Optimization Study Report for the Yanfolila Gold Project, Mali

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1.0 EXECUTIVE SUMMARY

1.1 Introduction

Hummingbird Resources plc (HUM) acquired the Yanfolila Project (Project) from Gold Fields (GF) in July, 2014. GF acquired the Project from Glencar Mining in December 2009 and developed the Project for four years to a point where HUM now plans to begin final development and construction. Once complete, the Project will mine and process over one million tonnes of gold bearing ore per year for approximately seven years and generate approximately 514,000 ounces of gold doré product. HUM is currently in the process of due diligence prior to financing. Once financing is secured and all approvals and permits in place, HUM will commence a 12-month construction period before gold production begins.

Information contained in this Yanfolila Project Optimisation Study Report (Study) is derived from numerous investigations and studies performed by HUM and its consultants, along with historical information GF developed while they owned the Project.

1.2 Location

The Yanfolila Cercle, located in Mali's Sikasso Region, hosts the Project at latitude 11°12'00" and longitude 8°30'00"E, is shown in Figure 1.1.

The Project lies 40 km to the west of the town of Yanfolila, the nearest town. The Project extends over approximately 50 km north and south and 25 km to the east and west. The Project includes seven exploration permits with consolidation requests pending. The access route follows a paved road from Bamako to Yanfolila, then a maintained gravel road to the village of Bougoudalé. A slightly revised road route will provide future access to the Project, as described in the Project Infrastructure section. The Project currently maintains a base camp, Komana Camp, located near Bougoudalé. HUM plans to expand Komana Camp as the Project advances.

1.3 Ownership

HUM owns 85% of Glencar Mali SARL (Glencar) and Société Malienne de la Petite Mine, a private Malian company, owns 5%. The government of Mali owns 10% of the Project, with an option to increase its ownership to 20%.

1.4 Climate and Physiography

The Project is located in the Malian Sudano-Guinean area, which has a seven month dry season (October to May), and a five month rainy season (May to October). Average annual rainfall is 1,100 mm, but with significant variability across the year. Maximum temperatures (around 40°C) are in the months of March and April, and minimum temperatures (around 15°C) in December and January.

The Project area is comprised of low-rolling hills at altitudes between approximately 315 meters above seal level (masl) on the shore of the Sankarani River, to approximately 455 masl at Komana West (KW), and approximately 440 masl at Komana East (KE). The relief is a lateritic plateau rising from the banks of the Sankarani River towards the east and the south across most of the southern and eastern permit area. The winding valley of the Sankarani River is an important component of the landscape.

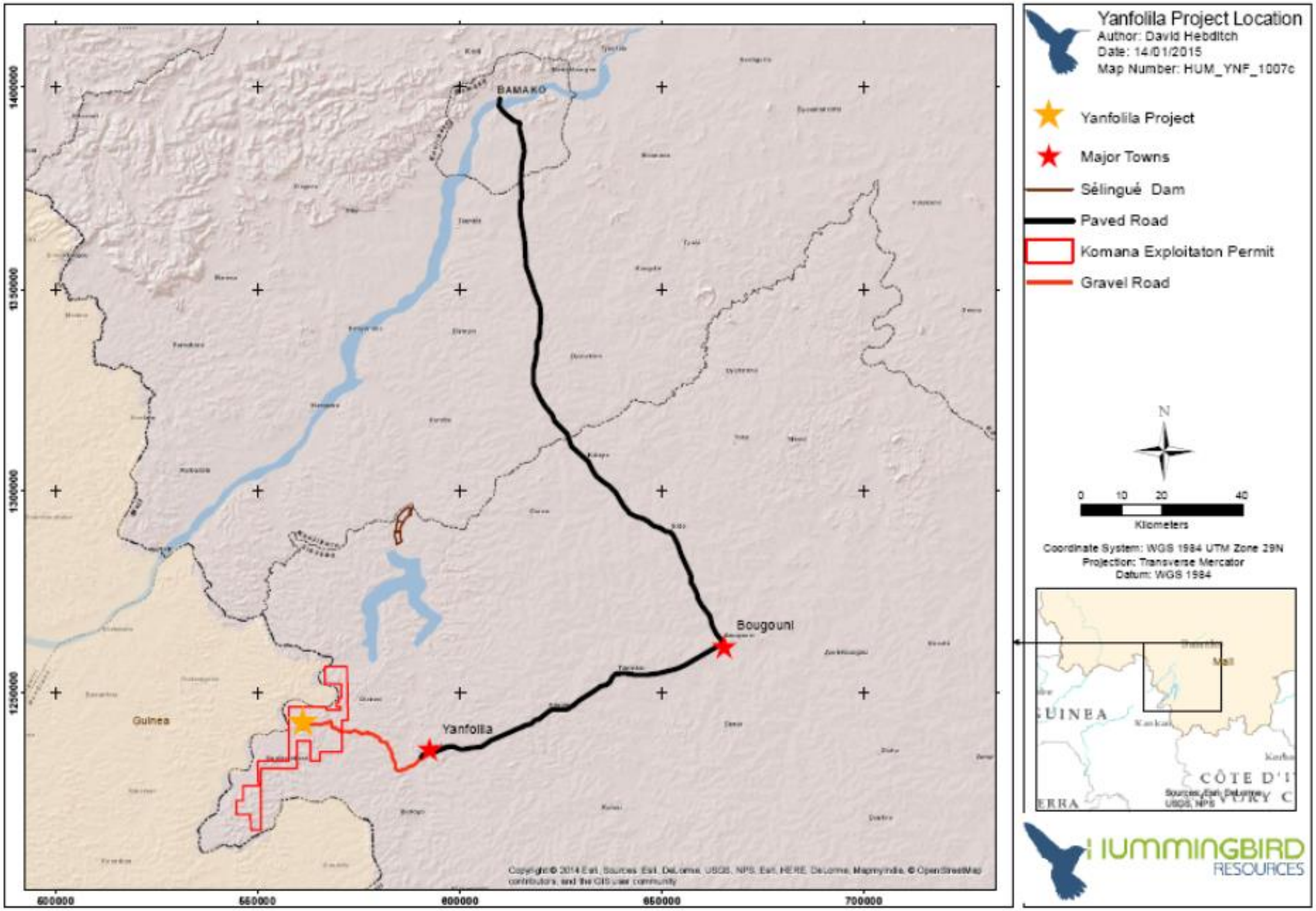

Figure 1.1 Project Location Map

1.5 History

Mali was once part of three famed West African empires controlling trans-Saharan trade in gold and other commodities. Mali fell under the control of France during the late 19th century, becoming part of French Sudan, which later united with Senegal to become the Mali Federation. This Federation gained independence from France in 1960. Mali split from Senegal within months and became the Republic of Mali on September 22, 1960.

In 1991 Mali relaxed its mining codes to attract renewed foreign investment in the industry. In 2012, Malian miners produced 50 tonnes of gold in Mali's south, making Mali the third largest producer in Africa, behind Ghana and South Africa.

HUM acquired the Project from GF in July, 2014. GF acquired the Project from Glencar Mining in December 2009, and developed the Project for four years to a point where HUM now plans to begin final development and construction.

1.6 Geological Setting

The Yanfolila Greenstone Belt, which hosts the Project, is orientated north-south on the eastern margin of the greater Siguiri Basin. It forms part of the Birimian Volcano-Sedimentary series of the West African Craton. The belt contains several sub-basins including the Komana Mafic Sub-Basin (KMSB) and the Kabaya Sub-Basin (KSB). These sub-basins, and the eastern margin of the wider Siguiri Basin, are bound to the east by the Sankarani Shear Zone (SSZ) and the Siekerolé Granite. The SSZ is the major basin-bounding structure in the region. Mineralisation is typical lode gold-style deposits structurally reminiscent of a foreland fold and thrust belt with a deformational framework similar to the Ghanaian Gold Province.

The KMSB hosts the majority of the Yanfolila gold deposits and targets currently defined. It has a stratigraphic sequence of basalt, polymict conglomerate, feldspathic sandstone, silt-shale, and a lithic-dominated greywacke. Mafic and felsic (porphyry, granodiorite, and diorite) intrusives crosscut the stratigraphy. Splays of the SSZ and regional-scale structures, which crosscut strata, control gold mineralisation. Gold is associated with rheological contrasting rock types that have been structurally deformed to provide fluid pathways from where any combination of disseminated, vein, breccia, intrusive, and replacement styles may evolve.

Outcrop is sparse throughout the basin and is mostly extensive lateritic and depositional terrains.

1.7 Deposit Types

Lithologies at KW comprise interbedded siltstone and sandstone, with basalts and porphyry intrusives. Regional scale cross-cutting faults and structural lineaments, as well as rheologically contrasting rock types, appear to control the localisation of gold. Mineralisation appears to be related to basalt at the western margin along the main north-south shear zone as well as in the contact zone with metasediment. Feldspar porphyry has locally developed quartz-vein stockwork associated with pyrite and arsenopyrite. Main alteration minerals are sericite, silica, feldspar, chlorite, and epidote. Weathering and oxidation of the sulphide component generally extends to a depth of 50 to 90 m, with a well-developed regolith zone.

The host rocks at KE comprise a mafic dolerite intrusion in the south and a siltstone unit in the north. Mineralisation is quartz-albite-carbonate breccia veins and related pyrite-hematite altered wall rocks associated with brittle-ductile shear zones. The orebody geometry strikes north-south to north-northeast and dips steeply to the west and is transected by numerous northeast striking faults, some of which are mineralised. Weathering and oxidation of the sulphide component generally extends to a depth of 30 to 50 m, with a well-developed regolith zone.

At Sanioumale East (SE), the main lithologies include a sequence of basalt and metasediment (interbedded siltstone, sandstone, and polymict conglomerate). Mafic dyke (dolerite) as well as felsic porphyry dykes occur in the system. Mineralisation is hosted mainly in basalt and the dolerite dyke proximal to the contact with metasediment. The mineralisation style comprises quartz-albite-carbonate breccia veins and related pyrite-hematite altered wall rocks associated with brittle-ductile shear zones. The orebody geometry is strikes north-south to north-northeast and dips steeply to the west, and is transected by numerous northeast striking faults, some of which are mineralised. Weathering generally extends to a depth of 30 to 50 m, with a well-developed regolith zone.

Sanioumale West (SW) lithologies include metasediments made up of sandstone and siltstone, as well as volcanoclastics with coarse grained feldspar and clasts. Mineralisation occurs in a deformed stratigraphic sequence of sandstone and siltstone including the volcanoclastics. Regional scale cross-cutting faults and structural lineaments, as well as rheologically contrasting rock types, appear to control the localisation of gold. Mineralisation styles are dominated by quartz-feldspar-carbonate veins in shear zones with disseminated pyrite grains in the numerous fissures. Weathering generally extends to a depth of 30 to 60 m, with a well-developed regolith zone.

Guirin West (GW) lithologies comprise metasediments of inter-bedded siltstone and sandstone, with basalt; mafic, and felsic intrusives. Mineralisation appears to be related to a felsic intrusive and volcanoclastics hosting a stockwork of quartz-feldspar-carbonate veins (10 to 50 cm in width). Weathering generally extends to a depth of 50 to 70 m, with a well-developed regolith zone.

1.8 Exploration

The Project encompasses areas mined by artisanal and small scale miners (ASM), believed to have been active in the area for the past 50 years.

BHP started exploring the Project area in 1992 and identified anomalies at KW and KE. In 1997 Randgold took over BHP’s exploration permits and furthered exploration at the Project, transferring its permits to Glencar Mining in 2004, which advanced the Project until 2009 when the Project was sold to GF. GF advanced the Project for four years, until selling the Project to HUM in July of 2014.

1.9 Mining

The Project will be developed by progressively mining five deposits: KE, KW, GW, SE, and SW, shown in Figure 1.2. The Gonka deposit is also shown as it may be included in the Project at a later date. HUM will contract with a mining service provider to apply industry standard open pit mining methods to remove waste and ore from the deposits according to a mine production schedule. Mining methods will include excavating the soft oxidized ore near the surface, and drilling and blasting harder transitional and fresh rock. Haul trucks will follow haul roads to transport waste rock to engineered waste rock disposal facilities (WRD) .

