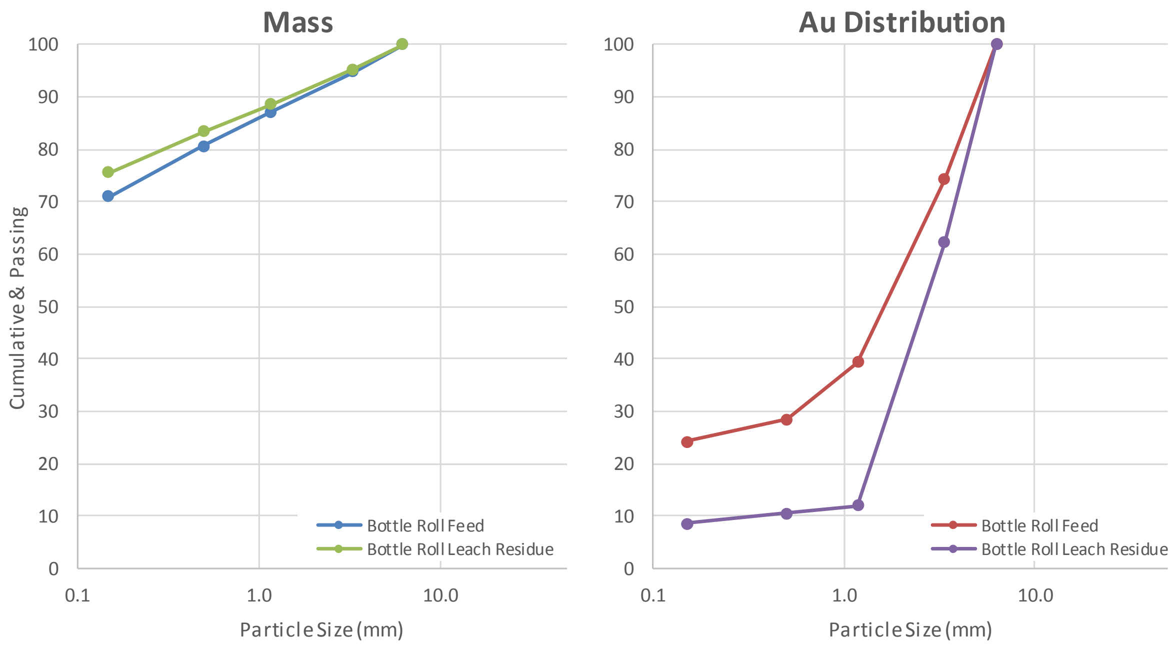


Figure 3.6: Mass & Gold Distribution, SD0005 -6.3mm Bottle Roll Feed & Leach Residue

The D_{80} of the bottle roll feed was 0.46mm which compared with a value of 0.29mm within the leach residue. Again, these figures were considerably finer than that of the intended particle top size.

With respect to gold distribution, the data showed this to be comparatively coarse with a D_{80} value of 4.00mm in the feed and 4.74mm in the leach residue which indicated preferential leaching of the gold within the finer size fractions.

3.4.1.2 SD0006

-20.0mm Crush Size

The distribution of mass and gold within the SD0006 -20.0mm bottle roll feed and leach residue is summarised below in Table 3.7 and Figure 3.7.

Table 3.7: Particle & Gold Size Analysis, SD0006 -20.0mm, Bottle Roll Feed & Leach Residue

Particle Size (mm)	Bottle Roll Feed				Bottle Roll Leach Residue			
	Mass		Au Distribution		Mass		Au Distribution	
	%	∑ % Passing	%	∑ % Passing	%	∑ % Passing	%	∑ % Passing
-20+12.5	8.46	91.54	5.22	94.78	3.67	96.33	6.06	93.94
-12.5+9.5	4.89	86.65	1.92	92.86	2.41	93.92	0.23	93.70
-9.5+6.3	6.74	79.91	13.84	79.02	3.67	90.25	64.17	29.53
-6.3+3.35	6.97	72.95	8.30	70.72	4.40	85.84	1.78	27.75
-3.35+1.18	7.95	64.99	13.25	57.46	4.35	81.49	22.37	5.38
-1.18+0.50	6.17	58.83	3.73	53.73	2.84	78.65	2.14	3.24
-0.50+0.15	8.40	50.43	2.23	51.50	4.92	73.73	1.04	2.21
-0.15	50.43	0.00	51.50	0.00	73.73	0.00	2.21	0.00
D_{80}		**6.40**		**6.53**		**0.74**		**8.82**

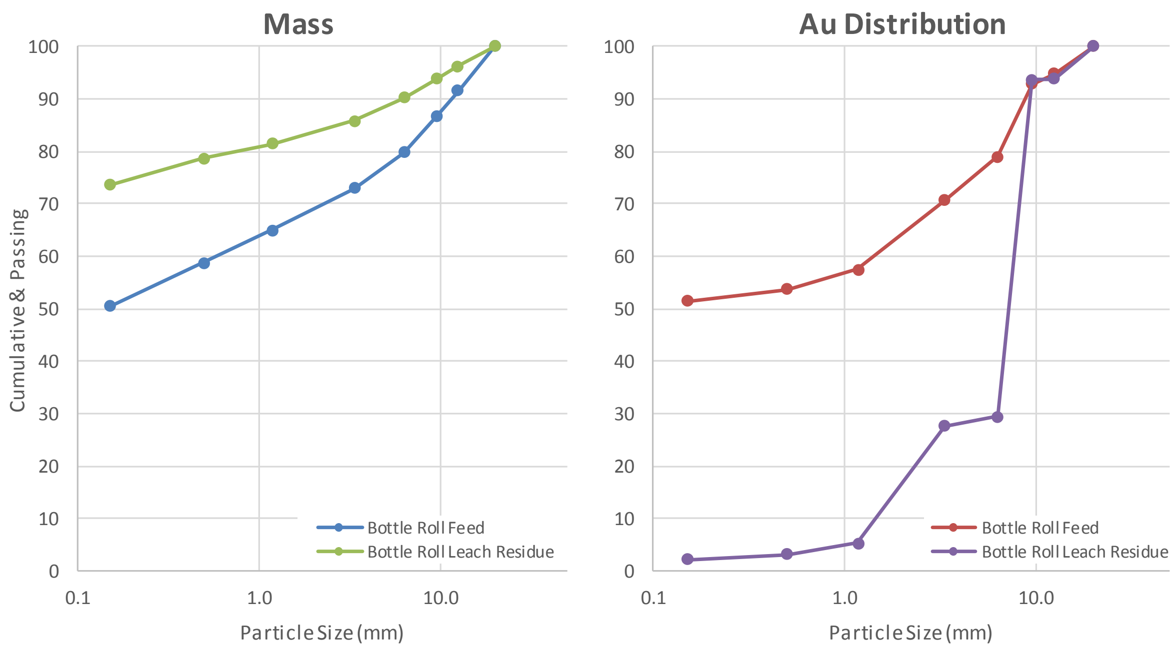


Figure 3.7: Mass & Gold Distribution, SD0006 -20.0mm Bottle Roll Feed & Leach Residue

The data showed the bottle roll feed to have a D_{80} of 6.40mm which was considerably coarser than that of the SD0005 sample at the same intended crush size. However, the distribution of gold was finer with 80% passing a particle size of 6.53mm.

Following leaching, particle size analysis showed the D_{80} of the leach residue to be 0.74mm indicating a high degree of material breakup during the leaching process. The D_{80} of the gold within the leach residue was 8.82mm which was coarser than that of the feed which demonstrates preferential leaching of the fine gold.

-12.5mm Crush Size

The distribution of mass and gold within the SD0006 -12.5mm bottle roll feed and leach residue is summarised in Table 3.8 and Figure 3.8.

Table 3.8: Particle & Gold Size Analysis, SD0006 -12.5mm, Bottle Roll Feed & Leach Residue

Particle Size (mm)	Bottle Roll Feed				Bottle Roll Leach Residue			
	Mass		Au Distribution		Mass		Au Distribution	
	%	∑ % Passing	%	∑ % Passing	%	∑ % Passing	%	∑ % Passing
-12.5+9.5	7.34	92.66	1.98	98.02	3.08	96.92	3.35	96.65
-9.5+6.3	7.98	84.67	37.90	60.13	4.64	92.28	18.01	78.65
-6.3+3.35	7.33	77.34	30.51	29.63	4.57	87.71	66.22	12.42
-3.35+1.18	8.04	69.31	3.87	25.75	4.90	82.81	4.82	7.60
-1.18+0.50	6.37	62.93	4.66	21.09	3.32	79.48	3.69	3.92
-0.50+0.15	9.83	53.10	4.52	16.56	5.27	74.21	0.89	3.02
-0.15	53.10	0.00	16.56	0.00	74.21	0.00	3.02	0.00
D_{80}		**4.21**		**7.98**		**0.57**		**6.54**

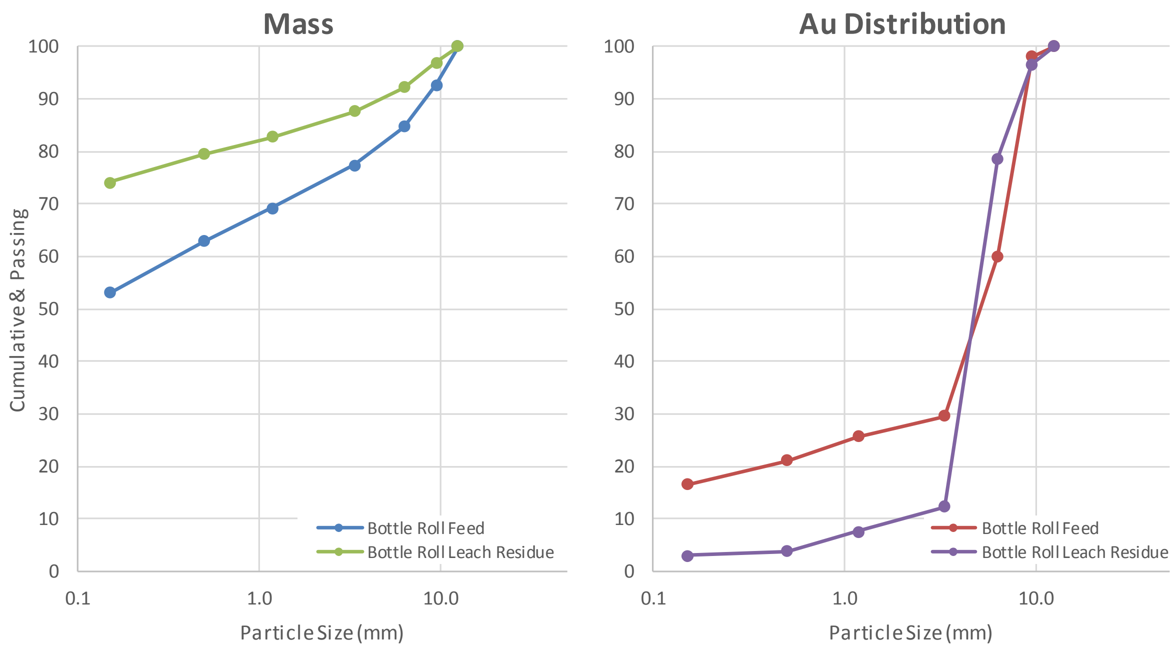


Figure 3.8: Mass & Gold Distribution, SD0006 -12.5mm Bottle Roll Feed & Leach Residue

The particle size analysis showed the leach feed to have a D_{80} of 4.21mm, reducing to 0.57mm within the leach residue. The data also showed a reduction in the amount of gold contained within the fractions below 3.35mm with 29.6% of the gold present in material below this size in the feed compared with a figure of 12.4% for the leach residue.

-6.3mm Crush Size

The distribution of mass and gold within the SD0006 -6.3mm bottle roll feed and leach residue is summarised in Table 3.9 and Figure 3.9.

Table 3.9: Particle & Gold Size Analysis, SD0006 -6.3mm, Bottle Roll Feed & Leach Residue

Particle Size (mm)	Bottle Roll Feed				Bottle Roll Leach Residue			
	Mass		Au Distribution		Mass		Au Distribution	
	%	∑ % Passing	%	∑ % Passing	%	∑ % Passing	%	∑ % Passing
-6.3+3.35	11.47	88.53	27.93	72.07	5.50	94.50	46.87	53.13
-3.35+1.18	13.13	75.40	27.03	45.03	9.49	85.01	38.56	14.57
-1.18+0.50	7.06	68.34	8.09	36.94	4.79	80.22	6.62	7.96
-0.50+0.15	9.60	58.74	10.42	26.52	6.62	73.60	2.88	5.08
-0.15	58.74	0.00	26.52	0.00	73.60	0.00	5.08	0.00
D_{80}		**1.75**		**4.19**		**0.48**		**5.04**

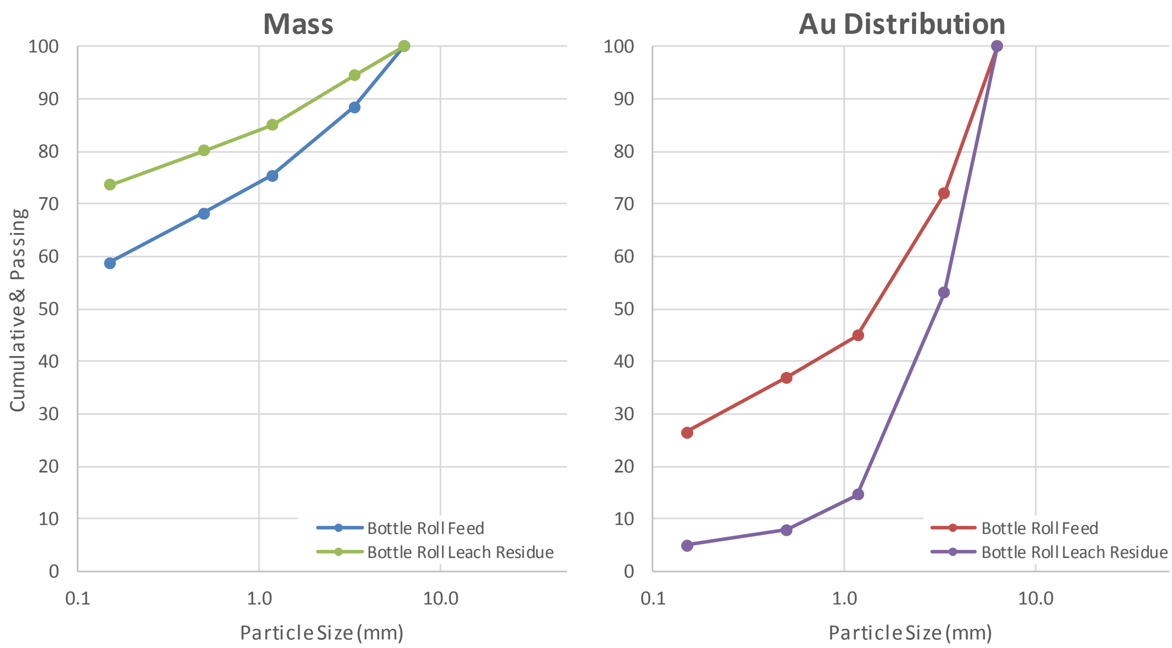


Figure 3.9: Mass & Gold Distribution, SD0006 -6.3mm Bottle Roll Feed & Leach Residue

The data showed the leach feed to have a D_{80} of 1.75mm reducing to 0.48mm within the leach residue.

Separate analysis of the gold distribution again showed preferential leaching within the finer fractions with the amount of gold distributed below 1.18mm reducing from 45.0% in the feed to 14.6% in the leach residue.

3.4.2 Gold Recovery by Particle Size

3.4.2.1 SD0005

The calculated recovery of gold by particle size for each of the bottle roll tests performed on the SD0005 sample are given in Figure 3.10 through Figure 3.12.

The figures reported are based on the difference between the gold content in the leach feed and that present in the leach residue allowing for any changes in mass distribution during testing.

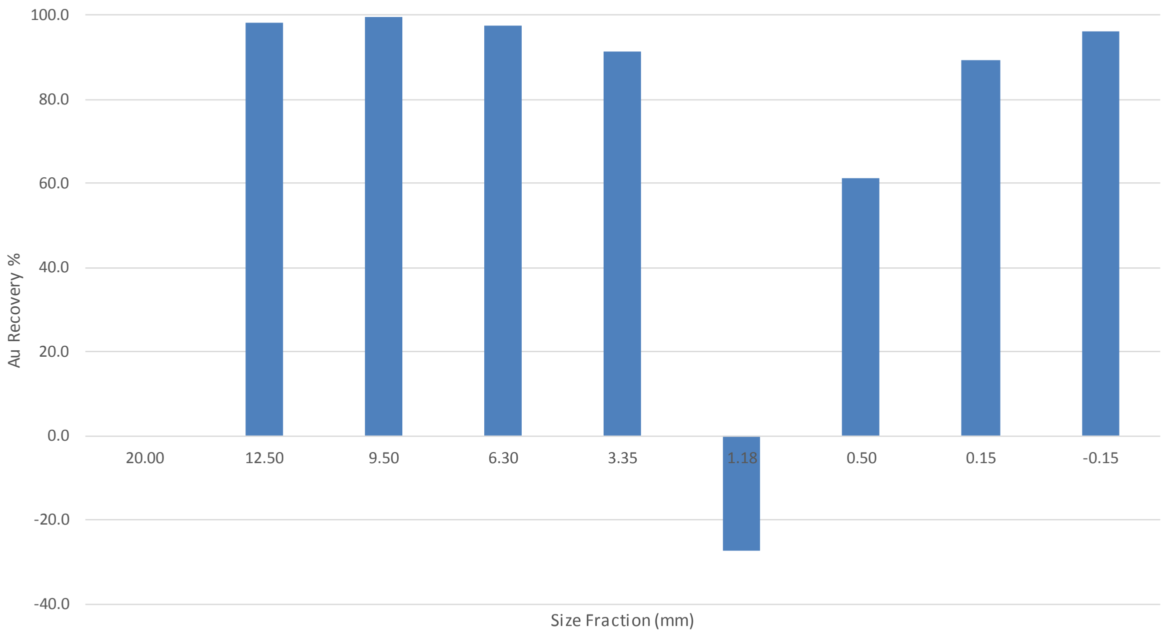


Figure 3.10: Gold Recovery by Particle Size, SD0005 -20.0mm Bottle Roll Test

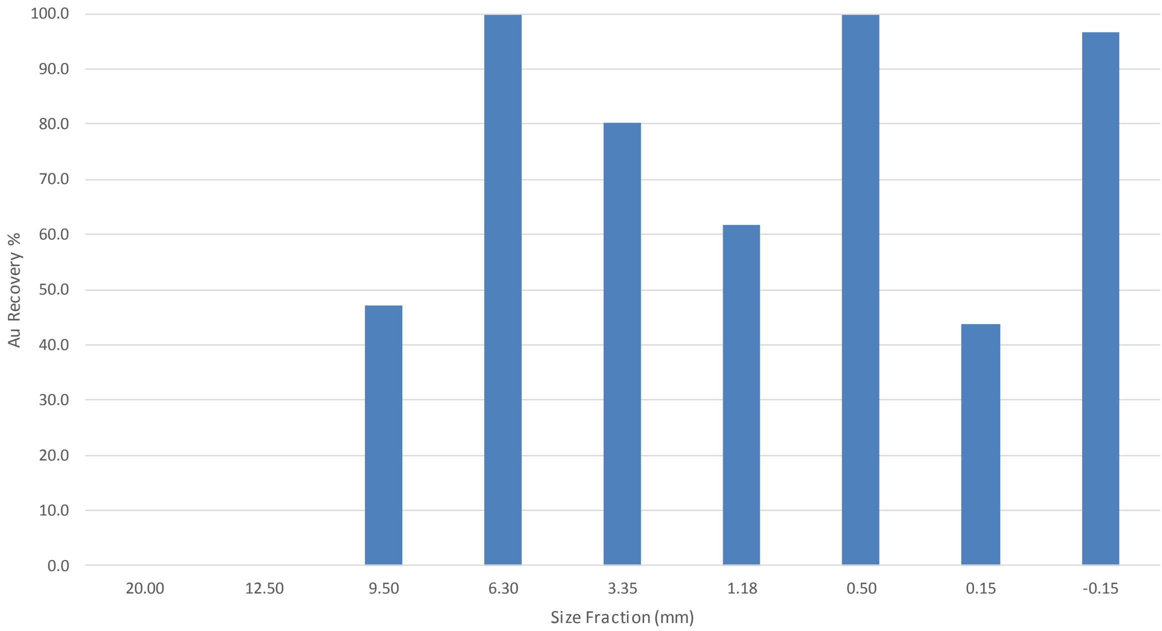


Figure 3.11: Gold Recovery by Particle Size, SD0005 -12.5mm Bottle Roll Test

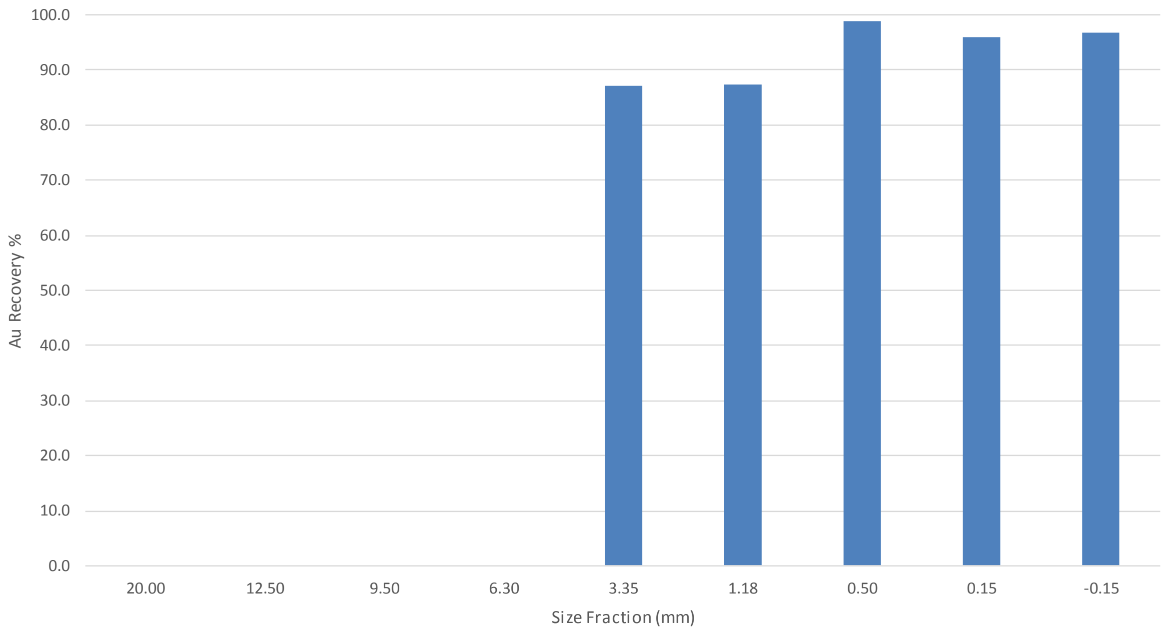


Figure 3.12: Gold Recovery by Particle Size, SD0005 -6.3mm Bottle Roll Test

Calculated gold recoveries by particle size were generally high (>80%), particularly within the tests conducted on the -20.0mm and -6.3mm material although a slightly higher degree of variation was observed in the test conducted on the -12.5mm material.

A negative gold recovery figure was calculated for the -3.35+1.18mm fraction of the test conducted on -20.0mm material however, this is most likely due to the shift in sample mass distribution during testing resulting in a skewed result.

3.4.2.2 SD0006

The calculated gold recoveries by particle size for the SD0006 bottle roll tests are given in Figure 3.13 through Figure 3.15.

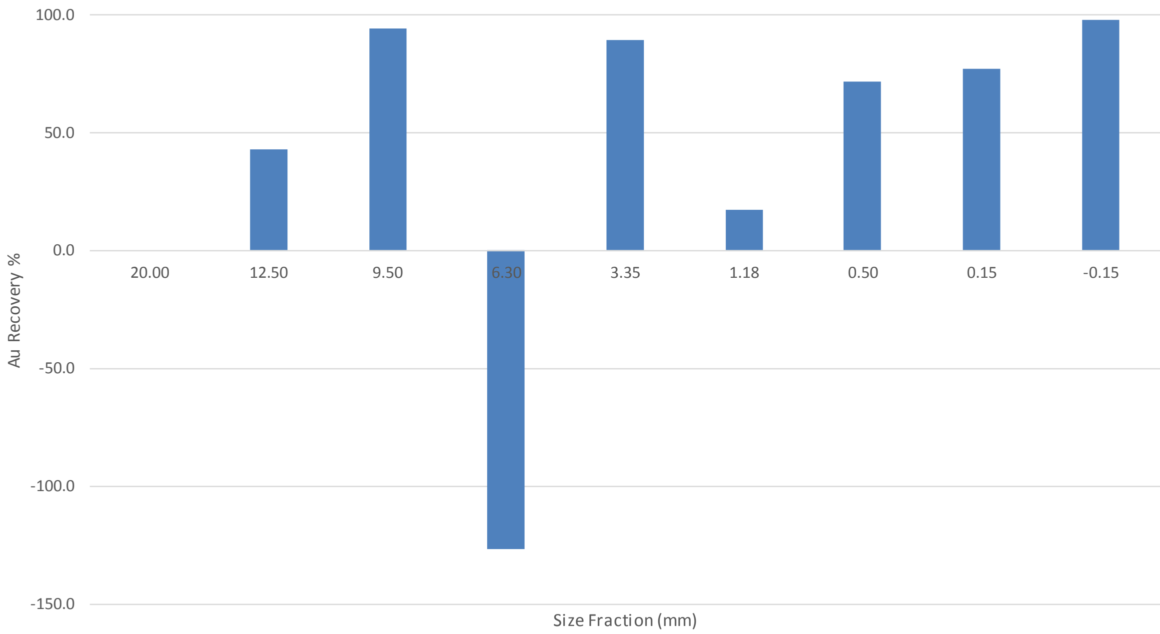


Figure 3.13: Gold Recovery by Particle Size, SD0006 -20.0mm Bottle Roll Test

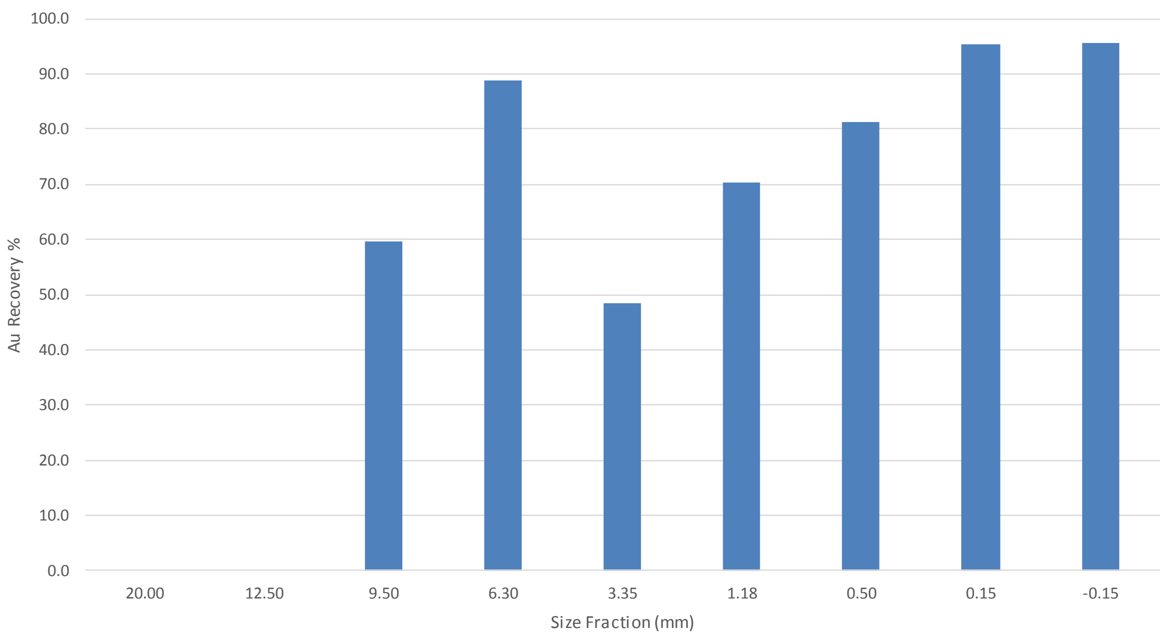


Figure 3.14: Gold Recovery by Particle Size, SD0006 -12.5mm Bottle Roll Test

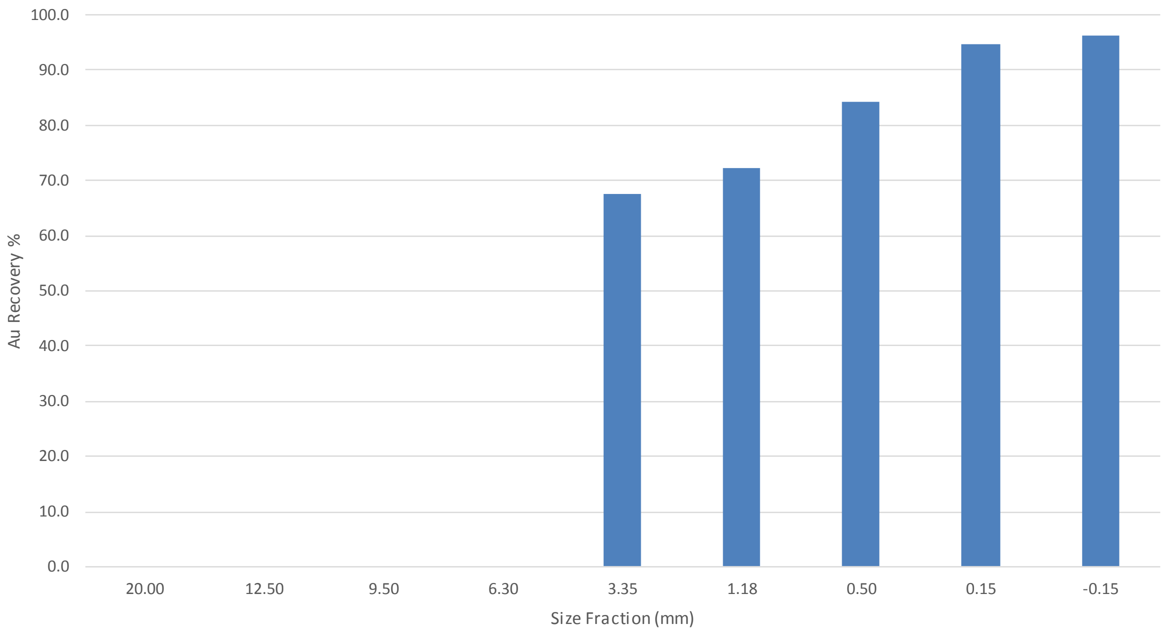


Figure 3.15: Gold Recovery by Particle Size, SD0006 -6.3mm Bottle Roll Test

Results from the tests conducted on the -12.5mm and -6.3mm material showed a much clearer trend of increased gold recovery with reduced particle size than was observed for the SD0005 sample reflecting the fact that the overall sample was slightly coarser and that gold liberation improved with particle size.

Calculated recoveries for the test performed on the -20.0mm material was slightly more mixed with gold recoveries ranging from -126.8% in the -9.5+6.3mm fraction to 97.9% in the -0.15mm fraction. Again, the indicated negative gold recovery within the one particular size fraction is unlikely to be accurate and again most probably due to the shift in the distribution of mass within the sample during testing.

3.5 Whole Ore Leach Testing

In order to provide a comparison of the gold and silver recoveries achievable under agitated leach conditions typical of a CIL type operation, a single kinetic cyanide leach test was performed on each of the samples.

Testing was conducted according to the following standard conditions:

- Sample Size: 2kg,
- Method: Stirred Leach,
- Percent Solids: 40% w/w,

- Grind Size 80% passing 75µm,
- Leach Duration: 48 Hours,
- Cyanide Concentration: 2.0g/L (maintained),
- pH: 10.5 – 11.0 (maintained),
- Solution Assays: Au & Ag,
- Solution Intervals: 2, 4, 8, 24, 32 & 48 Hours,
- Tailings Assay: Au & Ag.

The results of the tests are summarised below in Table 3.10 and Figure 3.16. Full results can be found in Appendix 5.

Table 3.10: Phase 1, Whole Ore Cyanide Leach Test Results

Sample	Reagent Consumption (kg/t)		Recovery (%)	
	Lime	Cyanide	Au	Ag
SD0005	0.72	1.35	97.4	32.7
SD0006	1.49	1.50	96.7	67.3

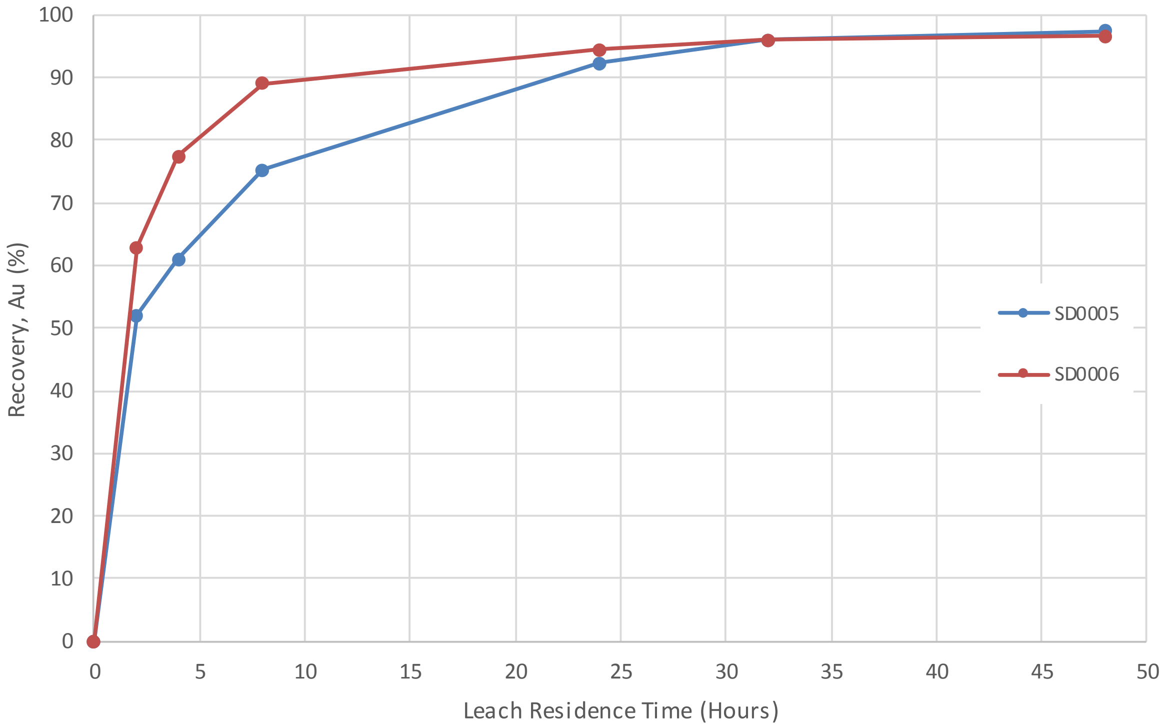


Figure 3.16: Gold Recovery vs. Residence Time, Phase 1, Whole Ore Cyanide Leach Tests

The test results showed final gold recoveries after 48 hours of leaching of 97.4% for the SD0005 sample and 96.7% for the SD0006 sample. Silver recoveries were 32.7% and 67.3% respectively.

Analysis of the rate at which the gold leached into solution showed slightly faster rate kinetics for the SD0006 sample than the SD0005 sample during the first 24 hours of leaching which is potentially due to the higher grade of gold within the feed.

After the first eight hours, data showed that 92.1% of the total gold recovered had been extracted from the SD0006 sample compared with 77.3% for the SD0005 sample.

Despite the higher head grade, the level of cyanide consumption was broadly similar in both tests, averaging 1.43kg/t although lime consumption was observed to double from 0.72kg/t for SD0005 to 1.49kg/t for SD0006.

4 PHASE 2 TESTWORK

Following completion of the first phase of testing, a Master Composite of the two samples was prepared and subjected to a programme of testing consisting of:

- Head Assay;
- Grind Calibration;
- Gravity-Leach Testing;
- Whole Ore Leach Testing;
- Agglomeration & Percolation Testing; and
- Column Leach Testing

Details of the test programme are given below.

4.1 Head Assay

A representative sub-sample of the Master Composite was pulverised to 100% passing and submitted for head assay for gold and silver. Results are given in the Table below.

Table 4.1: Master Composite Head Assay Result		
Element	**Units**	**Assay**
$Au_{(AR)}$	ppm	2.74
Ag	ppm	0.8

4.2 Grind Calibration

A multi-point grind calibration was conducted on the Master Composite to again determine the length of time required to achieve a particular product size distribution.

The grind calibration was performed using the same methodology as detailed previously (Section 3.2) with the range of target grind sizes increased to cover 80% passing values ranging from 212µm to 75µm.

The grind calibration curve for the Master Composite is shown overleaf in Figure 4.1. Full details are given in Appendix 6.

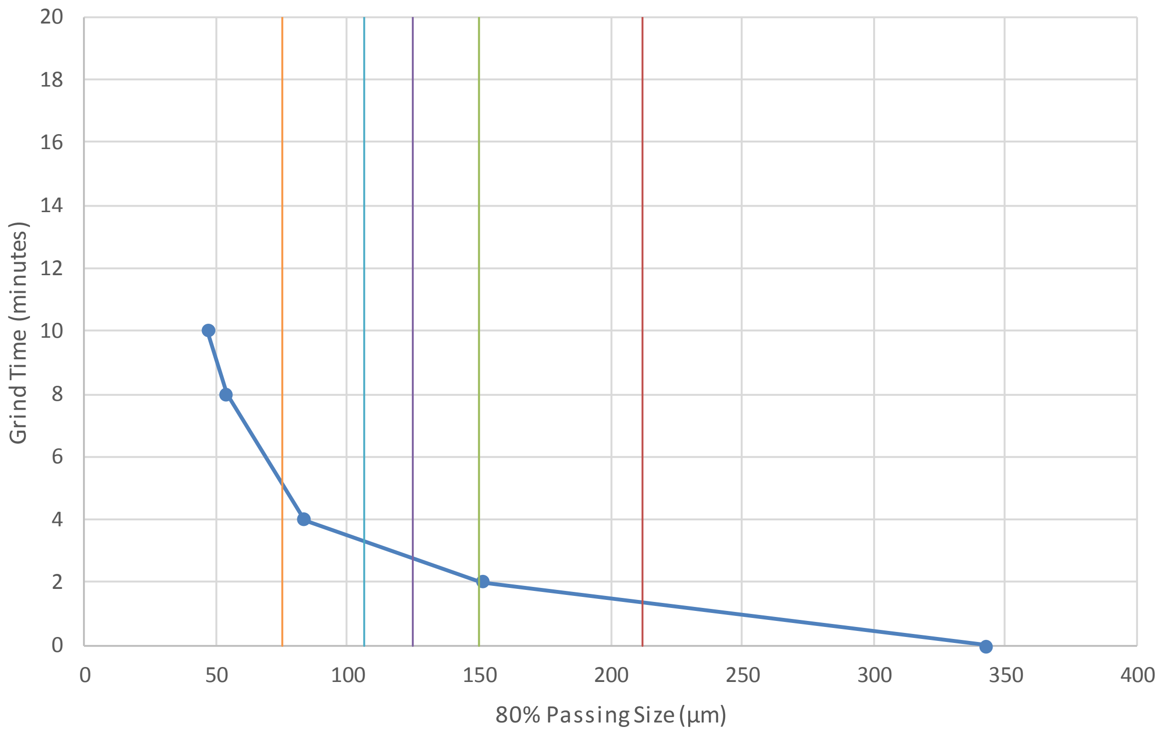


Figure 4.1: Master Composite Grind Calibration Curve

4.3 Gravity-Leach Testing

Gravity-leach testing was undertaken to investigate the amount of gold present in the Sanankoro material that was amenable to recovery by gravity concentration.

Testing was also performed to determine how much gold could be recovered from the subsequent gravity tailings through conventional cyanide leaching.

4.3.1 Gravity Concentration

The Master Composite was subjected to two stages of gravity concentration using a Knelson MD3 centrifugal gravity concentrator in order to investigate the amount of gold that could be recovered using a gravity concentration process.

Testing was conducted at an initial grind size of 80% passing 212µm after which, the tailings material from the first stage of processing were reground to the finer size of 80% passing 75µm and subjected to a second stage of processing.

Once complete, the two gravity concentrates, along with a sub-sample of the gravity tailings, were submitted for assay for gold.

The results of the two-stage gravity concentration test are summarised in the Table below.

Table 4.2: Master Composite Gravity Concentration Test Results

Product	Stage	Grind Size (µm)	Mass (%)	Assay, Au (ppm)	Recovery, Au (%)
Concentrate	1	212	0.34	406	50.65
	2	75	0.36	167	22.37
	Total	-	*0.70*	*282*	*73.02*
Tailings	-	-	99.30	0.74	26.98
Feed	-	-	100.00	2.71	-

The test results showed that after two stages of processing, 73.0% of the gold was recovered to a gravity concentrate grading 282ppm Au.

Of the total gold reporting to the gravity concentrate, 50.7% was recovered during the first stage of processing to a grade of 406ppm Au with the remaining 22.4% recovered during the second stage of processing to a grade of 167ppm Au.

4.3.2 Cyanide Leaching of Gravity Tailings

Upon completion of the gravity concentration testing, two kinetic cyanide leach tests were performed to determine the amount of gold and silver that could be recovered from the gravity tailings.

Each test was conducted using the conditions summarised below.

- Sample Size: 2kg,
- Method: Stirred Leach,
- Percent Solids: 40% w/w,
- Grind Size 80% passing 75µm,
- Leach Duration: 48 Hours,
- Cyanide Concentration: i) 1.0g/L (maintained),
  ii) 0.5g/L (maintained),
- pH: 10.5 – 11.0 (maintained),
- Solution Assays: Au & Ag,
- Solution Intervals: 2, 4, 8, 24, 32 & 48 Hours,
- Tailings Assay: Au & Ag.

The results of the tests are summarised in Table 4.3 and Figure 4.2. Full results are given in Appendix 7.