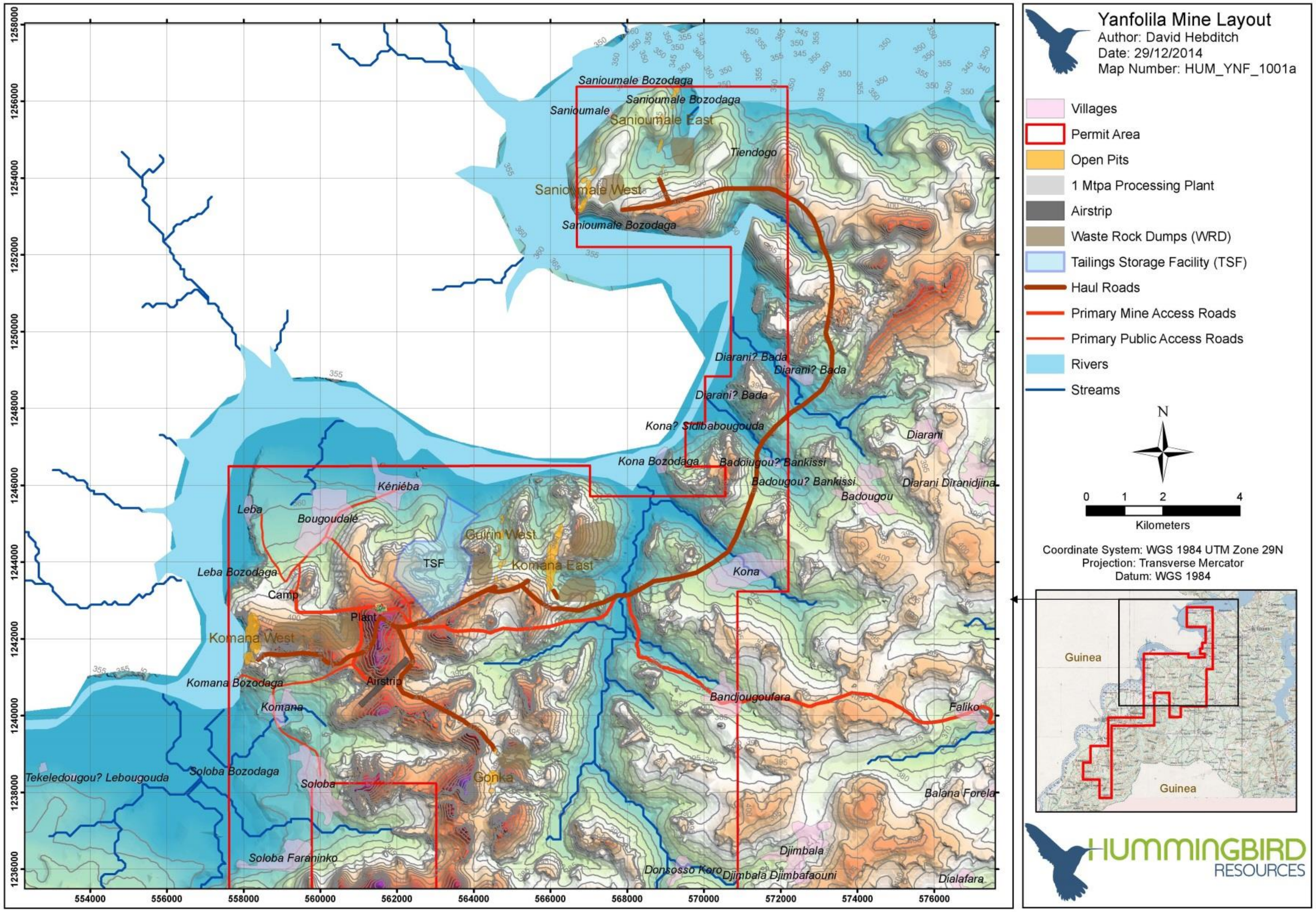

Figure 1.2 Yanfolila Mine Layout

The mine fleet will transport ore to the plant via haul roads which will be constructed onsite. The mining rate will support the process production plant capacity by feeding the plant at least 1 Mtpa of gold bearing ore for the life of mine (LOM).

1.10 Processing

Once the ore is fed into the process plant, it will undergo the following process steps to produce gold doré as shown in Figure 1.3.

Soft run of mine (ROM) ore will be fed into the process and initially crushed by a mineral sizer. The ball mill will grind the feed into finer particle size to enable gold recovery, with 80% of the mill product passing through a 106 µm screen. A select fraction of the mill output will be separated and fed into a gravity gold circuit which will apply centrifugal force to concentrate heavy, gold bearing particles for intensive leaching. Once leached, the gold bearing pregnant leach solution from the gravity circuit will be piped into an electrowinning cell. The gravity tails will be returned to the mill stream.

The main mill product will flow to the carbon in leach (CIL) process. The CIL process will apply controlled cyanide bearing leach solution to the finely ground ore to remove gold from the ore and deposit it on carbon. Approximately four times per day the most heavily loaded carbon will flow to elution and electrowinning processes to produce gold bearing electrode sludge. The cathode sludge will be filtered, dried, and mixed with flux in the secure gold room. It will then be smelted at high temperature to produce molten gold, which will be poured into gold doré bullion and shipped to a refinery.

The reagents and consumables used at the Project will consist of the following:

- Steel grinding balls
- Rubber grinding mill liners
- Sodium cyanide for cyanidation
- Lime for pH control
- Caustic soda for pH control
- Hydrochloric acid for washing carbon
- Carbon to recover gold during CIL processing
- Copper sulfate to detoxify tailings
- Sodium metabisulfite to detoxify tailings
- Lead nitrate for improved cyanidation
- Smelting reagents
- Diesel fuel

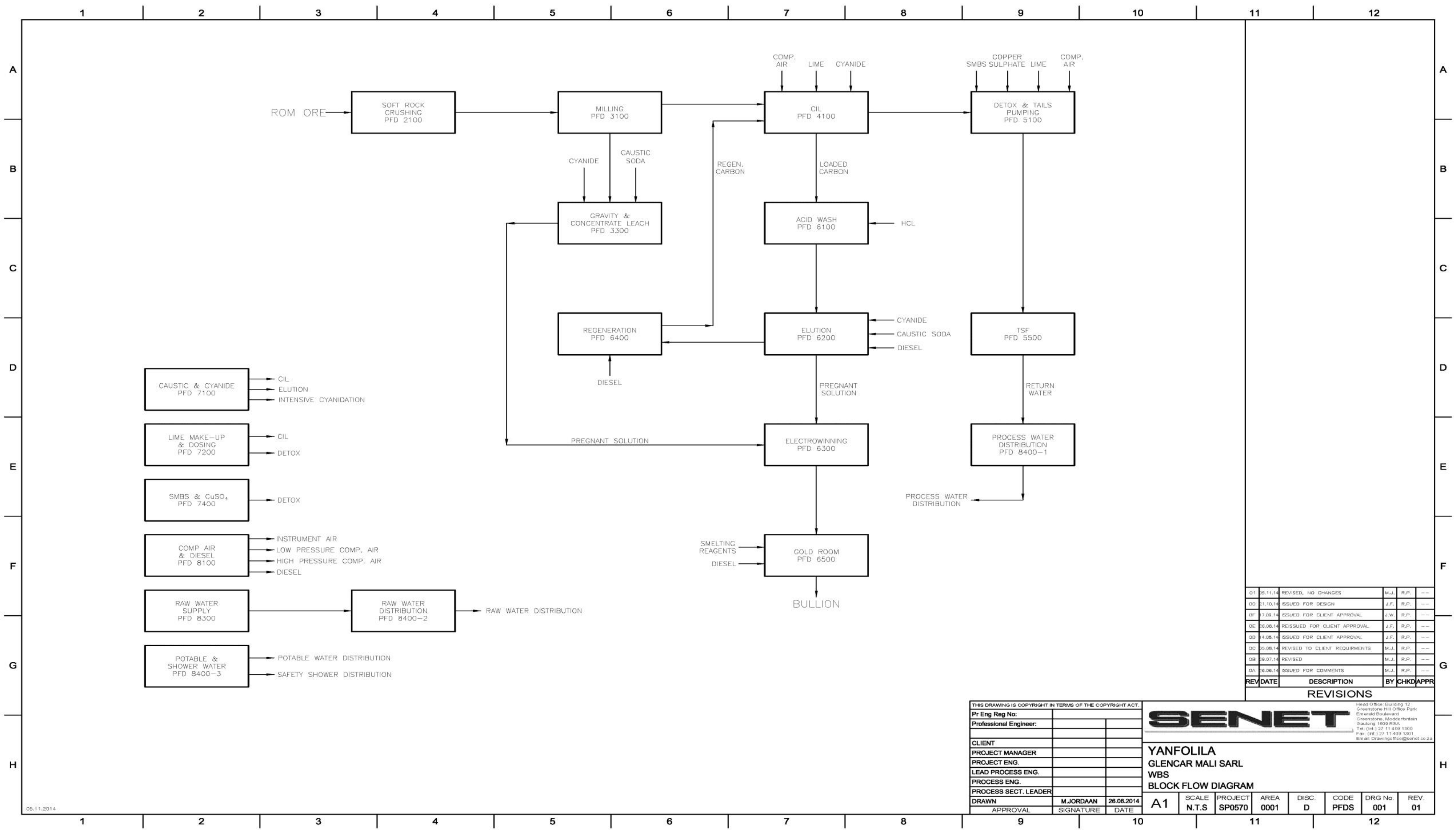

Figure 1.3 Process Block Flow Diagram

The Project will store reagents onsite in a secure containment area to ensure safekeeping. Aside from hydrochloric acid and small quantities of laboratory reagents, the bulk chemical reagents will be delivered in the dry powder or briquette form in bags packed on pallets.

1.11 Infrastructure

The overall site plan contains all facilities required to mine and process 1 Mtpa. These facilities include:

- Site Access Road
- Owner's Camp
- Haul Roads
- Process Plant and Infrastructure
- Waste Water Treatment Plant
- Potable Water Treatment Plant
- Tailings Storage Facility
- Waste Rock Disposal Facilities
- Airfield
- Power Plant
- Fuel Depot
- Auxiliary Buildings and Other Infrastructure

Site Access Road: HUM plans to realign the site access road to avoid communities and mining interference. The new alignment will begin at Kona Bridge and continue to the west to the plant site and Komana Camp. The existing main access road from Yanfolila to the Kona Bridge will be upgraded.

Haul Roads: Haul roads will be designed and built to efficiently transport waste to storage facilities and ore to the process plant. Laterite surfaced haul roads will be built with mine waste rock and have berms on either side at the height of the largest vehicle's axle. Haul traffic will be controlled by spotters where haul roads cross local light vehicle roads.

Komana Camp: HUM will expand the existing camp capacity to house a peak of 128 people, consisting of HUM operating personnel, construction management staff, and long term contractors (power plant, fuel depot, onsite laboratory, and mining), expatriate and/or senior staff. The peak requirement will occur during the last several months of construction when the operators' workforce is complete and the construction staff have not yet been reduced. The camp expansion will include: additional accommodation units, an expanded kitchen and dining facility, water supply and treatment, wastewater treatment, and an expanded medical clinic. The camp will provide room and board, a recreation area, laundry facilities, a store, and a first aid clinic.

Process Plant Infrastructure: This area will include: a 6 megawatt (MW) diesel generator power plant and a substation; reagent storage; change house; plant workshop; laboratory; control room, and an event pond to capture and store precipitation runoff from process areas or spillage of process solutions. The layout also provides area for mining contractor's facilities, administrative offices, and a fuel storage facility.

Process Plant: The plant area will contain a soft rock ROM tip, air compressor structure, ball mill ball storage, ball mill area, intensive cyanidation area, CIL area, carbon regeneration kiln, cyanide detoxification area, raw and process water tanks, gold room, and tower crane to service the CIL agitator shafts and interstage screens.

Tailings Storage Facility (TSF): The TSF will be situated midway between KE and KW pits, to the northeast of the processing plant. The TSF will contain and store residual tailings after they are treated. It is designed to accommodate an average 1 Mtpa tailings deposition equivalent to 740,741 m^3 (in situ (dry) tailings density estimated at 1.5 t/m^3). The TSF is a valley impoundment embankment utilizing waste rock to build the outer shell. In Phase 1 (initial 1 year of production) the embankment will be raised to a crest elevation of 372 masl, and in Phase 2 with multiple centerline lifts to a final crest elevation of 378 masl. Topsoil across the basin will be stripped to 0.15 m depth and stockpiled for use in rehabilitation work. Tailings will be deposited nearest the embankment edge and to the west of the facility, with the principle being to keep the water pond as far from the embankment as possible. Water from the supernatant pond will be recycled to the process plant for make-up via barge-mounted reclaim pump and a pipeline. The basin will not have a synthetic liner, as the in situ materials have very low permeabilities. The upstream slope of the embankment, a seepage intercept trench, and sump placed under the embankment wall will be lined with a geomembrane. A stormwater drainage diversion canal and an access road will ring the perimeter of the facility. The TSF will be fenced and warning signs erected.

Waste Rock Disposal Facility: Waste rock will be hauled to one of seven WRDs on the Project. The preliminary locations and footprints of these facilities are sketched in Figure 1.2 above.

Airfield: The Project will provide a landing strip for small aircraft. HUM intends to use an aircraft to deliver gold doré to Bamako, where it can be readily and safely transported to a refinery. The aircraft will also be used to transport HUM staff and visitors between Bamako and the mine.

The Project will also construct a warehouse, workshops, administration building, medium voltage substation, and laboratory. Overhead power lines will connect the substation with the power plant and with satellite facilities, such as the administration building, Komana Camp, and remote pumping locations. In cases where remote pumping requirements are small, local diesel generators will supply electrical power, or pumps will be directly driven by diesel engines.

1.12 Environmental

HUM will develop the Project sustainably, creating shared value for all stakeholders as a central commercial priority. HUM's policy is to work in the most socially and environmentally responsible way possible.

As a result, the Project team will implement environmental standards meeting or exceeding Malian regulations, international best practices, and HUM internal policies and procedures. This plan applies to all HUM staff and contract personnel, HUM subcontractors including SENET, and HUM management control, visitors to the Project, and personnel residing at HUM premises or working under HUM systems.

1.13 Project Economics

The results of this optimization study show a payback to occur early in the mine life, approximately 2.6 years after start of production. The base case financial model was developed from information described in Table 1.1.

Table 1.1 Economic Model Inputs

Description	Values
Construction Period	12 months
Life of Mine (LOM)	6.5
LOM Ore (tonnes)	6,414,000
LOM Processing Plant Feed Grade	2.65 g/tAu
LOM Gold Production (kozs)	514
Average Gold Recovery	94.00%
Average Annual Gold Production (ozs)	73,373
Gold Price	$1,250/oz (flat)
Inflation/Currency Fluctuation	None
Leverage	100% Equity
Income Tax	35% min.; 0.75% of Revenues
Carry Forward Tax Losses	3 years
Withholding Tax	15%
Stamp Duties on Mineral Product Export	0.60%
Depreciation	Straight line
Value Added Tax (VAT)	15%
VAT Payment/Recovery	Included (to commence 3 years after start of production)
Government Royalty ("Special tax on Certain Products")	3%
LPMDO Royalty	1%
Transportation and Refining Charges	$4.807/oz

Based upon this information, the Yanfolila Project is estimated to have an after-tax IRR of 35.1%. Assuming a discount rate of eight percent over an estimated mine life of 6.5 years, the after-tax NPV is estimated to be $72.4M.

2.0 LOCATION AND CLIMATE

2.1 Climate and Environmental Conditions

2.1.1 Climate

The Project is located in the Malian Sudano-Guinean area, which has a seven month dry season (October to May), and a five month rainy season (May to October). Average annual rainfall is 1,100 mm, but with significant variability across the year. Maximum temperatures (around 40°C) are in the months of March and April, and minimum temperatures (around 15°C) in December and January. Prevailing winds are northerly, north-easterly, and easterly between December and March, and southerly, south-westerly, and westerly from April to October.

2.1.2 Geomorphology and Landscape

The Project is comprised of areas of low-rolling hills at altitudes between approximately 315 masl on the shore of the Sankarani River, to approximately 455 masl at KW, and approximately 440 masl at KE. The relief is a lateritic plateau rising from the banks of the Sankarani River towards the east and the south across most of the southern and eastern permit area. The winding valley of the Sankarani River is an important component of the landscape.

2.1.3 Water Resources

The main surface water body in the area is the Sankarani River, which flows northwards into Mali. The Sankarani River drains a watershed of about 34,000 km^2 at the Selingue Dam, located approximately 50 km downstream of the Project area. Its flow rate is seasonally variable, with a reported monthly average varying between a minimum of 53 m^3/s in March at the end of the dry season and 964 m^3/s in September at the end of the rainy season. The Project area and adjacent land are crossed by several seasonal streams (dry during part of the dry season).

The north of Guinea and the south of Mali have relatively low populations and extremely low industrialization; it is likely that there are no significant inputs of industrial contaminants or sewage into the Sankarani River upstream of the Project area.

Three types of aquifer systems have been recognized in the Project area. These are:

- A shallow aquifer contained in laterites near to surface above a less permeable substratum (clay)
- A weathered aquifer in the transition zone above the fresh rock
- A fractured rock aquifer in the fresh rock (sandstone and basalt)

2.2 Physiography

The landform of the region is flat to gently undulating with minor drainage channels feeding into the Sankarani River, a tributary of the Niger River, that wraps around the west side of the KW deposit. This river marks the international border between Mali and Guinea. The Sankarani River and the channels both feed the Selingue Hydroelectric Dam.

Cuirasse laterite plateaus cover a significant portion of the land holding and are typically 3 to 5 m thick but in areas can reach 20 m thick. Scarp erosional windows and drainages are present along the edge of these plateaus. The areas adjacent to the Sankarani River are dominated by shallow floodplains which are filled with colluvium after the rainy season. River volumes are controlled by the Selingue Dam causing these areas to become dry from December to May when the dam levels fall.

Vegetation is sparse and consists of short seasonal grasses, sporadic small bushes, and trees. More succulent species grow along the banks of drainage features. In the summer rainy season, crops are grown on Quaternary sediments in the depositional regimes.

3.0 GEOLOGY, MINERALIZATION, RESOURCE, AND EXPLORATION

3.1 Regional Geology

The Yanfolila Greenstone Belt, which is orientated north-south on the eastern margin of the greater Siguiri Basin hosts the Project. This belt forms part of the Birimian Volcano-Sedimentary series of the West African Craton. The belt contains several sub-basins including the KMSB and the KSB. These sub-basins, and the eastern margin of the wider Siguiri Basin, are bound to the east by the SSZ and the Siekerolé Granite. The SSZ is the major basin-bounding structure in the region. Mineralisation is typical lode gold-style deposits structurally reminiscent of a foreland fold and thrust belt with a deformational framework similar to the Ghanaian Gold Province.

The KMSB hosts the majority of the Yanfolila gold deposits and targets currently defined. It has a stratigraphic sequence of basalt, polymict conglomerate, feldspathic sandstone, silt-shale, and a lithic-dominated greywacke. Mafic and felsic (porphyry, granodiorite, and diorite) intrusives crosscut the stratigraphy. Splays of the SSZ and regional-scale structures, which crosscut strata, control gold mineralisation. Gold is associated with rheological contrasting rock types that have been structurally deformed to provide fluid pathways from where any combination of disseminated, vein, breccia, intrusive, and replacement styles may evolve.

Outcrop is sparse throughout the basin and is mostly extensive lateritic and depositional terrains.

3.2 Resource Parameters

3.2.1 Interpolation Method (OK vs UC)

The constrained resources reported by GF in 2012 were estimated using Ordinary Kriging (OK). The unreported resource updates completed as part of GF's De-Risking Study (DRS) had the main resources estimated by Conditional Simulation (CS) to provide a better definition of high grade zones and a recoverable local selective mining unit (SMU) estimate.

HUM initially considered using OK for their GF resource update, applying a constraint to the high grade zones to prevent grade smearing. It became obvious that defining a high grade domain based on geology in oxide was impossible. CSA Global (CSA) evaluated another recoverable local estimator, Uniform Conditioning (UC), comparing resource estimations using OK (with high grade domaining), CS, and UC on domains in KE. The global resource numbers generated by each estimator differed little. Geologists decided to use UC as it is considered a better estimator to a SMU scale when using widely spaced data, which is the prevailing situation for the Yanfolila deposits.

3.2.2 Density

GF completed an extensive review of dry oxide density values determined from the core tray method, weight-in-air/weight-in-water, and laboratory testwork prior to finalising on densities to be used in the DRS. The main issue with the oxide density data is that it was not representative of oxide material associated with the mineralisation domains. As a check, GF initiated four vertical diamond core holes, two each at KE and KW, to have laboratory density determinations on the recovered oxide material. The laboratory results suggest that the DRS density for saprolite may be over-called by 5 to 10% and that the density for transitional may be under-called by 5 to 10%.

CSA used the same dry oxide densities recommended by GF for the DRS in the recently completed Yanfolila Project Mineral Resource estimates (MREs). A recent review of oxide densities, based on core tray densities collected from the recent 2014 drilling programmes and from pitting undertaken on all six deposits, has confirmed the density values assigned in the MRE for the upper saprolite zone and for the transitional zone.

3.2.3 Quality Assurance/Quality Control Review

CSA reviewed of all available Quality Assurance/Quality Control (QAQC) data for each deposit as part of the MRE process and have not identified any material concerns.

Umpire assay results from the recent 2014 drilling programme suggest the comparative results are accurate but imprecise, indicative of no bias but large random errors. Quantile-Quantile (Q-Q) plots indicate good correlation. The order of magnitude of the differences is not material. The QAQC standard reference material (SRM) highlight a negative bias in the results from ALS over the grade range of the SRMs submitted. This would explain the higher SGS values generally over this grade range in the comparison of the umpire results.

3.2.4 Topography

Due to time and budget limitations, GF elected to use satellite data in the Komana area. This data was processed using the UTM 29 GEOIDAL Model EGM 96 elevation datum and provided as the 2013 DEM. Any local accuracy issues were resolved using available ground surveys and drill hole collar surveys.

In the Sanioumale area, geologists based the topography on drill hole collars and if available other survey control such as spot heights and data related to geophysical surveys. An issue occurred with the data generated for SE when matching the resource/pit with the hydrology modelling. To overcome this issue it was decided to use space shuttle LiDAR contours. There are discrepancies between the DGPS collars and the LiDAR averaging three meters, which is estimated to possibly lead to a 10% over-call on resource tonnes.

3.2.5 Komana East

3.2.5.1 Local Geology and Mineralisation

Massive basalts are structurally emplaced on top of a mature, polymict conglomerate grading to a cyclical sequence of siltstone and sandstone. Stratigraphy strikes from 010 to 030° with moderate to steep dips to the west. A series of mafic intrusives intrude the sedimentary rocks and are the dominant gold host. Mineralisation is controlled by a north-south structure corridor and hosted by breccia zones within a mafic dyke swarm, which extends into a siltstone unit at the flexure of the structure to 020°. Veining is quartz-albite-carbonate with associated hematite-pyrite alteration. The mineralisation forms shallow, northerly plunging shoots. Weathering extends to depths of 30 to 50 m, with a well-developed regolith zone.

3.2.5.2 CSA Review of GF DRS Resource

CSA completed a review of GF's CS resource in early July 2014 for the oxide material and in August 2014 for the fresh material. CSA generally honoured domains generated by GF, except where they were implicitly modelled using the Leapfrog software. This specifically related to the northern sediment hosted domain (5950). The implicit modelling produced odd shapes and terminations, as well as generating a number of grade domains based on grade-only assumptions. CSA completed standard section domain modelling, inclusive of the implicitly modelled domains, which had the overall effect of increasing tonnage.

All domains in the weathered and fresh material have been re-estimated using UC. The entire oxide and transitional resource were re-categorised as Indicated, negating the need to undertake any resource conversion drilling.

3.2.5.3 Depletion and Final MRE

CSA used one metre composites to align with the dominant sampling interval. They applied a topcut of 40 g/tAu to control the influence of extreme grades. The variograms modelled were robust, and considered suitable for use in change of support required for UC. The parent block size was 20m x 20m x 3m which is approximately half the average drill spacing. The SMU size was 2.5m x 5m x 3m, in line with planned selectivity dimensions. The grade interpolation was conducted in four passes; the effective search distances for the reported model ranging from 30m x 20m x 10m to 200m x 100m x 15m. The data criteria used in each search pass were progressively relaxed from an initial minimum of 15 samples to a final minimum of six samples. No octant search was used. A minimum of three holes were required to estimate blocks in the first search, with one hole required in the final search. A recoverable MRE was produced using UC. This allowed for the estimation to an SMU scale while using widely spaced data ranging from 20m x 20m to 80m x 80m.

Material classified as Indicated has been drill tested on grids ranging from 20m x 20m to 40m x 40m to 40m x 80m. Statistical analysis of the samples and estimation of the grade and volume for a range of gold cut-offs has been verified by previous estimates, using CS, which showed very similar results.

The block model was depleted for mining using a 2D cookie cut approach applied by GF based on recorded orpaillage data. Depletion equated to 5,336 ozs.

The MRE for the weathered material at an economic cut-off of 0.6 g/tAu, compared with the GF DRS resource, is shown below in Table 3.1.

Table 3.1 Komana East - CSA and GF MRE at 0.6 g/tAu Cut-off for Weathered Material

Classification	Regolith State	CSA Tonnes	CSA Grade	CSA Metal	GF Tonnes	GF Grade	GF Metal
Indicated	Oxide	1,026,161	3.20	105,419	811,625	3.27	85,330
	Transitional	571,923	3.00	55,072	387,965	3.33	41,477
	Total	1,598,084	3.12	160,491	1,199,589	3.29	126,807
Inferred	Oxide	0	0.00	0	110,960	1.76	6,264
	Transitional	0	0.00	0	13,316	1.69	722
	Total	0	0.00	0	124,275	1.75	6,987
Totals	Oxide	1,026,161	3.20	105,419	922,584	3.09	91,594
	Transitional	571,923	3.00	55,072	401,280	3.27	42,199
	Total	**1,598,084**	**3.12**	**160,491**	**1,323,864**	**3.14**	**133,793**

The MRE for fresh material at an economic cut-off of 0.8 g/tAu, compared with the GF DRS resource, is shown below in Table 3.2.