Table 4.3: Master Composite Gravity Tailings Cyanide Leach Test Results

Cyanide Concentration (g/L)	Reagent Consumption (kg/t)		Recovery (%)	
	Lime	Cyanide	Au	Ag
1.0	0.49	1.21	94.1	34.1
0.5	0.87	0.53	92.1	36.2

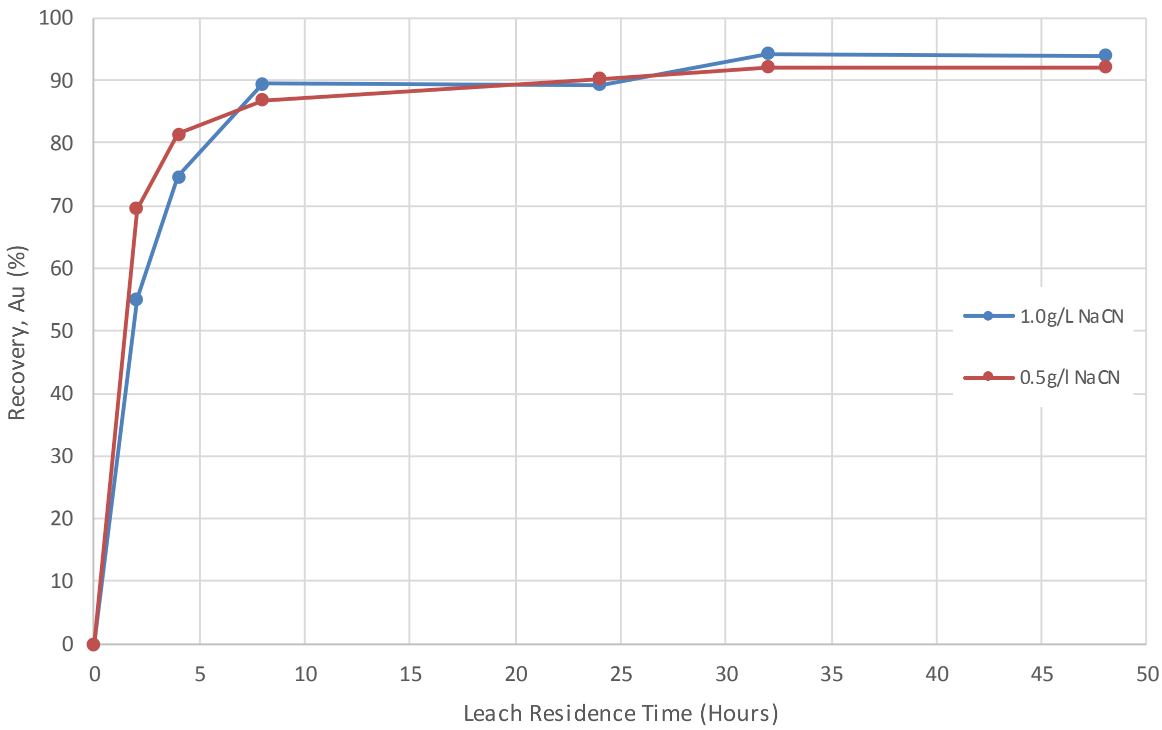


Figure 4.2: Gold Recovery vs. Residence Time, Master Composite Gravity Tailings Cyanide Leach Tests

The results showed a total gold recoveries of 94.1% for the test conducted using 1.0g/L cyanide reducing slightly to 92.1% for the test conducted using 0.5g/L cyanide. Silver recoveries were 34.1% and 36.2% respectively.

In addition to the high recoveries, analysis of the leach kinetics showed the gold to be readily leachable with an average of 95% of the total gold recovered extracted after the first 8 hours of leaching.

4.4 Whole Ore Leach Testing

A series of whole ore leach tests were conducted to investigate the amount of gold and that could be recovered from the Master Composite through direct cyanide leaching.

In total, six leach tests were performed during which, the effect of two key variables was investigated; grind size and cyanide strength.

The conditions applied to each test are summarised below

- Sample Size: 2kg,
- Method: Stirred Leach,
- Percent Solids: 40% w/w,
- Grind Size i) 80% passing 150µm,
  ii) 80% passing 125µm,
  iii) 80% passing 106µm,
  iv – vi) 80% passing 75µm,
- Leach Duration: 48 Hours,
- Cyanide Concentration: i-iv) 1.0g/L (maintained),
  v) 0.5g/L (maintained),
  vi) 0.25g/L (maintained),
- pH: 10.5 – 11.0 (maintained),
- Solution Assays: Au & Ag,
- Solution Intervals: 2, 4, 8, 24, 32 & 48 Hours,
- Tailings Assay: Au & Ag.

A summary of the first four tests which investigated the effect of grind size on leach response is given below in Table 4.4 and Figure 4.3. Full results are given in Appendix 8.

Table 4.4: Master Composite Whole Ore Cyanide Leach Test Results (Effect of Grind Size)

Grind Size (µm)	Reagent Consumption (kg/t)		Recovery (%)	
	Lime	Cyanide	Au	Ag
150µm	0.88	1.21	95.0	58.3
125µm	1.10	1.09	98.0	62.4
106µm	1.02	1.17	95.3	53.6
75µm	1.00	0.98	98.0	59.2

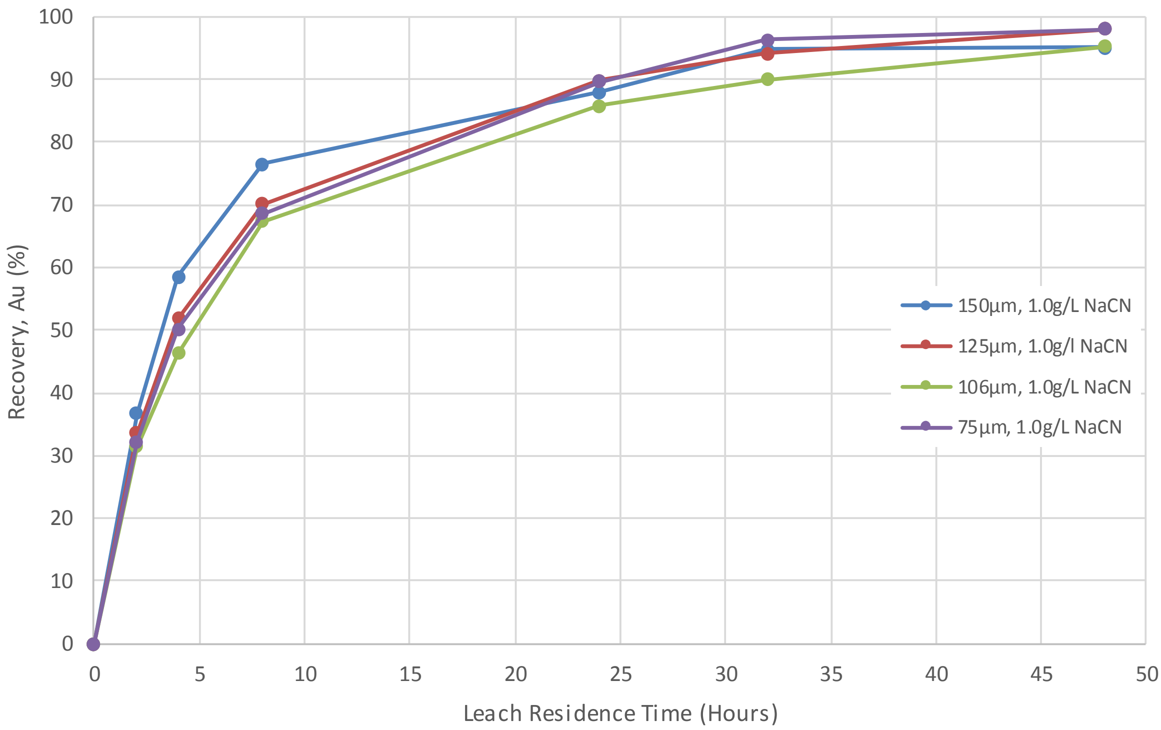


Figure 4.3: Gold Recovery vs. Residence Time, Master Composite Whole Ore Cyanide Leach Tests (Effect of Grind Size)

The results showed gold recoveries to range from 95.0% at a grind size of 80% passing 150µm to 98.0% at grind sizes of 80% passing 125µm and 75µm. Silver recoveries ranged from 53.6% at a grind size of 80% passing 106µm to 62.4% at a grind size of 80% passing 125µm.

Regardless of the grind size trialled, the rate at which the gold leached into solution was moderate with data showing an average gold recovery of 70.7% after the first 8 hours of leaching increasing to an average of 88.3% after 24 hours.

With respect to reagent consumption, the data showed the amount of cyanide consumed during testing to range from 0.98kg/t (75µm) to 1.21kg/t (150µm), averaging 1.11kg/t across the four tests. Average lime consumption was 1.00kg/t.

Based on the results, the grind size of 80% passing 75µm was selected for further testing as this test had shown the joint highest total gold recovery and was the size at which the gold was likely best liberated.

Consequently, a further two tests were performed to investigate the effect of reduced cyanide concentration on leach performance. Results are summarised overleaf in Table 4.5 and Figure 4.4 with full results again given in Appendix 8. The results of the previous test conducted using 1.0g/L cyanide and included for comparative purposes.

Table 4.5: Master Composite Whole Ore Cyanide Leach Test Results (Effect of Cyanide Concentration)

Cyanide Concentration (g/L)	Reagent Consumption (kg/t)		Recovery (%)	
	Lime	Cyanide	Au	Ag
1.0	1.00	0.98	98.0	59.2
0.5	1.22	0.49	97.3	56.6
0.25	1.53	0.08	92.9	62.4

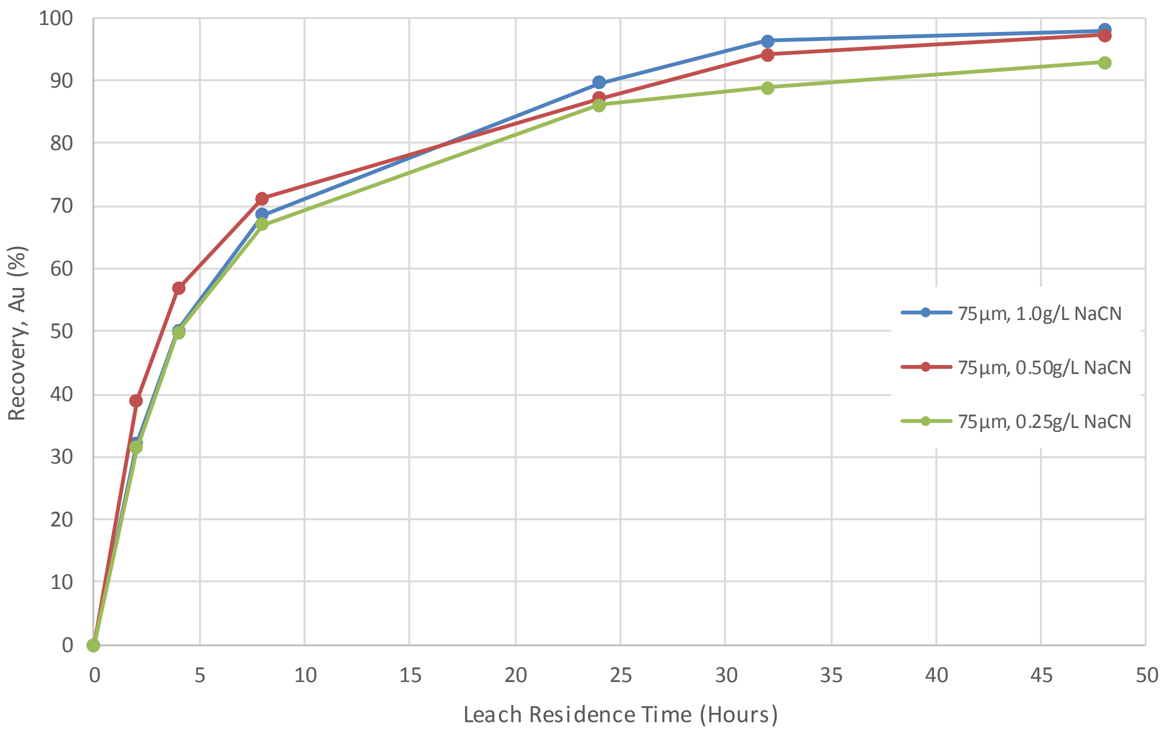


Figure 4.4: Gold Recovery vs. Residence Time, Master Composite Whole Ore Cyanide Leach Tests (Effect of Cyanide Concentration)

The test results showed a trend of reduced gold recovery with lower cyanide concentration with values falling from 98.0% when using 1.0g/L cyanide to 92.9% when using 0.25g/L cyanide. In contrast, no clear trend was observed for the silver recoveries with values ranging from 56.6% (0.5g/L NaCN) to 62.4% (0.25g/L NaCN).

The leach kinetics showed a similar trend to those observed previously during the grind size optimisation tests with an average of 72.7% gold recovered after 8 hours increasing to 91.2% after 24 hours. Overall, the leach kinetics were lowest in the test conducted using 0.25g/L NaCN which is most likely due to the low cyanide concentration.

Cyanide consumption during testing ranged from 0.49kg/t when leached using the 0.5g/L solution to 0.08kg/t when leached using the 0.25g/L solution. This compared with a figure of 0.98kg/t during previous testing at 1.0g/L NaCN.

4.5 Agglomeration & Percolation Testing

Agglomeration and percolation testing was undertaken to determine the drainage characteristics of the Master Composite and to identify whether it was necessary to agglomerate with cement prior to subsequent column leach testing.

The sample was subjected to a total of five percolation tests, four on material agglomerated with different quantities of cement and one on un-agglomerated material.

The agglomeration testing was undertaken by mixing the desired quantity of cement with 2.5kg of sample in a mixing drum to which water was added until agglomerated pellets had started to form. The pellets were then allowed to cure for a total of 48 hours prior to the percolation testing.

All testing was conducted the uncrushed (-20.0mm) Master Composite material with cement dosages ranging from 5 – 22.5kg/t trialled[1].

Once agglomerated, the samples were subjected to percolation testing to determine a range of characteristics including slumpage and maximum, minimum and average permeability flowrates. To determine whether the sample would require agglomeration or not, a target average drainage flowrate of 10,000l/m^2/hr was used.

The results of the percolation testing are summarised below in Table 4.6 and Figure 4.5. Full results can be found in Appendix 9.

Table 4.6: Master Composite Percolation Test Results

Cement Addition (kg/t)	Drainage Flowrate (l/m^2/hr)		
	Minimum	Maximum	Average
0	14	45	**25**
5	587	965	**780**
10	2,542	3,940	**3,136**
15	4,724	11,630	**7,208**
22.5	10,002	18,773	**12,794**

[1] It should be noted that whilst the uncrushed material had a nominal top size of -20.0mm, analysis of the Coarse Ore Bottle Roll feed as part of the first phase of testing (Section 3.4.1) had shown the SD0005 sample to have a D_{80} of 1.28mm whilst the SD0006 sample had a D_{80} of 6.40mm. On this basis, the D_{80} of the Master Composite at the -20.0mm crush size was in the order of 3.84mm.

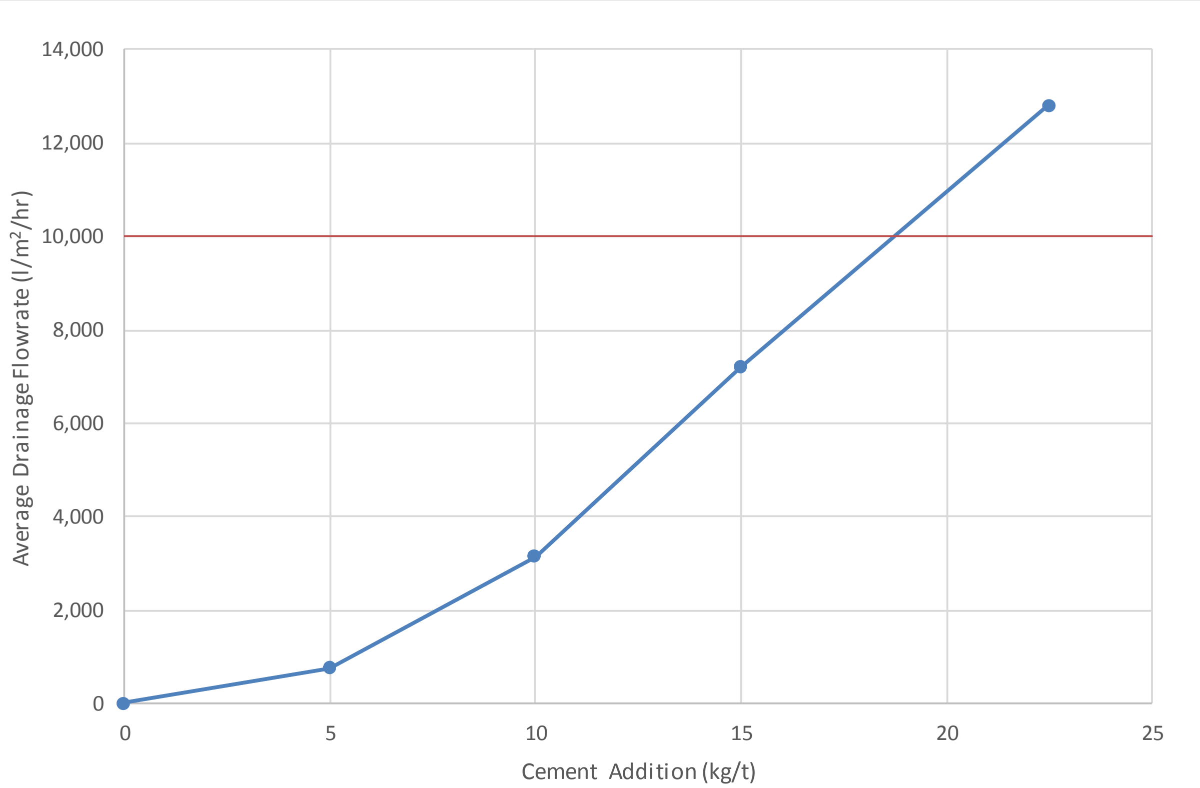


Figure 4.5: Cement Addition vs. Average Drainage Rate, Master Composite

The result of the testing conducted on un-agglomerated material showed the Master Composite to have exceptionally poor drainage characteristics with an average free drainage rate of just 25l/m^2/hr.

Of the tests conducted on agglomerated material, results showed the average drainage flowrates to be below the target level at all cement dosages with the exception of the highest value (22.5kg/t) which achieved an average drainage rate of 12,794l/m^2/hr. On this basis, the data indicates that a cement dosage of approximately 19kg/t would be required to achieve the target average percolation flowrate of 10,000l/m^2/hr.

Based on these results, the Master Composite was agglomerated with 22.5kg/t of cement prior to column leach testing.

4.6 Column Leach Testing

Column leach testing was undertaken to provide confirmation of the achievable metal recoveries and leach rates from the Master Composite sample under heap leach conditions.

The previous programme of coarse ore bottle roll testing had provided an indication of maximum gold recoveries however, it was felt that the relatively high degree of material breakup during testing may have skewed the recoveries figures reported and so column leach testing was required to provide confirmation of the gold and silver recoveries under static leaching conditions.

A single column leach test was conducted using 40kg of the Master Composite which had been agglomerated with 22.5kg/t of cement. A summary of the column leach test conditions is given below.

- Sample Size: 40kg,
- Method: Column leach with carbon trap,
- Column Size: 150mm x 2m,
- Retention Time: 105 days (95 days irrigation, 3 days drain, 7 days wash + drain),
- Irrigation Rate: 10l/m^2/hr,
- Solution Intervals: Daily (Week 1), Twice-weekly (Week 2-14)
- Solution Assays: Au, Ag, NaCN & pH,
- pH: 10.5 – 11.0 (maintained),
- Cyanide Concentration: 1.0g/L (maintained),
- Agglomeration: 22.5kg/t Cement,
- Water: Tap Water,
- Tailings Assays: Au & Ag (duplicate)

The results of the column leach test are summarised below in Table 4.7 and Figure 4.6. Full results are given in Appendix 10.

Table 4.7: Master Composite Column Leach Test Results

Reagent Consumption (kg/t)		Recovery to PLS (%)		Recovery to Carbon (%)	
Lime	Cyanide	Au	Ag	Au	Ag
0.12	0.62	56.0	37.0	56.3	42.9

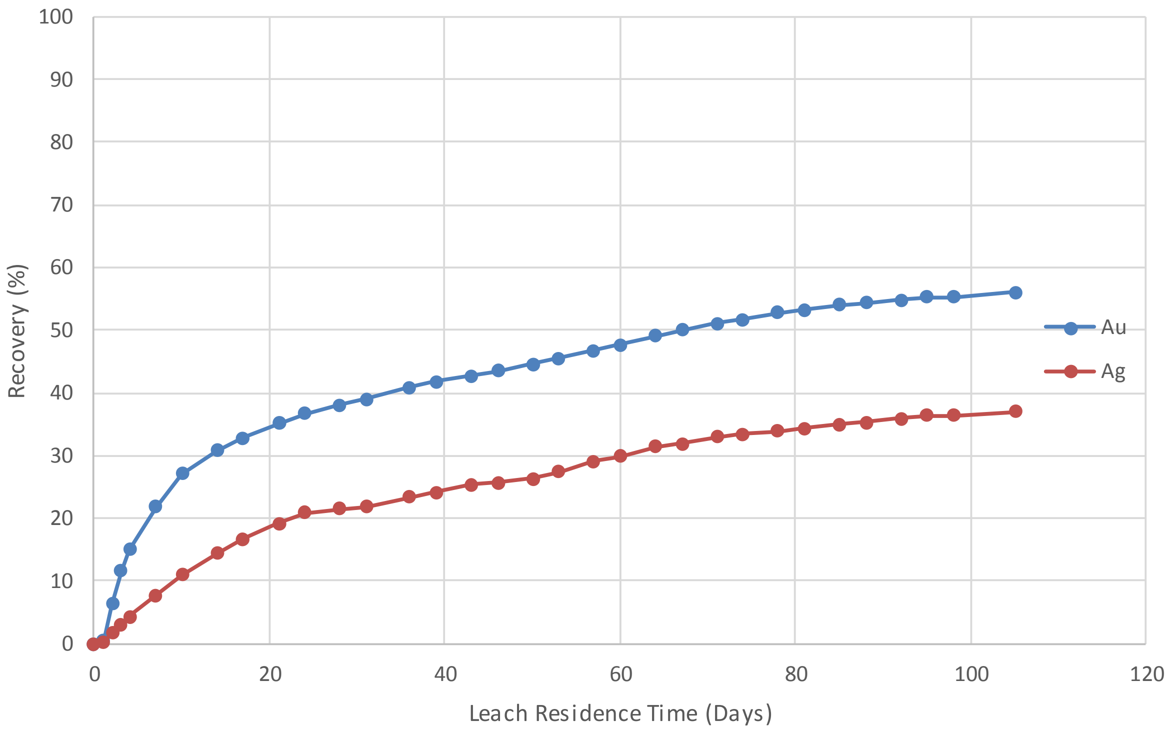


Figure 4.6: Gold Recovery to PLS vs. Residence Time, Master Composite Column Leach Test

The results of the column leach test showed gold and silver recoveries of 56.0% and 37.0% respectively to the pregnant leach solution (PLS) or 56.3% and 42.9% respectively to the loaded carbon. Whilst the calculated gold recoveries show good agreement, the calculated recovery of silver to the loaded carbon is higher than that to the PLS which is most likely due to silver levels within the PLS being near the detection limit of the analytical technique used (AAS).

When compared with the results of the previous coarse ore bottle roll tests, which had achieved gold recoveries of 78.4% (SD0005) and 66.8% (SD0006), the calculated gold recovery from the Master Composite was seen to be lower by approximately 16.6% (to PLS), based on the average of the coarse ore bottle roll test results, despite the significantly longer leach residence time.

Analysis of the column leach kinetics indicates that the recovery of both metals was continuing to increase at the time testing was suspended and that higher metal recoveries would have been achieved if testing had been extended beyond 95 days. However, the lower recoveries may also be indicative of some blinding of the particle surfaces due to the high amount of cement used during agglomeration, resulting in reduced solution contact, or alternatively be a result of the lack of material breakup, and subsequent liberation of the gold, as a result of the static testing methodology.

5 CONCLUSIONS AND RECOMMENDATIONS

5.1 Conclusions

5.1.1 Phase 1 Testwork

Head Assay

- Two samples of gold mineralisation from the Sanankoro deposit, Mali were received for testing;
- Head assay of the samples showed the SD0005 sample to contain 0.61ppm Au whilst the SD0006 sample contained 3.35ppm Au based on analysis by Fire Assay. Silver grades were <0.5ppm Ag and 1.2ppm Ag respectively;
- Base metal levels within the samples were low with copper values of 0.005 – 0.008% Cu, lead values of 0.002 – 0.003% Pb and zinc values of 0.004 – 0.010% Zn; and
- Total sulphur levels ranged from 0.022 – 0.046% $S_{(TOT)}$ whilst total carbon levels ranged from 0.027 – 0.13% $C_{(TOT)}$.

Coarse Ore Bottle Roll Testing

- Coarse ore bottle roll testing to investigate the effect of crush size on leach response showed gold recoveries ranging from 78.4% (-20.0mm) to 97.6% (-6.3mm) for the SD0005 sample and 66.8% (-20.0mm) to 93.1% (-6.3mm) for the SD0006 sample;
- Silver recoveries ranged from 14-8 – 21.9% (SD0005) and 35.1 – 47.9% (SD0006);
- Analysis of the leach kinetics showed continued gold recovery from the SD0005 samples throughout the duration of testing whereas results for the SD0006 sample showed gold recoveries to have reached maximum by around the tenth day of leaching;
- Cyanide consumption in the tests conducted on the SD0005 sample averaged 0.56kg/t and were lower than those of the SD0006 sample which averaged 0.65kg/t. This increase in cyanide consumption is likely due to the higher gold content in the SD0006 sample;
- Particle size analysis of the bottle roll feed showed all of the samples to have a considerably finer particle size distribution than that intended based on the target crush size. For the SD0005 sample, D_{80} values ranged from 1.28mm (-20.0mm) to 0.46mm (-6.3mm) whilst for the SD0006 sample, D_{80} values ranged from 6.40mm (-20.0mm) to 1.75mm (-6.3mm);
- Subsequent particle size analysis of the bottle roll leach residues showed a reduction in particle size distribution for all of the samples tested indicating a breakup of material during leaching. For the SD0005 sample, D80 values were observed to have reduced to 0.31 – 0.29mm whilst for the SD0006 sample, D80 values reduced to 0.74 – 0.48mm; and

- Calculation of the gold recovery by particle size showed a general trend for increased gold recovery by reduced particle size, up to 98% in selected size fractions, as would be expected given improved liberation within the finer size classes. Some discrepancies were observed with certain size fractions indicating negative gold recoveries however, this is most likely due to the change in the particle size distribution of the material during testing as previously mentioned.

Whole Ore Leach Testing

- Whole ore leach testing showed that gold recoveries of 97.4% and 96.7% could be achieved from the SD0005 and SD0006 samples respectively following 48 hours of leaching using a 2.0g/L cyanide solution; and
- Cyanide consumption during both tests was broadly similar, averaging 1.43kg/t.

5.1.2 Phase 2 Testwork

Head Assay

- Head assay of the Master Composite, which had been prepared by blending the SD0005 and SD0006 samples, showed it to contain 2.74ppm Au and 0.8ppm Ag.

Gravity Leach Testing

- Gravity testing using a Knelson centrifugal concentrator showed that 73.0% of the gold could be recovered from the Master Composite to a grade of 282ppm Au following two stages of processing;
- Of the gold recovered by gravity, over two thirds (50.7%) was recovered during the first stage of processing which had been performed at a grind size of 80% passing 212µm;
- Subsequent cyanide leaching of the gravity tailings showed that 94.1% of the gold remaining in the tailings could be recovered after 48 hours of leaching using a 1.0g/L cyanide solution. If the cyanide concentration was reduced to 0.5g/L, gold recovery fell slightly to 92.1%; and
- The combined recovery of gold to the gravity concentrate and gravity tailings leach solution was 98.4%. If it is assumed that 98% of the gold in the gravity concentrate can be successfully leached, overall gold recovery to the combined PLS would be in the order of 96.9%.

Whole Ore Leach Testing

- Testing of the Master Composite to investigate the effect of grind size on leach response showed gold recoveries to range from 95.0% at a grind size of 80% passing 150µm to 98.0% at grind sizes of 80% passing 125µm and 75µm. Silver recoveries ranged from 53.6% to 62.4%;
- Further testing to investigate the effect of cyanide concentration on leach response showed a consistent reduction in gold recovery with reduced cyanide concentration with values falling from 98.0% at 1.0g/L to 92.9% at 0.25g/L. No such trend was observed for silver with recoveries ranging from 56.6% (0.5g/L) to 62.4% (0.25g/L); and
- When compared against the results of the gravity-leach testing, the results indicate that higher overall gold recoveries could potentially be achieved through direct leaching of the ore without the inclusion of a gravity preconcentration stage. Further testing would however, be required to verify this conclusion.

Agglomeration & Percolation Testing

- Percolation testing showed the un-agglomerated Master Composite material to have exceptionally poor drainage characteristics with an average drainage flowrate of just 25l/m^2/hr;
- Testing of material which had been agglomerated with cement showed that a cement dosage of 22.5kg/t was required to achieve an average drainage flowrate of 12,794l/m^2/hr, above the target value of 10,000l/m^2/hr; and
- Based on the agglomeration testing, a cement dosage of approximately 19kg/t would be required to achieve the target drainage flowrate with the high cement requirement likely due to a combination of the nature of the material and the high degree of fines present in the ore.

Column Leach Testing

- Column leach testing of the Master Composite at a nominal crush size of -20.0mm achieved a gold recovery of 56.0% to the PLS after 95 days of irrigation. Overall recovery of gold to a sample of loaded carbon was 56.3%;
- Silver recovery to the PLS was 37.0% increasing to 42.9% to the loaded carbon with the increase likely attributable to the low grades of silver within the PLS resulting in some analytical variance;
- The leach kinetics for both metals showed that they continued to leach through the duration of the test and had not reached equilibrium by the time the test was stopped. It is therefore highly probable that higher gold and silver recoveries would have been achieved had the test been allowed to continue; and

- Overall, the amount of gold recovered from the column leach test was approximately 16.6% lower than that which had been indicated by the previous coarse ore bottle roll tests. This reduction in recovery may be due to the lack of material breakup (attrition) during static column testing;

5.2 Recommendations

Based on the testing that has been completed, Wardell Armstrong International would recommend that the following additional testwork be considered:

- Further gravity-leach testing to both confirm the amount of gold that can be recovered from the Sanankoro material by means of gravity processing and to quantify the amount of gold that can be recovered from the gravity concentrate by cyanide leaching;
- Comminution testing to determine ore hardness and energy requirements to grind the Sanankoro material to sizes suitable for whole ore / gravity-leach processing;
- Variability testing to investigate any variation in gravity and/or leach response across the deposit;
- Additional coarse ore bottle roll and column leach testing to further investigate the gold recoveries achievable through heap leaching of the ore;
- Dewatering testing to investigate the settling characteristics of the Sanankoro material post-leaching; and
- Environmental testing to characterise tailings material with respect to standard parameters such as metal solubility and acid rock drainage potential.

APPENDIX 1: Sample Inventory

Wardell Armstrong International

Client: Cora Gold
Project Name:
Project Number: ZT64-0703
Date Received: 23/01/2019
Log in Date: 23/01/2019
Carrier: Air France/UCH Logistics
Waybill no. 057 96070800

Sample ID		WAI Wt (kg)	Comments
SD0005	30-31	1.817	Sack ID: SD0005 30-50m
SD0005	31-32	1.749	
SD0005	32-33	0.660	
SD0005	33-34	1.673	
SD0005	34-35	1.475	
SD0005	35-36	2.352	
SD0005	36-37	1.732	
SD0005	37-38	0.872	
SD0005	38-39	2.242	
SD0005	39-40	1.937	
SD0005	40-41	2.050	
SD0005	41-42	1.460	
SD0005	42-43	2.174	
SD0005	43-44	1.641	
SD0005	44-45	2.227	
SD0005	45-46	1.414	
SD0005	46-47	2.157	
SD0005	47-48	2.593	
SD0005	48-49	1.910	
SD0005	49-50	1.246	
SD0005	50-51	1.718	
SD0005	51-52	2.376	
SD0005	52-53	1.753	
SD0005	53-54	1.825	
SD0005	54-55	1.384	
SD0005	55-56	0.911	
SD0005	56-57	1.418	

Wardell Armstrong International

Client:	Cora Gold
Project Name:	
Project Number:	ZT64-0703
Date Received:	23/01/2019
Log in Date:	23/01/2019
Carrier:	Air France/UCH Logistics
Waybill no.	057 96070800

Sample ID		WAI Wt (kg)	Comments
SD0005	57-58	1.597	Sack ID: SD0005 51-70m
SD0005	58-59	1.757	
SD0005	59-60	2.449	
SD0005	60-61	2.455	
SD0005	61-62	2.486	
SD0005	62-63	2.478	
SD0005	63-64	1.803	
SD0005	64-65	2.229	
SD0005	65-66	1.050	
SD0005	66-67	1.809	
SD0005	67-68	1.825	
SD0005	68-69	1.506	
SD0005	69-70	1.822	
SD0005	70-71	2.046	Sack ID: SD0005 71-76.2m
SD0005	71-72	3.457	
SD0005	72-73	2.371	
SD0005	73-74	1.810	
SD0005	74-75	1.854	
SD0005	75-76.2	1.927	
Total	46	85.497	

Wardell Armstrong International

Client: Cora Gold
Project Name:
Project Number: ZT64-0703
Date Received: 18/01/2019
Log in Date: 21/01/2019
Carrier: Air France/UCH Logistics
Waybill no. 057 96065115

Sample ID		WAI Wt (kg)	Comments
SD0006	9-10	2.541	Sack ID: Batch 1
SD0006	10-11	2.194	
SD0006	11-12	2.441	
SD0006	12-13	2.200	
SD0006	13-14	1.664	
SD0006	14-15	1.245	
SD0006	15-16	1.839	
SD0006	16-17	2.080	
SD0006	17-18	2.584	
SD0006	18-19	0.759	
SD0006	19-20	1.050	
SD0006	20-21	1.956	
SD0006	21-22	1.298	
SD0006	22-23	2.092	
SD0006	23-24	1.675	
SD0006	24-25	2.285	
SD0006	25-26	1.351	
SD0006	26-27	1.750	
SD0006	27-28	2.127	
SD0006	28-29	2.216	
SD0006	29-30	2.293	
SD0006	30-31	2.276	
SD0006	31-32	2.795	
SD0006	32-33	2.426	
SD0006	33-34	2.408	
SD0006	34-35	1.187	
SD0006	35-36	1.764	

Wardell Armstrong International

Client: Cora Gold
Project Name:
Project Number: ZT64-0703
Date Received: 18/01/2019
Log in Date: 21/01/2019
Carrier: Air France/UCH Logistics
Waybill no. 057 96065115

Sample ID		WAI Wt (kg)	Comments
SD0006	36-37	1.447	Sack ID: Batch 2
SD0006	37-38	1.919	
SD0006	38-39	1.798	
SD0006	39-40	1.697	
SD0006	40-41	1.333	
SD0006	41-42	2.153	
SD0006	42-43	2.859	
SD0006	43-44	2.837	
SD0006	44-45	2.407	
SD0006	45-46	2.385	
SD0006	46-47	1.130	
SD0006	47-48	1.576	
SD0006	48-49	2.624	
SD0006	49-50	2.618	Sack ID: Batch 3
SD0006	50-51	2.387	
SD0006	51-52	2.217	
SD0006	52-53	2.054	
SD0006	53-54	2.379	
SD0006	54-55	2.135	
SD0006	55-56	2.477	
SD0006	56-57	2.822	
SD0006	57-58	2.149	
SD0006	58-59	2.219	
SD0006	59-60	2.450	
SD0006	60-61	1.514	
SD0006	61-62	2.149	
SD0006	62-63	1.907	

Wardell Armstrong International

Client: Cora Gold
Project Name:
Project Number: ZT64-0703
Date Received: 18/01/2019
Log in Date: 21/01/2019
Carrier: Air France/UCH Logistics
Waybill no. 057 96065115

Sample ID		WAI Wt (kg)	Comments
SD0006	63-64	1.042	
SD0006	64-65	2.687	
SD0006	65-66	2.438	
SD0006	66-67	1.692	
SD0006	67-68	0.917	
SD0006	68-69	1.499	
SD0006	69-70.2	1.720	
Total	61	122.133	

Wardell Armstrong International

Client: Cora Gold

SD0006 Sample Selection

Hole_ID	From_m	To_m	Wt (kg)	LabAuppm (FA)
SD0006	12	15	5.11	0.79
SD0006	15	18	6.5	0.11
SD0006	18	21	3.77	0.63
SD0006	21	24	5.07	0.78
SD0006	24	27	5.39	1.3
SD0006	27	30	6.64	0.33
SD0006	30	33	7.5	2.13
SD0006	33	36	5.36	0.37
SD0006	36	39	5.16	10.9
SD0006	39	42	5.18	24.8
SD0006	42	45	8.1	0.19
SD0006	45	48	5.09	0.12
SD0006	48	51	7.63	0.14
SD0006	57	60	6.82	0.29
SD0006	60	63	5.57	1.41

APPENDIX 2: Grind Calibration Data (Phase 1)

Wardell Armstrong International Ltd

Job:	ZT640703	**Mill:**	4
Client:	Cora Gold	**Rods:**	15
Project:	Sanankoro	**% Solids:**	50
Sample:	SD0005		

Grind Time 5 Minutes

Size µm	Weight grams	% Retained Individual	% Retained Cumulative	Cumulative % Passing
300	1.5	0.7	0.7	99.3
212	5.2	2.5	3.2	96.8
150	13.1	6.3	9.6	90.4
106	15.9	7.7	17.2	82.8
75	11.6	5.6	22.8	77.2
53	11.1	5.4	28.2	71.8
38	7.6	3.7	31.8	68.2
-38	141.3	68.2	100.0	0.0
Total	**207.3**	**100.0**	-	-
d80	**91**			
d80$_{(Check)}$	91			

Grind Time 10 Minutes

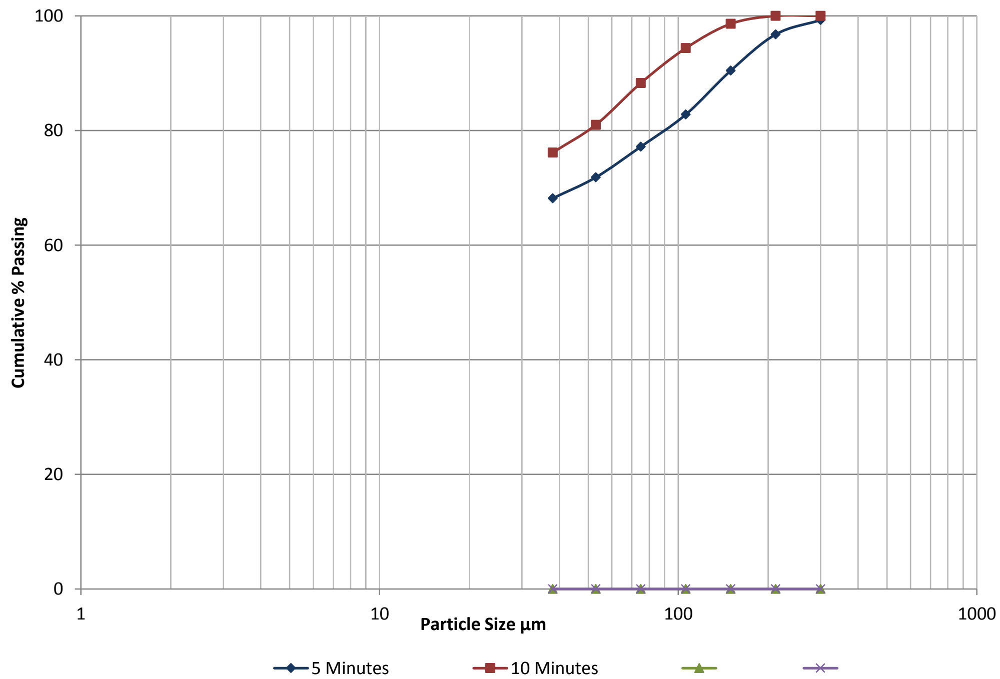
Size µm	Weight grams	% Retained Individual	% Retained Cumulative	Cumulative % Passing
300	0.0	0.0	0.0	100.0
212	0.0	0.0	0.0	100.0
150	2.6	1.4	1.4	98.6
106	8.0	4.3	5.6	94.4
75	11.4	6.1	11.7	88.3
53	13.7	7.3	19.0	81.0
38	9.1	4.8	23.9	76.1
-38	143.0	76.1	100.0	0.0
Total	**187.8**	**100.0**	-	-
d80	**50**			
d80$_{(Check)}$	#DIV/0!			

Grind Time

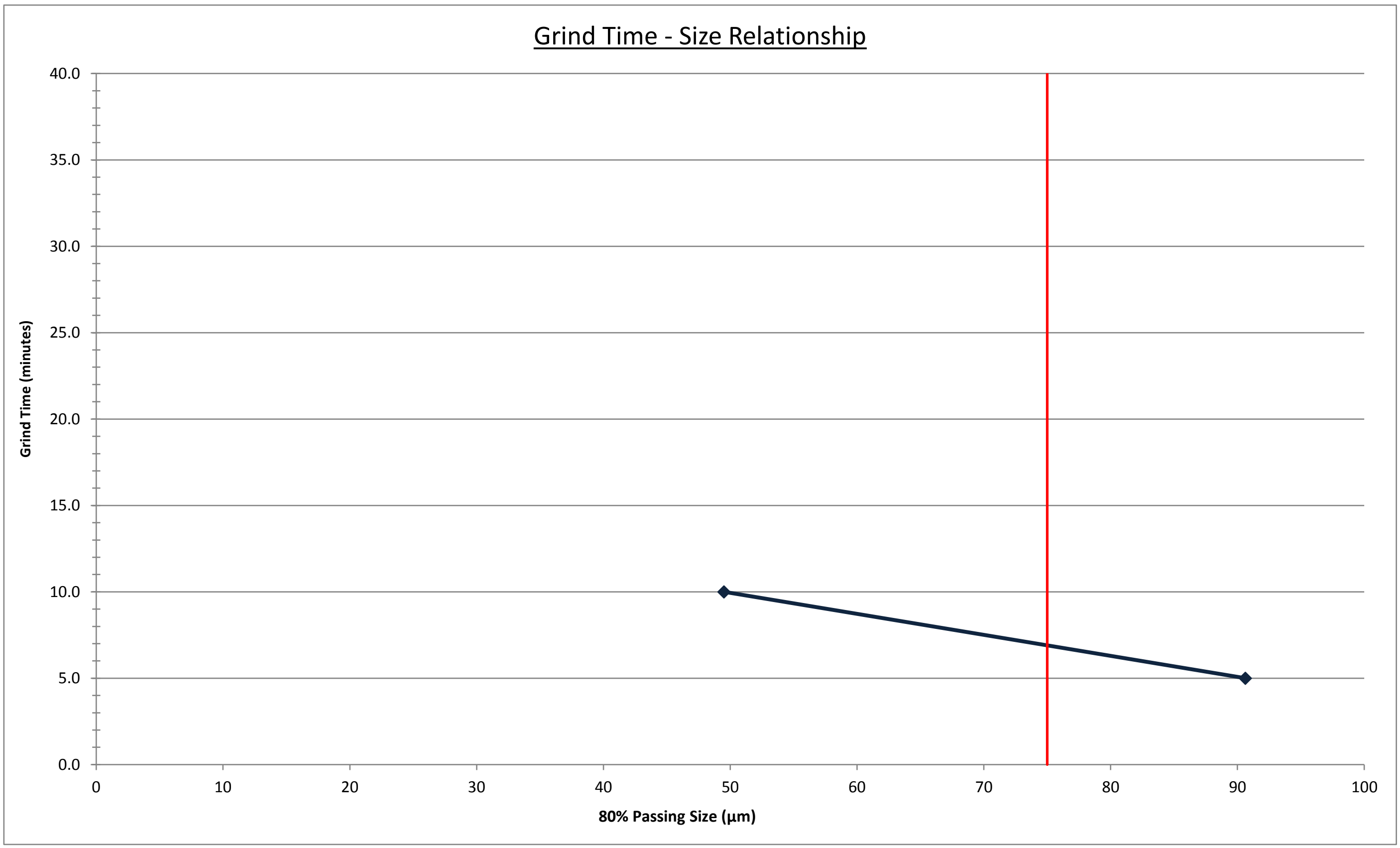
Size µm	Weight grams	% Retained Individual	% Retained Cumulative	Cumulative % Passing
300		#DIV/0!	#DIV/0!	#DIV/0!
212		#DIV/0!	#DIV/0!	#DIV/0!
150		#DIV/0!	#DIV/0!	#DIV/0!
106		#DIV/0!	#DIV/0!	#DIV/0!
75		#DIV/0!	#DIV/0!	#DIV/0!
53		#DIV/0!	#DIV/0!	#DIV/0!
38		#DIV/0!	#DIV/0!	#DIV/0!
-38		#DIV/0!	#DIV/0!	#DIV/0!
Total	**0.0**	**#DIV/0!**	-	-
d80	**#N/A**			
d80$_{(Check)}$	#N/A			

Grind Time

Size µm	Weight grams	% Retained Individual	% Retained Cumulative	Cumulative % Passing
300		#DIV/0!	#DIV/0!	#DIV/0!
212		#DIV/0!	#DIV/0!	#DIV/0!
150		#DIV/0!	#DIV/0!	#DIV/0!
106		#DIV/0!	#DIV/0!	#DIV/0!
75		#DIV/0!	#DIV/0!	#DIV/0!
53		#DIV/0!	#DIV/0!	#DIV/0!
38		#DIV/0!	#DIV/0!	#DIV/0!
-38		#DIV/0!	#DIV/0!	#DIV/0!
Total	**0.0**	**#DIV/0!**	-	-
d80	**#N/A**			
d80$_{(Check)}$	#N/A			



Job: ZT640703
Client: Cora Gold
Project: Sanankoro
Sample: SD0005



Target d80	Grind Time
75µm	6 Minutes 54 Seconds

Wardell Armstrong International Ltd

Job: ZT640703
Client: Cora Gold
Project: Sanankoro
Sample: SD0006

Mill: 4
Rods: 15
% Solids: 50

Grind Time 5 Minutes

Size µm	Weight grams	% Retained Individual	% Retained Cumulative	Cumulative % Passing
300	2.7	1.2	1.2	98.8
212	7.0	3.0	4.2	95.8
150	15.8	6.8	11.0	89.0
106	19.0	8.2	19.2	80.8
75	15.7	6.8	26.0	74.0
53	16.1	7.0	33.0	67.0
38	10.1	4.4	37.3	62.7
-38	145.0	62.7	100.0	0.0
Total	**231.4**	**100.0**	-	-
d80	**102**			
d80$_{(Check)}$	#DIV/0!			