Table 3.2 Komana East - CSA and GF MRE at 0.8 g/tAu Cut-off for Fresh Material

Classification	Regolith State	CSA Tonnes	CSA Grade	CSA Metal	GF Tonnes	GF Grade	GF Metal
Indicated	Fresh	5,694,247	3.09	565,856	4,256,511	3.36	459,990
Inferred	Fresh	1,171,359	2.29	86,086	1,527,849	2.65	129,960
Potential	Fresh	0	0.00	0	823,326	3.25	85,960
	Total	**6,865,606**	**2.95**	**651,942**	**6,607,686**	**3.18**	**675,911**

The JORC (Joint Ore Reserves Committee) Table 1 for KE is located in Appendix 1.

3.2.6 Komana West

3.2.6.1 Local Geology and Mineralisation

The dominant stratigraphy is fine-grained siltstone, shale, wacke, and sandstone interlayered with basalt. Also present are minor graphitic schist, siliceous chert, and a porphyritic intrusive. Contacts between the volcanic and sedimentary units are interpreted to be tectonic. Stratigraphy dips to the east and folding repetition is common. Mineralisation is within a north-trending shear zone parallel to the sub-vertical to east-dipping basalt, straddling zones of high competency contrast. A series of narrow, north-dipping gold dilation shoots occur at the intersection of the north-south shear zone and north-northeast structures. High-grade gold zones occur within a stockwork system of conjugate vein sets mostly striking northeast or northwest. Meta-sedimentary rocks represent the main host protolith, containing a significant proportion of the disseminated sulphides and quartz-gold veins. Main alteration minerals are quartz, albite, sericite, chlorite, and epidote with ubiquitous arsenopyrite, pyrite, and pyrrhotite. Weathering extends to depths of 50 to 90 m, with a well-developed regolith zone.

3.2.6.2 CSA Review of GF DRS Resource

A review of GF CS resource was completed in late July 2014 for the weathered and fresh zones. CSA generally honoured the 45 wireframes generated by GF but, as for KE, all those modelled through Leapfrog's implicit modelling have issues that were considered material to the resource estimation. These include insufficient volumes modelled to mineralised intercepts, unrealistic wireframing out of waste intervals, and potential for over-domaining based on grade assumptions alone. The concern was that these would result in lower tonnage and higher grade estimates. CSA undertook re-wireframing of the domains in order to model the volumes adequately.

KW also proved to be more challenging than KE due to the number and size of domains, and as a result poorer variography. This led to problems verifying GF variograms and subsequent poor domain reconciliations. CSA grouped domains in order to overcome these problems.

The overall result of this work was to increase oxide/transitional tonnes by around 60% compared with those generated by GF with only a minor corresponding reduction in grade.

3.2.6.3 Resource Conversion Drilling, Depletion, and Final MRE

Project geologists developed a drill plan to convert Inferred oxide and transitional resources that were identified following the review of the GF DRS Mineral Resource. A total of 1,373 m was completed in 17 rotary core (RC) holes. The domains were updated with the results of the drilling.

While updating models and surfaces based on the recent drilling CSA found that the regolith surfaces used by GF for the DRS resources did not match the regolith data in the database. A re-definition of regolith surfaces based on logging in the database and checking of selected holes onsite resulted in a shallower base of transitional surface than was previously modelled.

CSA used one metre composites for the final MRE, aligning with the dominant sampling interval. Topcutting was applied to data to control the influence of extreme grades prior to estimation, and was set for each of the separate mineralisation domains ranging from 18 up to 90 g/tAu. Variograms were modelled for each domain and were characterised by moderate to high nuggets and ranges in the region of 60 m in the main direction. They were considered suitable for use in change of support required for UC. The parent block size was 20m x 20m x 3m which is approximately half the average drill spacing in the north direction, and equal to the drill spacing in the east direction. The SMU size was 2.5m x 5m x 3m. The grade interpolation was conducted in four passes, the effective search distances for the reported model ranging from 80m x 30m x 10m to 200m x 60m x 20m. The data criteria used in each search pass were progressively relaxed from an initial minimum of 15 samples to a final minimum of one sample. No octant search was used. A minimum of three holes were required to estimate blocks in the first search, a minimum of two holes in the second search, and no restriction on minima for searches three and four. CSA produced a recoverable MRE using UC. This allowed for the estimation at an SMU scale while using wider spaced data.

Material classified as Indicated has been drill tested on a 40m x 20m spaced grid. The quality of geological logging and subsequent lithology boundary interpretation, and quality of gold grade determination from drill hole sampling and chemical assay is adequate for this level of classification.

Depletion has been modelled using GF recorded orpaillage data, plotted as ‘collar’ points with depth estimates of workings. It has been estimated using a nearest neighbour (NN) estimation approach which allows the orpaillage ‘down hole samples’ to inform neighbouring blocks. The true extent of underground workings is unknown, so there is uncertainty over the true volume of depletion. Estimated depletion equated to 34,592 ozs.

The MRE for the weathered material at an economic cut-off of 0.6 g/tAu, compared with the GF DRS resource, is shown in Table 3.3.

Table 3.3 Komana West – CSA and GF MRE at 0.6 g/tAu for Weathered Material

Classification	Regolith State	CSA Tonnes	CSA Grade	CSA Metal	GF Tonnes	GF Grade	GF Metal
Indicated	Oxide	2,152,717	2.33	161,510	727,180	2.68	62,664
	Transitional	357,283	2.21	25,333	576,939	2.57	47,583
	Total	2,510,000	2.32	186,842	1,304,119	2.63	110,247
Inferred	Oxide	0	0.00	0	345,419	1.70	18,899
	Transitional	0	0.00	0	170,346	1.56	8,537
	Total	0	0.00	0	515,765	1.65	27,437
Total	Oxide	2,152,717	2.33	161,510	1,072,598	2.37	81,564
	Transitional	357,283	2.21	25,333	747,285	2.34	56,120
	Total	**2,510,000**	**2.32**	**186,842**	**1,819,883**	**2.35**	**137,684**

The MRE for fresh material at an economic cut-off of 0.8 g/tAu, compared with the GF DRS resource, is shown in Table 3.4.

Table 3.4 Komana West – CSA and GF MRE at 0.8 g/tAu for Fresh Material

Classification	Regolith State	CSA Tonnes	CSA Grade	CSA Metal	GF Tonnes	GF Grade	GF Metal
Indicated	Fresh	3,234,077	2.48	257,705	908,455	3.80	111,121
Inferred	Fresh	751,266	2.06	49,659	896,168	2.53	72,844
Potential	Fresh	0	0.00	0	770,945	2.73	67,581
	Total	**3,985,343**	**2.40**	**307,364**	**2,575,568**	**3.04**	**251,546**

KW's JORC Table 1 is located in Appendix 1.

3.2.7 Guirin West

3.2.7.1 Local Geology and Mineralisation

This deposit was initially considered to be depositional in a paleo-channel, but suffers from a lack of holes with intersections below the base of weathering. Local lithologies include north-south to north-northeast trending, interbedded siltstone and sandstone metasediments, and basalt. Based on recent drilling, the mineralisation now appears to be related to a porphyry intrusive and volcanoclastics hosting a stockwork of quartz-feldspar-carbonate veining. Small, discontinuous mineralised zones occur in shear zones and are characterised by albite alteration with associated quartz veining and pyrite. Weathering generally extends to a depth of 50 to 70 m, with a well-developed regolith zone.

3.2.7.2 CSA Review of GF DRS Resource

CSA completed a review of GF OK resource in early September 2014. The review and update to domains completed by CSA came up with essentially the same numbers as GF. Material issues relating to the review estimate are:

- Both the GF and CSA estimates use OK due to the poor variograms generated by the data. The nugget for KE was assigned as no nugget could be determined from the GW data; and

- The entire resource is still classified as Inferred pending the air core (AC) drilling review.

3.2.7.3 Air Core Evaluation

AC data has the potential to make up 80% of the dataset used in the MRE. Review of the different hole types in the dataset suggests AC and RC/DC data have similar grade distributions and mean grades within mineralised composites (≥0.3 g/tAu over 3 m with outliers >20 g/tAu removed). This correlation provides confidence that the AC results are representative of the grade distribution seen with the DC/RC assay results and therefore can be considered for MRE grade interpolation.

3.2.7.4 Resource Conversion Drilling and Final MRE

A drill plan was developed to convert Inferred resources that were identified following the review of the GF DRS Mineral Resource. A total of 2,095 m was completed in 33 RC holes. The domains were updated with the drilling results.

With the majority of the drilling being in oxide and transitional material, the use of sulphide logging (e.g. alteration, veins, texture etc.) was not possible (unlike for other deposits). The controls on mineralisation are less well understood and occur as extremely narrow and very discontinuous zones that are difficult to model. As a result it was decided to model one domain, a zone of potential mineralisation, defining a broad zone around mineralised intercepts, including a large amount of waste. This resulted in a better structured variogram being defined and the ability to interpolate the grades into an SMU using UC.

For the final MRE, one metre composites were used, aligning with the dominant sampling interval. Five percent of the data was based on three metre composite samples; therefore compositing to one meter had the effect of splitting these samples. The effect of such splitting on the spatial continuity models was investigated and material effects are considered low. A topcut of 30 g/tAu was applied to control the influence of extreme grades. One variogram was modelled for the estimate and change of support required for UC. The parent block size is 20m x 50m x 3m and the SMU size is 2.5m x 5m x 3m, in line with planned selectivity. The grade interpolation was conducted in three passes, the effective search distances for the reported model ranging from 70m x 30m x 5m to 80m x 55m x 10m. The data criteria used in each search pass were progressively relaxed from an initial minimum of 12 samples to a final minimum of four samples. No octant search was used. A minimum of three holes were required to estimate blocks in the first and second searches, with one hole required in the final search. A maximum of 32 samples was used in the neighbourhood. A recoverable MRE was produced using UC. This allowed for estimation at an SMU scale while using widely spaced data ranging from 20m x 20m to 80m x 80m.

Material classified as Indicated was based on adequately detailed and reliable exploration, sampling, and testing to assume geological and grade continuity between drill hole samples. The material has been drill tested on grids ranging from 20m x 20m to 40m x 40m to 40m x 80m.

The MRE at an economic cut-off of 0.6 g/tAu for oxide and 0.7 g/tAu for transitional, compared with the GF DRS resource, is shown below in Table 3.5.

Table 3.5 Guirin West – CSA and GF MRE at 0.6 g/tAu Cut-off for Oxide and 0.7 g/tAu for Transitional Materials

Classification	Regolith State	CSA Tonnes	CSA Grade	CSA Metal	GF Tonnes	GF Grade	GF Metal
Indicated	Oxide	873,382	1.97	55,184	0	0.00	0
	Transitional	287,969	2.02	18,730	0	0.00	0
	Total	1,161,351	1.98	73,913	0	0.00	0
Inferred	Oxide	0	0.00	0	856,784	1.94	53,516
	Transitional	0	0.00	0	132,017	1.54	6,536
	Total	0	0.00	0	988,801	1.89	60,052
Total	Oxide	873,382	1.97	55,184	856,784	1.94	53,516
	Transitional	287,969	2.02	18,730	132,017	1.54	6,536
	Total	**1,161,351**	**1.98**	**73,913**	**988,801**	**1.89**	**60,052**

The JORC Table 1 for GW is found in Appendix 1.

3.2.8 Sanioumale West

3.2.8.1 Local Geology and Mineralisation

A sequence of metasediments dominates the deposit, including coarse feldspathic sandstone, finer siltstone, occasional black shale, and some localised polymict conglomerate. Mafic intrusives of varying composition are peripheral. An overall north-south strike and steeply west-dipping stratigraphy contrasts with the identified mineralised structures, which dip steeply to the east. Mineralisation occurrence, controlled by regional scale cross-cutting structures and lineaments and rheological contrasts between rock types, is a stockwork of narrow quartz-albite-carbonate veins, brecciated in places, with pyrite. Weathering generally extends to a depth of 30 to 60 m, with a well-developed regolith zone.

3.2.8.2 CSA Review of GF DRS Resource

CSA completed a review of GF CS resource in late August 2014. The review honoured the wireframes and domains generated by GF and realised essentially the same resource numbers. The main difference is that CSA categorised some 60% of the resources as Indicated where GF had it all as Inferred.

3.2.8.3 Resource Conversion Drilling and Final MRE

Project geologists developed a drill plan to convert Inferred resources that were identified following the review of the GF DRS Mineral Resource. A total of 3,102 m was completed in 46 RC holes and one RCD hole. The domains were updated with the results of the drilling.

For the final MRE, one metre composites were used, aligning with the dominant sampling interval. A topcut was applied to data to control the influence of extreme grades and was applied based on estimation domains, of which there were three, ranging from 13 to 30 g/tAu. The variograms modelled were robust, and considered suitable for use in change of support required for UC. The parent block size was 20m x 50m x 3m which is approximately half the average drill spacing. The SMU size was 2.5m x 5m x 3m. The grade interpolation was conducted in three passes, the effective search distances for the reported model ranging from 100m x 40m x 10m to 300m x 100m x 30m. The data criteria used in each search pass were progressively relaxed from an initial minimum of 12 samples to a final minimum of four samples. No octant search was used. A minimum of three holes were required to estimate blocks in the first search, with one hole required in the final search. A recoverable MRE was produced using UC. This allowed for the estimation at an SMU scale while using widely spaced data ranging from 40m x 40m to 40m x 80m.

Material classified as Indicated is based on adequately detailed and reliable exploration, sampling and testing to assume geological and grade continuity between drill hole samples. The material has been drill tested on grids ranging from 40m x 40m to 40m x 80m. Statistical analysis of the samples and estimation of the grade and volume for a range of gold cut-off's has been verified by previous estimates using CS.

The MRE at an economic cut-off of 0.8 g/tAu, compared with the GF DRS resource, is shown below in Table 3.6.

Table 3.6 Sanioumale West – CSA and GF MRE at a Cut-Off of 0.8 g/tAu

	Regolith	CSA			GF		
Classification	**State**	**Tonnes**	**Grade**	**Metal**	**Tonnes**	**Grade**	**Metal**
Indicated	Oxide	1,579,458	2.03	103,316	0	0.00	0
	Transitional	270,042	1.59	13,848	0	0.00	0
	Total	1,849,501	1.97	117,164	0	0.00	0
Inferred	Oxide	257,172	2.40	19,841	1,556,541	2.21	110,388
	Transitional	6,009	3.98	770	340,610	1.84	20,202
	Total	263,181	2.44	20,610	1,897,151	2.14	130,590
Total	Oxide	1,836,630	2.09	123,157	1,556,541	2.21	110,388
	Transitional	276,052	1.65	14,617	340,610	1.84	20,202
	Total	**2,112,682**	**2.03**	**137,774**	**1,897,151**	**2.14**	**130,590**

The JORC Table 1 for SW is located in Appendix 1.

3.2.9 Sanioumale East

3.2.9.1 Local Geology and Mineralisation

North-south to north-northeast trending lithologies, dipping steeply west, include a sequence of basalt and metasediment (interbedded siltstone, sandstone, and polymict conglomerate). Numerous northeast striking faults and intrusives of mafic dyke (dolerite) as well as felsic porphyry occur. Mineralisation is hosted mainly in basalt and the dolerite dyke proximal to the contact with metasediment, and focused on competency contrasts. The mineralisation style comprises quartz-albite-carbonate breccia veins and related pyrite-hematite altered wall rocks associated with brittle-ductile shear zones. Weathering generally extends to a depth of 30 to 50 m, with a well-developed regolith zone.

3.2.9.2 Resource Evaluation Drilling and Estimation

GF expected a potential 150 kozs resource to occur in the oxide based on the work completed. A drill proposal was prepared to evaluate a strike length of 2.2 km on a collar spacing of 40m x 80 m. A total of 6,455.1 m was completed in 12 RCD holes and 41 RC holes.

CSA completed a MRE on the results of the drilling, based on a western mineralised trend, which was tested with the resource evaluation drilling, and an eastern mineralised zone that has a drill spacing averaging 50m x 300m. All of the resource was categorised as Inferred, with drilling planned for the resource conversion of the western zone only. Drilling in the eastern zone is not justifiable at this stage because of the expected amount of identified ounces.

3.2.9.3 Air Core Evaluation

AC data makes up 22% of the dataset used in the MRE. CSA compared the distribution of grades of close-spaced RC and AC data (samples within 2 m). The populations compared reasonably well up to 2 g/tAu, but RC reports higher grades than AC at grades greater than 2 g/tAu. The number of samples compared is low (61), and there is a possibility that the apparent disparity of mean grades is due to a lack of RC data available at the higher grade ranges. A comparison of a larger number of 'twinned' data available at Guirin West, a deposit located approximately 10 km to the south of SE was reviewed. This comparison supported the use of AC data in that MRE, and it has been assumed that AC data is also representative of mineralisation at SE because sampling procedures and drilling protocols were the same as those at GW.

3.2.9.4 Resource Conversion Drilling and Final MRE

A drill plan was developed to convert Inferred resources that were identified following the MRE completed following the resource evaluation drilling. A total of 644 m was completed in 12 RC holes. The domains were updated with the results of the drilling.

A 3D lithological model was developed for basalt, intercalating siltstone and sandstone, granite, felsic porphyry, conglomerate, and sediments, and was based on available logged lithology data and cross referenced to cross-sectional interpretation completed by HUM site geologists. The mineralisation is well constrained to the units modelled and has been tested by infill drilling, providing some degree of confidence in the geological interpretation.

For the final MRE, one metre composites were used, aligning with the dominant sampling interval. Five percent of the data was based on three metre composite samples; therefore compositing to one metre had the effect of splitting these samples. The effect of such splitting on the spatial continuity models was investigated and material effects are considered low. A topcut of 35 g/tAu was applied to control the influence of extreme grades. A single variogram was modelled for all mineralised domains for use in estimation and the change of support required for UC. A nugget of 37% and range of 140 m in the major direction was modelled. The parent block size was 20m x 60m x 3m which is approximately half the average drill spacing and the SMU size is 2.5m x 5m x 3m, in line with planned selectivity. The grade interpolation was conducted in three passes, the effective search distances for the reported model ranging from 150m x 60m x 20m to 450m x 180m x 40m. The data criteria used in each search pass were progressively relaxed from an initial minimum of nine samples to a final minimum of three samples. No octant search was used. A minimum of three holes were required to estimate blocks in the first search, with one hole required in the final search. A recoverable MRE was produced using UC. This allowed for estimation at an SMU scale while using widely spaced data which on average is 40m x 80m.

Material classified as Indicated was based on adequately detailed and reliable exploration, sampling, and testing to assume geological and grade continuity between drill hole samples. The material has been drill tested on a spacing of 40m x 80m.

The MRE at an economic cut-off of 0.8 g/tAu is shown below in Table 3.7.

Table 3.7 Sanioumale East MRE at a Cut-Off of 0.8 g/t Au

Classification	Regolith State	CSA Tonnes	CSA Grade	CSA Metal	GF Tonnes	GF Grade	GF Metal
Indicated	Oxide	497,082	3.06	48,884	0	0.00	0
	Transitional	138,174	2.97	13,203	0	0.00	0
	Total	635,256	3.04	62,088	0	0.00	0
Inferred	Oxide	0	0.00	0	0	0.00	0
	Transitional	0	0.00	0	0	0.00	0
	Total	0	0.00	0	0	0.00	0
Total	Oxide	497,082	3.06	48,884	0	0.00	0
	Transitional	138,174	2.97	13,203	0	0.00	0
	Total	**635,256**	**3.04**	**62,088**	**0**	**0.00**	**0**

The JORC Table 1 for SE is located in Appendix 1.

3.3 Project Resource Statement

Below is the MRE for the Project for this Study to three significant figures which reflects the accuracy of the estimation. For this reason, numbers may not total due to rounding. Table 3.8 shows the weathered zone MRE including oxide and transitional material.

Table 3.8 Project MRE for Oxide and Transitional Material

Deposit	Economic Cut-Off		Regolith Type	CSA Tonnes	CSA Grade	CSA Metal	GF Tonnes	GF Grade	GF Metal
Komana East	0.6	Ind	Oxide	1,030,000	3.20	105,000	812,000	3.27	85,000
	0.6		Transitional	572,000	3.00	55,000	388,000	3.33	41,000
			Total	1,600,000	3.12	160,000	1,200,000	3.29	127,000
	0.6	Inf	Oxide	0	0.00	0	111,000	1.76	6,260
	0.6		Transitional	0	0.00	0	13,300	1.69	722
			Total	0	0.00	0	124,000	1.75	6,990
	0.6	Total	Oxide	1,030,000	3.20	105,000	923,000	3.09	92,000
	0.6		Transitional	572,000	3.00	55,100	401,000	3.27	42,200
			Total	**1,600,000**	**3.12**	**160,000**	**1,320,000**	**3.14**	**134,000**
Komana West	0.6	Ind	Oxide	2,150,000	2.33	162,000	727,000	2.68	62,700
	0.6		Transitional	357,000	2.21	25,300	577,000	2.57	47,600
			Total	2,510,000	2.32	187,000	1,300,000	2.63	110,000
	0.6	Inf	Oxide	0	0.00	0	345,000	1.70	18,900
	0.6		Transitional	0	0.00	0	170,000	1.56	8,540
			Total	0	0.00	0	516,000	1.65	27,400
	0.6	Total	Oxide	2,150,000	2.33	162,000	1,070,000	2.37	81,600
	0.6		Transitional	357,000	2.21	25,300	747,000	2.34	56,100
			Total	**2,510,000**	**2.32**	**187,000**	**1,820,000**	**2.35**	**138,000**
Guirin West	0.6	Ind	Oxide	873,000	1.97	55,200	0	0.00	0
	0.7		Transitional	288,000	2.02	18,700	0	0.00	0
			Total	1,160,000	1.98	73,900	0	0.00	0
	0.6	Inf	Oxide	0	0.00	0	857,000	1.94	53,500
	0.7		Transitional	0	0.00	0	132,000	1.54	6,540
			Total	0	0.00	0	989,000	1.89	60,100
	0.6	Total	Oxide	873,000	1.97	55,200	857,000	1.94	53,500
	0.7		Transitional	288,000	2.02	18,700	132,000	1.54	6,540
			Total	**1,160,000**	**1.98**	**73,900**	**989,000**	**1.89**	**60,100**
Sanioumale West	0.8	Ind	Oxide	1,580,000	2.03	103,000	0	0.00	0
	0.8		Transitional	270,000	1.59	13,800	0	0.00	0
			Total	1,850,000	1.97	117,000	0	0.00	0
	0.8	Inf	Oxide	257,000	2.40	19,800	1,560,000	2.21	110,000
	0.8		Transitional	6,010	3.98	770	341,000	1.84	20,200
			Total	263,000	2.44	20,600	1,900,000	2.14	131,000
	0.8	Total	Oxide	1,840,000	2.09	123,000	1,560,000	2.21	110,000
	0.8		Transitional	276,000	1.65	14,600	341,000	1.84	20,200
			Total	**2,110,000**	**2.03**	**138,000**	**1,900,000**	**2.14**	**131,000**
Sanioumale East	0.8	Ind	Oxide	497,000	3.06	48,900	0	0.00	0
	0.8		Transitional	138,000	2.97	13,200	0	0.00	0
			Total	635,000	3.04	62,100	0	0.00	0
	0.8	Inf	Oxide	0	0.00	0	0	0.00	0
	0.8		Transitional	0	0.00	0	0	0.00	0
			Total	0	0.00	0	0	0.00	0
	0.8	Total	Oxide	497,000	3.06	48,900	0	0.00	0
	0.8		Transitional	138,000	2.97	13,200	0	0.00	0
			Total	**635,000**	**3.04**	**62,100**	**0**	**0.00**	**0**
Total		Ind	Oxide	6,130,000	2.41	474,000	1,540,000	2.99	148,000
			Transitional	1,630,000	2.41	126,000	965,000	2.87	89,100
			Total	7,750,000	2.41	600,000	2,500,000	2.94	237,000
		Inf	Oxide	257,000	2.40	19,800	2,870,000	2.05	189,000
			Transitional	6,010	3.98	770	656,000	1.71	36,000
			Total	263,000	2.44	20,600	3,530,000	1.99	225,000
		Total	Oxide	6,390,000	2.41	494,000	4,410,000	2.38	337,000
			Transitional	1,630,000	2.42	127,000	1,620,000	2.40	125,000
			Total	**8,020,000**	**2.41**	**621,000**	**6,030,000**	**2.38**	**462,000**

The fresh zone MRE including oxide and transitional material is shown in Table 3.9.