Grind Time 10 Minutes

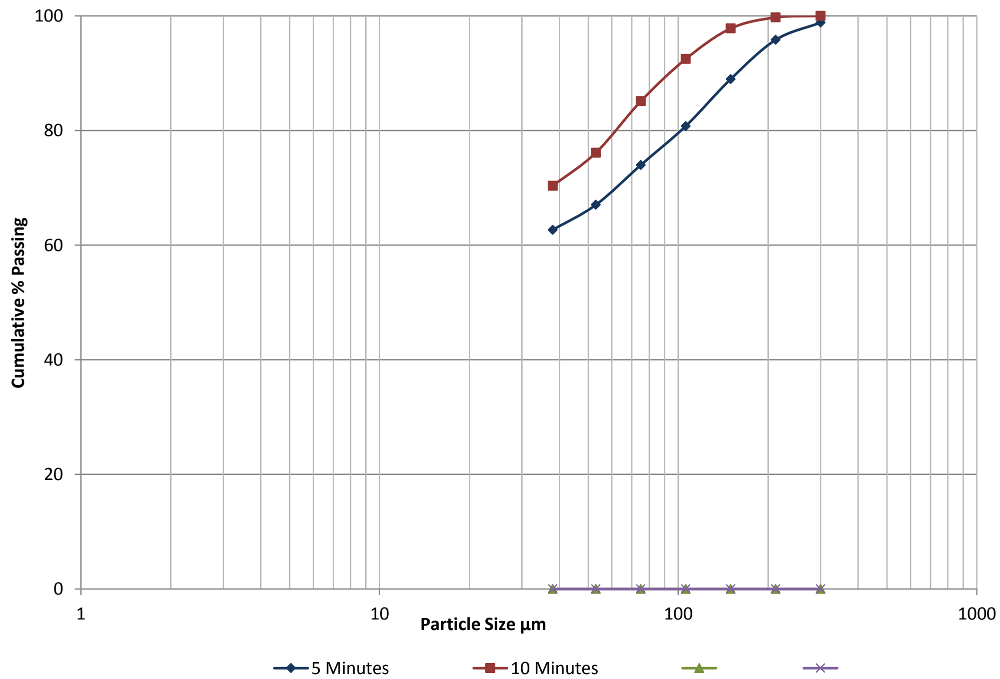
Size µm	Weight grams	% Retained Individual	% Retained Cumulative	Cumulative % Passing
300	0.0	0.0	0.0	100.0
212	0.7	0.3	0.3	99.7
150	4.9	1.9	2.2	97.8
106	13.6	5.3	7.5	92.5
75	18.8	7.4	14.9	85.1
53	23.0	9.0	23.9	76.1
38	14.7	5.8	29.7	70.3
-38	179.6	70.3	100.0	0.0
Total	**255.3**	**100.0**	-	-
d80	**63**			
d80$_{(Check)}$	63			

Grind Time

Size µm	Weight grams	% Retained Individual	% Retained Cumulative	Cumulative % Passing
300		#DIV/0!	#DIV/0!	#DIV/0!
212		#DIV/0!	#DIV/0!	#DIV/0!
150		#DIV/0!	#DIV/0!	#DIV/0!
106		#DIV/0!	#DIV/0!	#DIV/0!
75		#DIV/0!	#DIV/0!	#DIV/0!
53		#DIV/0!	#DIV/0!	#DIV/0!
38		#DIV/0!	#DIV/0!	#DIV/0!
-38		#DIV/0!	#DIV/0!	#DIV/0!
Total	**0.0**	**#DIV/0!**	-	-
d80	**#N/A**			
d80$_{(Check)}$	#N/A			

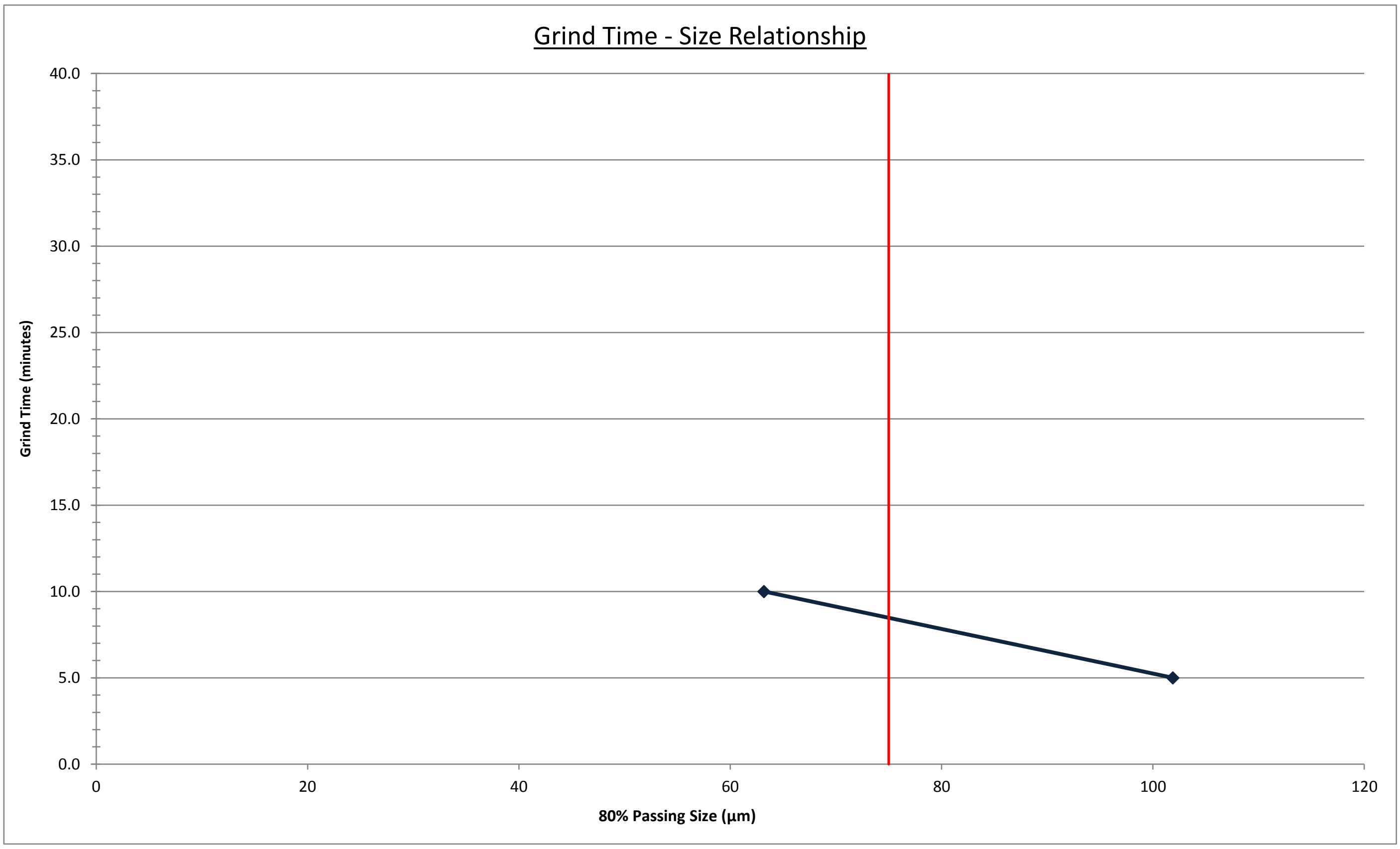
Grind Time

Size µm	Weight grams	% Retained Individual	% Retained Cumulative	Cumulative % Passing
300		#DIV/0!	#DIV/0!	#DIV/0!
212		#DIV/0!	#DIV/0!	#DIV/0!
150		#DIV/0!	#DIV/0!	#DIV/0!
106		#DIV/0!	#DIV/0!	#DIV/0!
75		#DIV/0!	#DIV/0!	#DIV/0!
53		#DIV/0!	#DIV/0!	#DIV/0!
38		#DIV/0!	#DIV/0!	#DIV/0!
-38		#DIV/0!	#DIV/0!	#DIV/0!
Total	**0.0**	**#DIV/0!**	-	-
d80	**#N/A**			
d80$_{(Check)}$	#N/A			



Wardell Armstrong International Ltd

Job: ZT640703
Client: Cora Gold
Project: Sanankoro
Sample: SD0006



Target d80	Grind Time
75µm	8 Minutes 29 Seconds

APPENDIX 3: Coarse Ore Bottle Roll Test Data

Kinetic Cyanide Leach Bottle Roll Test

Job: ZT640703
Client: Cora Gold
Project: Sanankoro
Sample: SD0005

Test A

Sample ID	Date	Pre-Treatment	NaCN Concentration (g/l) Initial	NaCN Concentration (g/l) Maintained	Initial Solids Weight (g)	Bottle Weight	Initial Solution (g)	% Solids	Feed Size (mm)	Initial Leach pH
BLT 1	28/01/2019	None	2.0	2.0	2000	325	3000	40.0	-20	8.60

Head Assay (g/t) Measured Au	Measured Ag	Back Calculated Au	Back Calculated Ag
0.64		0.70	0.55

Time (Days)	Reagent NaCN Added (g)	NaCN Consumed (g)	NaCN "Free" (%)	$Ca(OH)_2$ Added (g)	$Ca(OH)_2$ Consumed (kg/t)	Total Weight (g)	Measurement Solution Weight (g)	Extraction (ml)	Titre (ml)	pH	Liquor/Residue Assay Au (g/t)	Ag (g/t)	Recovery (%) wrt Head Au	wrt Head Ag	wrt Leach Residue Au	wrt Leach Residue Ag
0	6.00		0.20	0.6		5334.00	3009.0			10.50						
1	0.38	0.4	0.19	1.7	0.30	5334.00	3009.0	30	3.75	10.20	0.18	0.02	42.3	#DIV/0!	38.9	5.5
2	0.00	0.4	0.20	0	1.15	5325.00	3000.0	30	4.00	11.50	0.27	0.02	63.7	#DIV/0!	58.6	5.5
4	0.00	0.4	0.20	0	1.15	5326.00	3001.0	30	4.00	11.50	0.31	0.03	73.7	#DIV/0!	67.8	8.3
7	0.00	0.4	0.20	0	1.15	5325.00	3000.0	30	4.00	11.30	0.29	0.04	69.8	#DIV/0!	64.1	11.1
10	0.08	0.5	0.20	0	1.15	5325.00	3000.0	30	3.95	11.20	0.36	0.04	86.8	#DIV/0!	79.8	11.3
14	0.00	0.5	0.20	0.00	1.15	5325.00	3000.0	30	4.00	10.60	0.33	<0.01	80.6	#DIV/0!	74.1	3.1
17	0.00	0.5	0.20	0.2	1.15	5325.00	3000.0	30	4.00	10.47	0.34	0.03	83.8	#DIV/0!	77.0	8.7
21	0.00	0.8	0.19	0	1.25	5325.00	3000.0	50	3.75	10.55	0.35	0.03	86.9	#DIV/0!	79.9	8.7
Residue							2000.0				0.14	<0.5	21.9	#DIV/0!	20.1	91.3
Total (kg/t)	3.2	0.4			1.25											

Test B

Sample ID	Date	Pre-Treatment	NaCN Concentration (g/l) Initial	NaCN Concentration (g/l) Maintained	Initial Solids Weight (g)	Bottle Weight	Initial Solution (g)	% Solids	Feed Size (mm)	Initial Leach pH
BLT 2	28/01/2019	None	2.0	2.0	2000	334	3000	40.0	-20	8.70

Head Assay (g/t) Measured Au	Measured Ag	Back Calculated Au	Back Calculated Ag
0.64		1.51	0.63

Time (Days)	Reagent NaCN Added (g)	NaCN Consumed (g)	NaCN "Free" (%)	$Ca(OH)_2$ Added (g)	$Ca(OH)_2$ Consumed (kg/t)	Total Weight (g)	Measurement Solution Weight (g)	Extraction (ml)	Titre (ml)	pH	Liquor/Residue Assay Au (g/t)	Ag (g/t)	Recovery (%) wrt Head Au	wrt Head Ag	wrt Leach Residue Au	wrt Leach Residue Ag
0	6.00		0.20	0.6		5344.00	3010.0			10.55						
1	0.45	0.3	0.19	1.5	0.30	5344.00	3010.0	30	3.78	10.16	0.28	0.02	65.8	#DIV/0!	27.9	4.8
2	0.00	0.5	0.20	0	1.05	5334.00	3000.0	30	4.00	11.50	0.36	0.04	85.0	#DIV/0!	36.0	9.5
4	0.00	0.5	0.20	0	1.05	5334.00	3000.0	30	4.00	11.50	0.39	0.04	92.9	#DIV/0!	39.4	9.6
7	0.15	0.6	0.20	0	1.05	5334.00	3000.0	30	3.90	11.26	0.47	0.05	112.6	#DIV/0!	47.7	12.1
10	0.00	0.6	0.20	0	1.05	5334.00	3000.0	30	4.00	11.11	0.58	0.06	139.5	#DIV/0!	59.1	14.6
14	0.08	0.7	0.20	0.00	1.05	5333.00	2999.0	30	3.95	10.85	0.62	0.04	150.1	#DIV/0!	63.6	10.0
17	0.15	0.8	0.20	0.2	1.05	5334.00	3000.0	30	3.90	10.52	0.64	0.07	156.3	#DIV/0!	66.3	17.2
21	0.00	1.0	0.20	0	1.15	5334.00	3000.0	50	3.90	10.40	0.74	0.09	181.3	#DIV/0!	76.8	20.9
Residue							2000.0				0.35	<0.5	54.7	#DIV/0!	23.2	79.1
Total (kg/t)	3.4	0.5			1.15											

Wardell Armstrong International Ltd

Kinetic Cyanide Leach Bottle Roll Test

Job: ZT640703
Client: Cora Gold
Project: Sanankoro
Sample: SD0005

Test A

Sample ID	Date	Pre-Treatment	NaCN Concentration (g/l) Initial	NaCN Concentration (g/l) Maintained	Initial Solids Weight (g)	Bottle Weight	Initial Solution (g)	% Solids	Feed Size	Initial Leach pH
BLT 3	28/01/2019	None	2.0	2.0	2000	336	3000	40.0	-12.5	8.72

Head Assay (g/t) Measured Au	Measured Ag	Back Calculated Au	Back Calculated Ag
0.64		1.43	0.64

Time (Days)	Reagent NaCN Added (g)	NaCN Consumed (g)	NaCN "Free" (%)	$Ca(OH)_2$ Added (g)	$Ca(OH)_2$ Consumed (kg/t)	Total Weight (g)	Measurement Solution Weight (g)	Extraction (ml)	Titre	pH	Liquor/Residue Assay Au (g/t)	Ag (g/t)	Recovery (%) wrt Head Au	wrt Head Ag	wrt Leach Residue Au	wrt Leach Residue Ag
0	6.00		0.20	0.6		5343.00	3007.0			10.55						
1	1.50	0.7	0.18	1.5	0.30	5342.00	3006.0	30	3.50	10.13	0.29	0.02	68.1	#DIV/0!	30.5	4.7
2	0.00	1.5	0.20	0	1.05	5336.00	3000.0	30	4.00	11.51	0.39	0.04	92.1	#DIV/0!	41.2	9.4
4	0.00	1.5	0.20	0	1.05	5336.00	3000.0	30	4.00	11.51	0.41	0.05	97.7	#DIV/0!	43.7	11.9
7	0.00	1.5	0.20	0	1.05	5335.00	2999.0	30	4.00	11.30	0.48	0.05	115.0	#DIV/0!	51.5	12.0
10	0.00	1.5	0.20	0	1.05	5336.00	3000.0	30	4.00	11.19	0.56	0.06	134.9	#DIV/0!	60.4	14.4
14	0.00	1.5	0.20	0.00	1.05	5336.00	3000.0	30	4.00	11.07	0.58	0.04	140.9	#DIV/0!	63.1	9.9
17	0.30	1.8	0.19	0	1.05	5336.00	3000.0	30	3.80	10.78	0.61	0.07	149.3	#DIV/0!	66.8	17.0
21	0.00	1.8	0.20	0	1.05	5336.00	3000.0	50	4.00	10.45	0.68	0.09	167.2	#DIV/0!	74.8	21.9
Residue							2000.0				0.36	<0.5	56.3	#DIV/0!	25.2	78.1
Total (kg/t)	3.9	0.9			1.05											

Test B

Sample ID	Date	Pre-Treatment	NaCN Concentration (g/l) Initial	NaCN Concentration (g/l) Maintained	Initial Solids Weight (g)	Bottle Weight	Initial Solution (g)	% Solids	Feed Size	Initial Leach pH
BLT 4	28/01/2019	None	2.0	2.0	2000	350	3000	40.0	-12.5	8.57

Head Assay (g/t) Measured Au	Measured Ag	Back Calculated Au	Back Calculated Ag
0.64		1.11	0.64

Time (Days)	Reagent NaCN Added (g)	NaCN Consumed (g)	NaCN "Free" (%)	$Ca(OH)_2$ Added (g)	$Ca(OH)_2$ Consumed (kg/t)	Total Weight (g)	Measurement Solution Weight (g)	Extraction (ml)	Titre	pH	Liquor/Residue Assay Au (g/t)	Ag (g/t)	Recovery (%) wrt Head Au	wrt Head Ag	wrt Leach Residue Au	wrt Leach Residue Ag
0	6.00		0.20	0.6		5358.00	3008.0			10.50						
1	0.68	0.7	0.18	1.6	0.30	5358.00	3008.0	30	3.55	10.11	0.24	0.03	56.4	#DIV/0!	32.7	7.0
2	0.00	0.7	0.20	0	1.10	5350.00	3000.0	30	4.00	11.60	0.31	0.04	73.2	#DIV/0!	42.4	9.4
4	0.00	0.7	0.20	0	1.10	5350.00	3000.0	30	4.00	11.60	0.32	0.04	76.3	#DIV/0!	44.2	9.5
7	0.00	0.7	0.20	0	1.10	5350.00	3000.0	30	4.00	11.36	0.39	0.05	93.4	#DIV/0!	54.1	12.0
10	0.00	0.7	0.20	0	1.10	5350.00	3000.0	30	4.00	11.22	0.50	0.07	120.1	#DIV/0!	69.6	16.8
14	0.30	1.0	0.19	0.00	1.10	5350.00	3000.0	30	3.80	11.02	0.54	0.05	130.7	#DIV/0!	75.7	12.3
17	0.00	1.0	0.20	0	1.10	5350.00	3000.0	30	4.00	10.79	0.58	0.07	141.3	#DIV/0!	81.8	17.1
21	0.00	1.1	0.20	0	1.10	5350.00	3000.0	50	3.95	10.45	0.66	0.09	161.4	#DIV/0!	93.5	21.9
Residue							2000.0				0.072	<0.5	11.3	#DIV/0!	6.5	78.1
Total (kg/t)	3.5	0.5			1.10											

Kinetic Cyanide Leach Bottle Roll Test

Job: ZT640703
Client: Cora Gold
Project: Sanankoro
Sample: SD0005

Test A

Sample ID	Date	Pre-Treatment	NaCN Concentration (g/l) Initial	NaCN Concentration (g/l) Maintained	Initial Solids Weight (g)	Bottle Weight	Initial Solution (g)	% Solids	Feed Size	Initial Leach pH
BLT 5	28/01/2019	None	2.0	2.0	2000	341	3000	40.0	-6.3	8.72

Head Assay (g/t)			
Measured		Back Calculated	
Au	Ag	Au	Ag
0.64		0.98	0.62

Time (Days)	Reagent NaCN Added (g)	NaCN Consumed (g)	NaCN "Free" (%)	$Ca(OH)_2$ Added (g)	$Ca(OH)_2$ Consumed (kg/t)	Total Weight (g)	Measurement Solution Weight (g)	Extraction (ml)	Titre	pH	Liquor/Residue Assay Au (g/t)	Ag (g/t)	Recovery (%) wrt Head Au	wrt Head Ag	wrt Leach Residue Au	wrt Leach Residue Ag
0	6.00		0.20	0.5		5350.00	3009.0			10.53						
1	0.75	0.7	0.18	1.5	0.25	5350.00	3009.0	30	3.50	10.00	0.24	0.02	56.4	#DIV/0!	36.8	4.8
2	0.00	0.8	0.20	0	1.00	5341.00	3000.0	30	4.00	11.60	0.30	0.03	70.9	#DIV/0!	46.3	7.3
4	0.00	0.8	0.20	0	1.00	5341.00	3000.0	30	4.00	11.58	0.31	0.03	73.9	#DIV/0!	48.2	7.3
7	0.00	0.8	0.20	0	1.00	5340.00	2999.0	30	4.00	11.22	0.38	0.05	91.0	#DIV/0!	59.4	12.2
10	0.00	0.8	0.20	0	1.00	5341.00	3000.0	30	4.00	11.10	0.47	0.06	113.0	#DIV/0!	73.8	14.7
14	0.00	0.8	0.20	0.00	1.00	5341.00	3000.0	30	4.00	10.88	0.51	0.04	123.5	#DIV/0!	80.6	10.1
17	0.15	0.9	0.20	0	1.00	5341.00	3000.0	30	3.90	10.64	0.53	0.06	129.4	#DIV/0!	84.5	15.0
21	0.00	0.9	0.20	0	1.00	5341.00	3000.0	50	4.00	10.45	0.62	0.08	150.6	#DIV/0!	98.3	19.9
Residue							2000.0				0.017	<0.5	2.7	#DIV/0!	1.7	80.1
Total (kg/t)	3.5	0.5			1.00											

Test B

Sample ID	Date	Pre-Treatment	NaCN Concentration (g/l) Initial	NaCN Concentration (g/l) Maintained	Initial Solids Weight (g)	Bottle Weight	Initial Solution (g)	% Solids	Feed Size	Initial Leach pH
BLT 6	28/01/2019	None	2.0	2.0	2000	348	3000	40.0	-6.3	8.75

Head Assay (g/t)			
Measured		Back Calculated	
Au	Ag	Au	Ag
0.64		0.80	0.59

Time (Days)	Reagent NaCN Added (g)	NaCN Consumed (g)	NaCN "Free" (%)	$Ca(OH)_2$ Added (g)	$Ca(OH)_2$ Consumed (kg/t)	Total Weight (g)	Measurement Solution Weight (g)	Extraction (ml)	Titre	pH	Liquor/Residue Assay Au (g/t)	Ag (g/t)	Recovery (%) wrt Head Au	wrt Head Ag	wrt Leach Residue Au	wrt Leach Residue Ag
0	6.00		0.20	0.5		5357.00	3009.0			10.60						
1	0.83	0.8	0.17	1.5	0.25	5357.00	3009.0	30	3.45	10.08	0.22	0.02	51.7	#DIV/0!	41.6	5.1
2	0.00	0.8	0.20	0	1.00	5348.00	3000.0	30	4.00	11.60	0.30	0.04	70.8	#DIV/0!	57.0	10.1
4	0.00	0.8	0.20	0	1.00	5348.00	3000.0	30	4.00	11.52	0.30	0.04	71.5	#DIV/0!	57.5	10.2
7	0.00	0.8	0.20	0	1.00	5348.00	3000.0	30	4.00	11.20	0.36	0.05	86.3	#DIV/0!	69.4	12.9
10	0.00	0.8	0.20	0	1.00	5348.00	3000.0	30	4.00	11.10	0.43	0.06	103.5	#DIV/0!	83.3	15.5
14	0.00	0.8	0.20	0.00	1.00	5348.00	3000.0	30	4.00	10.95	0.42	0.04	102.2	#DIV/0!	82.2	10.6
17	0.08	0.9	0.20	0	1.00	5348.00	3000.0	30	3.95	10.72	0.42	0.04	103.2	#DIV/0!	83.0	10.7
21	0.00	1.2	0.19	0	1.00	5348.00	3000.0	50	3.80	10.45	0.49	0.06	120.6	#DIV/0!	97.0	15.9
Residue							2000.0				0.02	<0.5	3.8	#DIV/0!	3.0	84.1
Total (kg/t)	3.5	0.6			1.00											

Wardell Armstrong International Ltd

Kinetic Cyanide Leach Bottle Roll Test

Job: ZT640703
Client: Cora Gold
Project: Sanankoro
Sample: SD0006

Test A

Sample ID	Date	Pre-Treatment	NaCN Concentration (g/l) Initial	NaCN Concentration (g/l) Maintained	Initial Solids Weight (g)	Bottle Weight	Initial Solution (g)	% Solids	Feed Size (mm)	Initial Leach pH
BLT 7	28/01/2019	None	2.0	2.0	2000	338	3000	40.0	-20	8.75

Head Assay (g/t) Measured Au	Measured Ag	Back Calculated Au	Back Calculated Ag
2.70	1.20	2.55	0.91

Time (Days)	Reagent NaCN Added (g)	NaCN Consumed (g)	NaCN "Free" (%)	$Ca(OH)_2$ Added (g)	$Ca(OH)_2$ Consumed (kg/t)	Total Weight (g)	Measurement Solution Weight (g)	Extraction (ml)	Titre (ml)	pH	Liquor/Residue Assay Au (g/t)	Ag (g/t)	Recovery (%) wrt Head Au	wrt Head Ag	wrt Leach Residue Au	wrt Leach Residue Ag
0	6.00		0.20	0.6		5349.00	3011.0			10.55						
1	0.75	0.7	0.18	1.6	0.30	5349.00	3011.0	30	3.50	10.01	0.38	0.11	21.2	13.8	22.5	18.2
2	0.00	0.8	0.20	0	1.10	5338.00	3000.0	30	4.00	10.97	0.49	0.13	27.4	16.4	29.1	21.6
4	0.00	0.8	0.20	0.3	1.10	5338.00	3000.0	30	4.00	10.54	0.67	0.17	37.7	21.6	40.0	28.4
7	0.00	0.8	0.20	1	1.25	5338.00	3000.0	30	4.00	10.30	0.86	0.22	48.6	28.0	51.6	37.0
10	0.00	0.8	0.20	0	1.75	5338.00	3000.0	30	4.00	10.75	1.02	0.24	58.0	30.8	61.5	40.6
14	0.00	0.7	0.20	0.00	1.75	5339.00	3001.0	30	4.00	10.64	0.96	0.20	55.3	26.1	58.6	34.4
17	0.08	0.8	0.20	1	1.75	5338.00	3000.0	30	3.95	10.70	0.96	0.23	55.8	30.1	59.1	39.7
21	0.00	0.8	0.20	0	2.25	5337.00	2999.0	50	4.00	10.72	1.03	0.26	59.9	34.1	63.5	45.0
Residue							2000.0				0.93	<0.5	34.4	41.7	36.5	55.0
Total (kg/t)	3.4	0.4			2.25											

Test B

Sample ID	Date	Pre-Treatment	NaCN Concentration (g/l) Initial	NaCN Concentration (g/l) Maintained	Initial Solids Weight (g)	Bottle Weight	Initial Solution (g)	% Solids	Feed Size (mm)	Initial Leach pH
BLT 8	28/01/2019	None	2.0	2.0	2000	334	3000	40.0	-20	8.75

Head Assay (g/t) Measured Au	Measured Ag	Back Calculated Au	Back Calculated Ag
2.70	1.20	2.70	1.87

Time (Days)	Reagent NaCN Added (g)	NaCN Consumed (g)	NaCN "Free" (%)	$Ca(OH)_2$ Added (g)	$Ca(OH)_2$ Consumed (kg/t)	Total Weight (g)	Measurement Solution Weight (g)	Extraction (ml)	Titre (ml)	pH	Liquor/Residue Assay Au (g/t)	Ag (g/t)	Recovery (%) wrt Head Au	wrt Head Ag	wrt Leach Residue Au	wrt Leach Residue Ag
0	6.00		0.20	0.5		5345.00	3011.0			10.54						
1	0.60	0.6	0.18	1.6	0.25	5345.00	3011.0	30	3.60	10.02	0.44	0.12	24.5	15.1	24.5	9.6
2	0.00	0.6	0.20	0	1.05	5334.00	3000.0	30	4.00	10.90	0.61	0.16	34.1	20.2	34.1	12.9
4	0.00	0.6	0.20	0.3	1.05	5334.00	3000.0	30	4.00	10.48	0.86	0.21	48.4	26.6	48.3	17.0
7	0.00	0.6	0.20	1	1.20	5334.00	3000.0	30	4.00	10.33	0.97	0.28	55.0	35.6	54.9	22.8
10	0.00	0.6	0.20	0	1.70	5334.00	3000.0	30	4.00	10.79	1.15	0.27	65.5	34.7	65.4	22.2
14	0.30	0.9	0.19	0.00	1.70	5334.00	3000.0	30	3.80	10.71	1.11	0.24	63.9	31.3	63.8	20.1
17	0.08	1.0	0.20	0	1.70	5334.00	3000.0	30	3.95	10.60	1.09	0.27	63.4	35.4	63.3	22.6
21	0.00	1.4	0.19	0	1.70	5334.00	3000.0	50	3.70	10.60	1.20	0.30	70.1	39.4	70.0	25.3
Residue							2000.0				0.81	1.4	30.0	116.7	30.0	74.7
Total (kg/t)	3.5	0.7			1.70											

Kinetic Cyanide Leach Bottle Roll Test

Job: ZT640703
Client: Cora Gold
Project: Sanankoro
Sample: SD0006

Test A

Sample ID	Date	Pre-Treatment	NaCN Concentration (g/l) Initial	Maintained	Initial Solids Weight (g)	Bottle Weight	Initial Solution (g)	% Solids	Feed Size	Initial Leach pH
BLT 9	28/01/2019	None	2.0	2.0	2000	338	3000	40.0	-12.5	8.75

Head Assay (g/t) Measured Au	Ag	Back Calculated Au	Ag
2.70	1.20	1.70	0.91

Time (Days)	Reagent NaCN Added (g)	Consumed (g)	"Free" (%)	$Ca(OH)_2$ Added (g)	Consumed (kg/t)	Total Weight (g)	Measurement Solution Weight (g)	Extraction (ml)	Titre	pH
0	6.00		0.20	0.6		5345.00	3007.0			10.50
1	1.05	1.0	0.17	2.5	0.30	5345.00	3007.0	30	3.30	9.93
2	0.00	1.1	0.20	0	1.55	5338.00	3000.0	30	4.00	10.90
4	0.00	1.1	0.20	0	1.55	5338.00	3000.0	30	4.00	10.82
7	0.00	1.1	0.20	0	1.55	5337.00	2999.0	30	4.00	10.61
10	0.00	1.1	0.20	0.5	1.55	5338.00	3000.0	30	4.00	10.42
14	0.23	1.1	0.20	0.00	1.80	5338.00	3000.0	30	4.00	10.60
17	0.00	1.3	0.20	1	1.80	5338.00	3000.0	30	4.00	10.42
21	0.00	1.3	0.20	0	2.30	5338.00	3000.0	50	4.00	11.05
Residue							2000.0			
Total (kg/t)	3.6	0.6			2.30					

Time (Days)	Liquor/Residue Assay Au (g/t)	Ag (g/t)	Recovery (%) wrt Head Au	Ag	wrt Leach Residue Au	Ag
0						
1	0.58	0.16	32.3	20.0	51.3	26.4
2	0.73	0.19	40.9	24.0	64.9	31.5
4	0.77	0.20	43.5	25.4	69.1	33.5
7	0.82	0.23	46.7	29.4	74.1	38.7
10	0.98	0.25	56.1	32.2	89.0	42.4
14	0.92	0.22	53.3	28.8	84.6	37.9
17	0.90	0.24	52.7	31.6	83.6	41.5
21	0.97	0.26	57.1	34.4	90.6	45.2
Residue	0.16	<0.5	5.9	41.7	9.4	54.8

Test B

Sample ID	Date	Pre-Treatment	NaCN Concentration (g/l) Initial	Maintained	Initial Solids Weight (g)	Bottle Weight	Initial Solution (g)	% Solids	Feed Size	Initial Leach pH
BLT 10	28/01/2019	None	2.0	2.0	2000	335	3000	40.0	-12.5	8.72

Head Assay (g/t) Measured Au	Ag	Back Calculated Au	Ag
2.70	1.20	2.34	0.92

Time (Days)	Reagent NaCN Added (g)	Consumed (g)	"Free" (%)	$Ca(OH)_2$ Added (g)	Consumed (kg/t)	Total Weight (g)	Measurement Solution Weight (g)	Extraction (ml)	Titre	pH
0	6.00		0.20	0.6		5344.00	3009.0			10.50
1	0.98	1.0	0.17	2.1	0.30	5343.00	3008.0	30	3.35	10.00
2	0.00	1.0	0.20	0	1.35	5336.00	3001.0	30	4.00	11.09
4	0.00	1.0	0.20	0	1.35	5335.00	3000.0	30	4.00	10.71
7	0.00	1.0	0.20	0.5	1.35	5334.00	2999.0	30	4.00	10.42
10	0.00	1.0	0.20	0	1.60	5335.00	3000.0	30	4.00	10.70
14	0.00	1.0	0.20	0.50	1.60	5335.00	3000.0	30	4.00	10.40
17	0.00	1.0	0.20	0	1.85	5337.00	3002.0	30	4.00	10.89
21	0.00	1.2	0.19	0	1.85	5336.00	3001.0	50	3.85	10.80
Residue							2000.0			
Total (kg/t)	3.5	0.6			1.85					

Time (Days)	Liquor/Residue Assay Au (g/t)	Ag (g/t)	Recovery (%) wrt Head Au	Ag	wrt Leach Residue Au	Ag
0						
1	0.44	0.13	24.5	16.3	28.3	21.3
2	0.57	0.15	31.9	18.9	36.9	24.7
4	0.75	0.19	42.2	24.1	48.8	31.5
7	0.89	0.24	50.4	30.6	58.2	39.9
10	1.04	0.26	59.3	33.4	68.4	43.6
14	1.01	0.23	58.2	30.0	67.1	39.1
17	1.01	0.25	58.8	32.8	67.8	42.8
21	1.08	0.27	62.9	34.9	72.6	45.6
Residue	0.64	<0.5	23.7	41.7	27.4	54.4

Wardell Armstrong International Ltd

Kinetic Cyanide Leach Bottle Roll Test

Job: ZT640703
Client: Cora Gold
Project: Sanankoro
Sample: SD0006

Test A

Sample ID	Date	Pre-Treatment	NaCN Concentration (g/l) Initial	NaCN Concentration (g/l) Maintained	Initial Solids Weight (g)	Bottle Weight	Initial Solution (g)	% Solids	Feed Size	Initial Leach pH
BLT 11	28/01/2019	None	2.0	2.0	2000	335	3000	40.0	-6.3	8.73

Head Assay (g/t) Measured Au	Measured Ag	Back Calculated Au	Back Calculated Ag
2.70	1.20	2.18	0.99

Time (Days)	Reagent NaCN Added (g)	NaCN Consumed (g)	NaCN "Free" (%)	$Ca(OH)_2$ Added (g)	$Ca(OH)_2$ Consumed (kg/t)	Total Weight (g)	Measurement Solution Weight (g)	Solution Extraction (ml)	Titre	pH	Liquor/Residue Assay Au (g/t)	Liquor/Residue Assay Ag (g/t)	Recovery (%) wrt Head Au	wrt Head Ag	wrt Leach Residue Au	wrt Leach Residue Ag
0	6.00		0.20	0.6		5344.00	3009.0			10.55						
1	0.90	0.9	0.17	2.1	0.30	5344.00	3009.0	30	3.40	9.92	0.66	0.19	36.8	23.8	45.6	28.8
2	0.00	0.9	0.20	0	1.35	5335.00	3000.0	30	4.00	10.55	0.86	0.22	48.1	27.7	59.7	33.5
4	0.00	0.9	0.20	0.2	1.35	5335.00	3000.0	30	4.00	10.56	0.99	0.25	55.8	31.8	69.2	38.4
7	0.00	0.9	0.20	0	1.45	5334.00	2999.0	30	4.00	10.55	1.15	0.28	65.3	35.8	80.9	43.3
10	0.00	0.9	0.20	0.5	1.45	5335.00	3000.0	30	4.00	10.38	1.30	0.31	74.3	39.9	92.0	48.3
14	0.15	1.1	0.20	0.00	1.70	5335.00	3000.0	30	3.90	10.60	1.26	0.27	72.8	35.3	90.1	42.7
17	0.23	1.3	0.19	0	1.70	5335.00	3000.0	30	3.85	10.64	1.22	0.30	71.2	39.4	88.3	47.6
21	0.00	1.5	0.19	0	1.70	5335.00	3000.0	50	3.85	10.62	1.31	0.31	76.6	41.0	95.0	49.6
Residue							2000.0				0.11	<0.5	4.1	41.7	5.0	50.4
Total (kg/t)	3.6	0.8			1.70											

Test B

Sample ID	Date	Pre-Treatment	NaCN Concentration (g/l) Initial	NaCN Concentration (g/l) Maintained	Initial Solids Weight (g)	Bottle Weight	Initial Solution (g)	% Solids	Feed Size	Initial Leach pH
BLT 12	28/01/2019	None	2.0	2.0	2000	357	3000	40.0	-6.3	8.75

Head Assay (g/t) Measured Au	Measured Ag	Back Calculated Au	Back Calculated Ag
2.70	1.20	1.84	0.93

Time (Days)	Reagent NaCN Added (g)	NaCN Consumed (g)	NaCN "Free" (%)	$Ca(OH)_2$ Added (g)	$Ca(OH)_2$ Consumed (kg/t)	Total Weight (g)	Measurement Solution Weight (g)	Solution Extraction (ml)	Titre	pH	Liquor/Residue Assay Au (g/t)	Liquor/Residue Assay Ag (g/t)	Recovery (%) wrt Head Au	wrt Head Ag	wrt Leach Residue Au	wrt Leach Residue Ag
0	6.00		0.20	0.6		5355.00	2998.0			10.55						
1	0.98	1.0	0.17	2	0.30	5354.00	2997.0	30	3.35	9.95	0.49	0.15	27.2	18.7	39.9	24.2
2	0.00	1.0	0.20	0	1.30	5357.00	3000.0	30	4.00	10.70	0.63	0.18	35.3	22.7	51.8	29.3
4	0.00	1.0	0.20	0.2	1.30	5358.00	3001.0	30	4.00	10.56	0.77	0.22	43.4	27.9	63.7	36.1
7	0.08	1.1	0.20	0.5	1.40	5357.00	3000.0	30	3.95	10.46	0.93	0.25	52.7	31.9	77.4	41.3
10	0.08	1.1	0.20	0	1.65	5357.00	3000.0	30	3.98	10.75	1.08	0.27	61.6	34.8	90.4	44.9
14	0.23	1.4	0.19	0.50	1.65	5357.00	3000.0	30	3.85	10.41	1.02	0.24	58.8	31.3	86.4	40.5
17	0.00	1.4	0.20	0	1.90	5357.00	3000.0	30	4.00	10.80	1.00	0.27	58.3	35.4	85.6	45.7
21	0.00	1.5	0.20	0	1.90	5357.00	3000.0	50	3.90	10.80	1.06	0.27	62.2	35.7	91.3	46.2
Residue							2000.0				0.16	<0.5	5.9	41.7	8.7	53.8
Total (kg/t)	3.7	0.8			1.90											

APPENDIX 4: Coarse Ore Bottle Roll Particle Size Analysis Data

Wardell Armstrong International

Job: ZT640703
Client: Cora Gold
Project: Sanankoro
Sample: SD0005

Fraction: -20.0mm Bottle Roll Feed

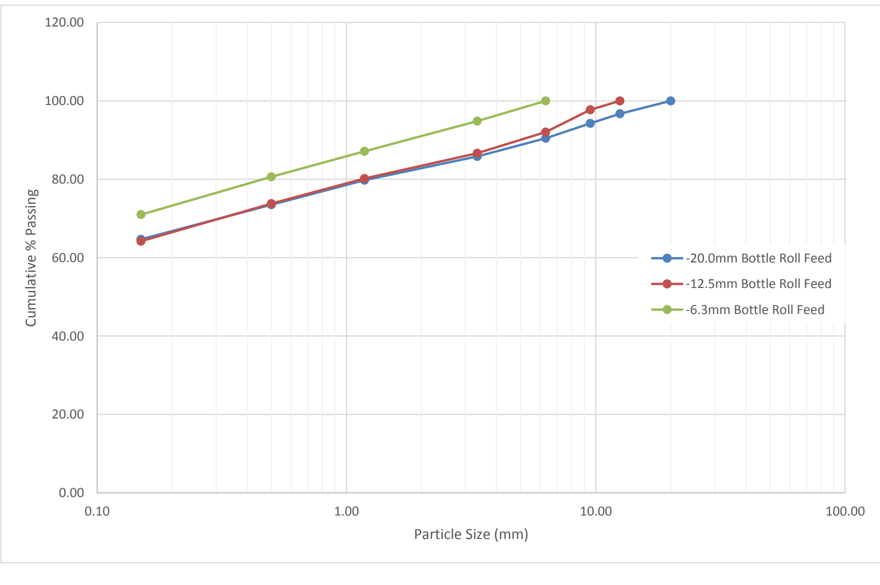
Size	Weight	% Retained		Cumulative %	Assay, Au	Distribution, Au (%)		Cumulative %
mm	grams	Individual	Cumulative	Passing	(ppm)	Individual	Cumulative	Passing
20.00	0.0	0.00	0.00	100.00		0.00	0.00	100.00
12.50	63.3	3.29	3.29	96.71	2.22	6.92	6.92	93.08
9.50	47.5	2.47	5.75	94.25	4.24	9.92	16.84	83.16
6.30	73.4	3.81	9.56	90.44	9.52	34.42	51.26	48.74
3.35	88.8	4.61	14.17	85.83	1.34	5.86	57.12	42.88
1.18	117.2	6.08	20.26	79.74	1.63	9.41	66.53	33.47
0.50	120.0	6.23	26.49	73.51	0.32	1.89	68.42	31.58
0.15	170.1	8.83	35.32	64.68	0.18	1.51	69.93	30.07
-0.15	1245.9	64.68	100.00	0.00	0.49	30.07	100.00	0.00
Total	**1926.2**	**100.00**	-	-	**1.05**		-	-
d80				**1.28**				**11.02**
d80$_{(Check)}$				1.27				9.21