Table 3.9 Project MRE for the Fresh Zone

Deposit	Economic Cut-Off		Regolith Type	CSA Tonnes	CSA Grade	CSA Metal	GF Tonnes	GF Grade	GF Metal
Komana East	0.8	Ind	Fresh	5,690,000	3.09	566,000	4,260,000	3.36	460,000
	0.8	Inf	Fresh	1,170,000	2.29	86,100	1,530,000	2.65	130,000
	0.8	Pos	Fresh	0	0.00	0	820,000	3.3	86,000
			Total	**6,870,000**	**2.95**	**652,000**	**6,610,000**	**3.18**	**676,000**
Komana West	0.8	Ind	Fresh	3,230,000	2.48	258,000	908,000	3.80	111,000
	0.8	Inf	Fresh	751,000	2.06	49,700	896,000	2.53	72,800
	0.8	Pos	Fresh	0	0.00	0	770,000	2.7	68,000
			Total	**3,990,000**	**2.40**	**307,000**	**2,580,000**	**3.04**	**252,000**
Total	0.8	Ind	Fresh	8,930,000	2.87	824,000	5,160,000	3.44	571,000
	0.8	Inf	Fresh	1,920,000	2.20	136,000	2,420,000	2.60	203,000
	0.8	Pos	Fresh				1,600,000	3.0	150,000
			Total	**10,900,000**	**2.75**	**959,000**	**9,180,000**	**3.14**	**927,000**

3.4 Exploration

Only the exploration potential for additional oxide resources in proximity to the Project processing plant and within the current exploitation permit is reviewed here. These are known at the time of reporting based on work completed by GF. The ability to locate additional oxide resources is dependent on a review and coverage of regional geochemical programmes.

3.4.1 Komana East - North Extension

A northern structural offset to the west, along a north-south trending shear corridor at the contact of basalt with siltstone/sandstone. Randgold trench dug in the area returned with 19 m at 1.58 g/tAu and Glencar RC drilling along strike returned 10 m at 1.17 g/tAu. Interpreted oxide resource strikes 400 m to a base of regolith depth of 40 m.

3.4.2 Gonka - Komana East Link

A northeast trending shear corridor runs between Gonka and KE. Previous trenching by Randgold returned 10 m at 1.51 g/tAu and 3 m at 1.59 g/tAu. Follow-up RCD drilling by Glencar/GF intersected 4 m at 1.64 g/tAu and 7 m at 1.66 g/tAu. Soil sampling on 20m x 200m returned a strong anomaly. Interpreted oxide anomaly strikes 3 km.

3.4.3 Komana West - North Extension

Extensive orpaillage workings cover this area. Recent RC drilling returned 19 m at 2.81 g/tAu and 14 m at 1.15 g/tAu. Interpreted oxide resource strikes 500 m, dipping west, to a base of regolith depth of 50 m.

3.4.4 Komana West - South Extension

Significant mineralisation intersected in previous RC drilling with 9 m at 2.32 g/tAu and RAB drilling intercepts of 4 m at 5.60 g/tAu, 4m at 1.80 g/tAu, and 4 m at 5.60 g/tAu. Interpreted oxide resource strikes 500 m to a base of regolith depth of 50 m.

3.4.5 Sanioumale East - Eastern Zone

An Inferred resource of 15 kozs is currently defined based on wide spaced drilling. A significant intercept of 5 m at 7.97 g/tAu returned from an RC drill hole. Northeast trending mineralisation associated with several AC intercepts over 1.2 km.

3.4.6 Soloba

Soloba is located 5.5 km south of KW on the same north-south shear corridor, located at the contact between basalt and metasediment. Associated orpaillage workings, and RAB drilling intercepts of 4 m at 2.45 g/tAu, 2 m at 2.05 g/tAu, and 4 m at 7.40 g/tAu. Interpreted oxide resource potential extends over 800 m.

3.4.7 Kama

Kama is 2.5 km south of Soloba, located on the KW north-south shear corridor. Host is north-northwest trending metasediment associated with feldspar alteration rich in tourmaline. RC drilling has intersected 4 m at 6.91 g/tAu and 6 m at 2.25 g/tAu. Interpreted oxide resource potential extends over 500 m, extending to 80 m depth.

4.0 MINING METHODS

The Yanfolila deposit consists of two main clusters of orebodies: in the South there are KW, GW, and KE, and in the North there are SW and SE. The mine will be developed using conventional drill and blast and load and haul mining methods. Whittle was used for strategic mine planning and scheduling and various geotechnical studies were done to determine the pit slope regime.

The mineable resources total approximately 6.4 Mt at an average LOM grade of 2.65 g/tAu, and at a LOM average stripping ratio of 10.6.

The mine production and mill feed rate was set 11 Mt of total material and 1 Mtpa of ore, respectively. A six months preproduction period will allow for personnel and equipment ramp-ups. Workforce and equipment requirements were estimated on the basis of working two 11-hour shifts per day, seven days per week, 365 days per year (allotting five days for weather and holiday-related interruptions). A three-rotating-crew work schedule will be required for continuous operator and maintenance coverage.

4.1 Geotechnical Recommendations

4.1.1 Background

Geotechnical recommendations were provided by Gordon Sweby in January 2015. The geotechnical work which forms the basis of these recommendations is summarised below:

- Geotechnical drilling and core logging (2012/13) – Gold Fields De-risking Study (DRS).
- Acoustic Televiewer downhole surveys (2012/13) – Gold Fields DRS.
- Laboratory Strength Tests (2012/13 and 2014) – Gold Fields DRS and Hummingbird Study.
- Slope Design Analysis (2012/13 and 2014) – Gold Fields DRS and Hummingbird Study.
- The fieldwork and analysis supporting the fresh rock recommendations is captured in AMC report 412040.
- Yanfolila Geotechnical Logging – Glencar Mali SARL, August 2013 and Gold Fields internal report.
- Geotechnical Design Report – Komana West and Komana East Pits and Waste Dumps: Yanfolila Project, August 2013. These reports focused solely on the KW and KE deposits.

Subsequent to the GF DRS work, HUM undertook further geotechnical drilling and sampling at the GW, SW, and SE deposits.

The pit slope recommendations were provided to Mr. William Rose of WLR Consulting, Inc. (WLRC) for input to pit planning studies.

4.1.2 Recommendations

Table 4.1 is from Gordon Sweby's geotechnical recommendations report and provides the overall slope angles, bench angles, berm widths, and bench heights for the pits. The complete geotechnical report can be found in Appendix 2.

Table 4.1 Pit Slope Recommendations

Pit	Material	Sector	Overall Slope Crest-to-Toe Angle (°)	Bench Face Angle (°)	Berm Width (m)	Bench Height (m)
Komana West (Main)	Oxide		31			
Komana West (South)	Oxide		35			
Komana West (Main)	Fresh	1		80	7.5	18
Komana West (Main)	Fresh	2		60	8.5	18
Komana East (North)	Oxide		32			
Komana East (Main)	Oxide		31			
Komana East (South)	Oxide		35			
Komana East	Fresh	1		60	8.5	18
Komana East	Fresh	2		65	10.5	18
Guirin West (1)	Oxide		42			
Guirin West (2)	Oxide		38			
Guirin West (3)	Oxide		33			
Guirin West (4)	Oxide		38			
Guirin West (5)	Oxide		35			
Sanioumale East (1)	Oxide		31			
Sanioumale East (2)	Oxide		35			
Sanioumale East (3)	Oxide		35			
Sanioumale East (4)	Oxide		38			
Sanioumale West (1)	Oxide		38			
Sanioumale West (2)	Oxide		38			
Sanioumale West (3)	Oxide		35			
Sanioumale West (4)	Oxide		38			
Sanioumale West (5)	Oxide		31			
Sanioumale West (6)	Oxide		35			

William Rose provided his interpretation of Gordon Sweby's work in the following tables and diagram, which he will use to perform the detailed mine design.

Table 4.2 Pit Slope Design Parameters – Saprolite Slope Without Ramps

15-Jan-15
Hummingbird Resources - Yanfolila Project
PIT SLOPE DESIGN PARAMETERS - Saprolite Slopes Without Ramp

	Rock
Road Width (w berm & ditch), m =	0.0
Bench Height (BH), m =	4.5

Description Sector / Rock -> Wall Ht, m:	Saprolite 18	Saprolite 27	Saprolite 36	Saprolite 45	Saprolite 54	Saprolite 63	Saprolite 72	Notes
Target Overall Angle (OSA), degrees	52.0	42.0	38.0	35.0	33.0	32.0	31.0	
Interramp Angle (IRA), degrees	38.0	38.0	35.5	33.0	31.0	30.5	29.5	Toe to toe
Bench Face Angle (BFA), degrees	68	68	68	68	68	68	68	
Catch Bench Interval (x BH)	1	1	1	1	1	1	1	
Ramp step out, m	0.0	0.0	0.0	0.0	0.0	0.0	0.0	
Catch Bench Width (CBW), m	3.94	3.94	4.49	5.11	5.67	5.82	6.14	3.9-m min. CBW target
Contour Offsets, m:								
Faces	1.82	1.82	1.82	1.82	1.82	1.82	1.82	
Across catch benches	5.76	5.76	6.31	6.93	7.49	7.64	7.95	
Roads on faces	1.82	1.82	1.82	1.82	1.82	1.82	1.82	
Roads across catch benches	5.76	5.76	6.31	6.93	7.49	7.64	7.95	
Switchbacks on faces	1.82	1.82	1.82	1.82	1.82	1.82	1.82	
Switchbacks across catch benches	5.76	5.76	6.31	6.93	7.49	7.64	7.95	
Computed OSA, degrees	43.3	41.4	38.1	35.0	32.7	31.9	30.7	Bottom toe to pit crest

Table 4.3 Pit Slope Design Parameters – Saprolite Slopes With One Internal Ramp

15-Jan-15

Hummingbird Resources - Yanfolila Project

PIT SLOPE DESIGN PARAMETERS - Saprolite Slopes With One Internal Ramp

	Rock
Road Width (w berm & ditch), m =	27.0
Bench Height (BH), m =	4.5

Description	Sector / Rock -> Wall Ht, m:	Saprolite 18	Saprolite 27	Saprolite 36	Saprolite 45	Saprolite 54	Saprolite 63	Saprolite 72	Notes
Target Overall Angle (OSA), degrees		52.0	42.0	38.0	35.0	33.0	32.0	31.0	
Interramp Angle (IRA), degrees		38.0	38.0	38.0	38.0	38.0	38.0	36.5	Toe to toe
Bench Face Angle (BFA), degrees		68	68	68	68	68	68	68	
Catch Bench Interval (x BH)		1	1	1	1	1	1	1	
Ramp step out, m		27.0	27.0	27.0	27.0	27.0	27.0	27.0	
Catch Bench Width (CBW), m		3.94	3.94	3.94	3.94	3.94	3.94	4.26	3.9-m min. CBW target
Contour Offsets, m:									
Faces		1.82	1.82	1.82	1.82	1.82	1.82	1.82	
Across catch benches		5.76	5.76	5.76	5.76	5.76	5.76	6.08	
Roads on faces		28.82	28.82	28.82	28.82	28.82	28.82	28.82	
Roads across catch benches		32.76	32.76	32.76	32.76	32.76	32.76	33.08	
Switchbacks on faces		55.82	55.82	55.82	55.82	55.82	55.82	55.82	
Switchbacks across catch benches		59.76	59.76	59.76	59.76	59.76	59.76	60.08	
Computed OSA, degrees		21.3	25.1	27.5	29.2	30.4	31.3	31.0	Bottom toe to pit crest

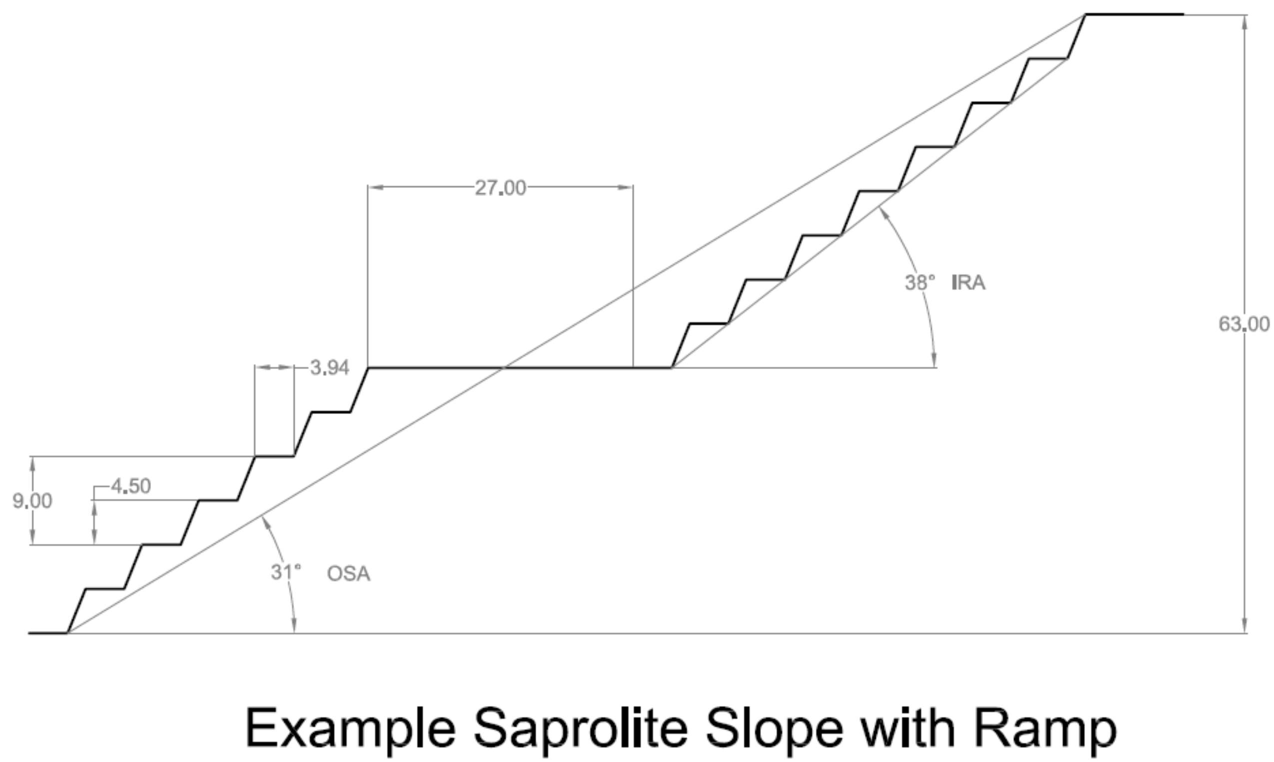

Figure 4.1 Example Saprolite Slope with Ramp

Table 4.4 Pit Slope Design Parameters

14-Jan-15

Hummingbird Resources - Yanfolila Project

PIT SLOPE DESIGN PARAMETERS

	Rock
Road Width (w berm & ditch), m =	27.0
Bench Height (BH), m =	9.0

Proposed
V

Description	Pit / Rock -> Sector:	KE Fresh 1	KE Fresh 2	KW Fresh 1	KW Fresh 2	KW Fresh 1 (MGS)	Notes
Interramp Angle (IRA), degrees	toe-to-toe	43.6	43.6	55.0	43.6	59.3	Toe to toe
Bench Face Angle (BFA), degrees		60.0	65.0	75.0	60.0	80.0	
Catch Bench Interval (x BH)		2	2	2	2	2	
Conventional minimum CBW, m		8.10	8.10	8.10	8.10	8.10	
Catch Bench Width (CBW), m		8.51	10.51	7.78	8.51	7.51	
Contour Offsets, m:							
Faces		5.20	4.20	2.41	5.20	1.59	
Across catch benches		13.71	14.71	10.19	13.71	9.10	
Roads on faces		32.20	31.20	29.41	32.20	28.59	
Roads across catch benches		40.71	41.71	37.19	40.71	36.10	
Switchbacks on faces		59.20	58.20	56.41	59.20	55.59	
Switchbacks across catch benches		67.71	68.71	64.19	67.71	63.10	
Double-bench expansions, m		18.90	18.90	12.60	18.90	10.69	

4.2 Hydrogeologic Studies

4.2.1 Pit Dewatering and Slope Depressurization

Conceptual and 3D numerical groundwater models have been developed for the Project area to encompass all open pits as presented in Schlumberger Water Services (SWS) report located in Appendix 3. The models were constructed in two separate parts to represent the Komana and Sanioumale areas respectively. Modelling has variably been performed to simulate groundwater inflow to the pits under conditions of 'passive flow' and to determine the yields which would be achieved in the event of implementation of active dewatering systems using peripheral wells. Results indicate that under passive pit inflow conditions, groundwater ingress (in addition to runoff into the pits from rainfall) that would require evacuation from the pit sumps would be:

- 36 l/s maximum and 15 l/s average for KW.
- 44 l/s maximum and 25 l/s average for KE.
- 30 l/s maximum and 14 l/s average for GW.
- 27 l/s maximum 8 l/s average for SW.
- 50 l/s maximum and 22 l/s average for SE.

Sump pump capacity of up to 150 l/s at a head of 195 m (about 450 kW) is proposed within each pit to achieve dry pit operation for 95% of operating time. While the volumes of water requiring evacuation under an active dewatering scenario are analogous to those indicated above, active dewatering is concluded within the Study to form a preferred approach due to range of inherent benefits of which the most crucial is the more effective depressurisation of wall rock achieved, the resultant increase of geotechnical stability of the pit walls, and the potential for reduced stripping ratios. Other key objectives likely to be met by the active dewatering comprise:

- Maintenance of groundwater levels below the advancing pit floor at all times during the mining operation.
- Underdrainage of the saprolite by advance pumping from the saprock. This will have the effect of lowering pore pressures in the pit slopes in advance of mining, allowing optimisation of slope designs.
- Maintenance of safer and more efficient working conditions for mining.
- Minimization of loss of access to flooded parts of the pit.
- Reduction of damage to equipment, tyre wear, and haulage costs.
- Production of "clean" water depending on the observed baseline quality of the groundwater abstracted which, if it complies with the relevant water quality guidelines for discharge or human consumption (based on expected use), could potentially be safely discharged without treatment.

For each pit, it is recommended that the installation of peripheral dewatering wells commences approximately six months prior to stripping such that groundwater levels can be lowered ahead of mining. This is predicted by numerical modelling to achieve good under-drainage and depressurization of the saprolite. Most saprolites can be adequately depressurized by under-drainage and the available test pumping data for Yanfolila support this.

Continuous (lagged) drawdown has been observed in all saprolite piezometers in response to test pumping the underlying saprock and fractured bedrock. Some additional in-pit wells or horizontal drainage holes may be required during later stages of mining to help dewater any identified deeper fracture zones.

Groundwater modelling indicates that maximum dewatering rates for each pit will be about 20 to 50 l/s (declining with time) and that dewatering wells will typically have initial yields of up to 10 l/s, decreasing with time to 2 to 3 l/s. Between 10 and 15 wells will be required at any one time for each pit. Wells will be of generally low-cost construction assuming application of the following methodology:

- Installation of RC pilot holes at favourable sites based on interpretation of the structural geology and considering the mine plan. The pilot holes will target the upper 20-30 m of fractured bedrock

- If good fractures are encountered, the pilot hole will be reamed to 254 mm diameter and completed with 200 mm diameter casing and screen. Well depths will be 80-140 m depending on the depth of the bedrock contact. It is anticipated that production pumps will be mostly 100 mm diameter (10 kW or less).

- If no fracturing is encountered and the yield of the pilot hole is poor, the site will be converted to an observation well.

Pore pressure modelling results suggest that early installation of dewatering wells will create a significant reduction in pore pressure within the saprolite and will thus allow slope angles to be increased relative to any passive dewatering scenario. The requirement for horizontal drains to supplement drainage from the dewatering wells is currently uncertain. A highly conservative allowance for 5,000 m per annum of horizontal drain drilling is therefore assumed within the Study.

Propagation of drawdown resulting from pit dewatering is estimated to remain within the Yanfolila property boundary. Little or no dewatering impact is therefore predicted with respect to existing community water supply wells and boreholes.

4.2.2 Waste Rock Disposal Facilities

Dedicated WRDs are planned for each productive pit area. All will be constructed in 10 m lifts with control ditches placed along the haul roads to promote surface water runoff and minimize infiltration. Geochemical test work and modelling indicates that the waste rock in all WRDs will possess greater neutralization capacity than acid generation potential hence acid rock drainage (ARD) hazards are not anticipated. All WRDs runoff will nonetheless be treated as ‘contact water’ and will be collected in peripheral toe-drains and routed to environmental control dams (ECDs) lined with compacted clay or laterite. The ECDs are nominally sized at 5,000 m^3 with a depth of at least two metres to allow removal of total suspended solids (TSS). Overflow from the ECDs will be mixed with diverted non-contact water and will report to the natural drainage system.

4.3 Mining Phase/Pit Designs

HUM retained BWF Mining Consultants, P.C. (BWF) to develop a strategic mine plan for the Project as part of the Study. Detailed mine designs and schedules required for pre-feasibility and feasibility level studies were not included in the scope of work. This work is currently on-going and was scheduled for completion in March 2015. BWF completed three phases of strategic planning:

- Phase 1 – based on the preliminary models from CSA and the GF DRS parameters, adjusted for the lower production rate. This phase provided direction for several components of the Study including the inclusion of sulphide material from KE and KW, the impact of ramps on the overall slope angles (OSA), contract mining costs, and geotechnical and hydrological studies.

- Phase 2 – Was based on the October CSA models which incorporated the infill drilling results and updated parameters for mining costs and pit slopes. Results from this phase were for used in developing the final Study assumptions.

- Phase 3 – Was based on the in situ October CSA models with dilution and loss added in Whittle™ and the production schedule. This also incorporated the January 2015 pit slope recommendations and the updated economic parameters. Results from this phase were incorporated into the Study financial model.

The Project is envisioned as an open-pit contract mining operation with an average production of 10.6 Mtpa to support a 1 Mtpa mill. In total 6.4 Mt of mill feed at an average grade of 2.65 g/t for a total of 547,000 contained ounces and 68.1 Mt of waste are mined. Table 1 shows the Phase 3 annual production schedule. Mining will be from multiple pits with at least two pits active at any time. Bench heights are planned at 4.5 m in the saprolite zone and 9 m the sulphide zone. Benches will be taken in 2 or 3 lifts respectively. Excavators and trucks will be used to load and haul the ore and waste. Drilling and blasting requirements will vary depending on the weathering: oxide material requires minimal blasting, all sulphide material requires blasting. RC grade-control drilling will be done ahead of mining to control dilution.

All phases of mine planning consisted of:

- Determining the Whittle™ input parameters,

- Generating shells for a range of gold prices,

- Skin analysis to select the ultimate pit shell for each deposit,

- Selection of shells for internal phases,

- Generation of mine and mill production schedules using the selected pit shells.

Table 4.5 Annual Production Schedule

		Periods	Years							
	Units	**Total**	**Yr. -1**	**Yr. 1**	**Yr. 2**	**Yr. 3**	**Yr. 4**	**Yr. 5**	**Yr. 6**	**Yr. 7**
Total – MINED										
Waste Mined	Mt	68.14	0.68	11.97	13.75	14.28	11.36	9.64	5.43	1.04
Ore Mined	Mt	6.41	0.12	0.87	0.96	0.97	1.00	1.00	1.00	0.49
Total Material Mined	Mt	74.55	0.80	12.84	14.71	15.24	12.36	10.64	6.43	1.54
Strip Ratio	w:o	10.6	5.9	13.7	14.3	14.8	11.4	9.6	5.4	2.1
Au Grade Mined	g/t	2.65	3.21	3.11	2.83	2.35	2.78	2.60	2.46	2.18
Au Cont. Mined	koz	547	12	87	88	73	89	84	79	35
Total - Mining Cost										
Mining Cost	MUSD	189.7	3.1	28.6	33.6	35.7	32.4	30.2	19.8	6.2
Mining Cost	USD/t-mined	2.54	3.88	2.23	2.28	2.34	2.62	2.84	3.08	4.05
Mining Cost	USD/t-ore	29.57	26.70	32.82	34.85	37.00	32.44	30.14	19.76	12.63
Total – MILLED										
Ore Milled	Mt	6.41	0.00	0.92	1.00	1.00	1.00	1.00	1.00	0.49
Au Grade	g/t	2.65	0.00	3.11	2.86	2.36	2.78	2.60	2.46	2.18
Au Cont.	koz	547	0	92	92	75	89	84	79	35
Recovery (Brittan Eq.)	%	94.0%	0.0%	94.3%	94.1%	93.7%	94.1%	93.9%	93.8%	93.6%
Au Recovered	koz	514	0	87	87	71	84	79	74	32

Note: This schedule includes Stockpiling of material so Mined and Milled tonnes and grades are different in some periods.

As part of Phase 1, ultimate pit designs were developed to determine the potential impact from the inclusion of ramps and to support the contract mining tender. Additionally, the DRS waste dump designs were revised to reflect the change in tonnages and mine layout.

The following conclusions and recommendations regarding mine planning were reached:

- Open-pit mining is a viable option for Yanfolila.
- Contract mining is an appropriate approach to the Project as it reduces the initial capital costs and risks associated with starting a mining operation in a country where HUM has not operated.
- Mining in GW, SE, and SW started in Q 12 because oxide material was needed to blend with the sulphide material so the maximum 50/50 blend and mill throughput requirements could be met.
- Limitations on the vertical rate of advance (18 m/qtr) did limit the production in all the deposits. This was a limitation on bringing higher grade ounces forward.
- Design of the internal phases is critical to the production schedule in terms of bringing ounces forward and smoothing the waste stripping.
- Multiple pits must be mined at the same time in order for there to be space for mining operations to proceed.
- Including sulphide material from KW and KE improves the Project economics significantly.
- Inclusion of sulphide material from other deposits such as Gonka should be considered in future production schedules.
- Detailed mine planning is still required prior to production and a Reserve statement.
- Once the detailed mine planning is completed a final contract mining tender should be completed.