Fraction: -12.5mm Bottle Roll Feed

Size	Weight	% Retained		Cumulative %	Assay, Au	Distribution, Au (%)		Cumulative %
mm	grams	Individual	Cumulative	Passing	(ppm)	Individual	Cumulative	Passing
12.50	0.00	0.00	0.00	100.00		0.00	0.00	100.00
9.50	45.10	2.27	2.27	97.73	3.12	1.98	1.98	98.02
6.30	113.10	5.70	7.98	92.02	35.09	55.92	57.90	42.10
3.35	106.70	5.38	13.36	86.64	1.74	2.62	60.52	39.48
1.18	128.4	6.47	19.83	80.17	2.33	4.22	64.73	35.27
0.50	126.7	6.39	26.22	73.78	14.45	25.80	90.53	9.47
0.15	190.1	9.58	35.80	64.20	1.26	3.37	93.90	6.10
-0.15	1273.3	64.20	100.00	0.00	0.34	6.10	100.00	0.00
Total	**1983.4**	**100.00**	-	-	**3.58**		-	-
d80				**1.15**				**108.97**
d80$_{(Check)}$				#DIV/0!				8.47

Fraction: -6.3mm Bottle Roll Feed

Size	Weight	% Retained		Cumulative %	Assay, Au	Distribution, Au (%)		Cumulative %
mm	grams	Individual	Cumulative	Passing	(ppm)	Individual	Cumulative	Passing
6.30	0.00	0.00	0.00	100.00		0.00	0.00	100.00
3.35	100.10	5.17	5.17	94.83	4.96	25.70	25.70	74.30
1.18	149.30	7.71	12.87	87.13	4.52	34.93	60.64	39.36
0.50	125.7	6.49	19.36	80.64	1.66	10.80	71.44	28.56
0.15	187.3	9.67	29.03	70.97	0.45	4.36	75.80	24.20
-0.15	1374.9	70.97	100.00	0.00	0.34	24.20	100.00	0.00
Total	**1937.3**	**100.00**	-	-	**1.00**		-	-
d80				**0.46**				**4.28**
d80$_{(Check)}$				#DIV/0!				4.00



Job: ZT640703
Client: Cora Gold
Project: Sanankoro
Sample: SD0005

Fraction: -20.0mm Bottle Roll Leach Residue

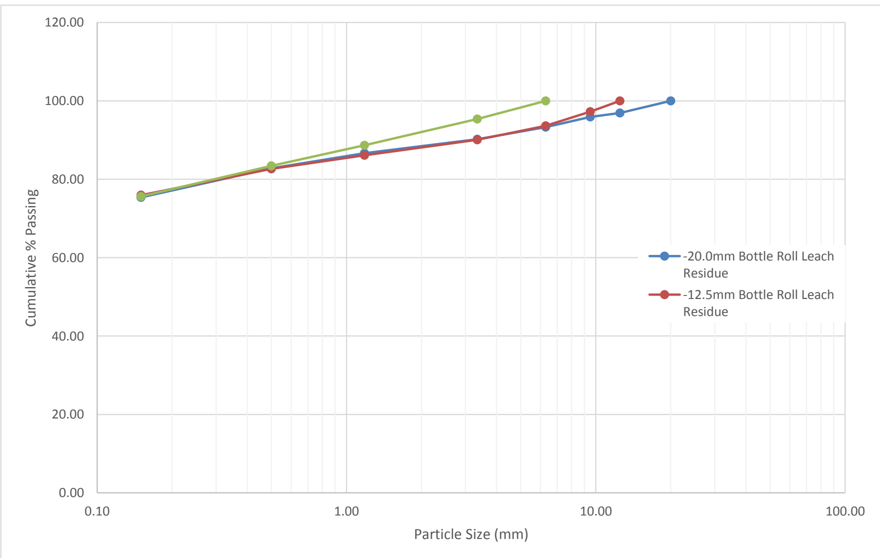
Size mm	Weight grams	% Retained Individual	% Retained Cumulative	Cumulative % Passing	Assay, Au (ppm)	Distribution, Au (%) Individual	Distribution, Au (%) Cumulative	Cumulative % Passing
20.00	0.0	0.00	0.00	100.00		0.00	0.00	100.00
12.50	60.1	3.10	3.10	96.90	0.042	0.79	0.79	99.21
9.50	19.3	1.00	4.09	95.91	0.065	0.39	1.18	98.82
6.30	50.3	2.59	6.69	93.31	0.36	5.66	6.84	93.16
3.35	59.9	3.09	9.78	90.22	0.17	3.18	10.03	89.97
1.18	69.1	3.57	13.34	86.66	3.54	76.49	86.52	13.48
0.50	74.8	3.86	17.20	82.80	0.2	4.68	91.20	8.80
0.15	143.9	7.42	24.63	75.37	0.023	1.03	92.23	7.77
-0.15	1460.7	75.37	100.00	0.00	0.017	7.77	100.00	0.00
Total	**1938.0**	**100.00**	-	-	**0.16**		-	-
d80				**0.31**				**32.49**
d80(Check)				0.37				3.07

Fraction: -12.5mm Bottle Roll Leach Residue

Size mm	Weight grams	% Retained Individual	% Retained Cumulative	Cumulative % Passing	Assay, Au (ppm)	Distribution, Au (%) Individual	Distribution, Au (%) Cumulative	Cumulative % Passing
12.50	0.00	0.00	0.00	100.00		0.00	0.00	100.00
9.50	53.24	2.76	2.76	97.24	1.36	19.22	19.22	80.78
6.30	69.61	3.61	6.38	93.62	0.11	2.03	21.25	78.75
3.35	68.36	3.55	9.93	90.07	0.52	9.44	30.69	69.31
1.18	76.7	3.98	13.91	86.09	1.45	29.52	60.21	39.79
0.50	66.2	3.43	17.34	82.66	0.068	1.19	61.40	38.60
0.15	129.5	6.72	24.06	75.94	1.01	34.72	96.12	3.88
-0.15	1462.7	75.94	100.00	0.00	<0.01	3.88	100.00	0.00
Total	**1926.2**	**100.00**	-	-	**0.20**		-	-
d80				**0.31**				**8.01**
d80(Check)				0.36				#DIV/0!

Fraction: -6.3mm Bottle Roll Leach Residue

Size mm	Weight grams	% Retained Individual	% Retained Cumulative	Cumulative % Passing	Assay, Au (ppm)	Distribution, Au (%) Individual	Distribution, Au (%) Cumulative	Cumulative % Passing
6.30	0.00	0.00	0.00	100.00		0.00	0.00	100.00
3.35	87.92	4.63	4.63	95.37	0.72	37.80	37.80	62.20
1.18	127.21	6.70	11.32	88.68	0.66	50.13	87.93	12.07
0.50	99.7	5.25	16.57	83.43	0.026	1.55	89.47	10.53
0.15	148.2	7.80	24.37	75.63	0.022	1.95	91.42	8.58
-0.15	1436.8	75.63	100.00	0.00	<0.01	8.58	100.00	0.00
Total	**1899.8**	**100.00**	-	-	**0.09**		-	-
d80				**0.29**				**4.82**
d80(Check)				0.35				4.74

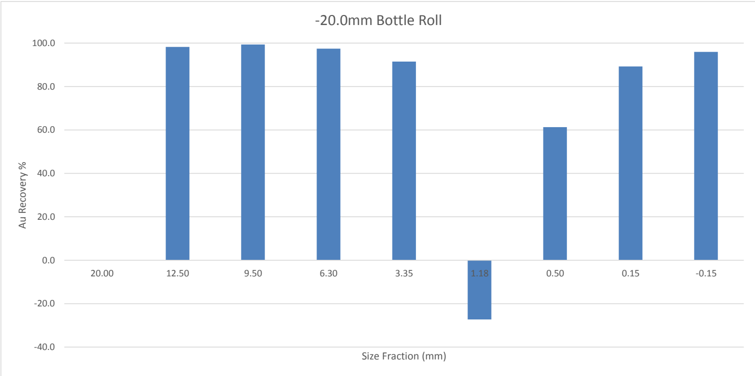


Wardell Armstrong International

Job: ZT640703
Client: Cora Gold
Project: Sanankoro
Sample: SD0005

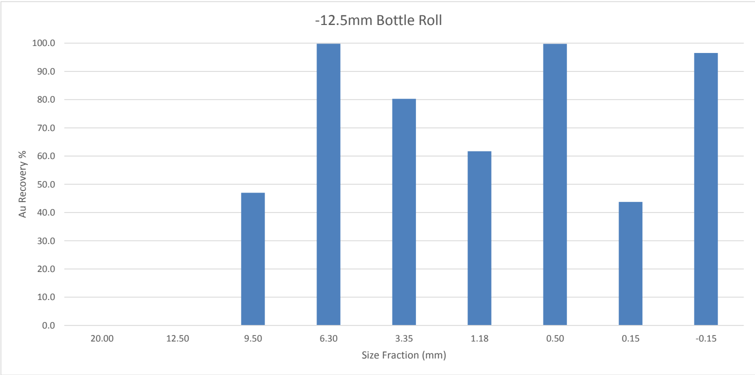
Fraction: -20mm Bottle Roll

Size (mm)	Bottle Roll Feed			Bottle Roll Residue			Au Recovery
	Mass (g)	Mass (%)	Au (ppm)	Mass (g)	Mass (%)	Au (ppm)	(%)
20.00							
12.50	63.3	3.3	2.22	60.1	3.1	0.042	98.22
9.50	47.5	2.5	4.24	19.3	1.0	0.065	99.38
6.30	73.4	3.8	9.52	50.3	2.6	0.36	97.43
3.35	88.8	4.6	1.34	59.9	3.1	0.17	91.50
1.18	117.2	6.1	1.63	69.1	3.6	3.54	-27.25
0.50	120.0	6.2	0.32	74.8	3.9	0.2	61.28
0.15	170.1	8.8	0.18	143.9	7.4	0.023	89.26
-0.15	1245.9	64.7	0.49	1460.7	75.4	0.017	95.96
Total	1926.2	100.0	1.05	1938.0	100.0	0.16	89.73



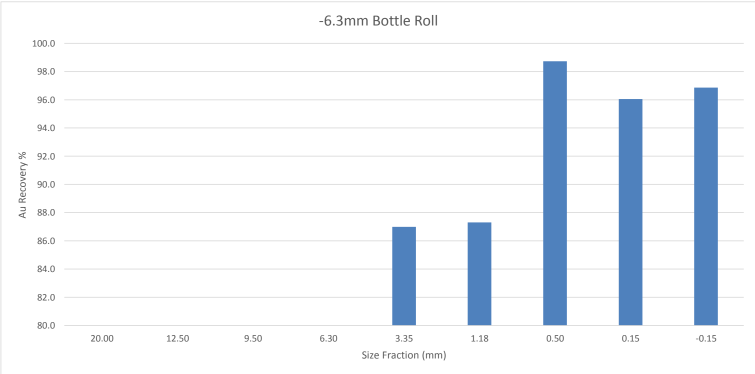
Fraction: -12.5mm Bottle Roll

Size (mm)	Bottle Roll Feed			Bottle Roll Residue			Au Recovery
	Mass (g)	Mass (%)	Au (ppm)	Mass (g)	Mass (%)	Au (ppm)	(%)
20.00							
12.50							
9.50	45.1	2.3	3.12	53.2	2.8	1.36	47.02
6.30	113.1	5.7	35.09	69.6	3.6	0.11	99.80
3.35	106.7	5.4	1.74	68.4	3.5	0.52	80.29
1.18	128.4	6.5	2.33	76.7	4.0	1.45	61.73
0.50	126.7	6.4	14.45	66.2	3.4	0.068	99.75
0.15	190.1	9.6	1.26	129.5	6.7	1.01	43.78
-0.15	1273.3	64.2	0.34	1462.7	75.9	0.01	96.52
Total	1983.4	100.0	3.58	1926.2	100.0	0.20	89.88



Fraction: -6.3mm Bottle Roll

Size (mm)	Bottle Roll Feed			Bottle Roll Residue			Au Recovery
	Mass (g)	Mass (%)	Au (ppm)	Mass (g)	Mass (%)	Au (ppm)	(%)
20.00							
12.50							
9.50							
6.30							
3.35	100.1	5.2	4.96	87.9	4.6	0.72	87.00
1.18	149.3	7.7	4.52	127.2	6.7	0.66	87.31
0.50	125.7	6.5	1.66	99.7	5.2	0.026	98.73
0.15	187.3	9.7	0.45	148.2	7.8	0.022	96.06
-0.15	1374.9	71.0	0.34	1436.8	75.6	0.01	96.87
	1937.3	100.0	1.00	1899.8	100.0	0.09	95.80



Wardell Armstrong International

Job: ZT640703
Client: Cora Gold
Project: Sanankoro
Sample: SD0006

Fraction: -20.0mm Bottle Roll Feed

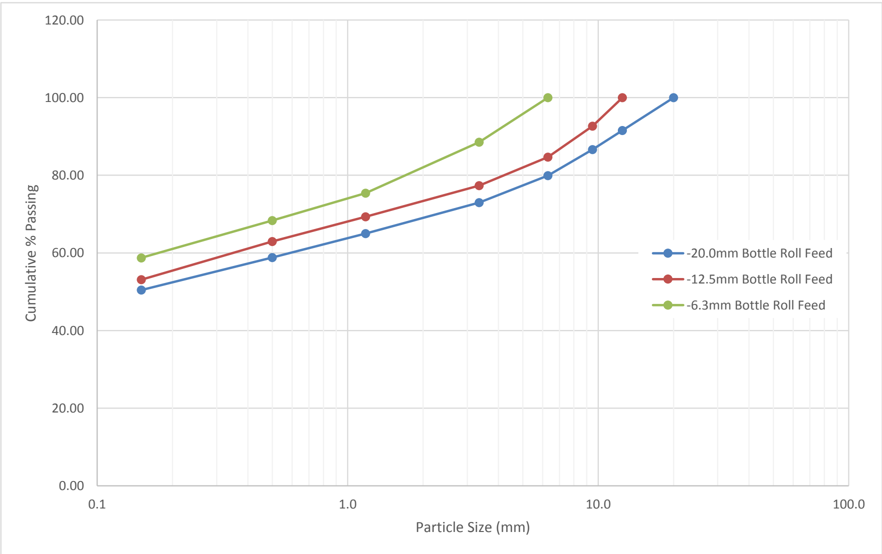
Size mm	Weight grams	% Retained Individual	% Retained Cumulative	Cumulative % Passing	Assay, Au (ppm)	Distribution, Au (%) Individual	Distribution, Au (%) Cumulative	Cumulative % Passing
20.0	0.0	0.00	0.00	100.00		0.00	0.00	100.00
12.5	161.5	8.46	8.46	91.54	1.56	5.22	5.22	94.78
9.50	93.3	4.89	13.35	86.65	0.99	1.92	7.14	92.86
6.30	128.6	6.74	20.09	79.91	5.19	13.84	20.98	79.02
3.35	133.0	6.97	27.05	72.95	3.01	8.30	29.28	70.72
1.18	151.8	7.95	35.01	64.99	4.21	13.25	42.54	57.46
0.50	117.7	6.17	41.17	58.83	1.53	3.73	46.27	53.73
0.15	160.3	8.40	49.57	50.43	0.67	2.23	48.50	51.50
-0.15	962.6	50.43	100.00	0.00	2.58	51.50	100.00	0.00
Total	**1908.8**	**100.00**	-	-	**2.53**		-	-
d80				**6.40**				**6.20**
d80(Check)				6.34				6.53

Fraction: -12.5mm Bottle Roll Feed

Size mm	Weight grams	% Retained Individual	% Retained Cumulative	Cumulative % Passing	Assay, Au (ppm)	Distribution, Au (%) Individual	Distribution, Au (%) Cumulative	Cumulative % Passing
12.5	0.00	0.00	0.00	100.00		0.00	0.00	100.00
9.5	144.40	7.34	7.34	92.66	1.00	1.98	1.98	98.02
6.30	156.90	7.98	15.33	84.67	17.66	37.90	39.87	60.13
3.35	144.10	7.33	22.66	77.34	15.48	30.51	70.38	29.62
1.18	158.0	8.04	30.69	69.31	1.79	3.87	74.25	25.75
0.50	125.3	6.37	37.07	62.93	2.72	4.66	78.91	21.09
0.15	193.3	9.83	46.90	53.10	1.71	4.52	83.44	16.56
-0.15	1044.0	53.10	100.00	0.00	1.16	16.56	100.00	0.00
Total	**1966.0**	**100.00**	-	-	**3.72**		-	-
d80				**4.21**				**9.49**
d80(Check)				4.42				7.98

Fraction: -6.3mm Bottle Roll Feed

Size mm	Weight grams	% Retained Individual	% Retained Cumulative	Cumulative % Passing	Assay, Au (ppm)	Distribution, Au (%) Individual	Distribution, Au (%) Cumulative	Cumulative % Passing
6.30	0.00	0.00	0.00	100.00		0.00	0.00	100.00
3.35	226.00	11.47	11.47	88.53	4.91	27.93	27.93	72.07
1.18	258.80	13.13	24.60	75.40	4.15	27.03	54.97	45.03
0.50	139.20	7.06	31.66	68.34	2.31	8.09	63.06	36.94
0.15	189.10	9.60	41.26	58.74	2.19	10.42	73.48	26.52
-0.15	1157.6	58.74	100.00	0.00	0.91	26.52	100.00	0.00
Total	**1970.7**	**100.00**	-	-	**2.02**		-	-
d80				**1.75**				**4.61**
d80(Check)				1.94				4.19



Wardell Armstrong International

Job: ZT640703
Client: Cora Gold
Project: Sanankoro
Sample: SD0006

Fraction: -20.0mm Bottle Roll Leach Residue

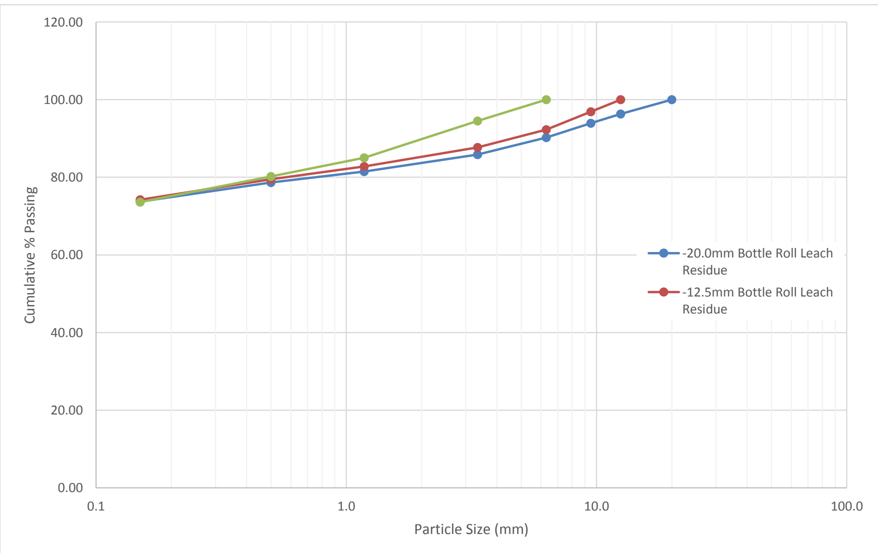
Size	Weight	% Retained		Cumulative %	Assay, Au	Distribution, Au (%)		Cumulative %
mm	grams	Individual	Cumulative	Passing	(ppm)	Individual	Cumulative	Passing
20.0	0.0	0.00	0.00	100.00		0.00	0.00	100.00
12.5	70.5	3.67	3.67	96.33	2.04	6.06	6.06	93.94
9.50	46.3	2.41	6.08	93.92	0.12	0.23	6.30	93.70
6.30	70.4	3.67	9.75	90.25	21.62	64.17	70.47	29.53
3.35	84.5	4.40	14.16	85.84	0.5	1.78	72.25	27.75
1.18	83.5	4.35	18.51	81.49	6.35	22.37	94.62	5.38
0.50	54.5	2.84	21.35	78.65	0.93	2.14	96.76	3.24
0.15	94.5	4.92	26.27	73.73	0.26	1.04	97.79	2.21
-0.15	1414.6	73.73	100.00	0.00	0.037	2.21	100.00	0.00
Total	**1918.7**	**100.00**	-	-	**1.24**		-	-
d80				**0.74**				**12.02**
d80(Check)				0.82				8.82

Fraction: -12.5mm Bottle Roll Leach Residue

Size	Weight	% Retained		Cumulative %	Assay, Au	Distribution, Au (%)		Cumulative %
mm	grams	Individual	Cumulative	Passing	(ppm)	Individual	Cumulative	Passing
12.5	0.00	0.00	0.00	100.00		0.00	0.00	100.00
9.5	59.38	3.08	3.08	96.92	0.96	3.35	3.35	96.65
6.30	89.44	4.64	7.72	92.28	3.43	18.01	21.35	78.65
3.35	88.14	4.57	12.29	87.71	12.8	66.22	87.58	12.42
1.18	94.4	4.90	17.19	82.81	0.87	4.82	92.40	7.60
0.50	64.1	3.32	20.52	79.48	0.98	3.69	96.08	3.92
0.15	101.6	5.27	25.79	74.21	0.15	0.89	96.98	3.02
-0.15	1430.3	74.21	100.00	0.00	0.036	3.02	100.00	0.00
Total	**1927.3**	**100.00**	-	-	**0.88**		-	-
d80				**0.57**				**7.45**
d80(Check)				0.61				6.54

Fraction: -6.3mm Bottle Roll Leach Residue

Size	Weight	% Retained		Cumulative %	Assay, Au	Distribution, Au (%)		Cumulative %
mm	grams	Individual	Cumulative	Passing	(ppm)	Individual	Cumulative	Passing
6.30	0.00	0.00	0.00	100.00		0.00	0.00	100.00
3.35	107.12	5.50	5.50	94.50	3.33	46.87	46.87	53.13
1.18	184.58	9.49	14.99	85.01	1.59	38.56	85.43	14.57
0.50	93.24	4.79	19.78	80.22	0.54	6.62	92.04	7.96
0.15	128.79	6.62	26.40	73.60	0.17	2.88	94.92	5.08
-0.15	1432.2	73.60	100.00	0.00	0.027	5.08	100.00	0.00
Total	**1945.9**	**100.00**	-	-	**0.39**		-	-
d80				**0.48**				**5.78**
d80(Check)				#DIV/0!				5.04

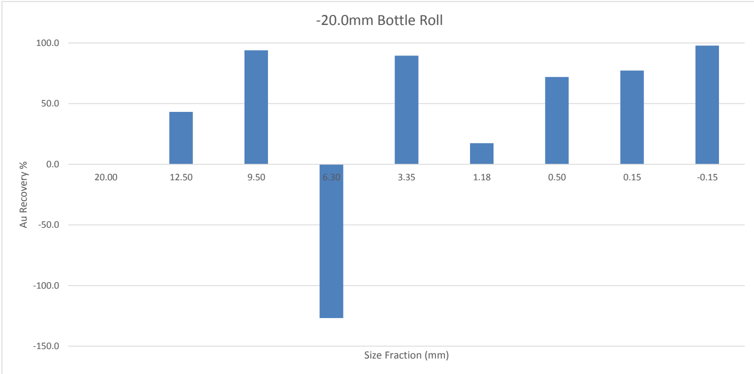


Wardell Armstrong International

Job: ZT640703
Client: Cora Gold
Project: Sanankoro
Sample: SD0006

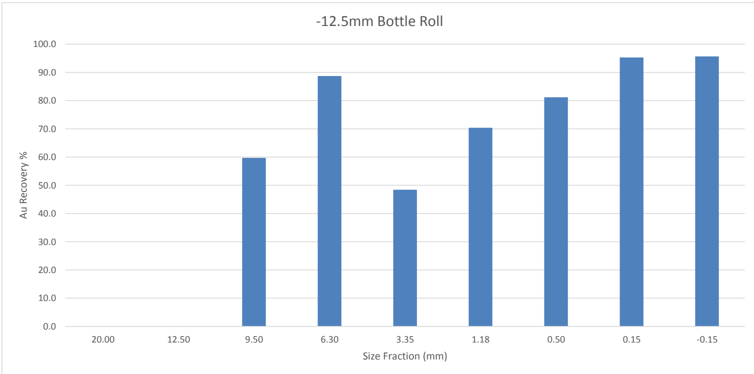
Fraction: -20mm Bottle Roll

Size (mm)	Bottle Roll Feed			Bottle Roll Residue			Au Recovery
	Mass (g)	Mass (%)	Au (ppm)	Mass (g)	Mass (%)	Au (ppm)	(%)
20.00							
12.50	161.5	8.5	1.56	70.5	3.7	2.04	43.23
9.50	93.3	4.9	0.99	46.3	2.4	0.12	94.02
6.30	128.6	6.7	5.19	70.4	3.7	21.62	-126.78
3.35	133.0	7.0	3.01	84.5	4.4	0.5	89.50
1.18	151.8	8.0	4.21	83.5	4.4	6.35	17.44
0.50	117.7	6.2	1.53	54.5	2.8	0.93	72.01
0.15	160.3	8.4	0.67	94.5	4.9	0.26	77.25
-0.15	962.6	50.4	2.58	1414.6	73.7	0.037	97.90
Total	1908.8	100.0	2.53	1918.7	100.0	1.24	81.94



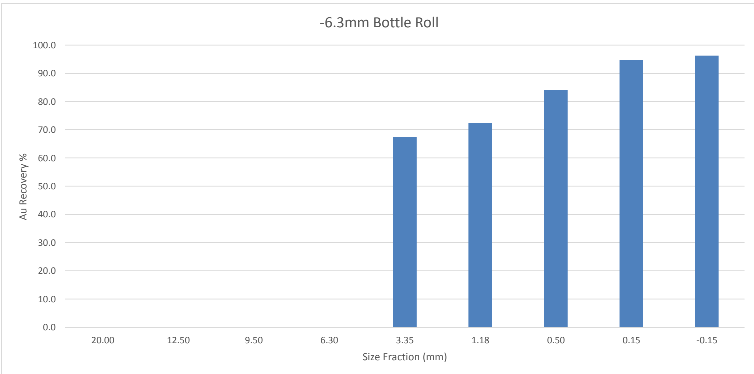
Fraction: -12.5mm Bottle Roll

Size (mm)	Bottle Roll Feed			Bottle Roll Residue			Au Recovery
	Mass (g)	Mass (%)	Au (ppm)	Mass (g)	Mass (%)	Au (ppm)	(%)
20.00							
12.50							
9.50	144.4	7.3	1.00	59.4	3.1	0.96	59.73
6.30	156.9	8.0	17.66	89.4	4.6	3.43	88.71
3.35	144.1	7.3	15.48	88.1	4.6	12.8	48.41
1.18	158.0	8.0	1.79	94.4	4.9	0.87	70.38
0.50	125.3	6.4	2.72	64.1	3.3	0.98	81.20
0.15	193.3	9.8	1.71	101.6	5.3	0.15	95.30
-0.15	1044.0	53.1	1.16	1430.3	74.2	0.036	95.66
Total	1966.0	100.0	3.72	1927.3	100.0	0.88	90.33



Fraction: -6.3mm Bottle Roll

Size (mm)	Bottle Roll Feed			Bottle Roll Residue			Au Recovery
	Mass (g)	Mass (%)	Au (ppm)	Mass (g)	Mass (%)	Au (ppm)	(%)
20.00							
12.50							
9.50							
6.30							
3.35	226.0	11.5	4.91	107.1	5.5	3.33	67.44
1.18	258.8	13.1	4.15	184.6	9.5	1.59	72.33
0.50	139.2	7.1	2.31	93.2	4.8	0.54	84.14
0.15	189.1	9.6	2.19	128.8	6.6	0.17	94.65
-0.15	1157.6	58.7	0.91	1432.2	73.6	0.027	96.28
	1970.7	100.0	2.02	1945.9	100.0	0.39	91.73



APPENDIX 5: Whole Ore Leach Test Data (Phase 1)

Kinetic Cyanide Leach Test

Client: Cora Gold
Project: Sanankoro
Sample: SD0005
Date: 12 February 2019
Grind: 80% passing 75 µm
Test: LT1

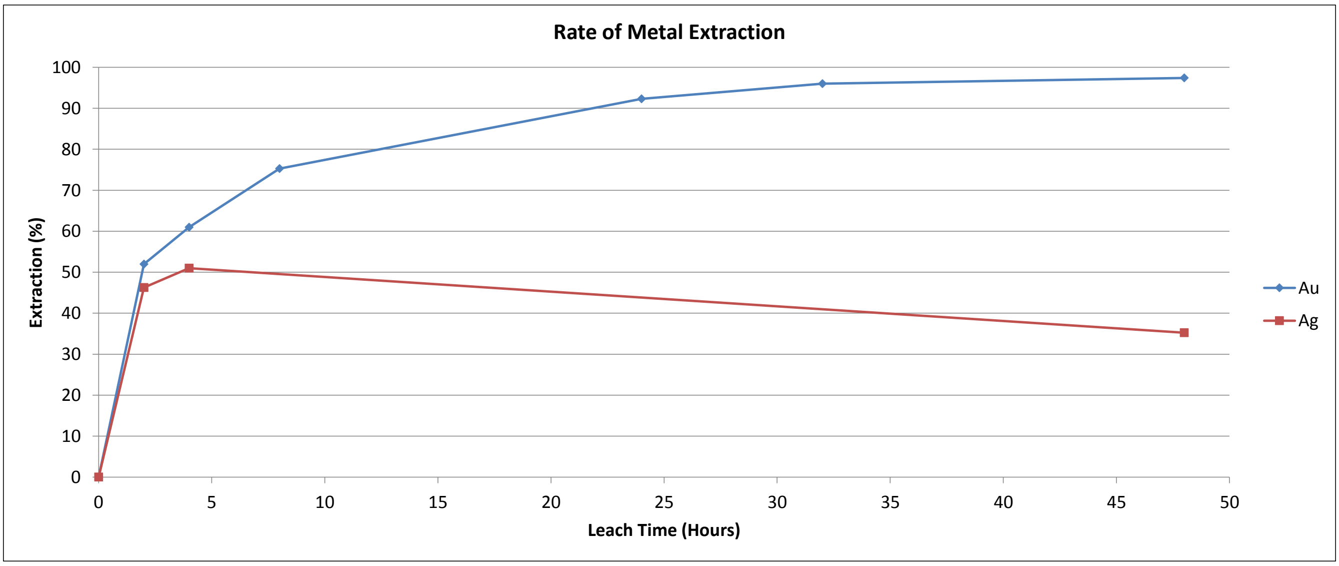
Time (Hours)	Additions				Solution				Sample				Metal Extracted			
	Solids (g)	Solution (ml)	NaCN (g)	Lime (g)	pH	NaCN (%)	Au (ppm)	Ag (ppm)	Vol (ml)	Titre	∑ Au (µg)	∑ Ag (µg)	Au (µg)	Au (%)	Ag (µg)	Ag (%)
	2000.0	3000.0			4.91											
0		3000.0	6.00	1.44	10.35	0.200										
2		2989.8	0.23		10.76	0.193	0.32	<0.23	30.0	3.85	10	7	957	52.0	688	46.3
4		3007.8			10.81	0.195	0.37	0.09	30.0	3.90	21	10	1122	61.0	278	18.7
8		2967.8	0.30		10.85	0.190	0.46	0.10	40.0	3.80	39	14	1386	75.3	306	20.6
24		2861.8	0.30		11.04	0.190	0.58	0.26	40.0	3.80	62	24	1699	92.3	758	51.0
32		2939.8	0.75		10.95	0.175	0.58	0.17	40.0	3.50	86	31	1767	96.0	524	35.2
48		2845.8			11.09	0.163	0.60	0.16	60.0	3.25	122	40	1793	97.4	486	32.7
Total																

Extraction Calculation

Product	Quantity	Gold			Silver		
		Assay (ppm)	Mass (µg)	Dist'n (%)	Assay (ppm)	Mass (µg)	Dist'n (%)
Solids (g)	2000.0	0.02	48	2.6	<0.50	1000	67.3
Solution (ml)	2845.8	0.60	1707	92.7	0.16	455	30.6
Solution Samples			86	4.6		31	2.1
Total Extraction			1793	97.4		486	32.7
Total			1841	100.0		1486	100.0
Calculated Head		0.92			0.74		
Assay Head							

Leach Conditions

NaCN Concentration	2.00	g/l
NaCN Addition	3.79	kg/t
NaCN Consumption	1.35	kg/t
Lime Consumption	0.72	kg/t
% Solids	40.0	
Grind Size d_{80}	75	µm
Aeration Gas	Air	
Aeration Rate	3.0	l/min



Kinetic Cyanide Leach Test

Client: Cora Gold
Project: Sanankoro
Sample: SD0006
Date: 12 February 2019
Grind: 80% passing 75 µm
Test: LT2

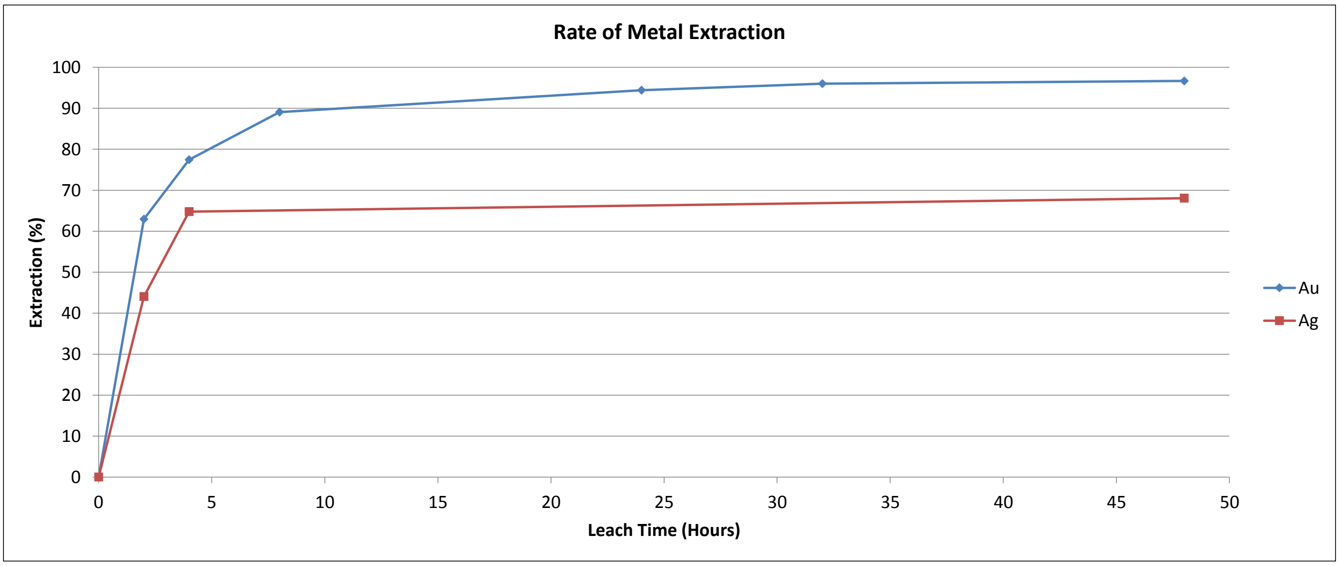
Time (Hours)	Additions				Solution				Sample				Metal Extracted			
	Solids (g)	Solution (ml)	NaCN (g)	Lime (g)	pH	NaCN (%)	Au (ppm)	Ag (ppm)	Vol (ml)	Titre	∑ Au (µg)	∑ Ag (µg)	Au (µg)	Au (%)	Ag (µg)	Ag (%)
	2000.0	3000.0			7.96											
0		3000.0	6.00	1.83	10.31	0.200										
2		2996.3	0.45	1.14	10.19	0.185	1.17	0.45	30.0	3.70	35	14	3506	62.9	1348	44.1
4		2992.3			10.81	0.200	1.43	0.49	30.0	4.00	78	28	4314	77.4	1480	48.4
8		2978.3	0.45		10.82	0.185	1.64	0.51	40.0	3.70	144	49	4962	89.1	1547	50.6
24		2842.3	0.60		11.13	0.180	1.80	0.68	40.0	3.60	216	76	5260	94.4	1981	64.8
32		2950.3			11.02	0.200	1.74	0.68	40.0	4.00	285	103	5349	96.0	2082	68.1
48		2898.3			11.09	0.150	1.76	0.68	60.0	3.00	391	144	5386	96.7	2059	67.3
Total																

Extraction Calculation

Product	Quantity	Gold			Silver		
		Assay (ppm)	Mass (µg)	Dist'n (%)	Assay (ppm)	Mass (µg)	Dist'n (%)
Solids (g)	2000.0	0.09	185	3.3	<0.50	1000	32.7
Solution (ml)	2898.3	1.76	5101	91.6	0.68	1956	63.9
Solution Samples			285	5.1		103	3.4
Total Extraction			5386	96.7		2059	67.3
Total			5571	100.0		3059	100.0
Calculated Head		2.79			1.53		
Assay Head							

Leach Conditions

NaCN Concentration	2.00	g/l
NaCN Addition	3.75	kg/t
NaCN Consumption	1.50	kg/t
Lime Consumption	1.49	kg/t
% Solids	40.0	
Grind Size d_{80}	75	µm
Aeration Gas	Air	
Aeration Rate	3.0	l/min



APPENDIX 6: Grind Calibration Data (Phase 2)

Wardell Armstrong International Ltd

Job:	ZT640703	**Mill:**	2
Client:	Cora Gold	**Rods:**	15
Project:	Sanankoro	**% Solids:**	50
Sample:			

Grind Time Minutes

Size µm	Weight grams	% Retained Individual	% Retained Cumulative	Cumulative % Passing
1400	11.9	2.3	2.3	97.7
1000	22.4	4.4	6.7	93.3
710	20.8	4.1	10.7	89.3
500	23.7	4.6	15.3	84.7
300	31.4	6.1	21.5	78.5
212	17.5	3.4	24.9	75.1
150	16.1	3.1	28.0	72.0
106	16.7	3.2	31.3	68.7
75	13.4	2.6	33.9	66.1
53	18.0	3.5	37.4	62.6
38	15.4	3.0	40.4	59.6
-38	305.9	59.6	100.0	0.0
Total	**513.2**	**100.0**	-	-
d80	**343**			
d80$_{(Check)}$	348			

Grind Time 2 Minutes

Size µm	Weight grams	% Retained Individual	% Retained Cumulative	Cumulative % Passing
500	1.7	0.6	0.6	99.4
300	18.9	6.3	6.8	93.2
212	21.8	7.2	14.1	85.9
150	17.8	5.9	20.0	80.0
106	13.8	4.6	24.6	75.4
75	9.5	3.1	27.7	72.3
53	13.0	4.3	32.0	68.0
38	8.8	2.9	34.9	65.1
-38	196.0	65.1	100.0	0.0
Total	**301.3**	**100.0**	-	-
d80	**151**			
d80$_{(Check)}$	#DIV/0!			

Grind Time 4 Minutes

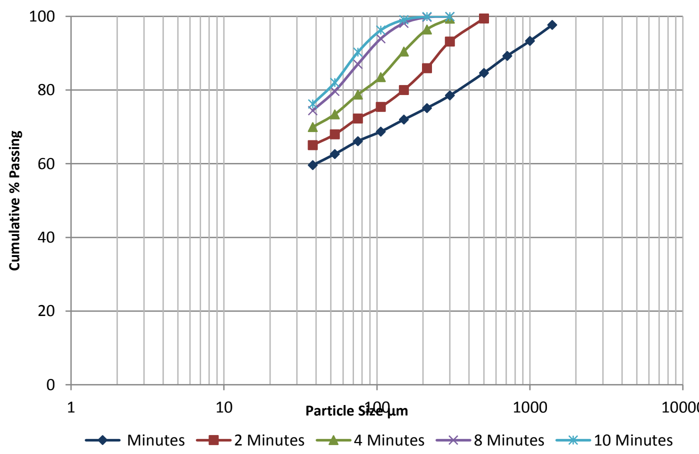
Size µm	Weight grams	% Retained Individual	% Retained Cumulative	Cumulative % Passing
300	1.8	0.6	0.6	99.4
212	8.7	2.9	3.5	96.5
150	17.6	6.0	9.5	90.5
106	20.5	7.0	16.5	83.5
75	14.1	4.8	21.3	78.8
53	15.7	5.3	26.6	73.4
38	10.3	3.5	30.0	70.0
-38	206.5	70.0	100.0	0.0
Total	**295.2**	**100.0**	-	-
d80	**84**			
d80$_{(Check)}$	83			

Grind Time 8 Minutes

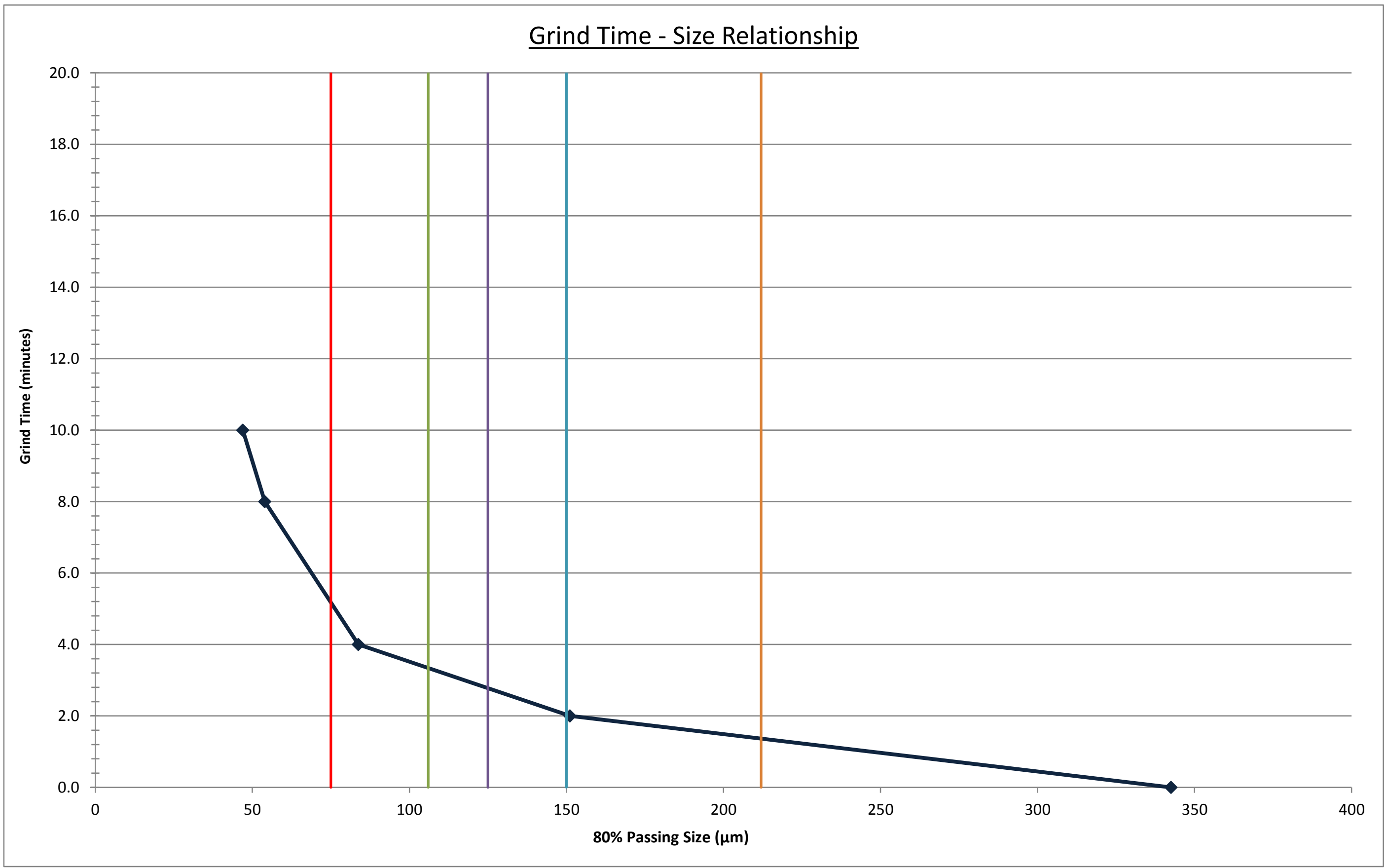
Size µm	Weight grams	% Retained Individual	% Retained Cumulative	Cumulative % Passing
300	0.1	0.0	0.0	100.0
212	0.6	0.2	0.2	99.8
150	4.7	1.6	1.8	98.2
106	12.3	4.2	6.0	94.0
75	20.3	7.0	13.0	87.0
53	21.4	7.3	20.3	79.7
38	15.3	5.2	25.6	74.4
-38	217.6	74.4	100.0	0.0
Total	**292.4**	**100.0**	-	-
d80	**54**			
d80$_{(Check)}$	54			

Grind Time 10 Minutes

Size µm	Weight grams	% Retained Individual	% Retained Cumulative	Cumulative % Passing
300	0.1	0.0	0.0	100.0
212	0.1	0.0	0.1	99.9
150	2.2	0.8	0.9	99.1
106	7.3	2.8	3.7	96.3
75	15.5	6.0	9.7	90.3
53	21.3	8.2	18.0	82.0
38	15.0	5.8	23.8	76.2
-38	197.3	76.2	100.0	0.0
Total	**258.8**	**100.0**	-	-
d80	**47**			
d80$_{(Check)}$	48			



Job: ZT640703
Client: Cora Gold
Project: Sanankoro
Sample:



Target d80	Grind Time
75µm	5 Minutes 11 Seconds
106µm	3 Minutes 20 Seconds
125µm	2 Minutes 47 Seconds
150µm	2 Minutes 2 Seconds
212µm	1 Minute 23 Seconds

APPENDIX 7: Gravity Tailings Leach Test Data

Wardell Armstrong International Ltd

Kinetic Cyanide Leach Test

Client:	Cora Gold
Project:	Sanankoro
Sample:	Master Composite - Gravity Tailings
Date:	09 April 2019
Grind:	80% passing 75 µm
Test:	KLT1

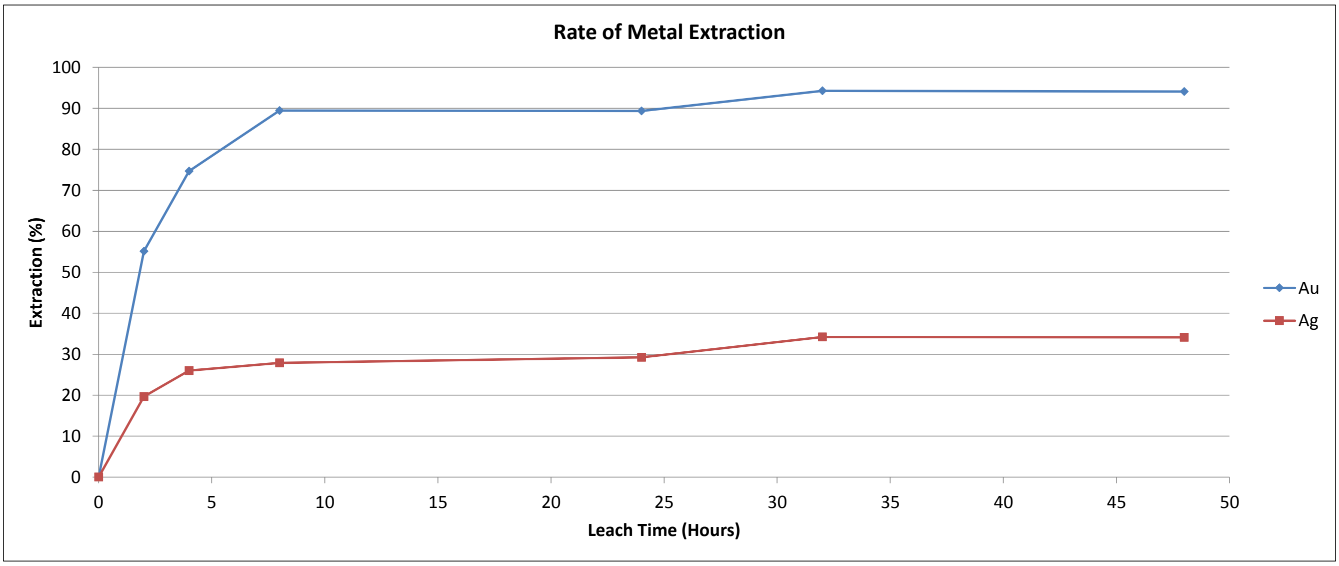
Time (Hours)	Additions				Solution				Sample				Metal Extracted			
	Solids (g)	Solution (ml)	NaCN (g)	Lime (g)	pH	NaCN (%)	Au (ppm)	Ag (ppm)	Vol (ml)	Titre	∑ Au (µg)	∑ Ag (µg)	Au (µg)	Au (%)	Ag (µg)	Ag (%)
	2000.0	3000.0			8.20											
0		3000.0	3.00	0.97	10.52	0.100										
2		2985.8	0.53		10.51	0.083	0.33	0.10	30.0	1.65	10	3	985	55.1	299	19.7
4		3011.8			10.57	0.100	0.44	0.13	30.0	2.00	23	7	1335	74.7	395	26.0
8		2973.8			10.73	0.100	0.53	0.14	40.0	2.00	44	13	1599	89.5	423	27.9
24		2875.8	0.90		10.97	0.070	0.54	0.15	40.0	1.40	66	19	1597	89.3	444	29.2
32		2943.8	0.53		10.97	0.083	0.55	0.17	40.0	1.65	88	25	1685	94.3	519	34.2
48		2897.8			11.02	0.085	0.55	0.17	60.0	1.70	121	36	1682	94.1	518	34.1
Total																

Extraction Calculation

Product	Quantity	Gold			Silver		
		Assay (ppm)	Mass (µg)	Dist'n (%)	Assay (ppm)	Mass (µg)	Dist'n (%)
Solids (g)	2000.0	0.05	106	5.9	<0.50	1000	65.9
Solution (ml)	2897.8	0.55	1594	89.2	0.17	493	32.5
Solution Samples			88	4.9		25	1.7
Total Extraction			1682	94.1		518	34.1
Total			1788	100.0		1518	100.0
Calculated Head		0.89			0.76		
Assay Head		0.74					

Leach Conditions

NaCN Concentration	1.00	g/l
NaCN Addition	2.48	kg/t
NaCN Consumption	1.21	kg/t
Lime Consumption	0.49	kg/t
% Solids	40.0	
Grind Size d_{80}	75	µm
Aeration Gas	Air	
Aeration Rate	3.0	l/min



Kinetic Cyanide Leach Test

Client: Cora Gold
Project: Sanankoro
Sample: Master Composite - Gravity Tailings
Date: 09 April 2019
Grind: 80% passing 75 µm
Test: KLT2

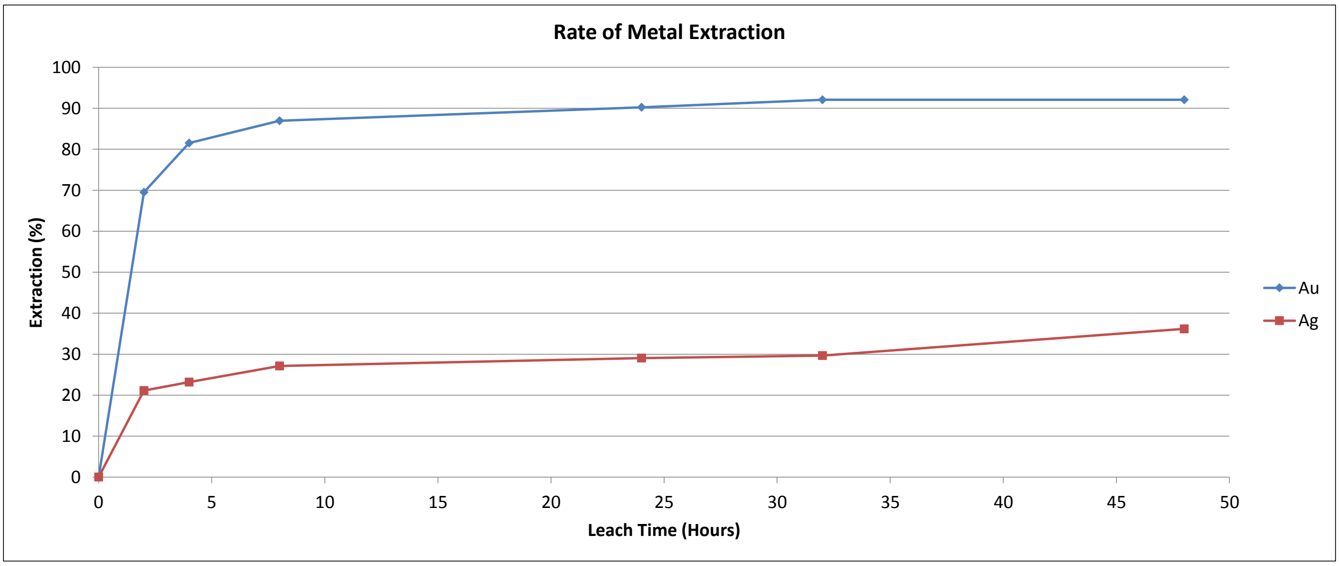
Time (Hours)	Additions				Solution				Sample				Metal Extracted			
	Solids (g)	Solution (ml)	NaCN (g)	Lime (g)	pH	NaCN (%)	Au (ppm)	Ag (ppm)	Vol (ml)	Titre	∑ Au (µg)	∑ Ag (µg)	Au (µg)	Au (%)	Ag (µg)	Ag (%)
	2000.0	3000.0			8.13											
0		3000.0	1.50	1.25	10.50	0.050										
2		3010.3	0.23		10.53	0.043	0.42	0.11	30.0	0.85	13	3	1264	69.5	331	21.1
4		3000.3			10.53	0.048	0.49	0.12	30.0	0.95	27	7	1483	81.5	363	23.2
8		2988.3			10.48	0.050	0.52	0.14	40.0	1.00	48	13	1581	86.9	425	27.1
24		2950.3	0.38		10.48	0.038	0.54	0.15	40.0	0.75	70	19	1641	90.2	455	29.0
32		2972.3	0.38	0.49	10.42	0.038	0.54	0.15	40.0	0.75	91	25	1675	92.1	464	29.6
48		2932.3			10.84	0.048	0.54	0.19	60.0	0.95	124	36	1675	92.1	567	36.2
Total																

Extraction Calculation

Product	Quantity	Gold			Silver		
		Assay (ppm)	Mass (µg)	Dist'n (%)	Assay (ppm)	Mass (µg)	Dist'n (%)
Solids (g)	2000.0	0.07	144	7.9	<0.50	1000	63.8
Solution (ml)	2932.3	0.54	1583	87.1	0.19	542	34.6
Solution Samples			91	5.0		25	1.6
Total Extraction			1675	92.1		567	36.2
Total			1819	100.0		1567	100.0
Calculated Head		0.91			0.78		
Assay Head		0.74					

Leach Conditions

NaCN Concentration	0.50	g/l
NaCN Addition	1.25	kg/t
NaCN Consumption	0.53	kg/t
Lime Consumption	0.87	kg/t
% Solids	40.0	
Grind Size d_{80}	75	µm
Aeration Gas	Air	
Aeration Rate	3.0	l/min



APPENDIX 8: Whole Ore Leach Test Data (Phase 2)

Wardell Armstrong International Ltd

Kinetic Cyanide Leach Test

Client:	Cora Gold
Project:	Sanankoro
Sample:	Master Composite
Date:	19 March 2019
Grind:	80% passing 150 µm
Test:	LT150

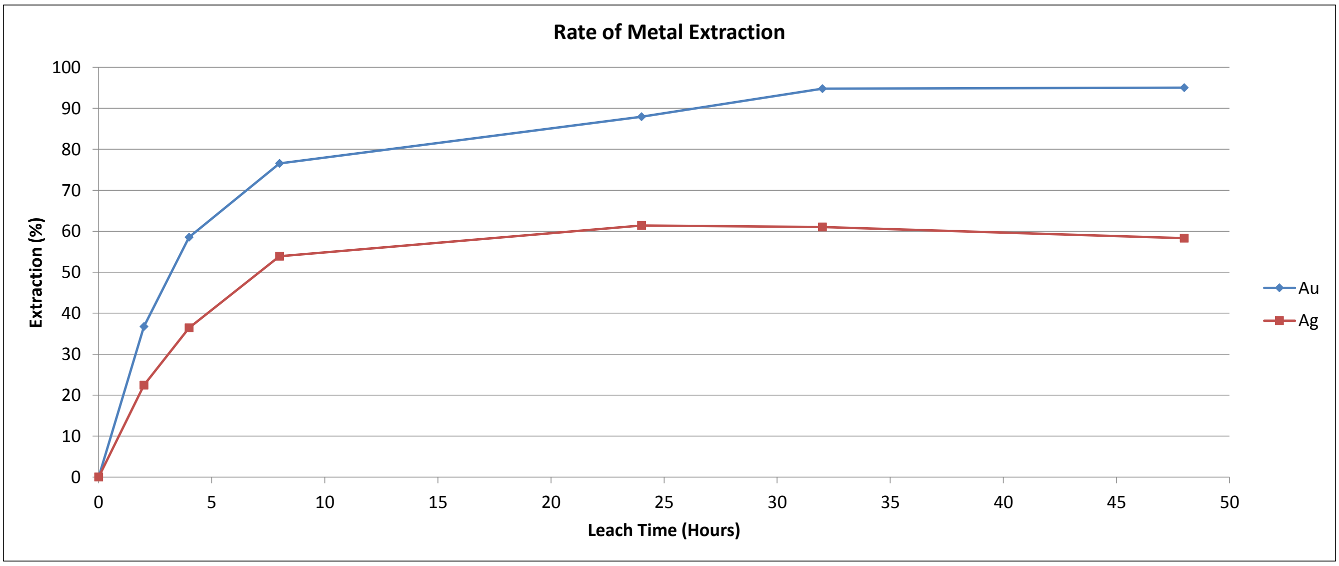
Time	Additions				Solution				Sample				Metal Extracted			
(Hours)	Solids (g)	Solution (ml)	NaCN (g)	Lime (g)	pH	NaCN (%)	Au (ppm)	Ag (ppm)	Vol (ml)	Titre	∑ Au (µg)	∑ Ag (µg)	Au (µg)	Au (%)	Ag (µg)	Ag (%)
	2000.0	3000.0			7.41											
0		3000.0	3.00	1.76	10.39	0.100										
2		2991.3	0.23		10.44	0.093	0.84	0.18	30.0	1.85	25	5	2513	36.7	538	22.4
4		2993.3	0.30		10.49	0.090	1.33	0.29	30.0	1.80	65	14	4006	58.6	873	36.4
8		2973.3	0.23		10.60	0.093	1.74	0.43	30.0	1.85	117	27	5239	76.6	1293	53.9
24		2891.3	0.60		10.83	0.080	2.04	0.50	40.0	1.60	199	47	6016	87.9	1473	61.4
32		2951.3	0.30		10.88	0.090	2.13	0.48	40.0	1.80	284	66	6485	94.8	1464	61.0
48		2865.3			10.95	0.075	2.17	0.47	60.0	1.50	414	94	6502	95.0	1399	58.3
Total																

Extraction Calculation

Product	Quantity	Gold			Silver		
		Assay (ppm)	Mass (µg)	Dist'n (%)	Assay (ppm)	Mass (µg)	Dist'n (%)
Solids (g)	2000.0	0.17	340	5.0	<0.50	1000	41.7
Solution (ml)	2865.3	2.17	6218	90.9	0.47	1332	55.5
Solution Samples			284	4.2		66	2.8
Total Extraction			6502	95.0		1399	58.3
Total			6842	100.0		2399	100.0
Calculated Head		3.42			1.20		
Assay Head		2.74					

Leach Conditions

NaCN Concentration	1.00	g/l
NaCN Addition	2.33	kg/t
NaCN Consumption	1.21	kg/t
Lime Consumption	0.88	kg/t
% Solids	40.0	
Grind Size d_{80}	150	µm
Aeration Gas	Air	
Aeration Rate	3.0	l/min



Kinetic Cyanide Leach Test

Client: Cora Gold
Project: Sanankoro
Sample: Master Composite
Date: 19 March 2019
Grind: 80% passing 125 µm
Test: LT125

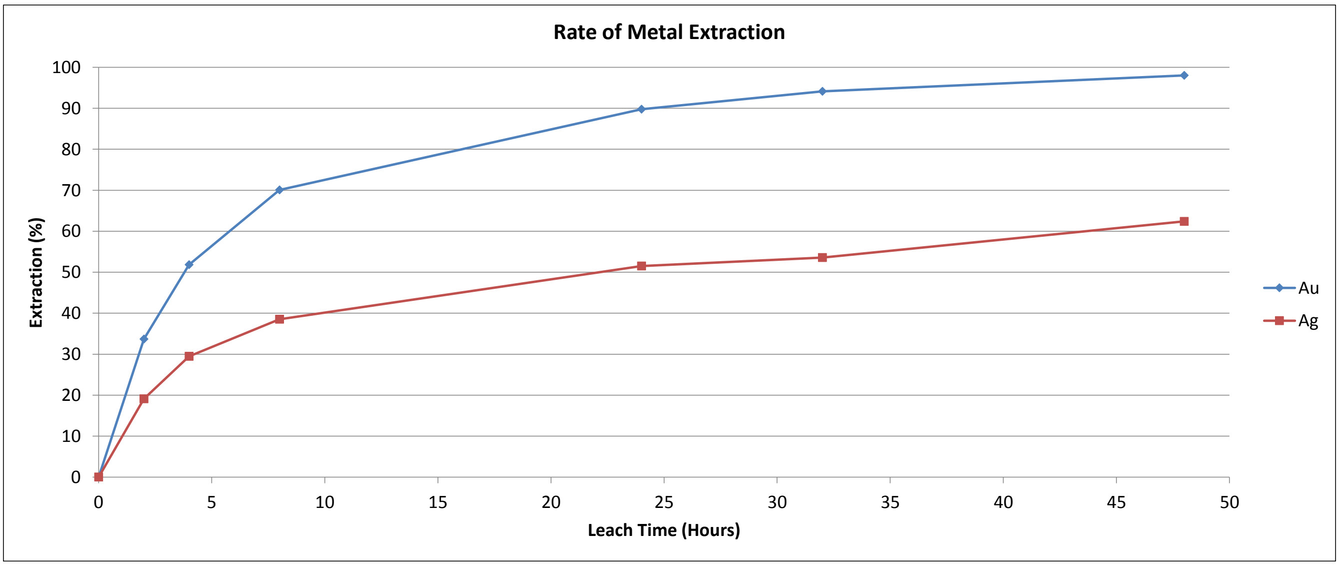
Time (Hours)	Additions				Solution				Sample				Metal Extracted			
	Solids (g)	Solution (ml)	NaCN (g)	Lime (g)	pH	NaCN (%)	Au (ppm)	Ag (ppm)	Vol (ml)	Titre	∑ Au (µg)	∑ Ag (µg)	Au (µg)	Au (%)	Ag (µg)	Ag (%)
	2000.0	3000.0			7.56											
0		3000.0	3.00	2.20	10.38	0.100										
2		2990.9	0.15		10.79	0.095	0.74	0.17	30.0	1.90	22	5	2213	33.7	508	19.1
4		2996.9			10.76	0.098	1.13	0.26	30.0	1.95	56	13	3409	51.8	784	29.5
8		2974.9	0.38		10.76	0.088	1.53	0.34	30.0	1.75	102	23	4608	70.1	1024	38.5
24		2928.9	0.60		10.78	0.080	1.98	0.46	40.0	1.60	181	42	5901	89.8	1370	51.5
32		2944.9	0.30		10.91	0.090	2.04	0.47	40.0	1.80	263	60	6189	94.1	1426	53.6
48		2908.9			10.90	0.075	2.13	0.55	60.0	1.50	390	93	6444	98.0	1660	62.4
Total																

Extraction Calculation

Product	Quantity	Gold			Silver		
		Assay (ppm)	Mass (µg)	Dist'n (%)	Assay (ppm)	Mass (µg)	Dist'n (%)
Solids (g)	2000.0	0.07	130	2.0	<0.50	1000	37.6
Solution (ml)	2908.9	2.13	6181	94.0	0.55	1600	60.1
Solution Samples			263	4.0		60	2.3
Total Extraction			6444	98.0		1660	62.4
Total			6574	100.0		2660	100.0
Calculated Head		3.29			1.33		
Assay Head		2.74					

Leach Conditions

NaCN Concentration	1.00	g/l
NaCN Addition	2.22	kg/t
NaCN Consumption	1.09	kg/t
Lime Consumption	1.10	kg/t
% Solids	40.0	
Grind Size d_{80}	125	µm
Aeration Gas	Air	
Aeration Rate	3.0	l/min



Kinetic Cyanide Leach Test

Client: Cora Gold
Project: Sanankoro
Sample: Master Composite
Date: 19 March 2019
Grind: 80% passing 106 µm
Test: LT106

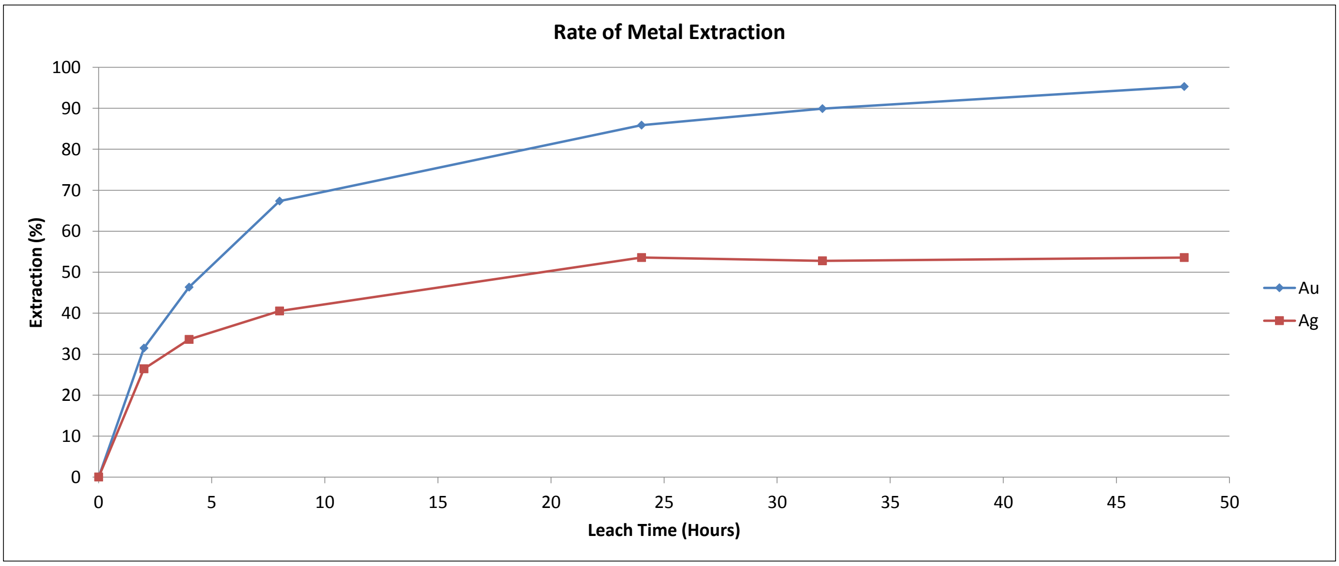
Time (Hours)	Additions				Solution				Sample				Metal Extracted			
	Solids (g)	Solution (ml)	NaCN (g)	Lime (g)	pH	NaCN (%)	Au (ppm)	Ag (ppm)	Vol (ml)	Titre	∑ Au (µg)	∑ Ag (µg)	Au (µg)	Au (%)	Ag (µg)	Ag (%)
	2000.0	3000.0			7.51											
0		3000.0	3.00	2.04	10.36	0.100										
2		2991.7	0.23		10.65	0.093	0.67	0.19	30.0	1.85	20	6	2004	31.5	568	26.4
4		2989.7			10.72	0.100	0.98	0.24	30.0	2.00	50	13	2950	46.4	723	33.6
8		2963.7	0.30		10.74	0.090	1.43	0.29	40.0	1.80	107	25	4288	67.4	872	40.5
24		2895.7	0.68		10.89	0.078	1.85	0.39	40.0	1.55	181	40	5464	85.9	1154	53.6
32		2963.7	0.23		10.86	0.093	1.87	0.37	40.0	1.85	256	55	5723	89.9	1137	52.8
48		2889.7			10.94	0.070	2.01	0.38	60.0	1.40	376	78	6064	95.3	1153	53.6
Total																

Extraction Calculation

Product	Quantity	Gold			Silver		
		Assay (ppm)	Mass (µg)	Dist'n (%)	Assay (ppm)	Mass (µg)	Dist'n (%)
Solids (g)	2000.0	0.15	300	4.7	<0.50	1000	46.4
Solution (ml)	2889.7	2.01	5808	91.3	0.38	1098	51.0
Solution Samples			256	4.0		55	2.5
Total Extraction			6064	95.3		1153	53.6
Total			6364	100.0		2153	100.0
Calculated Head		3.18			1.08		
Assay Head		2.74					

Leach Conditions

NaCN Concentration	1.00	g/l
NaCN Addition	2.22	kg/t
NaCN Consumption	1.17	kg/t
Lime Consumption	1.02	kg/t
% Solids	40.0	
Grind Size d_{80}	106	µm
Aeration Gas	Air	
Aeration Rate	3.0	l/min



Wardell Armstrong International Ltd

Kinetic Cyanide Leach Test

Client:	Cora Gold
Project:	Sanankoro
Sample:	Master Composite
Date:	19 March 2019
Grind:	80% passing 75 µm
Test:	LT75

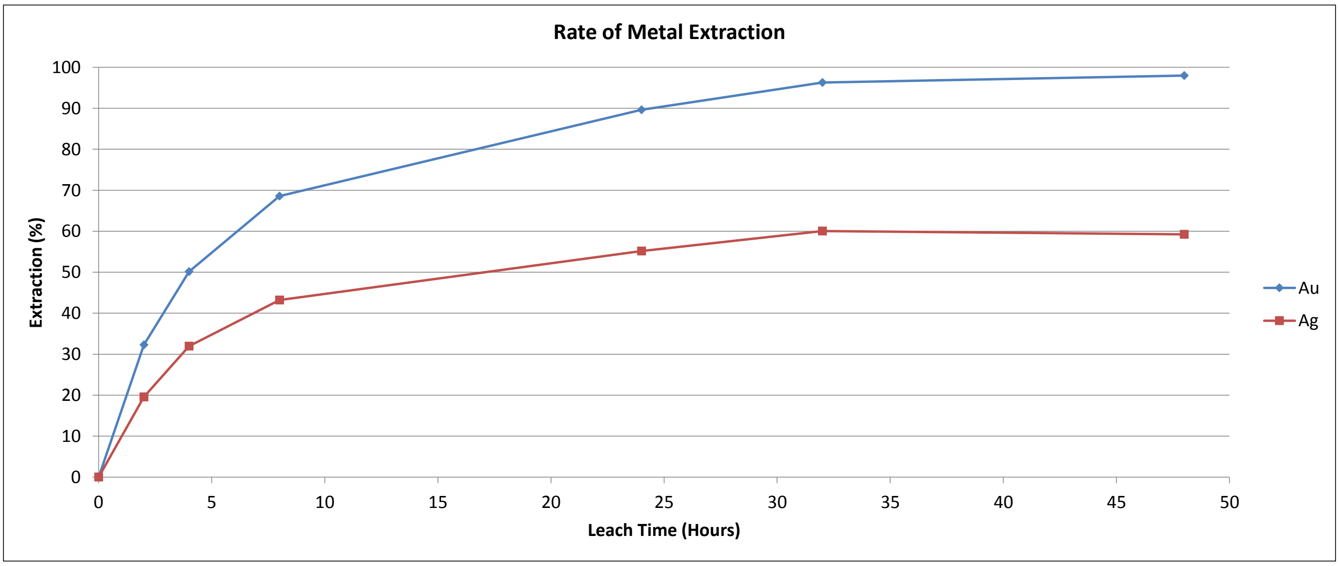
Time	Additions				Solution				Sample				Metal Extracted			
(Hours)	Solids (g)	Solution (ml)	NaCN (g)	Lime (g)	pH	NaCN (%)	Au (ppm)	Ag (ppm)	Vol (ml)	Titre	∑ Au (µg)	∑ Ag (µg)	Au (µg)	Au (%)	Ag (µg)	Ag (%)
	2000.0	3000.0			7.64											
0		3000.0	3.00	1.99	10.40	0.100										
2		3002.1	0.23		10.55	0.093	0.75	0.16	30.0	1.85	23	5	2252	32.3	480	19.6
4		2996.1	0.23		10.54	0.093	1.16	0.26	30.0	1.85	57	13	3498	50.1	784	32.0
8		2992.1	0.15		10.52	0.095	1.58	0.35	30.0	1.90	105	23	4785	68.6	1060	43.2
24		2956.1	0.45		10.62	0.085	2.08	0.45	40.0	1.70	188	41	6253	89.6	1353	55.2
32		2982.1	0.15		10.64	0.095	2.19	0.48	40.0	1.90	276	60	6719	96.3	1473	60.0
48		2962.1			10.64	0.075	2.22	0.47	60.0	1.50	408	89	6837	98.0	1452	59.2
Total																

Extraction Calculation

Product	Quantity	Gold			Silver		
		Assay (ppm)	Mass (µg)	Dist'n (%)	Assay (ppm)	Mass (µg)	Dist'n (%)
Solids (g)	2000.0	0.07	141	2.0	<0.50	1000	40.8
Solution (ml)	2962.1	2.22	6561	94.0	0.47	1392	56.8
Solution Samples			276	3.9		60	2.5
Total Extraction			6837	98.0		1452	59.2
Total			6978	100.0		2452	100.0
Calculated Head		3.49			1.23		
Assay Head		2.74					

Leach Conditions

NaCN Concentration	1.00	g/l
NaCN Addition	2.11	kg/t
NaCN Consumption	0.98	kg/t
Lime Consumption	1.00	kg/t
% Solids	40.0	
Grind Size d_{80}	75	µm
Aeration Gas	Air	
Aeration Rate	3.0	l/min



Kinetic Cyanide Leach Test

Client: Cora Gold
Project: Sanankoro
Sample: Master Composite
Date: 09 April 2019
Grind: 80% passing 75 µm
Test: LT0.5

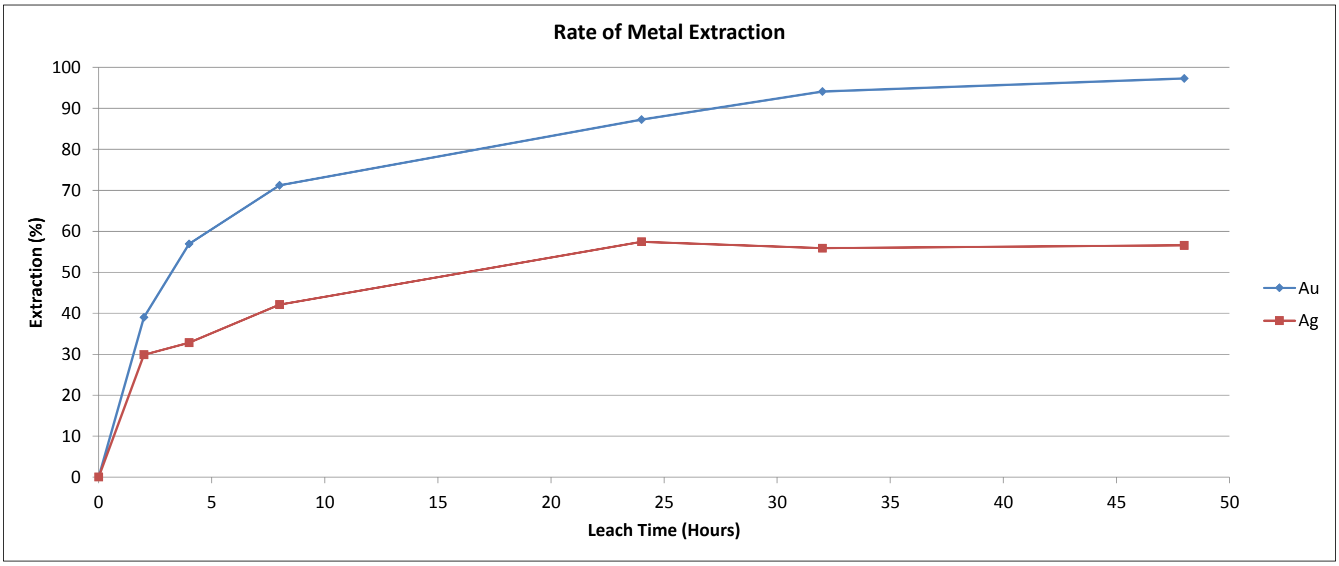
Time (Hours)	Additions				Solution				Sample				Metal Extracted			
	Solids (g)	Solution (ml)	NaCN (g)	Lime (g)	pH	NaCN (%)	Au (ppm)	Ag (ppm)	Vol (ml)	Titre	∑ Au (µg)	∑ Ag (µg)	Au (µg)	Au (%)	Ag (µg)	Ag (%)
	2000.0	3000.0			7.58											
0		3000.0	1.50	2.43	10.53	0.050										
2		2988.1			10.96	0.050	0.92	0.23	30.0	1.00	28	7	2749	39.0	687	29.8
4		2994.1			10.91	0.050	1.33	0.25	30.0	1.00	68	14	4010	56.9	755	32.8
8		2982.1			10.81	0.050	1.66	0.32	40.0	1.00	134	27	5018	71.2	969	42.1
24		2878.1	0.30		10.73	0.040	2.09	0.45	40.0	0.80	218	45	6149	87.2	1322	57.4
32		2956.1	0.38		10.56	0.038	2.17	0.42	40.0	0.75	304	62	6632	94.1	1287	55.9
48		2886.1			10.67	0.040	2.27	0.43	60.0	0.80	441	88	6856	97.3	1303	56.6
Total																

Extraction Calculation

Product	Quantity	Gold			Silver		
		Assay (ppm)	Mass (µg)	Dist'n (%)	Assay (ppm)	Mass (µg)	Dist'n (%)
Solids (g)	2000.0	0.10	192	2.7	<0.50	1000	43.4
Solution (ml)	2886.1	2.27	6551	93.0	0.43	1241	53.9
Solution Samples			304	4.3		62	2.7
Total Extraction			6856	97.3		1303	56.6
Total			7048	100.0		2303	100.0
Calculated Head		3.52			1.15		
Assay Head		2.74					

Leach Conditions

NaCN Concentration	0.50	g/l
NaCN Addition	1.09	kg/t
NaCN Consumption	0.49	kg/t
Lime Consumption	1.22	kg/t
% Solids	40.0	
Grind Size d_{80}	75	µm
Aeration Gas	Air	
Aeration Rate	3.0	l/min



Kinetic Cyanide Leach Test

Client: Cora Gold
Project: Sanankoro
Sample: Master Composite
Date: 09 April 2019
Grind: 80% passing 75 µm
Test: LT75

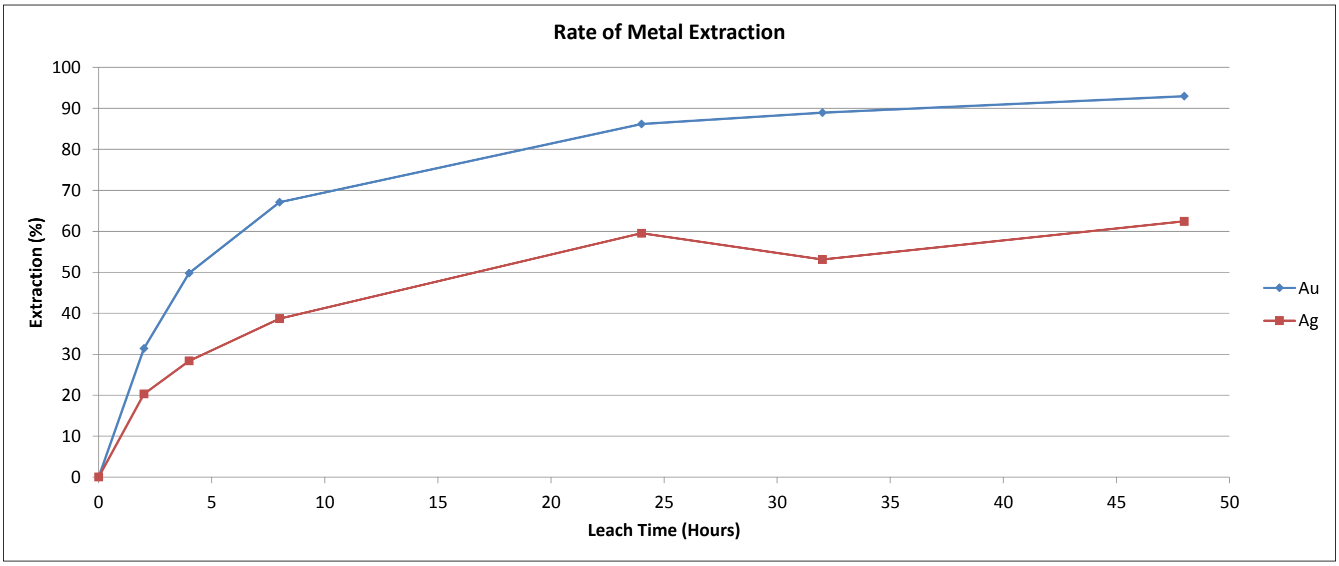
Time	Additions				Solution				Sample				Metal Extracted			
(Hours)	Solids (g)	Solution (ml)	NaCN (g)	Lime (g)	pH	NaCN (%)	Au (ppm)	Ag (ppm)	Vol (ml)	Titre	∑ Au (µg)	∑ Ag (µg)	Au (µg)	Au (%)	Ag (µg)	Ag (%)
	2000.0	3000.0			7.61											
0		3000.0	0.75	2.82	10.54	0.025										
2		2999.6			11.30	0.025	0.83	0.18	30.0	0.50	25	5	2490	31.4	540	20.3
4		2995.6			11.23	0.025	1.31	0.25	30.0	0.50	64	13	3949	49.8	754	28.3
8		2987.6			11.06	0.025	1.76	0.34	40.0	0.50	135	27	5322	67.1	1029	38.6
24		2939.6			10.66	0.025	2.28	0.53	40.0	0.50	226	48	6837	86.2	1584	59.5
32		2969.6		0.23	10.41	0.025	2.30	0.46	40.0	0.50	318	66	7056	88.9	1414	53.1
48		2927.6			10.40	0.020	2.41	0.55	60.0	0.40	462	99	7373	92.9	1662	62.4
Total																

Extraction Calculation

Product	Quantity	Gold			Silver		
		Assay (ppm)	Mass (µg)	Dist'n (%)	Assay (ppm)	Mass (µg)	Dist'n (%)
Solids (g)	2000.0	0.28	560	7.1	<0.50	1000	37.6
Solution (ml)	2927.6	2.41	7055	88.9	0.55	1596	59.9
Solution Samples			318	4.0		66	2.5
Total Extraction			7373	92.9		1662	62.4
Total			7933	100.0		2662	100.0
Calculated Head		3.97			1.33		
Assay Head		2.74					

Leach Conditions

NaCN Concentration	0.25	g/l
NaCN Addition	0.38	kg/t
NaCN Consumption	0.08	kg/t
Lime Consumption	1.53	kg/t
% Solids	40.0	
Grind Size d_{80}	75	µm
Aeration Gas	Air	
Aeration Rate	3.0	l/min



APPENDIX 9: Agglomeration & Percolation Test Data

Wardell Armstrong International

2hr Pre-SOAK PERCOLATION TEST DATA : PRE-COLUMN LEACH TESTWORK

Project:	Cora Gold	**No:**	ZT64-0703	**Test Date:**	**08-Mar-19**
Sample:	Sanankoro			**Test ID:**	SKN-0kg/t

Agglomeration Data:								
Sample Weight		**(kg)**		**Initial:**	2.500			
Cement	**Kg/t**	0	**Lime**	**Kg/t**	0	**Water**	**(g)**	0
	(g)	0.0		**(g)**	0.0		**Kg/t**	0.0

1. Initial Dry Height (mm) H_1	485	**Pellet Characteristics**	
(Add water to cover sample - Soak for 2 hours)		**Initial Agglomerate (kg):**	
2. Initial Wet Height (mm) H_2	495	**Cured Wt. (kg):**	
(Tap sample to settle down, 4-5 hard hits)		**Calculated (kg):**	2.500
3. Tapped Height (mm) H_3	495	**Cured %Moisture:**	-100.0
4. Mark & Maintain Water Height @ 50mm above Initial Tapped Height			

(Determine water flow at constant head)

Percolation Test Flowrate							
Column:	**ID**	**1**	Diameter Ø (mm)		75	Xsectional Area (m^2)	0.004418
Free Draining Flowrate:	Time (min)	1		Weight (kg)	3.09	Flow ($l/m^2/hr$)	41,981
		1			3.09		
Bucket (g)			640				**1 hr**

DRAINAGE SAMPLE				
Time (min)	**Bucket + Water (g)**	**Flow (l)**	**Flowrate ($l/m^2/hr$)**	**Quality**
15.17	690	0.05	45	Cloudy
30.33	656	0.02	14	
45.00	660	0.02	19	
60.00	664	0.02	22	
		-0.64	145	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	

DRAINAGE CHARACTERISTICS		
Flowrate $l/m^2/hr$	Maximum	45
	Minimum	14
	Average	25
5. Final Height (mm)		
Undrained H_{UD}		465
Drained H_D		455
6. Column Sample Weight (kg):		
Wet Wt (kg):		3.568
Dry Wt (kg):		2.435
Test Water pH:		8.20
Drainage Water pH:		6.55
SLUMPAGE CHARACTERISTICS		
Slumpage (%)	Auto	-2.1
	Tapped	-2.1
	$Final_{UD}$	4.1
	$Final_D$	6.2
Moisture Content (%)		31.8
Solids Migration (%)		2.6

Code for Pellets		
1	[EXCELLENT]	
2	[GOOD]	
3	[FAIR]	
4	[POOR]	
5	[NO PELLETS (Total Collapse)]	

Notes:

V.Cloudy	Very Cloudy
Sl.Cloudy	Slightly Cloudy

Wardell Armstrong International

2hr Pre-SOAK PERCOLATION TEST DATA : PRE-COLUMN LEACH TESTWORK

Project:	Cora Gold	**No:**	ZT64-0703	**Test Date:**	**08-Mar-19**
Sample:	Sanankoro			**Test ID:**	SKN-5kg/t

Agglomeration Data:

Sample Weight		**(kg)**		**Initial:**	2.500		
Cement	**Kg/t**	5	**Lime**	**Kg/t**	0	**Water**	**(g)** 430
	(g)	12.5		**(g)**	0.0		**Kg/t** 172.0

1. Initial Dry Height (mm) H_1	460
(Add water to cover sample - Soak for 2 hours)	
2. Initial Wet Height (mm) H_2	470
(Tap sample to settle down, 4-5 hard hits)	
3. Tapped Height (mm) H_3	455

Pellet Characteristics	
Initial Agglomerate (kg):	2.867
Cured Wt. (kg):	2.858
Calculated (kg):	2.943
Cured %Moisture:	13.8

4. Mark & Maintain Water Height @ 50mm above Initial Tapped Height

(Determine water flow at constant head)

Percolation Test Flowrate

Column:	**ID**	**2**	Diameter Ø (mm)	75	Xsectional Area (m²)	0.004418
Free Draining Flowrate:	Time (min)	1	Weight (kg)	3.39	Flow (l/m²/hr)	45,824
		1		3.36		
Bucket (g)			640			**1 hr**

DRAINAGE SAMPLE

Time (min)	Bucket + Water (g)	Flow (l)	Flowrate (l/m²/hr)	Quality
15.53	1,744	1.10	965	Cloudy
30.57	1,578	0.94	847	Clear
45.33	1,424	0.78	721	Clear
60.00	1,274	0.63	587	Clear
		-0.64	145	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	

DRAINAGE CHARACTERISTICS

Flowrate l/m²/hr	Maximum	965
	Minimum	587
	Average	780

5. Final Height (mm)	
Undrained H_{UD}	450
Drained H_D	435

6. Column Sample Weight (kg):	
Wet Wt (kg):	3.116
Dry Wt (kg):	2.418
Test Water pH:	8.20
Drainage Water pH:	11.11

SLUMPAGE CHARACTERISTICS

Slumpage (%)	Auto	-2.2
	Tapped	1.1
	$Final_{UD}$	2.2
	$Final_D$	5.4

Moisture Content (%)	22.4
Solids Migration (%)	17.8

Code for Pellets

1	[EXCELLENT]	
2	[GOOD]	
3	[FAIR]	
4	[POOR]	
5	[NO PELLETS (Total Collapse)]	

Notes:

V.Cloudy	Very Cloudy
Sl.Cloudy	Slightly Cloudy

Wardell Armstrong International

2hr Pre-SOAK PERCOLATION TEST DATA : PRE-COLUMN LEACH TESTWORK

Project:	Cora Gold	**No:**	ZT64-0703	**Test Date:**	**08-Mar-19**
Sample:	Sanankoro			**Test ID:**	SKN-10kg/t

Agglomeration Data:								
Sample Weight		**(kg)**		**Initial:**	2.502			
Cement	**Kg/t**	10	**Lime**	**Kg/t**	0	**Water**	**(g)**	452
	(g)	25.0		**(g)**	0.0		**Kg/t**	180.7

1. Initial Dry Height (mm) H_1	495	**Pellet Characteristics**	
(Add water to cover sample - Soak for 2 hours)		**Initial Agglomerate (kg):**	2.964
2. Initial Wet Height (mm) H_2	500	**Cured Wt. (kg):**	2.950
(Tap sample to settle down, 4-5 hard hits)		**Calculated (kg):**	2.979
3. Tapped Height (mm) H_3	490	**Cured %Moisture:**	16.9
4. Mark & Maintain Water Height @ 50mm above Initial Tapped Height			

(Determine water flow at constant head)

Percolation Test Flowrate

Column:	**ID**	**3**	Diameter Ø (mm)		75	Xsectional Area (m^2)	0.004418
Free Draining Flowrate:	Time (min)	1		Weight (kg)	3.25	Flow ($l/m^2/hr$)	45,613
		1			3.43		
					3.40		
Bucket (g)			640				**1 hr**

DRAINAGE SAMPLE

Time (min)	Bucket + Water (g)	Flow (l)	Flowrate ($l/m^2/hr$)	Quality
15.67	5,186	4.55	3,940	Cloudy
30.55	4,106	3.47	3,162	Clear
45.33	3,912	3.27	3,006	Clear
60.00	3,386	2.75	2,542	Clear
		-0.64	145	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	

DRAINAGE CHARACTERISTICS		
Flowrate $l/m^2/hr$	Maximum	3,940
	Minimum	2,542
	Average	3,163
5. Final Height (mm)		
Undrained H_{UD}		485
Drained H_D		470
6. Column Sample Weight (kg):		
Wet Wt (kg):		3.256
Dry Wt (kg):		2.511
Test Water pH:		8.20
Drainage Water pH:		11.32
SLUMPAGE CHARACTERISTICS		
Slumpage (%)	Auto	-1.0
	Tapped	1.0
	$Final_{UD}$	2.0
	$Final_D$	5.1
Moisture Content (%)		22.9
Solids Migration (%)		15.7

Code for Pellets		
1	[EXCELLENT]	
2	[GOOD]	
3	[FAIR]	
4	[POOR]	
5	[NO PELLETS (Total Collapse)]	

Notes:

V.Cloudy	Very Cloudy
Sl.Cloudy	Slightly Cloudy

Wardell Armstrong International

2hr Pre-SOAK PERCOLATION TEST DATA : PRE-COLUMN LEACH TESTWORK

Project:	Cora Gold	**No:**	ZT64-0703	**Test Date:**	**13-Mar-19**
Sample:	Sanankoro			**Test ID:**	SKN-15kg/t

Agglomeration Data:								
Sample Weight		**(kg)**		**Initial:**	2.500			
Cement	**Kg/t**	15	**Lime**	**Kg/t**	0	**Water**	**(g)**	464
	(g)	37.5		**(g)**	0.0		**Kg/t**	185.6

1. Initial Dry Height (mm) H_1	490	**Pellet Characteristics**	
(Add water to cover sample - Soak for 2 hours)		**Initial Agglomerate (kg):**	2.928
2. Initial Wet Height (mm) H_2	485	**Cured Wt. (kg):**	2.916
(Tap sample to settle down, 4-5 hard hits)		**Calculated (kg):**	3.002
3. Tapped Height (mm) H_3	465	**Cured %Moisture:**	15.1

4. Mark & Maintain Water Height @ 50mm above Initial Tapped Height

(Determine water flow at constant head)

Percolation Test Flowrate

Column:	**ID**	**1**	Diameter Ø (mm)		75	Xsectional Area (m^2)	0.004418
Free Draining Flowrate:	Time (min)	1		Weight (kg)	3.06	Flow (l/m^2/hr)	42,504
		1			3.14		
					3.19		
Bucket (g)			640				**1 hr**

DRAINAGE SAMPLE

Time (min)	Bucket + Water (g)	Flow (l)	Flowrate (l/m^2/hr)	Quality
8.50	7,920	7.28	11,630	Cloudy
17.00	6,112	5.47	8,742	Clear
27.33	6,318	5.68	7,462	Clear
36.50	4,476	3.84	5,683	Clear
47.00	4,512	3.87	5,008	Clear
60.00	5,162	4.52	4,724	Clear
		-0.64	145	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	

DRAINAGE CHARACTERISTICS		
Flowrate l/m^2/hr	Maximum	11,630
	Minimum	4,724
	Average	7,208
5. Final Height (mm)		
Undrained H_{UD}		465
Drained H_D		465
6. Column Sample Weight (kg):		
Wet Wt (kg):		3.094
Dry Wt (kg):		2.457
Test Water pH:		7.53
Drainage Water pH:		11.52
SLUMPAGE CHARACTERISTICS		
Slumpage (%)	Auto	1.0
	Tapped	5.1
	$Final_{UD}$	5.1
	$Final_D$	5.1
Moisture Content (%)		20.6
Solids Migration (%)		18.1

Code for Pellets		
1	[EXCELLENT]	
2	[GOOD]	
3	[FAIR]	
4	[POOR]	
5	[NO PELLETS (Total Collapse)]	

Notes:

V.Cloudy	Very Cloudy
Sl.Cloudy	Slightly Cloudy

Wardell Armstrong International

2hr Pre-SOAK PERCOLATION TEST DATA : PRE-COLUMN LEACH TESTWORK

Project:	Cora Gold	**No:**	ZT64-0703	**Test Date:**	**13-Mar-19**
Sample:	Sanankoro			**Test ID:**	SKN-22.5kg/t

Agglomeration Data:									
Sample Weight		**(kg)**		**Initial:**	2.500				
Cement	**Kg/t**	22.5	**Lime**	**Kg/t**	0	**Water**	**(g)**	484	
	(g)	56.3		**(g)**	0.0		**Kg/t**	193.6	

1. Initial Dry Height (mm) H_1	525	**Pellet Characteristics**	
(Add water to cover sample - Soak for 2 hours)		**Initial Agglomerate (kg):**	
2. Initial Wet Height (mm) H_2	525	**Cured Wt. (kg):**	
(Tap sample to settle down, 4-5 hard hits)		**Calculated (kg):**	3.040
3. Tapped Height (mm) H_3	495	**Cured %Moisture:**	-102.3
4. Mark & Maintain Water Height @ 50mm above Initial Tapped Height			

(Determine water flow at constant head)

Percolation Test Flowrate

Column:	ID	**2**	Diameter Ø (mm)	75	Xsectional Area (m^2)		0.004418
Free Draining Flowrate:	Time (min)	1	Weight (kg)	3.36		Flow (l/m²/hr)	45,564
		1		3.37			
		1		3.34			
Bucket (g)		640					**1 hr**

DRAINAGE SAMPLE				
Time (min)	**Bucket + Water (g)**	**Flow (l)**	**Flowrate (l/m²/hr)**	**Quality**
6.50	9,626	8.99	18,773	Cloudy
13.83	9,424	8.78	16,266	Sl. Cloudy
23.00	9,196	8.56	12,675	Sl. Cloudy
33.50	9,104	8.46	10,946	Clear
45.00	9,110	8.47	10,002	Clear
56.00	9,094	8.45	10,436	Clear
60.00	3,722	3.08	10,463	Clear
		-0.64	145	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	
		-0.64	#DIV/0!	

DRAINAGE CHARACTERISTICS		
Flowrate l/m²/hr	Maximum	18,773
	Minimum	10,002
	Average	12,794

5. Final Height (mm)	
Undrained H_{UD}	490
Drained H_D	490

6. Column Sample Weight (kg):	
Wet Wt (kg):	3.148
Dry Wt (kg):	2.517
Test Water pH:	7.53
Drainage Water pH:	11.64

SLUMPAGE CHARACTERISTICS		
Slumpage (%)	Auto	0.0
	Tapped	5.7
	$Final_{UD}$	6.7
	$Final_D$	6.7

Moisture Content (%)	20.0
Solids Migration (%)	17.2

Code for Pellets		
1	[EXCELLENT]	
2	[GOOD]	
3	[FAIR]	(highlighted)
4	[POOR]	
5	[NO PELLETS (Total Collapse)]	

Notes:

V.Cloudy — Very Cloudy

Sl.Cloudy — Slightly Cloudy

APPENDIX 10: Column Leach Test Data

Wardell Armstrong International Ltd

Column Leach Test Report

Client : Cora Gold
Project: Sanankoro
Sample: Master Comp

TS:TS

Date	Day	Feed Solution													
		Vol (l).					pH	Temp (°C)	$Ca(OH)_2$	Barren Titre	NaCN Additio	NaCN Concentration		Au	Ag
		Water	Barren	Rem.	Applied	Cum.			(g)	(ml)	(g)	g/l	%	(ppm)	(ppm)
18-Mar	0	20.00				0.00	10.85	9.3	2.14		20.00	1.00	0.10		
19-Mar	1					0.00			0.00			1.00	0.10		
20-Mar	2			11.80	8.20	8.20	11.01	10.7	0.00			1.00	0.10		
21-Mar	3	20.00		7.84	3.96	12.16	10.60	11.7	0.54		12.16	1.00	0.10		
22-Mar	4														
25-Mar	7	15.00		8.34	11.66	23.82	10.60	10.6	0.54		6.66	1.00	0.10		
28-Mar	10	20.00		3.30	11.70	35.52	10.60	9.8	1.61		16.70	1.00	0.10		
01-Apr	14		15.00	5.10	14.90	50.42	11.80	10.7	0.00	1.60	3.00	1.00	0.10		
04-Apr	17		20.00	3.30	11.70	62.12	11.87	8.2	0.00	1.45	5.50	1.00	0.10		
08-Apr	21		15.06	4.56	15.44	77.56	11.87	10.5	0.00	1.80	1.51	1.00	0.10		
11-Apr	24		20.00	3.90	11.16	88.72	11.86	10.0	0.00	1.85	1.50	1.00	0.10		
15-Apr	28		15.00	5.02	14.98	103.70	11.94	8.4	0.00	1.90	0.75	1.00	0.10		
18-Apr	31		25.00	3.66	11.34	115.04	11.83	12.0	0.00	1.90	1.25	1.00	0.10		
23-Apr	36		16.00	0.22	24.78	139.82	11.34	14.7	0.00	1.95	0.40	1.00	0.10		
26-Apr	39		20.00	3.10	12.90	152.72	11.54	11.6	0.00	1.90	1.00	1.00	0.10		
30-Apr	43		15.00	2.94	17.06	169.78	11.53	11.9	0.00	1.85	1.13	1.00	0.10		
03-May	46		20.00	1.98	13.02	182.80	11.43	11.3	0.00	1.90	1.00	1.00	0.10		
07-May	50		16.00	2.68	17.32	200.12	11.21	10.4	0.00	1.80	1.60	1.00	0.10		
10-May	53		20.00	3.58	12.42	212.54	11.25	10.6	0.00	1.85	1.50	1.00	0.10		
14-May	57		16.00	2.70	17.30	229.84	11.35	13.0	0.00	1.65	2.80	1.00	0.10		
17-May	60		20.00	3.10	12.90	242.74	11.30	13.6	0.00	1.70	3.00	1.00	0.10		
21-May	64		16.00	2.80	17.20	259.94	10.90	13.6	0.00	1.70	2.40	1.00	0.10		
24-May	67		20.00	3.40	12.60	272.54	10.79	14.5	0.00	1.60	4.00	1.00	0.10		
28-May	71		16.00	3.18	16.82	289.36	11.00	13.9	0.00	1.80	1.60	1.00	0.10		
31-May	74		20.00	4.82	11.18	300.54	10.95	15.8	0.00	1.80	2.00	1.00	0.10		
04-Jun	78		16.00	2.78	17.22	317.76	10.90	15.0	0.00	1.50	4.00	1.00	0.10		
07-Jun	81		20.00	2.80	13.20	330.96	11.13	13.5	0.00	1.75	2.50	1.00	0.10		
11-Jun	85		16.00	2.58	17.42	348.38	10.79	13.6	0.00	1.60	3.20	1.00	0.10		
14-Jun	88		20.00	2.86	13.14	361.52	10.79	13.9	0.00	1.80	2.00	1.00	0.10		
18-Jun	92		16.00	2.54	17.46	378.98	10.85	15.7	0.00	1.70	2.40	1.00	0.10		
21-Jun	95			3.06	12.94	391.92									
24-Jun	98	20.00			0.00	391.92					0.00				
01-Jul	105			0.00	20.00	411.92									
		95.00	**413.06**		**411.92**				**4.82**						
		508.1													

Date	Day	Pregnant Solution								Metal Recovery - Measured Head		Metal Recovery - Back Calculated Head		Ratio	NaCN		Exit Flowrate		
		Vol (l).		pH	Temp (°C)	NaCN Titre	NaCN Conc.	Au	Ag	Au	Ag	Au	Ag	Solids: Solution	Daily**	Cum	Daily**	Average Cum.	
		Daily**	Cum.			(ml)	%	(ppm)	(ppm)	(%)	(%)	(%)	(%)			(kg/t)	(l/hr/m²)	(l/hr/m²)	(ml/min)
18-Mar	0									0.0%	0.0%	0.0%	0.0%	0.00					
19-Mar	1	0.94	0.9	11.79	11.2	0.30	0.02	0.83	0.04	0.7%	0.1%	0.6%	0.1%	0.02	0.021	0.021	2.22	2.22	0.65
20-Mar	2	3.84	4.8	12.17	10.7	0.30	0.02	1.97	0.14	7.9%	1.9%	6.4%	1.9%	0.12	0.084	0.105	9.05	5.64	1.66
21-Mar	3	3.82	8.6	12.32	11.3	0.75	0.04	1.82	0.10	14.5%	3.1%	11.6%	3.1%	0.22	0.062	0.167	9.01	6.76	1.99
22-Mar	4	3.00	11.6	12.32	10.7	1.15	0.06	1.49	0.12	18.7%	4.3%	15.1%	4.3%	0.30		0.167	7.07	6.84	2.01
25-Mar	7	8.66	20.3	11.60	9.9	1.50	0.08	1.02	0.12	27.0%	7.6%	21.8%	7.7%	0.52	0.056	0.223	5.10	6.82	2.01
28-Mar	10	11.16	31.4	11.91	9.4	1.60	0.08	0.63	0.09	33.7%	10.9%	27.1%	11.0%	0.81	0.058	0.280	8.77	7.41	2.18
01-Apr	14	14.66	46.1	11.62	10.2	1.75	0.09	0.33	0.07	38.2%	14.2%	30.8%	14.4%	1.19	0.047	0.328	8.64	7.76	2.29
04-Apr	17	11.52	57.6	11.66	8.8	2.00	0.10	0.23	0.06	40.7%	16.4%	32.8%	16.7%	1.49	0.000	0.328	9.05	7.99	2.35
08-Apr	21	15.42	73.0	12.28	10.1	2.30	0.12	0.20	0.05	43.7%	18.9%	35.2%	19.2%	1.89	-0.060	0.268	9.09	8.20	2.41
11-Apr	24	10.80	83.8	12.18	9.8	1.95	0.10	0.19	0.05	45.6%	20.6%	36.7%	21.0%	2.17	0.007	0.275	8.49	8.23	2.43
15-Apr	28	14.78	98.6	12.25	8.2	1.95	0.10	0.12	<0.01	47.3%	21.1%	38.1%	21.4%	2.55	0.010	0.285	8.71	8.30	2.45
18-Apr	31	10.70	109.3	12.08	11.4	2.00	0.10	0.11	0.01	48.4%	21.5%	39.0%	21.8%	2.82	0.000	0.285	8.41	8.31	2.45
23-Apr	36	24.60	133.9	11.63	14.3	1.95	0.10	0.10	0.02	50.7%	23.1%	40.9%	23.4%	3.46	0.016	0.300	11.60	8.77	2.58
26-Apr	39	12.38	146.3	11.75	11.6	1.95	0.10	0.10	0.02	51.9%	23.9%	41.8%	24.2%	3.78	0.008	0.308	9.73	8.84	2.60
30-Apr	43	16.76	163.0	11.73	11.3	1.95	0.10	0.07	0.02	53.0%	24.9%	42.7%	25.3%	4.21	0.011	0.319	9.88	8.94	2.63
03-May	46	12.82	175.9	11.74	11.3	1.90	0.10	0.08	<0.01	54.0%	25.4%	43.5%	25.7%	4.55	0.017	0.336	10.08	9.01	2.65
07-May	50	17.24	193.1	11.79	10.0	1.85	0.09	0.08	<0.01	55.3%	25.9%	44.5%	26.3%	4.99	0.033	0.369	10.16	9.11	2.68
10-May	53	11.70	204.8	11.68	10.6	2.00	0.10	0.11	0.03	56.5%	27.0%	45.5%	27.5%	5.29	0.000	0.369	9.20	9.11	2.68
14-May	57	17.00	221.8	11.86	12.3	1.90	0.10	0.10	0.03	58.1%	28.7%	46.8%	29.1%	5.73	0.022	0.391	10.02	9.17	2.70
17-May	60	12.10	233.9	11.71	13.6	1.70	0.09	0.09	0.02	59.1%	29.5%	47.6%	29.9%	6.05	0.047	0.438	9.51	9.19	2.71
21-May	64	16.02	249.9	11.54	13.3	1.90	0.10	0.12	0.03	60.9%	31.0%	49.1%	31.5%	6.46	0.021	0.459	9.44	9.21	2.71
24-May	67	12.72	262.6	12.08	14.4	1.95	0.10	0.10	0.01	62.1%	31.4%	50.1%	31.9%	6.79	0.008	0.467	10.00	9.24	2.72
28-May	71	16.82	279.5	12.02	13.9	1.95	0.10	0.08	0.02	63.4%	32.5%	51.1%	33.0%	7.22	0.011	0.478	9.91	9.28	2.73
31-May	74	11.36	290.8	12.09	15.4	1.95	0.10	0.07	0.01	64.2%	32.9%	51.7%	33.4%	7.52	0.007	0.485	8.93	9.27	2.73
04-Jun	78	16.24	307.1	12.16	15.0	1.95	0.10	0.09	0.01	65.5%	33.4%	52.8%	33.9%	7.94	0.010	0.496	9.57	9.28	2.73
07-Jun	81	13.04	320.1	12.25	13.5	1.85	0.09	0.05	0.01	66.2%	33.8%	53.3%	34.3%	8.27	0.025	0.521	10.25	9.32	2.74
11-Jun	85	17.24	337.3	12.04	13.6	2.00	0.10	0.05	0.01	67.0%	34.4%	54.0%	34.9%	8.72	0.000	0.521	10.16	9.36	2.76
14-Jun	88	13.10	350.4	12.03	13.5	1.75	0.09	0.04	0.01	67.5%	34.8%	54.4%	35.3%	9.06	0.042	0.563	10.30	9.39	2.77
18-Jun	92	17.24	367.7	12.11	15.5	1.90	0.10	0.04	<0.01	68.1%	35.4%	54.9%	35.9%	9.50	0.022	0.586	10.16	9.42	2.78
21-Jun	95	12.98	380.7	12.06	15.1	1.80	0.09	0.04	<0.01	68.6%	35.8%	55.3%	36.3%	9.84	0.034	0.619	10.20	9.45	2.78
24-Jun	98	0.76	381.4	10.26	16.4	0.20	0.01	0.04	<0.01	68.6%	35.8%	55.3%	36.4%	9.86		0.619			
01-Jul	105	19.06	400.5	11.96	17.5	0.90	0.05	0.05	<0.01	69.5%	36.4%	56.0%	37.0%	10.35		0.619			
		400.5										**56.0%**	**37.0%**		**0.619**				

Closed circuit started
Pumps switched off for Column drain down
Tap Water Wash - Open Circuit

1. Preparation Parameters

Column diam. (mm)	150.0
Solution Rate (ml/min)	3.0
Solution Rate (l/m²/hr)	**10.0**
Sample Wet Wt kg	44.7
Sample Dry Wt kg	38.7
Moisture (%)	13.43
Lime (kg/t)	0.00
Cement (kg/t)	22.50

2. Grading Analysis

	Gold		Silver	
	g/t	mg Au	g/t	mg Ag
Head Assay g/t	2.74	105.82	0.8	31.0
Extracted g/t	1.90	73.58	0.29	11.28
Tails Assay g/t	1.50	57.74	0.50	19.21
Calculated Head g/t	3.39	131.32	0.788	30.49
Recovery on Measured Head	70%		36%	
Rec on Calculated Head	**56%**		**37%**	
Rec on Meas. Head and Residue	45%		38%	
Rec on Calc. Head and Carbon	57%		47%	
Rec on Carbon and Residue	**56%**		**43%**	

3. Height Parameters

Column height mm	1905
Initial space at top mm	36
Final space at top mm	50
Slump mm	14.0
Slump (%)	0.75

4. Residue

Wet Wt (kg)	50.2
% Solids (wt/wt)	78.3
Moisture (%)	21.7
Dry Wt (kg)	38.4
Moisture Content Determination	
Mass of wet residue (kg)	50.200
Mass of Dry residue (kg)	39.300

5. Carbon

Wt (g)		215.9
Assay (g/t)		mg
Au	344.0	74.27
Ag	67.0	14.47

6. Reagent Consumption

	kg/t
NaCN	0.62
Lime	0.12

Flowrate	
Col Area (m²)	0.0177
l/m²/hr	10
ml/hr	176.71
ml/min	2.95

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APPENDIX 4: Environmental and Social Scoping Study for the Sanankoro Gold Prospect

(Completed by Digby Wells Environmental)

Environmental and Social Scoping Study for the Sanankoro Gold Prospect

Scoping Report

Project Number:

CGL5913

Prepared for:

Cora Gold Limited

October 2019

Digby Wells Environmental (Jersey) Limited. Co. No. 115951. Suite 10, Bourne House, Francis Street, Jersey, JE2 4QE
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Directors: D Pettit and M Radyn

This document has been prepared by Digby Wells Environmental.

Report Type:	**Scoping Report**
Project Name:	**Environmental and Social Scoping Study for the Sanankoro Gold Prospect**
Project Code:	**CGL5913**

Name	**Responsibility**	**Signature**	**Date**
Anita Gutu	Report Writer		September 2019 Updated October 2019
Mamadou Dembélé	Report Writer		August 2019
Soumaila Traoré	Report Writer		August 2019
Duncan Pettit	Report Reviewer		September 2019

EXECUTIVE SUMMARY

Cora Gold Limited (Cora Gold) is undertaking gold exploration activities associated with the Sanankoro Gold Prospect located in southern Mali. Digby Wells Environmental (Digby Wells) was appointed to undertake a Scoping Study to characterise the biophysical and socio-economic environment of the project area, provide early indication of potential environmental and social risks and determine the Terms of Reference (ToR) for the Environmental and Social Impact Assessment (ESIA) process that will be required as part of the environmental permitting process. The ESIA will be undertaken in accordance with Malian Law No. 2012-015 of 27 February 2012 (the Mining Code) and associated Decrees, among others. In addition, International Best Practice will also be considered.

Biophysical Environment and Risks

The local vegetation is predominantly distributed grasslands. Due to Artisanal and Small Scale Mining (ASM) activities throughout the project area, much of the land associated with the resource target areas, and areas identified for the processing plant and other administrative infrastructure within the project area has been disturbed. Isolated forest galleries which are associated with floodplains are also present within the project area. Furthermore, land (including floodplains and wetland areas) associated with the project area are extensively utilised for agricultural activities while nearby streams are used for ASM activities. Medium and large mammals in the Project area are rare and hunting is a practised to a limited extent. Several Nationally Protected and Red Data Plant species are expected to occur regionally.

The streams and drainage lines in the project area are predominantly ephemeral, including the Fie and Niger Rivers which traverse the west and north boundaries of the project area. Surface water resources are utilised by the communities for economic activities (agricultural and ASM) while groundwater is used for potable and domestic uses. A total of six water samples were collected upstream and downstream of the project area which indicate no issues in terms of water quality. This could be due to dilution as aesthetically there are many impacts from ASM activities. Generally groundwater quality results were found to be good.

Water management in the project design and during operations is important as these areas may intersect with various water resources. To this end, detailed floodplain determinations will be required to delineate the floodplains, as well as determine potential surface water volumes during extreme rainfall events. Groundwater modelling will also be an important task to determine the potential impact dewatering may have on surface water resources, as well as to determine potential contamination plumes from the project’s waste deposits. The potential pit areas, plant and other administrative infrastructure are however largely disturbed by existing ASM activities along the targeted ore structures.

Socio-Economic Environment and Risks

The project is located within the Kangaba Cercle of the Koulikoro Region and spans over the Séléfougou and Maramadougou rural communes. A total of six rural communes are located within the project area, namely: Séléfougou, Sanankoro, Bokoro (hamlet), Sélin, Faragouagnan and Kignèlen (hamlet).

The primary economic activities in the project area comprise ASM, cultivation, livestock breeding, and limited small trade which includes the exploitation of natural resources. Agricultural activities are located within and near the communities and maintained by the respective villagers. Livestock rearing in the project area includes large and small livestock, such as cattle, sheep, goats and poultry. ASM activities are practiced throughout the regions and the population undertaking ASM activities is increasing, as well as attracting individuals from neighbouring regions and countries.

Several communities and their associated economic activities are located within 500 m of potential pit areas. Communities' households, agricultural fields and ASM activities within 500 m of proposed pits will result in economic and physical displacement. This, together with the expected increased influx of people into the area as a result of the presence of the project, is expected to be the key socio-economic implications.

A Resettlement Action Plan (RAP) and Livelihood Restoration Plan (LRP) will be required for the economic and physical displacement associated with project land acquisition. The RAP and LRP will need to have a clear entitlement framework to address any potential challenges as it is expected that resettlement will be widely contested due to the extent and reliance of this activity currently. ASM management, and the loss of livelihood, will have significant impacts in the area and will need to be managed carefully and in cooperation with the technical and administrative authorities.

It is recommended that baseline socio-economic surveys are undertaken in the affected communities to determine the baseline of affected communities and the extent of potential resettlement prior to any population influx. It is important to manage community expectations and potential resettlement should not be communicated until a final layout plan is complete.

Population influx is expected because of the project as individuals from surrounding regions and neighbouring countries move to the project area in search of employment. The population influx will also place additional pressure on the already stressed natural resources as well as social services and infrastructure in the project area.

Conclusion

No immediate fatal flaws were identified for the project. However, the identified project risks will require careful planning and management. These risks and key impacts can be managed throughout the ESIA process and include:

- Economic and physical displacement;
- Population influx and the resulting impacts, including increase in ASM; and
- Water management.

The project area is already largely disturbed, however, natural habitats (including potential protected species and wetland areas) exist which should be avoided as far as possible. It is recommended that the environmental and social studies are undertaken in collaboration with the engineering design and feasibility studies to feed into project decision making. The ESIA process takes approximately 12 to 16 months, depending on the level of collaboration between the respective feasibility teams. It should be noted that the above timing considers two season surveys for biotic studies (wet and dry season) but excludes potential resettlement and livelihood restoration as this process is independent to the ESIA and environmental permitting.

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1 Introduction

Digby Wells Environmental Jersey Limited (hereinafter Digby Wells) was appointed by Cora Gold Limited (Cora Gold) to undertake an environmental and social screening assessment (Scoping Study) for the Sanankoro Gold Prospect located in southern Mali.

Cora Gold is a West African gold exploration company and has initiated an extensive exploration programme with the intent to develop its Sanankoro Gold Discovery (or "the project") along the Yanfolila Gold Belt. The targeted project is located within the Kangaba Cercle in the Koulikoro Region.

This report constitutes an Environmental and Social Scoping Study for the project which aims to provide a high level baseline of the project area and an early indication of the potential environmental and social risks associated with developing the Sanankoro mining project. For the purpose of this report, the project area is defined by the proposed exploration tenement area which covers an approximate area of 320 square kilometres (km^2). The tenement area, which is known as Sanankoro, consists of four contiguous permits namely, Sanankoro, Bokoro Est, Bokoro II and Dako.

1.1 Project Background

Prior to Cora Gold's interest in Sanankoro, considerable historical exploration activity was carried out by Gold Fields Limited between 2000 and 2012. These activities included extensive soil geochemistry as well as Rotary Air Blast (RAB), Air Core (AC) and Reverse Circulation (RC) drilling. Progressively gold mineralised structures, including Zone A and Zone B with a total distance of approximately 14 km as well as the Sélin structure were determined. Furthermore, extensive surface artisanal workings exist at Sanankoro which confirm the continuity of mineralisation of these structures.

Cora Gold subsequently acquired the interests to Sanankoro and continued exploration metallurgical testwork which primarily focussed on the high potential Zone A and Sélin structures. Through its exploration, Cora Gold confirmed continuous oxide gold mineralisation with significant grades up to 4.48 grams per ton of gold (g/t Au) and 2.83 g/t Au at the Sélin and Zone A prospects respectively.

In 2019, Cora Gold initiated a drilling campaign with SRK and Wardell Armstrong to establish a maiden gold oxide mineral resource estimate for the delineation of feasible pit areas. During August 2019, results based on three out of four core holes completed confirmed the continuity of gold sulphide mineralisation which was intersected at a vertical depth up to 170 m below surface. These results have demonstrated the potential to significantly expand scale of discovery at Sanankoro.

Cora Gold intends to start a small mine (approximately 30,000 – 50,000 ounces of gold per annum) with a processing plant with an approximate capacity of 0.5 – 1 million tonnes per annum (Mtpa) to generate revenue and to use this funding to further expand the facilities and to enable a better understanding of the deposit.

1.2 Purpose of this Report

The aim of an environmental and social scoping study is to establish the biophysical and socio-economic characteristics of the project area as well as identify potential impacts that could arise as a result of the development at a high-level.

The objectives of this report are therefore as follows:

- Provide a high-level description of the background physical environment, social, cultural and economic conditions of the project area;
- Identify the primary potential positive and negative impacts associated with the project;
- Provide mitigation types for the identified impacts;
- Determine any potential risks and fatal flaws that may inhibit the project's development; and
- Provide a Terms of Reference (ToR) which outlines the required legislative requirements to obtain the relevant environmental permits to commence the operation as well as the environmental and socio-economic assessments studies deemed relevant for the environmental permitting process.

2 Project Description

2.1 Project Location and Access

Mali is divided into eight Regions and one capital district, namely Bamako. These eight Regions are subdivided into 49 Cercles which represent a second level administrative unit in Mali.

Sanankoro is located in the Koulikoro Region which is divided into seven Cercles encompassing 106 rural communes. The project falls within the Kangaba Cercle and spans across the Séléfougou and Maramadougou rural communes. A total of six rural communities/villages are located within the project area, namely: Séléfougou, Sanankoro, Bokoro (hamlet), Sélin, Faragouagnan and Kignèlen (hamlet).

The Boucle du Baoulé National Park as well as several nature reserves including Fina, Kongossambougou and Badinko are located in the Koulikoro Region. The Fié and Niger Rivers are located west and north of the project area respectively.

The Region of Koulikoro is bordered by Mauritania on the north and by Guinea to the south. The capital city, Bamako, is located approximately 110 km northwest of the project area. From Bamako, Sanankoro is accessible via paved road to either the Sélingué (National Road 27) or Kangaba (National Road 16) communes and then various undeveloped road options are available to the project site.

The regional and local setting of the project area are shown in Plan 1 and Plan 2 respectively below.

Plan 1: Regional Setting

Plan 2: Local Setting

2.2 Mining Method and Associated Infrastructure

Through progressive exploration activities, the resource estimate will be determined to inform a detailed mine plan for the project. At this stage, Cora Gold only intends to exploit the oxide ore with the potential of expanding this to sulphide treatment at a later stage if required. The ore is planned to be mined via conventional open-pit mining methods.

It is expected that the project will consist of several open pits along the Zone A and Sélin vertical structures, along with ancillary mining infrastructure, including:

- Waste Rock Dumps (WRDs);
- Tailings Storage Facility (TSF);
- Run of Mine (ROM) pad;
- Gold ore processing plant;
- Administration offices and warehouses;
- Mine camp (existing); and
- Power generation facility.

The potential processing of the oxide ore would be either through heap leach (1 million tonnes per annum) or gravity and Cyanide-In-Leach (CIL) methods (0,5 million tonnes per annum). Interim metallurgical results indicated up to 97% gold recovery.

In addition to this, there is also the option of treating the substantial amount of residue tailings material within the project area which have been generated by artisanal mining activity and deposited along the local river courses. It is estimated that this could amount to up to 1.5 million tonnes.

Plan 3 depicts a preliminary mine layout plan which indicates locations of potential pit areas, TSF and processing plant (inclusive of administrative infrastructure). It is noted that the mine layout is subject to refinement as more information on the resource estimate, mine plan and site sensitivities becomes available.

Plan 3: Preliminary Infrastructure Layout

3 Methodology

The Scoping Study was undertaken through a desktop review and infield assessment to the project area. The methodology implemented is detailed in the subsections below.

3.1 Desktop Assessment

A desktop review was undertaken to broadly determine the biophysical and social environment of the project area. In addition, applicable laws and permitting processes were reviewed to inform the ToR provided in Section 7 below and ensure that an understanding of the compliance requirements is achieved.

The desktop review included information provided by Cora Gold, namely: Environmental Study of the Sanankoro Permit Wildlife and Flora Inventory (2017), Preliminary Socio-Economic Study of the Influence Zone of the Sanankoro Permit (July, 2019), Drill Results at Depth at Sanankoro (August, 2019) and provisional mine layouts. In addition, secondary/publically accessible information such as aerial imagery, environmental and social research reports relevant to the project region as well as government documents including national census data was reviewed to inform the assessment.

The desktop review aimed to obtain a characterisation of the following environmental aspects:

- Abiotic:
  - Climate;
  - Surface water;
  - Groundwater; and
  - Soils and land use.
- Biotic:
  - Fauna and Flora;
  - Wetlands; and
  - Aquatic Ecosystems.
- Social and Cultural:
  - Socio-economic;
  - Heritage and archaeology; and
  - Determine the potential need for resettlement.

3.2 Infield Assessment

Following the desktop review, Digby Wells undertook an infield assessment to the project area between 15 July and 19 July 2019. The visit was conducted by a social specialist and an environmental generalist consultant from Digby Wells' Bamako office. The infield assessment served to orientate the project team and to gather additional information as well as ground-truth existing biophysical and socio-economic environmental information.

From a biophysical perspective, the potential pit areas (along the Zone A and Sélin structures and their immediate surrounds were surveyed to identify high-level vegetation habitats, animals, land uses and existing water resources among others. The existing artisanal mining sites within the project area were also surveyed. Consultations with the accompanying Cora Gold personnel as well traditional authorities from surrounding communities was undertaken to incorporate local knowledge and experiences of the biophysical characteristics into the assessment.

From a socio-economic perspective, initial consultation with the accompanying Cora Gold representative and several local administrative authorities were undertaken. No direct consultation with communities was undertaken to avoid raising expectations during this early stage of the project. Primary data collection focussed on the six communities within Sanankoro which are likely to be directly influenced by the project. These villages are Séléfougou, Sanankoro, Bokoro (hamlet), Sélin, Faragouagnan and Kignèlen. Individual discussions with the village authorities of each of these localities were undertaken as part of the socio-economic assessment. In addition, a meeting with the Séléfougou Commune Authority was conducted during the infield assessment.

The infield assessment further aided in identifying/confirming issues and risks associated with the project. These risks were determined based on environmental sensitivities identified, the socio-economic characterisation of the project area, the potential location of the pits and the anticipated project activities.

4 Biophysical Characterisation

The subsections below provide a summary of the environmental characterisation of the project area.

4.1 Regional Climate

The project area is located within the Sudano-Guinean zone which experiences hot summers and mild winters. According to the Köppen and Geiger classification, the Koulikoro region is classified as *Aw.* savannah climate which is characterised by annual average temperatures ranging from 24 Degrees Celsius (°C) to 30 °C. The maximum temperatures are recorded during March and April and the lowest temperatures experienced during December.

During the rainy season (May to October), the average rainfall varies between 1,100 millimetres (mm) and 1,400 mm while practically no rainfall is experienced during the dry season (November to April).

The table below summarises the average annual climatic information obtained for the Kangaba Cercle

Table 4-1: Average Temperature and Rainfall for Kangaba Cercle

	January	February	March	April	May	June	July	August	September	October	November	December
Temperature (°C)												
Average	24.2	27.5	30	30.9	30.9	28.4	26.7	25.6	26.3	27.2	26.5	24.5
Average Minimum	14.6	18.1	21.3	23.1	23.7	21.9	21.3	20.5	20.6	20.4	17.9	15.1
Average Maximum	33.8	37	38.7	38.8	38.2	34.9	32.1	30.8	32	34.1	35.1	34
Precipitation (mm)												
Average Rainfall	0	0	5	29	71	155	240	303	214	85	12	0

Source: Climate-Data.org

4.2 Topography

The relief of the project area is characterised as undulating lateritic plateaus with isolated hills throughout the landscape. The Fié and Niger Rivers flood the west and north of the project area respectively. The average altitude of the project area is 350 metres above sea level (mamsl) with a minimum of approximately 321 masml in the slopes and the maximum altitude is about 947 masml on the hills.

The land cover is severely impacted by anthropogenic activities resulting in soil degradation. The overall conditions of the area have however allowed the establishment of abundant vegetation characterised by mosaics of savannahs (shrubs, trees and grasslands) and cleared forest areas. The figures below provide evidence of the landscape present within the project area.

Figure 4-1: Undulating Landscape

Figure 4-2: Savannah Vegetation Cover in the Project Area

4.3 Soils and Land Use

The soil types present within the project area comprise lateritic soils and alluvial soils. Isolated hills associated with rock outcrops are also present throughout the project area. Lateritic soils were identified along the plateaus are common to tropical and subtropical climates and are typically rich in iron and/or aluminium oxides. Soils of colluvial and alluvial origin (loamy and sandy-clay soil types) are present along floodplains, valleys and depressions found within the project area. Alluvial soils are general characterised by high silt content and their fertility is characterised by the gradual deposition of plant debris and sediments on their surface.

In terms of land use, the primary land uses comprise agriculture and Artisanal and Small Scale Mining (ASM) activities. The project area is covered by arable land and is occupied by cultivated fields, fallow land and areas used for grazing which forms a diverse mosaic of cleared rural areas. These land uses are commonly found along the river floodplains toward the west and north of the project area and surrounding slopes (Figure 4-3). ASM is a key land use within the project area identified towards the south and centre of the project area. Notably, this has also resulted in increasing migration into the project area which places further pressure on the ecosystem. Secondary land uses include forest logging (collection of firewood, medicinal plants and wood to make canoes and furniture), hunting and fishing particularly in the Fié River.

Figure 4-3: Evidence of Grazing along the Fié River

4.4 Terrestrial Biodiversity

Several biodiversity inventories have been carried out for the Sanankoro area. Most recently, a Biodiversity Inventory Study was undertaken during 2017 by the Environment and Social Development Company (ESDCO) – SARL. The objective of this study was to provide an identification and description of the habitats present in the project area as well as provide an assessment of the conservation status of biodiversity. This study, together with the infield assessment carried out by Digby Wells was utilised in establishing the baseline environment in terms of biodiversity for the project area.

4.4.1 Floral Characteristics

4.4.1.1 Regional Vegetation

Mali is divided into five main ecosystems which present a wide range of agro-ecological environments which are further divided into 14 regions. The project area is located within the Upper Bani Niger Region (specifically the High Bani Niger) which is situated in the moist/humid Guinean zone. This region is bordered by the Mandingue Plateau to the north, by the Koutiala in the north east, and by Plateau de Foniokoulou, which continues into Guinea, to the east (United States Agency of International Development (USAID), 2008).

This sub-humid zone constitutes 6% of the country’s remaining forests and is considered an important production site comprised of wooded savannah and natural forest. In 2008, USAID reported that between 40% to 90% of the ground has grassland vegetation cover, with gallery forests in valleys creating continuous bands of dense vegetation.

Although vegetation is parts of the zone are well conserved, pockets of degradation do exist specifically in the High Bani Niger and southern part of the Mandungue Plateau. This is mainly as a result of ASM sites such as Siama, Finkolo and Fabouloa.

4.4.1.2 Site-Specific Vegetation Composition

The vegetation of the project area consists of several types of formations associated with the savannah and forest vegetation types. Savannah dominates the project area and is characterised by the presence of herbaceous layer and woody species including *Ostrioderich chevaleri, Pterocarpus sp Combretum sp., Vitex sp.,* and *Landolphia sp.* The forest vegetation is characterised by a low presence or absence of grassy vegetation and dominated by the presence of large trees.

These vegetation types are divided into the following distinctive formations within the project area (ESDCO, 2017):

- **Grassland Savannah** – this vegetation is found in the northeast and southeast of the project area and is characterised by the presence of grasses on shallow soils;
- **Shrub Savannah** – this vegetation is found in the southeast of the project area and is characterised by herbaceous plants which are under 7 m high. Dominant species include *Terminalia macroptera* and *Pterocarpus erinaceus*;
- **Wooded Savannah** – this vegetation type is found in the northwest and southwest of the project area and is characterised by trees over 7 m high with a less dense herbaceous layer. Dominant woody species include *Parinari Excelsa*, *Erythrophleum Guinense*, *Parkia Biglobosa*, *Isoberlia Doka* and *Daniela Oliveri*;
- **Bowé Areas** – Bowe represent degraded land where ferricrete is exposed and is found throughout the project area across the different vegetation types. These areas hardly bear woody vegetation due to the near-absence of a soil layer and are commonly associated with ASM sites;
- **Forest Galleries** – this vegetation type is found in the in the west of the project area along the rivers and is characterised by deciduous and/or evergreen species. Dominant species include *Daniela Oliveri*, *Mytragina Inermis*, *Uapaca Somon*, *Pterocarpus erinaceus*, *Cola cordifolia*, and *Oxytenantera Abyssinica*; and
- **Clear Forests** – this vegetation type is found in the south of the project area and is confined to the sparsely populated areas of the Fié River basin. Dominant species include *Antiaris Africana*, *Khaya Senegalensis*, *Afzelia Africana* and *Combretum sp*. ;

The natural habitat is also largely disturbed by agricultural land and residential areas. Cultivation and grazing activities are present along the floodplains and slopes, contributing to pressure of the forest galleries. In addition, artisanal mining continues to expand throughout the project area, contributing to the formation of bowé areas of the savannah habitats. Other activities resulting in the exploitation of forest resources include logging (for carpentry and firewood) and collection of medicinal plants as well as picking products, beekeeping and sourcing charcoal.

The figures below depict some transformed land as a result of agricultural and artisanal mining activities.

Figure 4-4: Grazing alone the Fié River Floodplain

Figure 4-5: Cleared Forest for Artisanal Mining Activities

Source: ESDCO, 2017

During the floristic inventory carried out by ESDCO (2017), a total of 79 plant species were recorded. The most common species included: *Combretum glutinosum, Pterocarpus erinaceus, Acacia macrostachya, Isoberlinia doka, Detaruim microcarpum, Entada africana, Lannea acida, Lannea microcarpa, Daniellia oliveri* and *Terminalia laxiflora*.

During Digby Wells' infield assessment in July 2019, a total of three red data species were recorded in the project area as detailed in the table below. It is noted that the surveyed areas, namely along the Zone A and Sélin structures, is largely degraded as a result of artisanal mining. The red data species occurred and are expected to occur in more natural areas within the project area.

Table 4-2: Red data species recorded at Sanankoro (Digby Wells, 2019)

Scientific Name	Habitats	Threats
Afzelia africana	Forest galleries	Exploitation, partly for the international market
Khaya senegalensis	Savannah and forest galleries	Logging, with local exploitation largely uncontrolled and poorly supervised.
Pavetta lasioclada	Forest galleries	Threatened by agricultural practices, logging for firewood and clearing for artisanal mining.

Table 4-3 provides a list of the fully protected plant species in terms of Decree No. 10-387/P-RM of 26 July 2010 which are expected to occur in the project area.

Table 4-3: Lists of fully protected species

Scientific Names	Bambara Names
Parkia biglobosa	*Nere*
Vitellaria paradoxum	*Shi*
Cordyla pinnata	*Dugura*
Cordyla pinnata	*Dugura*
Acacia Senegal	*dɔnkɔri*
Detaruim microcarpum	*Tabacoumba*
Tamarindus indica	*N'Tomi*
Acacia albida	*Balanzan*
Cordila pinnata	*Dougoura*

4.4.2 Faunal Characteristics

4.4.2.1 Regional Fauna

Mali is characterised by its faunal biodiversity throughout the country's various climatic zones. Within the Upper Bani Niger, abundant water resources create diverse habitats for a variety of mammals, rodents, reptiles and avifauna. Illegal trapping of small fauna populations and hunting of large mammals has greatly affected availability of species in the region (USAID, 2008). Abundant animal species include hippopotamus, python, baboon, and green and red monkeys. Other reptiles and rodents which are known to occur include cobra, green Mamba, ground squirrel, and Gambian rat. Rare species include antelopes, Grand Calao/hornbill, crocodile, vulture, common jackal, turtle, and tortoises. Lion has also historically been associated with the region, but is considered unlikely. The table below provides a list of key declining species in the region.

Table 4-4: Declining Species in the Region

Scientific Name	Common Name	Status
Panthera Leo	Lion	Rare
Crocodylus	Crocodile	Rare
Canis Aureus	Common jackal	Rare
Orycteropus Afer	Aardvark	Rare
Mellivora Capensis	Ratels/badger	Rare
Galago Senegalensis	Bush baby	Rare
Gyps Africanus	White-backed vulture	Rare
Testudines	Turtle	Rare
Testudinidae	Tortoise	Rare
Kobus Ellipsiprymmus	Defassa waterbuck	Threatened
Pamthera Pardus	Leopard	Near Extinction
Trichechus senegalensis	Manatee	Near Extinction
Pan Troglodytes	Chimpanzee	Near Extinction

Alcelaphus Bubalis	Hartebeest	Near Extinction
Manis Gigantea	Giant pangolin/reptile	Near Extinction
Potamochoerus Porcus	Bush pig	Near Extinction
bird species		
Strigiformes	Owl	Threatened
H. Vocifer	*African fish eagle*	Threatened
Bucorvus	Ground hornbill/grand calao	Rare
Gyps rueppellii	Vulture	Rare
Falco Peregrinus	Peregrine falcon	Near Extinction

Source: USAID, 2008

4.4.2.2 Site-Specific Fauna

The wildlife inventory carried out by ESDCO (2017) recorded a total of 673 faunal species within the project area, as detailed in the table below.

Table 4-5: Recorded Faunal Species at Sanankoro (ESDCO, 2017)

Scientific Name	Family	Common name	No.
Mammals			
Xerus Erythropus	Sciuridae	Striped ground squirrel	03
Heliosciurus Gambianus	Sciuridae	Gambian sun squirrel	01
Erythrocebus Patas	Hominids	Patas monkey	05
Atelerix Albiventris	Erinaceaidae	Four-toed hedgehog	01
Epomophorus Gambianus	Pteropodidae	Gambian Epauletted Fruit Bat	50
Lepus Victoriae	Leporidae	African Savanna Hare	01
Birds			
Tockus Erythrorhynchus	Bucerotidae	Red-billed hornbill	106
Lophoceros nasutus	Bucerotidae	African Grey Hornbill	60
Centropus Senegalensis	Cuculidae	Senegal coucal	03
Streptopelia Senegalensis	Columbidae	Laughing Dove	85
Ptilopachus Petrosus	Odontophoridae	Stone Partridge	01
Crinifer Piscator	Musophagids	Western plantain-eater	01
Coracias Abyssinica	Coraciidae	Abyssinian roller	04
Treron Waalia	Columbidae	Bruce's green pigeon	04
Streptopelia Semitorquata	Columbines	Red-eyed dove	20
Oena Capensis	Columbidae	Namaqua dove	02
Lagonosticta Senegalla	Estrildidae	Red-billed firefinch	01
Lamprotornis Caudatus	Sturnidae	Long-tailed glossy starling	70
Francolinus Bicalcaratus	Phasianidae	Double-spurred francolin	04
Numida Meleagris	Numididae	Helmeted guineafowl	20
Passer Griseus	Passerines	Northern grey-headed sparrow	50
Ardea Garzetta	Ardeidae	Little egret	100
Poicephalus Senegalus	Psittacidae	Senegal parrot	53
Lamprotornis Chalybeus	Sturnidae	Greater blue-eared starling	26
Reptiles			

Scientific Name	Family	Common name	No.
Agama Cristata	Agamidae	Insular agama	01
Invertebrates			
Apis Mellifera	Adansonii	Western honey bee	01

It is noted that majority of farmers in the area have hunting as their secondary activity. This demonstrates the anthropogenic pressure on the diversity of wildlife in the project area.

4.5 Presence of Wetlands

Several wetland systems exist within the project area. These wetland areas are predominantly associated with floodplains on the banks of the Niger and Fié Rivers as well as various other non-perennial tributary streams which run throughout the project area. The wetlands are flooded during the rainy season and consequently the saturation of wetland soils results in the presence of hydrophytic plants. Wetland areas are commonly used for agricultural (rice cultivation) as well as gold processing activities.

Table 4-6 provides a description of the wetlands which were encountered during the infield assessment. These wetlands are shown in Plan 4. It is noted that the infield investigations focussed on the potential pit areas and their immediate surroundings only and to confirm the presence of wetland systems. More wetlands, in areas such as the possible TSF location, may be directly or indirectly affected by the development are expected to occur.

Table 4-6: Identified Wetlands

Wetlands	Co-ordinates		Site Observations	Photographic Evidence
	North	West		
Wetland 1	11.738848	-8.458386	Wetland (depression) located in the east of the project area (downstream of the potential TSF and plant area). No anthropogenic pressure was observed to the wetland.	
Wetland 2	11.797901	-8.488515	A floodplain wetland identified along the north-western boundary of the project area along the banks of the Niger River. This floodplain is utilised by the surrounding communities to cultivate paddy rice.	

Wetlands	Co-ordinates		Site Observations	Photographic Evidence
	North	West		
Wetland 3	11.789321	-8.492404	A floodplain wetland identified along the north-western boundary of the project area along the banks of the Fié River. No agricultural anthropogenic activities were identified.	
Wetland 4	11.778107	-8.483781	A floodplain wetland identified along the north-western boundary of the project area along the banks of the Fié River (further downstream of Wetland 3). This wetland serves as a refuge for birds and a pasture for animals.	

Plan 4: Identified Wetland Areas

4.6 Surface Water

4.6.1 Hydrological Characteristics

The hydrological setting of the project area is illustrated in Plan 5 below. The project area intersects with the Niger River System, with its main tributaries Fié River and Niger River located in proximity to the potential development footprints. Lake Sélingué, an artificial lake utilised for hydroelectric production and commercial fishing, is located immediately south of the project area on the Fié River.

The Fié River (Figure 4-6) traverses the western boundary of the project area and is approximately 1,5 km from the potential pit areas while the Niger is located approximately 800 m outside of the northern boundary of the project area. The Fié River drains into the Niger River which ultimately joins the Niger River approximately 40 km upstream of Bamako. The ASM activities for the project area are associated mainly with the Fié River. Various non-perennial streams including Talétou Kô and Bokoro Kô also traverse the project area, draining into the Fié River.

Figure 4-6: The Fié River to the west of the Sanankoro permit

Plan 5: Hydrological Setting

4.6.2 Water Uses

Surface water uses in the project area comprise potable domestic use, market gardening and animal watering and ASM activities. Fishing is also extensively practiced on the Niger and Fié Rivers.

Generally, perennial streams are utilised for ASM activities during the dry season and consequently some drainage lines have been permanently destructed and considerable amounts of residue tailings material have been deposited along local river courses. Old quarries were also identified in the project area (Figure 4-7) which are utilised for clean stormwater storage ponds to serve artisanal mining activities during the rainy season. Sedimentation contributes to clogging or damming up which leads to drying of downstream areas.

Figure 4-7: Old ASM Quarry used as a Rainwater Storage Pond

4.6.3 Water Quality

Cora Gold appointed the National Water Laboratory, Mali to undertake a water quality analysis in January 2019. A total of three surface water samples were collected as detailed in the table below as well as depicted on Plan 6.

Table 4-7: Surface Water Sampling Locations

Label	Location	Co-ordinates	
		Easting	Northing
SW1	Niger River (Downstream)	558432	1307134
SW2	Fié River	555449	1303122
SW3	Niger River (upstream)	554362	1305926

The water quality results were compared to the Guidelines for Drinking Water Quality Standards published by the World Health Organisation (2017) to provide an indication of the level of pollution, as detailed in Table 4-8 below. With the exception of Chromium in all three samples, the laboratory results were found to be within acceptable limits for drinking water. It is noted that, according to ENSEEIHT (2018), BOD5 for water intended for human consumption is acceptable at a concentration of 0 milligram per litre (mg/l) and domestic waste water at 1.5 mg/l to 2 mg/l. Generally concentrations of a few mg/l for both BOD5 and COD (which is always greater than BOD5) represent good quality surface water. These concentrations are exceeded for all samples upstream and downstream of the project area. It is further noted that Cyanide concentrations (which are of concern with respect to the ASM activities) are well below 0.05 mg/l in all three samples collected.

Table 4-8: Surface Water Quality Results

Parameter	Units	WHO 2017 Drinking Water Quality Guideline Limits	Sample Results		
			SW1	SW2	SW3
Colour (TCU)	-	-	51	257	52
Turbidity (NTU)	-	-	6	29	4
Calcium (Ca2+)	mg/l	-	2.85	1.18	3.25
Magnesium (Mg^{2+})	mg/l	-	1.03	0.94	1.02
Sodium (Na+)	mg/l	-	1.17	1.92	3.32
Potassium (K +)	mg/l	-	0.157	0.48	1.27
Iron (Fe 2+)	mg/l	-	0.157	0.0840	0.143
Dissolved Oxygen (O_2)	mg/l	-	7,97	7.37	8.02
% saturation in O_2	mg/l	-	96.4	91	103.7
BOD5	mg/l	-	120	240	170
COD	mg/l	-	292	567	422
Lead (Pb)	mg/l	0.01	0.009	0.00	0.00
pH	-	-	7.46	6.5	7.15
Conductivity (25°C)	µS/cm	-	44	21	41
Hardness ($CaCO_3$)	mg/l	-	11.36	6.82	12
Alkalinity ($CaCO_3$)	mg/l	-	13.90	12.54	14
Bicarbonates (HCO_3^-)	mg/l	-	16.97	15.30	17.54
Sulphates (SO_4^{2-})	mg/l	-	2.62	0.93	1.96
Chloride (Cl-)	mg/l	-	2.15	0.37	1.18
Manganese (Mn)	mg/l	-	0.087	0.133	0.123
Copper (Cu^{2+})	mg/l	2	0.0	0.0	0.0
Ammonia ($NH+_4$)	mg/l	-	0.11	0.54	0.16
Nitrogen Dioxide (No_2)	mg/l	-	0.003	0.005	0.011
Cyanide (CN-)	mg/l	-	0.002	0.007	0.007
Arsenic (As)	mg/l	0.01	0.00	0.00	0.00
Nickel (Ni)	mg/l	0.07	0.0	0.0	0.0
Chromium (Cr6+)	mg/l	0.05	**0.06**	**0.23**	**0.26**
Total Dissolved Solids (105°C)	mg/l	-	60.06	28.66	55.96
Nitrate ($NO3^-$)	mg/l	50	0.92	0.17	0.93
Fluoride (F^-)	mg/l	1.5	0.219	0.220	0.253
Zinc (Zn)	mg/l	-	0.07	0.05	0.73
Orthophosphate	mg/l	-	<0.001	<0.001	<0.001

Source: National Water Laboratory Mali, 2019

Plan 6: Water Quality Sampling Points

4.7 Groundwater

The regime of the Niger River System is strongly influenced by groundwater flows. In general, groundwater is influenced by annual precipitation and permeability of soils.

Within the project area, groundwater is an important source of drinking water to the surrounding communities who report generally good quality of the water (based on its aesthetic characteristics). An inventory of groundwater source points was carried out during 2018 as detailed in Table 4-9 below and depicted in Plan 7. These sources comprise drilled boreholes as well as tradition and large-diameter wells. Water sources can be public or private depending on the locality. Groundwater sources are unequally distributed throughout the project area. Generally these sources are publically accessible from the support of technical and/or financial partners to provide communities with drinking water. This is with the exception of the drilled boreholes identified in the Bokoro hamlet which are privately owned by individuals.

Table 4-9: Location of Groundwater Sources

Location	Description of Source	Co-ordinates	
		X	Y
Sanankoro	Well	562390	1296455
	Drilled borehole	562655	1296364
	Drilled borehole	552686	1296563
Bokoro	Drilled borehole	557915	1299251
	Drilled borehole	558014	1299012
	Drilled borehole	557810	1298314
	Drilled borehole	557635	1297906
	Drilled borehole	557767	1298259
Sélin	Drilled borehole	558838	1307235
	Well	558656	1307046
	Well	558648	1307141
Faragouagnan	Drilled borehole	556037	1303722
Kignèlen	Drilled borehole	558927	1305910
	Well	558955	1305970
	Well	558965	1305983
	Well	558916	1305978

Plan 7: Groundwater Sources in the Project Area

4.7.1 Groundwater Quality

A total of three groundwater samples were collected for water quality analysis during January 2019. The table below provides details of the sampling points which are also depicted on Plan 6 above.

Permit	Location	UTM	
		Easting	Northing
GW1	Sanankoro Public School	562689	1296567
GW2	Sanankoro Nursery	562653	1296366
GW3	Cora Gold Camp	559815	1296872

The groundwater quality results were compared to the Malian National Standard for Drinking Water to provide an indication of the level of pollution, as detailed in Table 4-10 below. Generally, the laboratory results from the sampled boreholes indicate that groundwater is of acceptable quality. This was further confirmed by community members interviewed during the infield assessment. Notably, the physiochemical water quality at GW2 (Sanankoro Nursery) was deemed poor as it is slightly mineralised, soft and coloured. This was also observed in the sample collect at the Cora Gold camp (GW3) borehole. Cyanide concentrations (which are of concern with respect to the ASM activities) are well below 0.07 mg/l in all three samples collected.

Table 4-10: Groundwater Quality Results

Parameter	Units	Mali Drinking Water Standard	Sample Results		
			GW1	GW2	GW3
Colour (TCU)	-	25	0	**35**	0
Turbidity (NTU)	-	10	0	10	0
Calcium (Ca2+)	mg/l	400	16.10	8.40	3.01
Magnesium (Mg^{2+})	mg/l	100	6.48	2.34	1.33
Sodium (Na+)	mg/l	400	25.02	15.91	4.32
Potassium (K +)	mg/l	100	3.87	3.77	1.67
Iron (Fe 2+)	mg/l	0.3	0.015	0.014	0.014
Lead (Pb)	mg/l	0.01	0.0	0.0	0.0
pH	-	5.5≤ - ≤9	7.10	6.75	6.22
Conductivity (25°C)	µS/cm	1500	251	140	49
Hardness ($CaCO_3$)	mg/l	500	67	31	13
Alkalinity ($CaCO_3$)	mg/l	<150	113	62	17
Bicarbonates (HCO_3^-)	mg/l	-	138.49	75.58	20.53
Sulphates (SO_4^{2-})	mg/l	500	4.68	0.80	7.59
Chloride (Cl-)	mg/l	600	1.89	2.15	1.97
Manganese (Mn)	mg/l	0.5	0.020	0.045	0.09
Copper (Cu^{2+})	mg/l	1	0.0	0.0	0.0

Parameter	Units	Mali Drinking Water Standard	Sample Results		
			GW1	GW2	GW3
Ammonia (NH_4^+)	mg/l	0.5	0.11	0.09	0.06
Nitrogen Dioxide (No_2)	mg/l	0.02	0.002	0.003	0.003
Cyanide (CN^-)	mg/l	0.07	0.003	0.002	0.002
Arsenic (As)	mg/l	0.01	0.0	0.00	0.00
Nickel (Ni)	mg/l	0.02	0.00	0.0	0.0
Chromium (Cr6+)	mg/l	0.05	0.0	**0.21**	0.00
Total Dissolved Solids (105°C)	mg/l	1200	193.16	132.67	66.88
Nitrate ($NO3^-$)	mg/l	500	1.69	1.43	1.13
Fluoride (F^-)		0.5	0.178	0.288	0.250
Zinc (Zn)	mg/l	3	0.02	0.04	0.24
Orthophosphate	mg/l	0.005	<0.01	<0.01	<0.01
Ryznar Index	-	-	9.23	10.60	12.07

5 Socio-Economic Characterisation

The subsections below provide a summary of the socio-economic characterisation of the project area.

5.1 Administrative and Political Structure

Mali is a landlocked country in West Africa which covers an area of 1,240,192 m^2 of which 20,002 m^2 is occupied by water.

The governance system is characterised by Administrative Management (District, Region, Cercle and District) and Decentralised Management (Regional Assemblies, Council of Cercles and Communal Councils).

At Region level, a Governor is appointed as the executive of the Region and a Prefet is appointed to each Cercle. At the district level, sub-prefets who constitute support staff to the Prefet are in place and act as the interface between communes, villages and the Prefet.

Regional Assemblies for each Cercle are appointed at Regional level and their members are elected from the members of the Cercle Councils of the Region. At the level of the Cercles, administrative management is carried out through the Council of Cercles whose elected representatives are selected from the Communal Councils. Lastly, the Commune Council officials are elected from their representing villages.

The administrative and political organisation applicable to the project area is depicted in the figure below.

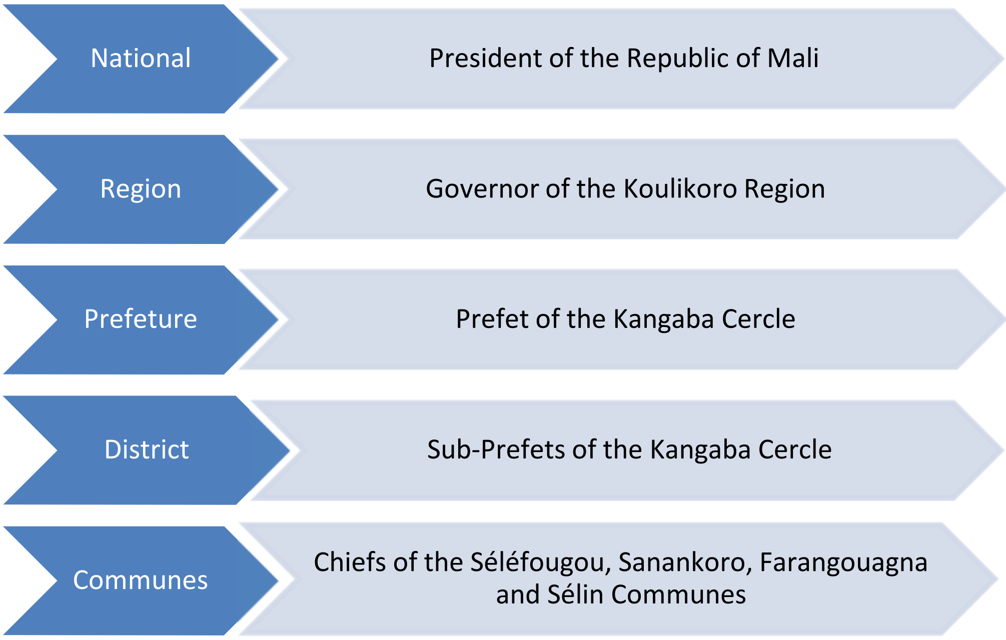

Figure 5-1: Administrative Structure of the Project Area

For the purposed of this socio-economic characterisation, the project is defined as follows:

- Primary Study Area:
  - Séléfougou Commune, comprising
    - Séléfougou Village;
    - Sanankoro Village; and
    - Bokoro Hamlet (belonging to the Séléfougou village).
  - Maramandougou Commune, comprising:
    - Sélin Village;
    - Faragouagnan Village; and
    - Kignèlen Hamlet (belonging to the Figuiratomo village).
- Secondary Study Area:
  - Remaining localities of the Séléfougou and Maramandougou Communes (inclusive of four and nine villages, respectively); and
  - Kangaba Cercle (covering 58 Communes).

The primary study area corresponds with the extent of the project area which, as indicated in Section 2.1, includes the abovementioned six villages which span over two rural communes. These localities are mostly likely to be affected by the project. The secondary study area extends to the reminder of the corresponding rural communes, extendable to the Kangaba Cercle and Koulikoro Region where applicable.

5.2 Demography

5.2.1 Population Size

The population statistics of the primary study area is detailed in the table below (General Census of Population and Housing (GCPH) of Mali, 2009).

Table 5-1: Demographic statistics of the main study area

Rural Commune	Locality	Number of households	Household size	Population		
				Male	Female	Total
Séléfougou	Sanankoro Village	45	9.6	232	198	430
	Séléfougou Village	442	7.2	1552	1621	3173
	Bokoro Hamlet	950	8	4550	3050	7600
Subtotal		**1437**	**8,2**	**6334**	**4869**	**11203**
Maramandougou	Sélin Village	50	8,5	209	216	425
	Kignèlen Hamlet	6	33	93	105	198
	Faragouagnan Village	54	6,7	178	170	348
Subtotal		**110**	**16**	**480**	**491**	**971**
Total		**1547**	**12,1**	**6814**	**5360**	**12 174**

Sources: GCPH (2009) in blue and data gathered in focus group meetings by Digby Wells (2019) in purple.

As shown in the table, the gender distribution indicates a higher dominance of men (55.9%) to women (44.1%) in the primary study area. This is contrary to the secondary study area and national statistics that show a higher number of women to men.

5.2.2 Age Distribution

Information gathered during the focus group meetings indicates that the population within the primary study area is mostly made up of young people (aged between 14-45 years old). This is followed by the group aged 46-59, while the age groups 0-13 and those aged above 60 make up the smallest part of the population (Figure 5-2). This age distribution is consistent with the second study area and country generally.

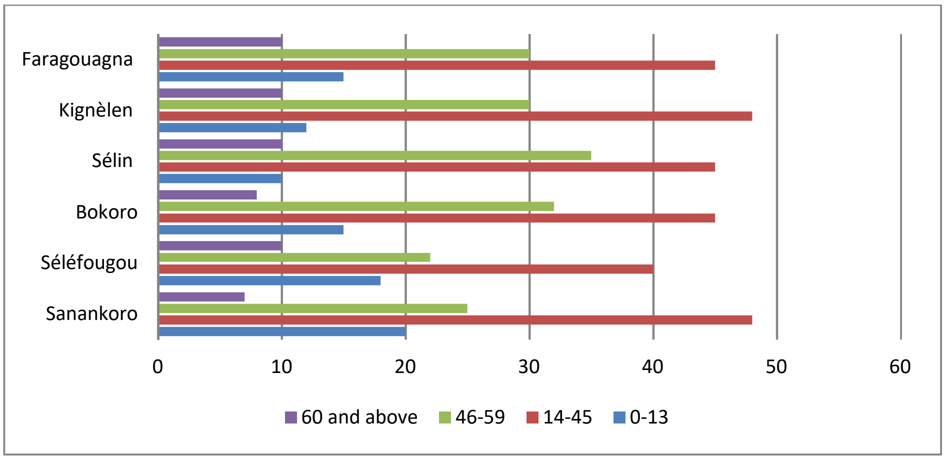

Figure 5-2: Age Distribution of the Primary Study Area

Source: Digby Wells, 2019 (estimation provided by village authorities)

5.2.3 Population Composition

The population of the primary study area is mainly composed of both indigenous people and Malian nationals of other regions. The presence of several African nationalities was also reported during focus group meetings. This was mainly attributed to artisanal mining opportunities present within the project area.

Migration to the project area was said to have accelerated from the year 2012 and is mainly centred around the Bokoro Hamlet where the largest ASM site in the project area (also recorded as one of the most productive in Mali) is found. To this end, Bokoro is the main host community for newcomers (national and foreign nationalities) in the region. It is estimated that newcomers make up over 90% of the population within Bokoro. The foreign nationalities whose presence has been reported in Bokoro mainly comprise people from Guinea, Burkina Faso, Nigerian, Cameroon and Senegal.

It should be noted that increasing migration into the Koulikoro Region has also been attributed to the political unrest experienced in the northern and central parts of Mali.

5.2.4 Ethnic Groups and Spoken Languages

With respect to the indigenous population, the predominant ethic group within the primary study area is Malinké (around 90%). The Fulani, Bambara, Bozos and Somonos together make up the remaining 10%. Due to ASM activity, ethnic groups in Mali have generally diversified overtime and are represented by several groups including Malinké, Bambara, Dogo, Senufo, Peulh, Miniyanka, Sonrhai, Tamasheq, Samoko, Bobo and Kassonke among others. In addition to ethnic groups in Mali, some ethnic groups from neighbouring countries are also represented, including: Mossis, Dioulas, Djermas, Haoussas, Wolofs etc.

In terms of languages, the most spoken language in the primary study area is Malinké, followed by Bambara and Fulani.

5.2.5 Religion

Religious practice in the primary study area is dominated by Islam (around 98%) with traditional religion and Christianity making up the remaining 2%.

5.3 Socio-economic Activities

The main socio-economic activities of the primary study area are agriculture, artisanal mining, livestock farming, fishing and small commercial trade. ASM is practiced along ore structures throughout the project area while agriculture is practiced within and around all villages which have expanded in proximity to the ASM sites due to the growth of the activity. As a result some socio-economic activity within the project area occurs directly or within 500 m of the proposed project activities.

Based on information gathered during the focus group meetings, the percentage of the economically active population per economic activity is presented in Figure 5-3. The percentage of local income derived per economic activity is presented in Figure 5-4.

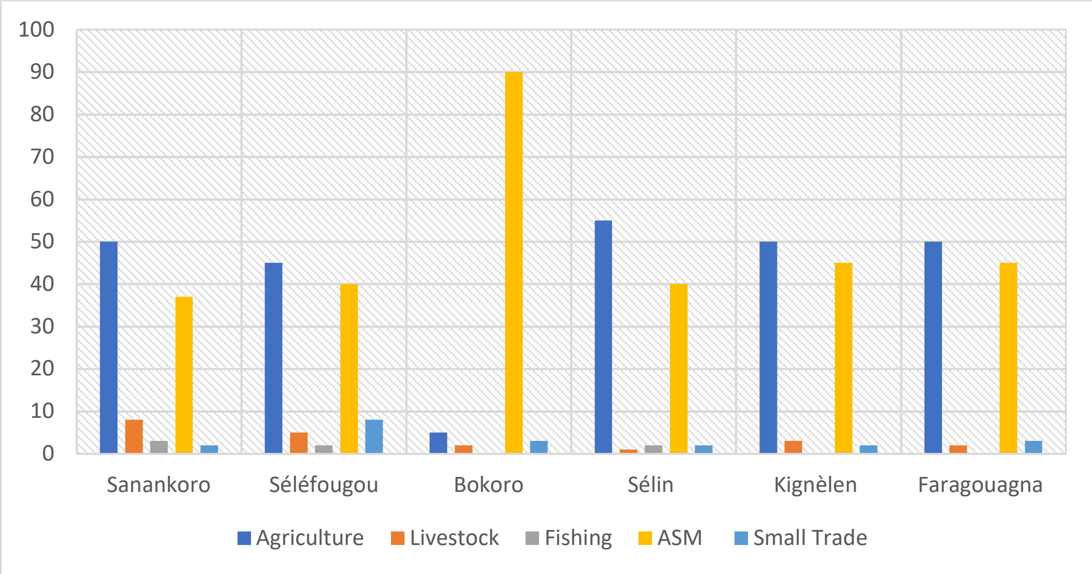

Figure 5-3: Percentage of Active Population by Economic Activity

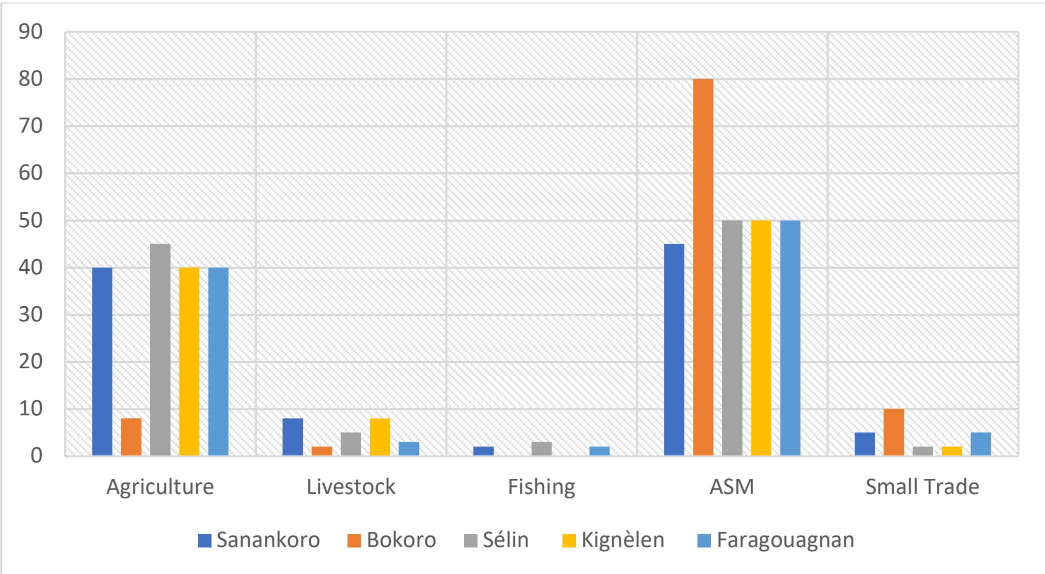

Figure 5-4: Percentage Income of Localities in the Project Area

Agriculture and artisanal mining are the most widely practised economic activities in the project area. On average artisanal mining accounts for 55% of the income derived in the project area, followed by agriculture below 35%. Livestock, small commercial trade and fishing contribute marginally to economic income at 5%, and 4.8% and 1.4% respectively.

These activities are discussed in further detail below.

5.3.1 Artisanal Mining

ASM is much more common in the southwestern part of Mali, particularly in the administrative Regions of Kayes, Koulikoro and Sikasso. The Kangaba Cercle is home to several mining sites, both operational and decommissioned. Notably, no active ASM sites are currently present in the Maramandougou Commune, however it is understood that many of its inhabitants practice the activity in adjacent communes. Through the Bokoro Hamlet, the Séléfougou Commune is home to one of the largest ASM sites in Mali.

In the project area, ASM is considered an essential activity to sustain livelihoods and widely practiced as a main or secondary activity throughout. Income generated through ASM is used for food, health care services, education, equipment and household infrastructure as well as to support secondary agricultural practices (i.e. purchase of seeds/fertilisers, livestock and equipment, supplement income of workers etc.).

Due to the continuous expansion of artisanal mining in the project area, growing environmental pressures are evident. This includes threats associated with deforestation, loss of plant cover and soil as well as pollution of the surrounding water resources which are utilised for gold processing activities. Another emerging consequence of artisanal mining which was noted is its effect on education. Young boys (aged between 14-18 years old) are said to often leave school in favour of ASM. Furthermore, younger populations are abandoning agricultural activities to participate In ASM.

Figure 5-5: ASM site in Bokoro

The ASM site in Bokoro is semi-mechanised and the use of modern equipment such as metal detectors, water pumps, spitters, transport trucks etc. is present. The use of traditional equipment such as pickers and shovels are however still common throughout the region.

5.3.2 Agriculture

Consistent with all Cercles in Mali, agriculture is extensively practiced in the Kangaba Cercle and associated communes and is most commonly subsistence agricultural practice. The main crops cultivated in the project area include sorghum, millet, maize, rice (with wetlands used to cultivate paddy rice), cowpeas, peanuts, cassava and sweet potato. Cotton was identified as the only crop cultivated for commercial markets within the Cercle and constitutes a significant source of income. It was noted however, that this is said to be declining in the last few years.

Yields per hectare remain relatively modest, but most households are able to meet their food needs or supplemented from local markets. The total area of dry crop and cash crop cultivation (including maize, millet and rice) is estimated at 497 ha of the project area and with an average yield of 825 kg/ha and 480 kg/ha for groundnut and cowpeas.

Market gardening and arboriculture are poorly developed in the Kangaba Cercle, particularly in the primary study area. Limited market gardening was identified at water points or along the banks of the Niger and Fiè Rivers. The area reserved for market gardening activities in the project area is estimated at 17 ha while arboriculture was estimated to be practiced on approximately 12 ha of land in the project area.

Traditional equipment and methods are still commonly used to cultivate fields. Only one tractor was identified for agricultural practice in the Bokoro Hamlet (Figure 5-6). According to the village authorities, and as mentioned above, artisanal mining is a source of financing for agriculture (buying equipment, fertilizer, seeds, etc.).

Figure 5-6: Traditional and Modern Ploughing observed in the Project Area

The following main constraints were identified for the agricultural sector in the project area:

- Limited equipment to expand practices;
- Insufficient access to fertilizers, other chemical inputs and adapted seeds;
- Abandonment of agriculture for artisanal mining;
- Increased land utilisation for artisanal mining; and
- Lack of technical and financial partnerships.

The following main strengths were identified for the agricultural sector in the project area:

- Support of State technical services;
- Availability of arable land and irrigable plains;
- Proximity and availability of water resources throughout the project area; and
- Availability to seed of appropriate quality.

5.3.3 Livestock Breeding

Livestock breeding is extensive throughout the Kangaba Cercle and significantly contributes to food security (meat, milk, eggs). Livestock breeding in the project area is estimated at 1,500 cattle; 770 sheep; 775 goats and 1,900 poultry. There is a transhumance (seasonal movement of livestock) corridor within the project area between the banks of the Fié River and the Bokoro Hamlet.

Figure 5-7: Grazing Livestock in the Project Area

The following main constraints were identified for the livestock breeding in the project area:

- Difficulties accessing vaccines and other quality veterinary services;
- Insufficient equipment and livestock infrastructure;
- Low technical and financial support;
- Increased utilisation of pasture land for artisanal mining;
- Presence of diseases (e.g. bovine pleuropneumonia and symptomatic); and
- Theft of livestock.

The following main strengths were identified for the livestock breeding in the project area:

- Availability of water resources;
- Presence of State technical services; and
- Presence of a transhumance corridor.

5.3.4 Fishing

Fishing is practiced in the Niger River, its tributaries as well as village ponds in the Kangaba Cercle. Fishing is mainly for domestic consumption and for sale in key markets in Kangaba, Bamako, Séléfougou and Figuiratomo. In the project area, fishing appears to be a restricted activity, practiced mainly by Bozos and Somonos fishermen along the Niger and Fiè Rivers. Figure 5-8 below shows the fishing nets and some catches in the Fiè.

Figure 5-8: Fishing nets and some catches in the Fiè

The presence of the Niger and Fiè Rivers as well as their associated tributaries is important for fishing in the project area. The following main constraints were identified for fishing in the project area:

- Potential pollution of the river courses associated with artisanal mining activities[1];
- Insufficient technical assistance; and
- Limited access to fishing equipment.

5.3.5 Trade

Due to the difficulty of accessing the project area (undeveloped transport routes), the level of purchasing power of the people of Kangaba Cercle for commercial activities is limited. Bulk trade activity mainly takes place in Bokoro due to the existence of the ASM activities and consequent greater financial potential. Commercial trade is mostly of agricultural products, manufactured products, condiments, construction equipment and materials, clothing, livestock, poultry, electronic equipment, fuel, spare parts and cosmetics.

5.4 Socio-economic Infrastructure

5.4.1 Access to Education

The management of education in the Kangaba Cercle is under the responsibility of Kati's Academy of Basic Education. In terms of school infrastructure, Sanankoro comprises one first cycle (first to sixth grade) public school in Sanankoro Village and two first cycle community schools in Sélin and Faragouagnan. A private first cycle school has been constructed in Bokoro and two first cycle madrasas (Arabic schools) are also available in Bokoro. There is no second cycle (seventh to ninth grade) in the area, and as a result students are obliged to continue their studies in second cycle schools available in the broader communes/Cercles. The table below details the school infrastructure available in the project area.

[1] Although sampling shows good quality (Section 4.6.3 above), this was a single sampling source and does not discount the potential impacts to water resources.

Table 5-2: School Infrastructure available in the Project Area

School infrastructure	Number of classrooms and type of construction material
Private School of Bokoro	2 straw classrooms
Ançardine of Bokoro (madrasas)	2 straw classrooms
'Dar'Hadis of Boroko (madrasas)	-
Public School of Sanankoro	6 cement and banco classrooms with 3 latrines
Centre for Education for Development (CED) of Sanankoro	-
Faragouagnan Community School	2 classrooms in straw
Sélin Community School	3 PADI straw classrooms

Source: ESDCO, 2019

Apart from Sanankoro Public School, the existing infrastructure was found to be inadequate, in addition to no second cycle schools being available in the area. Excluding the two madrasas, the school infrastructures mentioned in the table above accommodates up to 496 students. A total of 16 teachers serve these schools comprising five for the Public School of Sanankoro, five for the Private School of Bokoro, four for the Sélin Community School and two for the Faragouagnan Community School.

5.4.2 Access to Health

The table below details the health facilities available in the primary study area.

Table 5-3: Health infrastructure statistics in the project area

Village/Hamlet	Rural Communes	Existing health structures			Total
		Community Health Centres	Health Posts	Private Care Office	
Séléfougou	Séléfougou	1			1
Sanankoro			1		1
Bokoro				5	5
Sélin	Maramandougou		1		1
Faragouagnan			0		
Kignèlen			0		
Total		**1**	**2**	**5**	**8**

Source: ESDCO, 2019

A total of eight health facilities are available in the project area. Notably, private care is only available in Bokoro as a result of the presence of ASM activities. Generally it was found that staff from the public facilities are poorly trained and access to the facilities is said to be a challenge particularly during the rainy season.

The common diseases, in order of frequency indicated during the focus group meetings, are malaria, respiratory diseases, ulcers, waterborne diseases, malnutrition in children and some cases of trauma. The following main challenges were identified for health care in the project area:

- Poor access to community health centres and the health posts;
- Insufficient health care staff and resources; and
- General growing insalubrity in the area.

In terms of tertiary level health facilities, the Koulikoro Regional Hospital and/or the Kati Hospital are available to the project area.

5.4.3 Access to drinking water

Access to drinking water is a known challenge to most rural communes in Mali. This is mainly attributed to inadequate and uneven distribution of the hydraulic equipment to access groundwater resources. Drinking water in the project area is sourced from boreholes and wells detailed in Table 4-9 above. It is noted that the all boreholes in Bokoro are privately owned while other boreholes in the remaining villages and hamlet are publically access through the National Solidarity Fund, Mali Plan, Mali-Japanese Cooperation or Italian Financing. Equipment associated with these sources is said to be fully functional (ESDCO, 2019).

5.4.4 Electricity

The rural communes of Mali are not connected to the national electricity network, Energie Du Mali (EDM). Electricity generation sources used by these communities include solar energy (solar panels), generators and batteries.

Access to electricity in the project area is consistent with this reality in the country. This was confirmed during focus group meetings with the village authorities and infield observations. The Malian Agency for the Development of Domestic Energy and Rural Electrification (AMADER) is responsible for the electrification of the two commune capitals (Séléfougou and Figuiratomo) and was commissioned in 2003.

5.4.5 Housing Infrastructure

As in most rural areas in Mali, the main habitat infrastructures in the project area are of two types, namely round huts with straw roofs or square houses with sheet metal roofing. A few rare rectangular or square cement houses were identified in the project area. The photos below show the types of common housing infrastructure in the project area.

As a result of the proximity of villages to existing ASM sites, particularly Bokoro and Sanankoro, some housing infrastructure occurs within a 500 m buffer of the potential pit areas associated with the project. This will result in physical displacement of this housing infrastructure.

Figure 5-9: Dwelling Types in the Project Area

5.4.6 Land Tenure

In terms of land tenure, land in the Republic of Mali belongs to the State. However, communities have a customary right to the use of land. Authorisation to the use of the land is administered thorough the Commune Councils. It was established through the focus group meetings that land disputes between certain localities and / or certain families do exist in the project area.

5.4.7 Sanitation and Waste

No formal sanitation or waste management systems are available in the project area. This is consistent with many rural communes In Mali. Uncontrolled deposits of food and other waste and especially the flow of sewage water in the streets was observed throughout the project area.

5.5 Cultural Heritage

The worship of sacred trees, stones, and demarcated sacred places is commonly practiced in the area. Through the focus group discussions, various areas considered to be of cultural significance were highlighted in the project area. These comprise sacred trees (Sélin, Kignèlen), sacred objects or stones (Sélin), sacred woods " Djagatou " (Sélin, Sanankoro), old cemetery (Sanankoro), and the tomb of saint (Bokoro). Customary authorities and hunting associations are considered custodians of cultural heritage.

The table below provides the localities of scared places encountered during the infield assessment (Plan 8). Although the existence of various places and objects of cultural significance were described by people within the project area, not all locations were shared during the time of the infield assessment.

Table 5-4: Some sacred places and sacred trees of the project area

Objects and sacred trees	North	East
Kignèlen Sacred Tree	11.814604	-8.456083
Sanankoro Cemetery	11.707526	-8.480195
Sanankoro Djakatou	11.708242	-8.480903

Plan 8: Identified Heritage Resources

6 Preliminary Impacts Identification

This chapter aims to provide an overview of the environmental and social impacts that may occur should the project proceed. The potential impacts are discussed below per environmental aspects. The potential impacts were identified based on the high-level understanding of the environmental and socio-economic attributes of the project area as well as the preliminary indication of infrastructure areas (Plan 3 above). It is assumed that the processing plant, ancillary administrative infrastructure and workshops will be located in the same area.

6.1 Identified Potential Impacts

The potential impacts identified for the project as well as potential mitigation types are detailed in the table below.

Table 6-1: Identified Potential Impacts and Mitigation Types

Aspect	Potential Impact Description	Mitigation Type
Biophysical Environment		
Soils, Land Use and Land Capability	The development footprint will be cleared and topsoil will be stripped and stockpiled for the establishment of infrastructure. This is expected for the TSF and associated access routes only as the resource targets as well as areas identified for the plant and other administrative infrastructure have been disturbed by ASM activities. This will result in the exposure of soil resources and consequently could result in the following key impacts: ▪ Alteration of the physical and chemical soil structure resulting in the loss of soil capability and quality (agricultural potential); and ▪ Loss of valuable topsoil and subsoil through wind and water erosion. The use of heavy vehicles can lead to soil compaction which during high rainfall events could lead to sheet runoff, exacerbating soil erosion. Due to ASM activities, communities and associated activities such as subsistence cultivation and livestock grazing have developed/expanded in proximity to the ore targets. The development of the project is therefore expected to result in changes to land use for the establishment of infrastructure and loss of land capability where agricultural practices remain along the ore targets.	▪ Establishment of site clearing procedures to ensure that the footprint of disturbance is limited as far as possible; ▪ Establishment of a soil monitoring programme for soil stockpiles to prevent the loss of topsoil; ▪ Establishment of a revegetation programme immediate following construction to minimise unnecessary bare surfaces; and ▪ Establishment of appropriate access routes and traffic control measures to prevent the unnecessary movement of vehicles in undisturbed areas.
Terrestrial Biodiversity	Site clearance for the establishment of infrastructure will have a significant direct impact on biodiversity, including: ▪ Direct loss of floral species/vegetation types; ▪ Loss of habitat for faunal species (small mammals, rodents, reptiles and birds) as well as disturbance of their migration patterns; ▪ Loss of protected trees; and ▪ Compromised integrity of terrestrial biodiversity caused by the infestation of Alien Invasive Plants (AIPs) as a result of habitat fragmentation. It is acknowledged that the potential resource targets (Zone A and Selin) and areas currently identified for the plant and other administrative infrastructure are already disturbed due to ASM and exploration activities therefore biodiversity impacts are expected to be limited to areas such as the TSF and potential access routes. The presence of the project is likely to	▪ Establishment of site clearing procedures to ensure that the footprint of disturbance is limited as far as possible; ▪ Establishment of a Biodiversity Management Plan aimed at ensuring the preservation and sustainable use of natural resources in the project area; ▪ Establishment of a relocation programme for the conservation of protected floral and faunal species encountered in the development footprint; ▪ Establishment of an AIP management plan to prevent the spread of AIPs; and

Aspect	Potential Impact Description	Mitigation Type
	result in the influx of people into the project area seeking employment opportunities. This could lead to the following indirect impacts resulting in the loss of terrestrial biodiversity: ▪ Increased exploitation of natural resources (particularly in the forest habitat); and ▪ Increased use of land for the development of socio-economic infrastructure, agricultural areas and hunting activities.	▪ Implement of a Rehabilitation and Closure Plan (RCP) in line with established closure and final land use objectives.
Wetlands	Depending on the final project layout, direct impacts to wetland habitats as follows: ▪ Destruction of wetlands and consequently loss of habitat for associated faunal species; and ▪ Economic displacement of rice cultivation. Wetlands are present in the Project area and are predominantly associated with floodplains of streams and drainage lines. Other potential indirect impacts that could be experienced to wetlands include sedimentation (should soil erosion impacts be realised) and reduced water quality. This can consequently lead to reduced integrity of wetlands and establishment of AIPs which would adversely affect the ecological services they provide to the ecosystem. Wetlands play an important role in toxicant removal and ecosystem services. Furthermore, the wetlands in the project area associated with rice cultivation and therefore impacts to wetland will likely have significant biodiversity and socio-economic implications.	▪ Implementation of wetland buffer zones (at least 100 m) and avoiding the placement of infrastructure in wetlands and their associated buffer zones; ▪ Establishment of stormwater management structure to prevent the contaminated runoff entering surrounding water resources ▪ Establishment of a soil monitoring programme for soil stockpiles to prevent sedimentation of surrounding water resources; and ▪ Establishment of an AIP management plan to prevent the spread of AIPs to wetlands.
Surface Water	Surface water impacts that could occur relate to reduced water quality and reduced water quantity. In terms of water quality, the following possible impacts may be experienced: ▪ Sedimentation of water resources (should soil erosion impacts be realised) resulting in reduced water quality; and ▪ Water pollution as a result of hydrocarbon spills and/or contaminated runoff entering water resources. In terms of water quantity impacts, the establishment of the project will require the containment of water and stormwater management system which will result in reduced catchment yields. Impacts to both surface water quality and quantity will impact downstream users and surrounding biodiversity (including direct impacts to aquatic life).	▪ Establishment of a surface water quality management programme upstream and downstream of the project area to detect impacts from project activities; ▪ Establishment of stormwater management structure to prevent the contaminated runoff entering surrounding water resources; ▪ Establishment of a comprehensive water management plan (particularly if stream diversions are required) to minimise downstream impacts; and

Aspect	Potential Impact Description	Mitigation Type
	It is also noted that the various perennial and non-perennial streams associated with the Fié and Niger Rivers traverse the project area and could potentially coincide with infrastructure and pit areas. In this event, modifications of drainage patterns in the project area and possibly river diversions will be required. This will likely require major earthworks, high costs and substantial environmental implications.	▪ Establishment of appropriate compensation mechanisms in the event that significant water quantity reduction to downstream reaches is probable.
Groundwater	Groundwater impacts that could occur relate to reduced water quality and reduced water quantity. Open-pit mining will require dewatering activities to allow for safe mining. This results in lowering of the groundwater table and the formation of a cone of depression. The cone of depression may impact on water availability to surface water resources, wetlands and community wells and boreholes. In terms of groundwater quality, potential pollution could occur as a result of seepage of hazardous substances and runoff as well as leaching of contaminants from WRDs, ROM pad and TSF, creating a contamination plume. Any potential contamination plumes are likely to migrate to the pits during operation due to the cone of depression, however post closure may result in the mobilisation of contaminants into the downstream catchment.	▪ Establishment of a groundwater quality management programme upstream and downstream of the project area to detect impacts from project activities; ▪ Establishment and continuous update of a groundwater numerical model to determine the potential contamination plume and cone of depression; and ▪ Establishment of appropriate compensation mechanisms in the event that significant groundwater quantity reduction which affects surrounding public boreholes is probable.
Air Quality	Construction and operational activities have the potential to impact on air quality. The increase in traffic, use of earthmoving machinery, material handling, stockpiling and use of crushers and processing to take place is likely to result in the generation of dust, Particulate Matter (PM) and gaseous emissions. Generation of dust and gaseous emissions could lead to the degradation of air quality and potential nuisance impacts, reduction in visibility and potential respiratory illness for nearby sensitive receptors.	▪ Establishment of an air quality monitoring programme to detect impacts from project activities; and ▪ Establish of a grievance mechanism to address community complaints related to poor air quality.
Socio-Economic Environment		
Cultural Heritage	Sites of cultural significance area present throughout the project area, such as burial grounds and graves and sacred sites. The project may result in the direct destruction of such sites, or the degradation depending on the proximity of project activities.	▪ Implementation of heritage buffer zones to prevent the destruction of areas of cultural significance;

Aspect	Potential Impact Description	Mitigation Type
		▪ Establish of a grievance mechanism to address community concerns regarding impacts to heritage resources; ▪ Establishment of a Chance Find Procedure to manage unknown heritage resources may be encountered.
Traffic	The project will result in the establishment of access routes and haul roads which will result in an impact on traffic and community health and safety.	▪ Implementation of a traffic management plan (e.g. installation of signage and enforcement of speed limits).
Socio-Economic	A 500 m buffer is recommended around pit areas. This may result in the physical displacement of communities in proximity to the potential pit areas (particularly the Bokoro Hamlet). Furthermore, economic displacement is probable as agricultural and ASM activities are undertaken in these areas of displacement. Given the extent of ASM in the project area, this is likely to be a significant and contested impact.	▪ Implementation of a Resettlement Action Plan (RAP) to compensate the physical displacement of communities; ▪ Implementation of a Livelihood Restoration Plan (LRP) to compensate the economic displacement of communities; and ▪ Establishment of a clear communication strategy and Stakeholder Engagement Plan (SEP) to manage displacement related impacts.
	The project will provide formal employment opportunities through skilled, semi-skilled and unskilled labour requirements. In addition, opportunity for skills development and transfer to people within the project area and surrounding areas will be realised. This in turn will result in economic growth. The purchasing power within the project area will also be increased as employed personnel will have the means to purchase more goods and services as well as further invest in secondary economic activities. At regional and national level, the project will contribute increasing revenue through tax and local budget contributions.	▪ Establishment of procurement strategies/procedures which favour the employment of local people and use of local goods and services suppliers as far as possible to maximum local economic growth; ▪ Implementation of training programmes for maximise skills development.
	The presence of the project will likely exacerbate the influx of people from both within Mali and neighbouring countries who are seeking employment opportunity. This in turn can lead to the following adverse impacts:	▪ Establishment of a clear communication strategy and SEP to manage impacts related to influx;

Aspect	Potential Impact Description	Mitigation Type
	▪ Increased pressure on existing socio-economic infrastructure; ▪ Increased exploitation of natural resources and degradation due to land occupation; ▪ Increase in social pathologies and communicable diseases; and ▪ Adverse influences on cultural customs of the indigenous population.	▪ Establishment of procurement strategies/procedures which favour the employment of local people first to avoid conflict in communities; and ▪ Implementation of Local Development Initiatives to alleviate further pressures to existing socio-economic infrastructure.
	Often, those living in rural areas with few economic opportunities may have unrealistic expectations of a new project. As a result dissatisfaction and unmet expectations are probable. This presents risks of unrest in the project area (e.g. striking and damage of mine infrastructure). In addition, the expectations of better employment opportunity from the project may further deter young people from agricultural practice which will negative affect livelihoods in the project area.	▪ Establishment of a clear communication strategy and SEP to manage stakeholder expectations; ▪ Establishment of procurement strategies/procedures which favour the employment of local people first to avoid conflict; ▪ Implementation of Local Development Initiatives to alleviate further pressures to existing socio-economic infrastructure; and ▪ Establish of a grievance mechanism to address community concerns.
	The presence of the project can positively contribute the development of socio-economic infrastructure and skills development in the wider project area through strategic Local Development Initiatives. This in turn will result in local economic development.	▪ Establishment of a Community Development Plan (CDP) aimed at ensuring greater community spend and commitment.

7 Terms of Reference

This chapter therefore provides an overview of the applicable Legal Framework as well as ToR for the environmental and social assessment which will be required as part of the permitting process. The ToR has been tailored with specific consideration of the outcomes of this Scoping Study.

7.1 Legal Framework

The development and mining of the Sanankoro Gold Prospect will require compliance with national legislation and International Best Practice. The applicable legislative framework is summarised in the subsections below.

7.1.1 Mali Legislation

Cora Gold is required to apply for an Exploitation Permit for the project area in accordance with Law No. 2012-015 of 27 February 2012 (the Mining Code). In terms of Article 22, Decree No. 2012-311 of 21 June 2012 of the Mining Code, any application for exploitation must be accompanied by an application for an Environmental Permit. This requires for an Environmental and Social Impact Assessment (ESIA) Process to be undertaken in accordance with the rules and procedures set out under the following legislation:

- Law No. 01-020 P-RM of 30 May 2001 on Pollution and Nuisance; and
- Decree No. 2018-0991 / P-RM of 31 December 2018.

The Direction Nationale de l'Assainissement et du Controle des Pollutions et Nuisances (DNACPN) are the responsible authorities for environmental permitting in Mali. Broadly, the ESIA Process requires a detailed assessment of environmental and social impacts that could arise as a result of the development as well as develop an Environmental and Social Management Plan (ESMP) to mitigate and manage these impacts (refer to Section 7.2 below).

7.1.2 International Standards

The Equator Principles and International Finance Corporation's Performance Standards (IFC PS) widely recognised as effective tools for the sustainable management of environmental and social risks of a project to ensure projects were developed, operated and closed in a socially responsible manner and reflecting sound environmental management practices. These standards provide an approach to the determination, assessment and management of environmental and social risk in project financing.

To comply with International Best Practice, the Equator Principles and IFC Performance Standards should be utilised as the regulatory framework for the project. A summary of the Equator Principles and IFC Performance Standards is provided in the tables below.

Table 7-1: Equator Principles (2013)

Principle	Requirement
Principle 1	Under the Equator Principles, proposed developments are categorised depending on its potential environmental and social risks. The Sanankoro Gold Project will be classed as category A. Projects of this category are deemed to have potential significant adverse environmental and social risks and/or impacts that are diverse, irreversible or unprecedented.
Principle 2	For a category A project, a suitably comprehensive assessment process appropriate to the nature and scale of the project is required. The nature of the Sanankoro Gold Project necessitates a detailed ESIA and ESMP is prepared.
Principle 3	For projects taking place in Designated countries (generally first world countries), the applicable standard will be host country laws, regulations and permitting requirements that pertain to Environmental and Social matters. For projects taking place in Non-Designated countries, the Equator Principles requires compliance with the IFC Performance Standards and the World Bank Environmental, Health and Safety Guidelines (EHS Guidelines). In addition to the IFC Performance Standards and EHS Guidelines, compliance with in-country legislation is also required.
Principle 4	The Equator Principles will require a Category A project that an Environmental and Social Management System be composed of policies and procedures to manage environmental and social risks.
Principle 5	Category A projects require that effective stakeholder engagement is undertaken and is an ongoing process. Vulnerable and indigenous groups must be taken into consideration and all legal requirements of consultation met.
Principle 6	The Equator Principles require that a Category A project implement a grievance mechanism to record and document all concerns and issues raised by the communities, regarding the project.
Principle 7	The Equator Principles require a Category A project to undergo an independent review by a consultant.
Principle 8	The Equator Principles require the inclusion of covenants regarding the implementation of the Equator Principles III into legal documentation structuring the deal. This requirement gives the requirements of the Equator Principles III a legally binding nature between the contracting parties.
Principle 9	The Equator Principles requires a Category A project to appoint an independent environmental consultant to undertake the monitoring and reporting, or that applicable skills be retained in house.
Principle 10	The Equator Principles require a Category A project to make the ESIA available online. It is further required that a GHG emissions report be publicly released if

	emissions exceed, or are anticipated to exceed, 100 000 CO2 equivalent per annum. Once the likely mining and haulage scenarios are established the requirement for an emissions report will be evaluated.

Table 7-2: IFC Performance Standards (2012)

Performance Standard	**Requirement**
PS 1: Assessment and Management of Environmental and Social Risks and Impacts:	PS 1 underscores the importance of managing environmental and social performance throughout the life of a project. An effective Environmental and Social Management System (ESMS) is a dynamic and continuous process initiated and supported by management, and involves engagement between the project promoter, its workers, local communities directly affected by the project (the Affected Communities) and, where appropriate, other stakeholders. The ESMS entails a methodological approach to managing environmental and social risks and impacts in a structured way on an ongoing basis
PS 2: Labour and Working Conditions:	PS 2 recognises that the pursuit of economic growth through employment creation and income generation should be accompanied by protection of the fundamental rights of workers. Failure to establish and foster a sound worker-management relationship can undermine worker commitment and retention and can jeopardise a project. Conversely, through a constructive worker-management relationship, and by treating the workers fairly and providing them with safe and healthy working conditions, tangible benefits can be realised, such as enhancement of the efficiency and productivity of their operations.
PS 3: Resource Efficiency and Pollution Prevention:	PS recognises that increased economic activity and urbanisation often generate increased levels of pollution to air, water, and land, and consume finite resources in a manner that may threaten people and the environment. More efficient and effective resource use and pollution prevention and mitigation technologies and practices have become more accessible and achievable in virtually all parts of the world.
PS 4: Community Health, Safety, and Security:	PS 4 recognises that project activities, equipment, and infrastructure can increase community exposure to risks and impacts. In addition, communities that are already subjected to impacts from climate change may also experience an acceleration and/or intensification of impacts due to project activities. While acknowledging the public authorities' role in promoting the health, safety, and security of the public, this Performance Standard addresses the promoter's responsibility to avoid or minimise the risks and impacts to community health, safety, and security that may arise from project related-activities, with particular attention to vulnerable groups.

PS 5: Land Acquisition and Involuntary Resettlement:	PS 5 recognises that project-related land acquisition and restrictions on land use can have adverse impacts on communities and persons that use this land. Involuntary resettlement refers both to physical displacement (relocation or loss of shelter) and to economic displacement (loss of assets or access to assets that leads to loss of income sources or other means of livelihood) as a result of project-related land acquisition and/or restrictions on land use. Resettlement is considered involuntary when affected persons or communities do not have the right to refuse land acquisition or restrictions on land use that result in physical or economic displacement. This occurs in cases of (i) lawful expropriation or temporary or permanent restrictions on land use and (ii) negotiated settlements in which the buyer can resort to expropriation or impose legal restrictions on land use if negotiations with the seller fail.
PS 6: Biodiversity Conservation and Sustainable Management of Living Natural Resources:	PS 6 recognises that protecting and conserving biodiversity, maintaining ecosystem services, and sustainably managing living natural resources are fundamental to sustainable development. The requirements set out in this Performance Standard are guided by the Convention on Biological Diversity.
PS 7: Indigenous Peoples:	PS 7 recognises that Indigenous Peoples, as social groups with identities that are distinct from mainstream groups in national societies, are often among the most marginalised and vulnerable segments of the population. In many cases, their economic, social, and legal status limits their capacity to defend their rights to, and interests in, lands and natural and cultural resources, and may restrict their ability to participate in and benefit from development. Indigenous Peoples may be more vulnerable to the adverse impacts associated with project development than non-indigenous communities. This vulnerability may include loss of identity, culture, and natural resource-based livelihoods, as well as exposure to impoverishment and diseases.
PS 8: Cultural Heritage:	PS 8 recognises the importance of cultural heritage for current and future generations. Consistent with the Convention Concerning the Protection of the World Cultural and Natural Heritage, this Performance Standard aims to ensure the protection of cultural heritage in the course of project activities. In addition, the requirements of this Performance Standard on a project's use of cultural heritage are based in part on standards set by the Convention on Biological Diversity.

7.2 ToR for the ESIA

Based on the project's classification (Category A Project), national legislative requirements for an Environmental Permit as well as the potential impacts mentioned above, a comprehensive ESIA Process is required. The key stages for the ESIA will include:

- Scoping (and site selection);
- Stakeholder engagement;
- Baseline data collection (wet and dry seasons);
- Project description and interaction with the design and decision-making;
- Assessment of impacts and identification of mitigation measures;
- Integrated management system and plans; and
- Reporting and disclosure.

The subsections below provide the proposed Plan of Study for the ESIA Process with respect to the required in terms of the specialist studies and stakeholder engagement to inform the ESIA.

7.2.1 Specialist Studies

7.2.1.1 Biophysical Environment

The following section describes the biophysical studies that are required as part of the ESIA process.

7.2.1.1.1 Air Quality

The potential impacts on air quality need to be understood. Given ASM activities in the project it is expected that cumulative impacts may be realised. It is crucial that the impacts on air quality at mining activities (stockpiles, pits, waste and haul roads) are understood as it has the potential to adversely affect both people and the environment. The objectives and deliverables for the air quality assessment are summarised in Table 7-3.

Table 7-3: Objectives and Key Deliverable for the Air Quality Assessment

Objectives	Key Deliverables
▪ Determine the regional climate and assess the baseline conditions (dust deposition), as well as the local (site-specific) prevailing weather conditions and its influence on the climatic and atmospheric dispersion and dilution potential of pollutants released into the atmosphere; ▪ Identify existing sources of emissions and characterise the ambient air quality within the airshed;	▪ Site-specific meteorological data will be obtained and evaluated. In the absence of site specific meteorological data, modelled meteorological data. ▪ Dust fallout baseline.

Objectives	Key Deliverables
▪ Review of potential health effects associated with air pollutants; and ▪ Define the potential sensitive receptors, such as local communities, as well as environmental constraints relative to air quality.	
▪ Estimate the emissions from various sources within the proposed operations.	▪ Emissions Inventory
▪ Compute ambient concentrations as a function of source configurations, emission strengths and meteorological characteristics, to ascertain the spatial and temporal patterns in the ground level concentrations arising from the emissions of various sources; and ▪ Determine pollutants contribution from the operational phase.	▪ Dispersion Model
▪ Highlight predicted zones of maximum ground level impacts (particulate matter and gases), potential for human health and environmental impacts; and ▪ Recommendations of buffer zones and impact management zones.	▪ Air Quality Impact Assessment, inclusive of monitoring recommendations

7.2.1.1.2 Soils, Land Use and Land Capability

A Soil, Land Use and Land Capability study is required to determine the project's impact to the livelihoods (agricultural activities) of the communities in the project area. The objectives and deliverables of the study are provided in Table 7-4.

Table 7-4: Objectives and Key Deliverables for the Soils and Land Use Assessment

Objectives	Key Deliverables
▪ Identify the dominant soil forms, their distribution, the existing land capability and current land use within the project area.	▪ Literature review
▪ Determine soil type and depth on site and define soil acidity, fertility and texture	▪ Sample analysis of topsoil (0-0.3 m) and subsoil (0.3-0.6 m) of the dominant soil forms (40 samples)
▪ Identify and rate potential impacts on the soils, land capability and land use ▪ Identify management and mitigation measures to reduce impact significance	▪ Soils and Land Use Impact Assessment Report

7.2.1.1.3 Biodiversity

The biodiversity studies will consist of terrestrial flora and fauna, aquatic and wetland studies.

Fauna and Flora

A detailed vegetation study is required in the growing season to determine the different plant communities, species compositions, biodiversity as well as any potential Red Data plant species and Protected tree species.

Detailed faunal studies on the mammals, birds, reptiles, amphibians and invertebrates will be conducted to list all species present in Project area. This record will determine the biodiversity ranges and the likelihood of any potential Red Data or protected species in the Project area. The objectives and deliverables of the study are provided in Table 7-5.

Table 7-5: Objectives and Key Deliverables for the Fauna and Flora Assessment

Objectives	Key Deliverables
▪ Determine the actual flora species present on site and discuss these in context of plant communities within the ecosystem of the area; ▪ Discuss protected, endemic, exotic, alien invasive and culturally significant plant species; ▪ Identify any rare or protected species; ▪ Identify mammals, birds, amphibians and invertebrates potentially making use of the area	▪ Fauna and Flora Baseline
▪ Identify and map sensitive areas, as described by the provincial and national legislation	▪ Flora and Fauna Impact Assessment

Aquatics

An aquatics study of the Fié and Niger Rivers as well as any tributaries downstream of key infrastructure is necessary to determine the aquatic composition of the system prior to mining activities. A study is necessary for both the high and low flows to record the baseline. Although it is expected that there will be limited flows of smaller streams within the project area during the dry season, the instream habitat will still be surveyed to understand the seasonal characteristics of the aquatic habitat. The objectives and deliverables of the study are provided in the table below.

Table 7-6: Objectives and Key Deliverables for the Aquatics Assessment

Objectives	Key Deliverables
▪ Determine the actual aquatic species (fish and macroinvertebrates) present in the river and its tributaries and discuss these in context of the ecosystem of the area ▪ Identify and discuss any red data or protected species ▪ Determine existing surface water quality by taking water samples ▪ Determine the existing status of the river bed by taking sediment samples	▪ Aquatics Baseline
▪ Identify and map sensitive areas and determine the potential impacts from mining operations	▪ Aquatics Impact Assessment

Wetlands

The wetlands identified on site were predominantly floodplain wetlands, although the potential remains for further wetlands throughout the project area. The extent of the possible wetlands in the area will be confirmed by a wetland specialist. Wetlands are regarded as critical, sensitive habitats that must be conserved. Wetland areas are already impacted by agricultural and ASM activities. The wetlands study will work in conjunction and include findings from the fauna, flora, aquatics and hydrological studies. The objectives and deliverables of the study are provided in Table 7-7.

Table 7-7: Objectives and Key Deliverables for the Wetlands Assessment

Objectives	Key Deliverables
▪ Delineate the wetland areas of the project areas ▪ Classify the soil characteristics of the wetland areas ▪ Determine and classify the current health of the wetland systems ▪ Determine the impact already being exerted on the systems	▪ Wetlands Baseline
▪ Identify and map the wetland areas and their health ▪ Incorporate analysis from the fauna, flora, aquatics and hydrological studies to determine the potential impacts from mining operations	▪ Wetlands Impact Assessment

7.2.1.1.4 Surface Water

A hydrological investigation will define the catchment boundaries to comprehensively understand the hydrological characteristics of the project area and water management measures required. The objectives and deliverables of the study are provided in Table 7-8.

Table 7-8: Objectives and Key Deliverables for the Surface Water Assessment

Objectives	Key Deliverables
▪ Describe the baseline surface water environment	▪ The desktop assessment will include use of already available information from previous studies, land type data, land use map, 1:50 000 topographical data and climate information for the description of the surface water environment
▪ Determine existing surface water quality by conducting a site visit and take surface water quality samples; ▪ Delineate floodlines to determine impacts from the pits and required diversions.	▪ Surface water quality sample analysis
▪ Determine the potential impacts that could arise from the proposed Project on the surface water environment and the nearby rivers ▪ Assess potential impacts that the proposed development may have, and to provide mitigation measures for those identified impacts	▪ Surface water impact assessment report ▪ Surface water quality and streamflow monitoring programme
▪ Provide the expected volumes of water to ensure that onsite water is managed appropriately. This will depict water inflows, losses and outflows within the mine	▪ Water and salt balance

▪ Prepare a conceptual Stormwater Management Plan (SWMP) according to the recommended management standards. Ensuring that clean water is separated from dirty water. The SWMP will include the following:
  ▪ An outline of key SWMP principles;
  ▪ Delineation of clean and dirty water catchments indicated on a plan;
  ▪ Conceptual placement of clean and dirty water structures indicated on a plan; and
  ▪ Storm water management monitoring plan.
	▪ Conceptual SWMP (no engineering design will be included; this is for environmental purposes only)

7.2.1.1.5 Groundwater

A geohydrological investigation is important to understand the potential dewatering of the aquifer. It will also help to determine the potential impact from WRDs and TSF to contaminate the groundwater. An understanding of the system before the placement of infrastructure and waste storage facilities is necessary. A numerical model will be required to inform the decisions around the design of the mine infrastructure. The objectives and deliverables of the study are provided in Table 7-9.

Table 7-9: Objectives and Key Deliverables for the Groundwater Assessment

Objectives	Key Deliverables
▪ Establish a conceptual idea of the hydrogeological occurrence and dynamics.	▪ Desktop study and literature review
▪ Collect data pertaining to the current groundwater conditions and use, including localities of current groundwater abstraction points (boreholes, hand dug wells or springs), ownership, usage volumes and types, equipment and groundwater levels.	▪ Hydrocensus
▪ Establish a reference point against historical and future groundwater conditions ▪ Describe the baseline groundwater environment, identify potential impacts on groundwater and describe the ToR for the Groundwater Impact Assessment	▪ Analysis of hydro-chemical samples from selected boreholes
▪ Undertake a geochemical assessment of the waste rock and tailings material to characterise the wastes and identify contaminants of concern. Contaminants will be identified based on the processing method. Strontium should also be assessed to determine its potential for mobilisation.	▪ Geochemical assessment
▪ Delineate weathered zones and identify possible linear structures that could act as preferred groundwater flow paths and finalise the drilling targets necessary for the study ▪ Determine aquifer responses and calculate the parameters presenting the aquifer hydro-dynamics underlying the investigation area	▪ Geophysics and borehole sampling ▪ Aquifer tests
▪ Describe the complete groundwater system in terms of characterisation of aquifers, contaminant formation, boundaries, hydro-stratigraphic units, the	▪ Hydrogeological model

Objectives	Key Deliverables
groundwater flow system, precipitation, evapo-transpiration, runoff, hydraulic parameters, recharge and discharge rates and hydro-chemical data ▪ Determine potential contaminant transport from the proposed project	
▪ Compile a detailed groundwater impact assessment based on the outcome of the numerical model, with recommended mitigation measures that may be necessary to address groundwater impacts associated with the project ▪ Define a network of observation points and compile a monitoring program that would monitor groundwater conditions (levels and chemistry) before and after commencement of operations	▪ Groundwater Impact Assessment Report ▪ Groundwater quality monitoring programme

7.2.1.2 Social Studies

The following section details the socio-economic studies that are required as part of the ESIA.

7.2.1.2.1 Archaeological and Heritage

A phase one Archaeological and Heritage Impact Assessment (AHIA) is required to identify potentially significant cultural heritage or archaeological sites and resources. The objectives and deliverables of the study are provided in Table 7-10.

Table 7-10: Objectives and Key Deliverables for the Archaeological and Heritage Assessment

Objectives	Key Deliverables
▪ Provide an understanding of the location, nature and extent of the proposed Project in relation to archaeological and cultural heritage ▪ Stakeholder consultations on cultural heritage	▪ Heritage field investigations and baseline
▪ Identify heritage resources and potential impacts to stipulate any limitations or conditions of the development ▪ Determine the general protection required of these; provide mitigation measures or conditions for authorisation	▪ Heritage Impact Assessment
▪ Promote compliance with the Heritage Legislation by providing mitigation measures to be implemented	▪ Statutory Comment Feedback Reports (if required)

7.2.1.2.2 Noise

Given the proximity of communities to potential pit areas, a baseline noise assessment is required to measure the ambient noise level in the project area prior to mining activities. The study will focus on measuring where noise will be generated during construction and operations and where possible noise receptors are located. The objectives and deliverables of the noise study are provided in Table 7-11.

Table 7-11: Objectives and Key Deliverables for the Noise Assessment

Objectives	Key Deliverables
▪ Conduct baseline noise monitoring at selected sensitive receptors	▪ Noise Baseline
▪ Quantify expected noise levels for the construction and operation phases of the proposed project	▪ Dispersion mapping Isopleths
▪ Determine whether the proposed project will be in compliance with the relevant noise regulations and guidelines ▪ Model the noise propagation and the impact to sensitive receptors ▪ Include recommended mitigation measures as well as recommended action plans to minimise the impact of noise on the surrounding environment	▪ Noise Impact Assessment Report

7.2.1.2.3 Socio-Economic

A thorough and up to date understanding of the baseline socio-economic conditions of the communities around the Project area is critical to determine the potential impacts and possible implementation plans for community development.

It is important to complete the socio-economic survey as soon as possible before in-migration to the Project area occurs. The baseline is then used to assess the possible impacts of the project to the population and propose appropriate mitigation measures and action plans. The objectives and key deliverables for the socio-economic study are provided in Table 7-12.

Table 7-12: Objectives and Key Deliverables for the Social Assessment

Objectives	Key Deliverables
▪ Record relevant social spatial information (e.g. settlement patterns, location of public infrastructure and landmarks, etc.) ▪ Review and analyse the available Commune reports for the project area	▪ Socio-Economic Baseline.

Objectives	Key Deliverables
▪ Carry out household surveys, FGDs and one on one meetings ▪ Identify potential socio-economic impacts ▪ Compile the Terms of Reference for the Social Impact Assessment ▪ Gain an appreciation of the project area of impact and compile a detailed socio-economic baseline	
▪ Identify (verify), assess and rate likely socio-economic impacts of the proposed project	▪ Impact Assessment Report
▪ Identify cost-effective and practical mitigation measures aimed at reducing the severity of adverse impacts, and enhancement measures for potential benefits, and define practical steps for implementing the recommended mitigation measures	▪ Social Management Plan

7.2.1.2.4 Resettlement Action Plan and Livelihood Restoration Plan

Based on the potential pit locations, a RAP and LRP will be required to compensate for the physical and economic displacement. A Resettlement Policy Framework (RPF) should be compiled to outline the entitlement framework and resettlement scope, as well as confirming the resettlement legislation to be applied. The objectives and key deliverables for the socio-economic study are provided in Table 7-13.

Table 7-13: Objectives and Key Deliverables for the RAP and LRP

Objectives	Key Deliverables
▪ Determine the legislative requirements for involuntary resettlement ▪ Define the resettlement scope and entitlement framework ▪ Compile resettlement costs and options	▪ RPF
▪ Development of consultation materials and establish consultation structures ▪ Stakeholder engagement ▪ Resettlement survey planning ▪ Asset and census survey of households and economic activities ▪ Socio-economic verification survey ▪ Identification of replacement housing and livelihood options	▪ RAP and LRP

7.2.1.3 Preliminary Conceptual Rehabilitation and Closure Assessment

A preliminary conceptual Rehabilitation and Closure Plan is required to account for all proposed mine activities, and to comply with Article 154 of the Mining Code. A closure cost assessment must also be developed to determine the financial provision for closure at the end of the life of mine. The objectives and key deliverables for the Conceptual Rehabilitation and Closure Assessment are provided in Table 7-14.

Table 7-14: Objectives and Key Deliverables for the Conceptual Rehabilitation and Closure Assessment

Objectives	Key Deliverables
▪ Determine the closure objectives for the desired end land use ▪ Determine the specific rehabilitation actions to be undertaken during operation, decommissioning and closure phases of the mining operation ▪ Provide consolidated environmental maintenance and monitoring programmes for the project	▪ Rehabilitation and Closure Plan (RCP)

7.2.1.4 Other Assessments

A number of other studies may be required for the project, depending on the final project design and authority requirements (i.e. international funder requirements). These assessments could include:

- Climate Change;
- Community Health; and
- Traffic Impact Assessment.

7.2.2 Environmental and Social Impact Assessment

The results of the studies must be collated into a document which will identify the impacts that the project activities will have on the receiving environment. The impacts should be assessed in terms of their severity, duration, extent and significance.

The document will also consider the cumulative impacts associated with the project, in which other external factors or activities are considered and which may contribute to and exacerbate the mine’s impacts.

These findings along with a detailed project description will be captured in the ESIA. The ESIA will also provide the legal framework as a reference point for the project. Mitigation and management measures associated with each impact will be provided. These measures will be implemented to minimise the significant negative environmental and social impacts and enhance potential positive impacts.

7.2.3 Stakeholder Engagement

A Stakeholder Engagement process will be initiated prior to any activities commencing. The Stakeholder Engagement process focuses on consulting key Interested and Affected Parties (I&APs) to identify the issues and concerns that will inform the socio-economic and environmental aspects of the studies. The strategy for this phase is based on interviews with authorities and I&APs and it is proposed that public meetings in the nearby and affected villages take place. The final studies, which should include environmental investigations, must be distributed to registered I&APs to provide them with accurate information and create a basis from which they can raise issues and concerns.

8 Conclusion and Recommendations

This Scoping Study was undertaken to screen the biophysical and socio-economic characteristics of the Sanankoro Gold Project. The assessment aimed to provide early indication of potential environmental and social risks and determine the ToR for the ESIA process that will be required as part of the exploration/environmental permitting process.

The Scoping Study revealed anthropogenic activities within the project area have resulted in environmental degradation with respect to biodiversity as well as potentially adverse implications of water resources. Remaining natural areas should be preserved as far as possible. Cora Gold has already started considering alternative infrastructure locations to optimise the use of already disturbed areas within the project area. Based on the preliminary findings of this Scoping Study, the location of the plant and other potential administrative infrastructure which was originally located in an undisturbed area near the potential TSF site was reconsidered to an existing disturbed area. It is likely that only the TSF and associated access routes will result in disturbance of remaining natural areas in the project area. This is largely due to the topographical and space requirements for the TSF. Furthermore, Cora Gold intends to avoid ore structures which may be exploited in future. Where natural areas are impacted there is also the opportunity to compensate for these by improving existing impacted areas that will now fall within the mine project area and could be restricted from community access. To this end, impacts to biodiversity are expected to be limited and, if correctly implemented, the project could result in positive offsets to existing negative impacts determined within the project area.

In terms of the socio-economic environment, various communities whose livelihoods are mostly sustained through ASM and agricultural activities are present and growing within the project area. The project area is host to one of the most productive ASM sites in Mali which is increasing both through migrant workers and increasing numbers of local youth being attracted to ASM. The socio-economic infrastructure and services however remain relatively limited. Other notable environmental attributes include:

- The presence of various water resources (wetlands, possible aquatic biota, perennial and non-perennial streams) associated with the Fié and Niger River Systems;
- The presence of arable land throughout the project area;

- Good groundwater quality and quantities for potables uses by the surrounding communities;
- The presence of forest galleries and savannah with associated sensitive plant and tree species; and
- The presence of heritage resources and sacred places of cultural significance to surrounding communities.

No immediate fatal flaws were identified for the project. The most significant Project risks are likely to be associated with the following:

- Economic and physical displacement, particularly livelihoods associated with ASM;
- Population influx and the resulting impacts, including increase in ASM;
- River diversions and possible destruction of critical wetland habitats; and
- Potential cross-border water quality and quantity impacts.

Based on the preliminary layout, the potential pits along the Zone A and Selin ore targets currently located within 500 m of several communities and their economic activities, including directly within the Bokoro ASM site and community agricultural land. The project will therefore entail land acquisition which will result in physical and economic displacement of several communities. This impact is expected to be the most significant and potentially contested given the reliance of these economic activities. The displacement will need to be managed through a RAP and LRP. The RAP and LRP will need to have a clear entitlement framework to address any potential challenges associated with land ownership.

The project is located in underdeveloped but growing communes in Mali; population influx is expected as individuals from surrounding areas and neighbouring countries will likely migrate in search of employment. The population influx will place additional pressure on natural resources as well as the already stressed social services and infrastructure in the project area. This in turn may result in increased individuals pursuing ASM activities and conflict between ASM operations, the communities and Cora Gold. It is imperative that a clear and collaborative plan with the authorities is implemented to deal with ASM prior to the project commencing.

It is expected that the TSF and possibly some pit areas will be located within water resources (wetlands and streams associated with the Fié and Niger Rivers). Surface water resources are important for economic activities within the project area and possibly for other downstream users while groundwater is the primary source of potable water. The establishment of the mining related infrastructure may impact on water and drainage, affecting water availability to the surrounding and downstream users. The extent of this impact (including potential need for diversions) will need to be confirmed. As a result detailed floodplain and groundwater modelling to determine the extent of flooding and the effect that dewatering may have on the surface water resources.

The ToR detailed in Section 7 above, is strongly recommended to give a comprehensive understanding of the environmental and socio-economic characteristics as well as the potential impacts that could arise as a result of the project. These studies and ESIA will also fulfil national legislative requirements for the exploration/environmental permitting process as well as international best practice. An ESMP will be developed which will provide mitigation and management aimed at minimising negative impacts as far as possible and enhance possible positive impacts to ensure the sustainability of the development.

To ensure that the ESIA is undertaken is accordance with International Best Practice, several tasks are recommended to be initiated ahead of the ESIA Process. This will aid in ensuring a comprehensive understanding of the baseline environment as well as provide an early indications of key impacts for which detailed management will be required. To this end, Table 8-1 provides an action plan and recommendations to adequately manage project risks going into the ESIA Process.

Table 8-1: Recommendations and Proposed Immediate Action Plan

Number	Immediate Action – Exploration Phase	Frequency
1	A dust monitoring network should be installed to quantify the current, baseline dust deposition rates associated with existing activities within the project area.	Monthly monitoring.
2	Commence with the water monitoring program upstream and downstream of the project area. The monitoring should take place at least quarterly and should include a surface hydrology modelling programme to assist in designing the mine and associated pits. It will inform the mine on the design of water control structures	Quarterly
3	Determine the current groundwater depth and aquifer strength. This will allow the development of a conceptual groundwater model and aid in prefeasibility planning. A seasonal groundwater monitoring programme is recommended at the communities in this report.	Quarterly
4	Continue to maintain relationships and communication with the Project area authorities. This provides support for the management of potential problems with the local communities. Any change in staffing in the project area should undergo an extensive hand-over period to ensure relationships are maintained with authorities and communities.	Ad hoc
5	Undertake a socio-economic survey of the communities that will be physically and economically displaced. This will provide a baseline to monitor potential population influx into the project area. Compensation costs for economic and physical displacement should be included as part of the socio-economic survey to assist in feasibility planning. Communication regarding potential resettlement should be avoided until the mine layout is finalised.	Once off

Number	Immediate Action – Exploration Phase	Frequency
6	Identify all places of worship and sacred objects in the Project area and implement chance find procedures. This will aid in preventing the destruction of items or places of cultural significance, which could result in community unrest.	Continuous

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