4.3.1 Phase 3 Parameters

Phase 1 and 2 provided interim results used to guide the Phase 3 production schedule. Designed ultimate pits were developed from the Phase 1 results to determine the potential impact on the pits from the inclusion of ramps. Additionally, the Phase 1 results were used as the basis for the contract mining tenders. However, since these are not the basis for the detailed design or Study financial model they will not be discussed in any detail in this report. The remainder of this memorandum will focus on Phase 3 since it represents the current view of the Project.

Phase 3 shell generation and scheduling parameters are shown in Table 2. Shell generation used the Study results that were available January 22, 2015. Subsequent to this the dilution and oxide/sulphide blend parameters were modified. These modifications were included in the schedule but the shells were not revised since the previous parameters were more conservative.

Mining costs are based on the contract mining bids from November 2014 and the SFTP 2013 DRS bid. The SFTP 2014 prices were used except for drilling and blasting, haul road construction, and loading and hauling. SFTP provided a constant rate for drilling and blasting so the AMS bid was used since it had variable rates by material type and for pre-splitting. BCM was the only contractor to provide a cost/km for haul road construction. Load and haul costs were based on the 2013 SFTP bid because it was based on better haulage information and had detailed costs by bench and deposits that could be used as the basis. Owner mining costs were developed based on HUM staffing plans and labour rates. Mining costs were estimated by block using variable costs by deposit, rock type, depth and northing. Details on the derivation of the mining costs are contained in YNF_PH3_Mine_Cost_V6.xlsx located in Appendix 4. Some key points on the mining costs and Phase 3 assumptions are:

- Dilution – For the shell generation the average vein width and a 0.6 m rind (1.2 m total) were used to determine the average dilution. CSA developed the average widths using a block filling routine in Datamine. For the Phase 3 production schedule dilution was estimated using the mineable widths developed by CSA using Datamine plus a 1% grade control estimation error and a 0.4 m excavator tolerance (Table 4.2). In situ tonnes and grade were estimated by bench. Dilution was then applied by bench, including an adjustment to avoid negative tonnes, prior to scheduling. An ounce loss of 5% was applied at the same time. Details on the final schedule dilution and loss assumptions are in PH3_Dilution_Loss_Adj_CSA_30Jan15.xlsx located in Appendix 4.

- Capital Costs – Preliminary capital cost estimates were developed based on the Phase 2 results. These were used to determine the sustaining capital costs that were incorporated into the block mining costs. The development of this is contained in the mine cost file. Capital costs were then revised based on the Phase 3 production schedule. These are detailed in the file Phase3_Schedule_Capital.xlsx in Appendix 4.

- Loading and Hauling Costs – A base for loading and hauling costs was estimated from the SFTP 2013 bid. Hauling costs were then increased with depth at a rate of USD 0.04/9 m from the reference elevation which varied by pit. Due to the length of the deposits an adjustment was also made based on the northing using an overland rate of USD 0.19/BCMxkm. Ex-pit haulage distances for ore and waste were based on the Phase 2 layout which included a revised alignment for the SW/SE access road. Haulage costs were then estimated using the overland rate. An ore premium was included in the costs to cover the lower productivity and other costs associated with mining ore.

- Rehandle – Due to the dump pocket capacity, 100% rehandle has been applied to the mining costs. For KW, KE, and GW this is shown in the rehandle costs since it occurs at the ROM pad. The SW and SE rehandle costs are included in the ore premium since the material is stockpiled near the pits and then transferred to smaller trucks which can direct tip.

- Pre-split – Cost for pre-splitting the transition and sulphide material were estimated based on 89 mm holes, 11 m long and spaced 1 m apart. An average perimeter was estimated from the Phase 2 mid-bench for transition and sulphide for KW and KE. Costs for the selected bench were estimated and divided by the bench tonnage to get an average cost per tonne which was then included in the transition and fresh blasting cost.

- Clear & Grub, Topsoil – Clear and grub requirements were estimated based on the Phase 2 pit crest and Phase 1 waste dump toes. A 20 m buffer was added around these areas. The cost per hectare was then applied to estimate the total costs that was then divided by the Phase 2 tonnages which resulted in the costs per tonne. Topsoil costs were estimated in a similar manner based on a 150 mm depth.

- Grade Control – Costs for grade control were based on a RC drilling program ahead of mining. Details are included in the mine cost file. Ore spotters were included in the mine owner's staffing requirements.

- Dewatering – Active dewatering costs were included based on the Schlumberger capital and operating cost estimates. Pit dewatering costs were based on the SFTP rate of USD$170,000/year/pit. This rate assumes 5 months of dry season at USD$10,200/month and 7 months of wet season at USD$17,000/month. Details on the dewatering cost are in the mine cost file.

- Fuel Cost – Both the 2013 and 2014 bids were based on a fuel price of USD$1.00/L. Since then fuel prices have dropped to USD$0.723/L. Drilling and blasting costs were reduce by 4% to reflect this drop in price. Load and hauling costs were decreased based on 20% of the total cost being fuel.

- OSA Whittle™ - An estimate of the impact of ramps was made based on the Phase 1 designed ultimate pits. This was then applied to the IRA resulting in the OSA applied in Whittle. Table 4.2 shows the OSA allowance by deposit and sector.

- Recovery – A recovered gold grade was calculated by block based on the gold grade and the recovery equation provided by Brittan Process Consulting, LLC in December 2014.

- Transition – Since the transition material is variable with characteristics that range from oxide to sulphide the assumption for costing and scheduling was that 50% would be treated as oxide and 50% as sulphide.

- KW Floodplain – The maximum water elevation behind the Selingue dam is 349.1 m. This was used as the floodplain elevation. A 50 m offset was then applied to limit the KW pit on the west side. GF conducted a survey to set the location of this elevation.

Hydrological models have been developed for the Komana and Sanioumale areas. These models used the Phase 1 pit shells to develop the passive and active water surfaces. The resulting surfaces were used in the geotechnical analysis. Details on the hydrological modelling can be found in Appendix 3 (SWS Report).

Geotechnical drilling, sampling and testing has been completed for each of the deposits. More information is available for KE and KW because GF focused on these since they contain most of the defined gold and are where mining would start. Based on this information a series of geotechnical studies with inter-ramp slope angle (IRA) recommendations were completed. The most recent study for the Study can be found in Appendix 2: 15_01_12_Yanfolila Slope Design Parameters (DRAFT).pdf.

Table 4.6 Phase 3 Parameters

Parameter	Units	Phase 3 KE	KW	GW	SW	SE	Notes
Block Models							
Resource Model Categories		Ind., Inf.	Ind., Inf.	Ind., Inf.	Ind., Inf.	Ind., Inf.	Ind. = 2, Inf. = 3
Shell Generation Resource Categories		Ind.	Ind.	Ind.	Ind.	Ind.	
Metal Prices							
Gold Price	USD/t. oz	1,250	1,250	1,250	1,250	1,250	Per Tarus Contract. Provided in email from Tom Nov. 21.
Discount Rate	%	9.0%	9.0%	9.0%	9.0%	9.0%	Per Tarus Contract. Provided in email from Tom Nov. 21.
Royalties & Other USD/oz Costs							
Societe Malienne de la Petite Mine D'Or SAR	%	1.0%	1.0%	1.0%	1.0%	1.0%	per Nov. 21 email from Tom.
Tarus Royalty	%	1.0%	1.0%	1.0%	1.0%	1.0%	
Mali Royalty	%	3.0%	3.0%	3.0%	3.0%	3.0%	
Stamp Duty	%	0.6%	0.6%	0.6%	0.6%	0.6%	per Nov. 22 email from Tom.
Income Tax Revenue	%	1.00%	1.00%	1.00%	1.00%	1.00%	
Total Royalty - Au	%	6.6%	6.6%	6.6%	6.6%	6.6%	
Net Gold Price (After Refining, Royalties)	USD/t. oz	1,161.43	1,161.43	1,161.43	1,161.43	1,161.43	
Net Gold Price (After Refining, Royalties)	USD/g	37.34	37.34	37.34	37.34	37.34	
Unit Operating Cost							
Fixed Mining Costs							
Owner Mining Cost	USD/yr	1,529,096	1,529,096	1,529,096	1,529,096	1,529,096	
Contractor Fees	USD/yr	1,009,200	1,009,200	1,009,200	1,009,200	1,009,200	Changed to SFTP 2014 Bid Price for Phase 3.
Total Fixed Cost	USD/yr	2,538,296	2,538,296	2,538,296	2,538,296	2,538,296	
Average Annual Mine Production	Mt	10					
Owners Cost	USD/t-mined	0.153	0.153	0.153	0.153	0.153	
Contractor Monthly Fee	USD/t-mined	0.100	0.100	0.100	0.100	0.100	Change to SFTP rate is difference
Clear & Grub	USD/t-mined	0.014	0.020	0.049	0.030	0.044	
Topsoil Removal	USD/t-mined	0.028	0.038	0.095	0.058	0.084	
Dewatering	USD/t-mined	0.067	0.063	0.197	0.076	0.102	Combination owner and contractor.
Dayworks	USD/t-mined	0.040	0.040	0.040	0.040	0.040	
Total Non-Variable Cost	USD/t-mined	0.402	0.414	0.634	0.457	0.523	Base Mining Cost used in MCAF calculation.

Phase 3 Parameters Continued

Parameter	Units	Phase 3 KE	KW	GW	SW	SE	Notes
Variable Mining Costs							
Drilling and Blasting							Variable Mining Cost By Rock Type in MCAF & PCAF calculations.
Laterite Waste	USD/t-mined	0.450	0.450	0.450	0.450	0.450	Phase 3 used AMS rates as provided - adjusted for diesel price change.
Oxide Waste	USD/t-mined	0.090	0.090	0.090	0.090	0.090	
Transition Waste	USD/t-mined	0.477	0.560	0.252	0.252	0.252	Phase 3 includes pre-split
Fresh Waste	USD/t-mined	1.258	1.340	1.032	1.032	1.032	Phase 3 includes pre-split
Laterite Ore	USD/t-milled	0.517	0.517	0.517	0.517	0.517	
Oxide Ore	USD/t-milled	0.103	0.103	0.103	0.103	0.103	
Transition Ore	USD/t-milled	0.477	0.560	0.252	0.252	0.252	Phase 3 includes pre-split
Fresh Ore	USD/t-milled	1.258	1.340	1.032	1.032	1.032	Phase 3 includes pre-split
Incremental Ore Cost							Included in PCAF calculation.
Laterite Ore	USD/t-milled	0.067	0.067	0.067	0.067	0.067	
Oxide Ore	USD/t-milled	0.013	0.013	0.013	0.013	0.013	
Transition Ore	USD/t-milled	0.000	0.000	0.000	0.000	0.000	
Fresh Ore	USD/t-milled	0.000	0.000	0.000	0.000	0.000	
Loading and Hauling							
Laterite Base	USD/t-mined	1.206	1.206	1.206	1.206	1.206	Differences in cost by material type is due to moisture content.
Oxide Base	USD/t-mined	1.206	1.206	1.206	1.206	1.206	These costs are in dry tonnes to match the model.
Transition Base	USD/t-mined	0.986	0.986	0.986	0.986	0.986	
Fresh Base	USD/t-mined	0.894	0.894	0.894	0.894	0.894	
Overland Haulage Cost	USD/BCM*km	0.190					
Incremental Waste Haulage Distance							
Reference Waste Haul	km	1.000					
South - Strike	km	-0.200	1.000	-0.500	-0.300	0.000	
Middle - Strike	km	0.000	0.500	0.000	0.000	0.000	
North - Strike	km	0.200	1.000	0.500	0.500	0.500	
Incremental Waste Haulage Cost							In MCAF calculation USD/BCM is divided by Block S.G. to determine cost/tonne.
South - Strike	USD/BCM	0.152	0.380	0.095	0.133	0.190	
Middle - Strike	USD/BCM	0.190	0.285	0.190	0.190	0.190	
North - Strike	USD/BCM	0.228	0.380	0.285	0.285	0.285	
Incremental Ore Haulage Distance							Total waste haul length subtracted from total Ore haul length.
South - Strike	km	4.300	1.800	4.800	25.100	23.200	
Middle - Strike	km	0.000	2.100	0.000	0.000	24.400	
North - Strike	km	4.000	2.100	4.500	25.700	24.400	
Incremental Ore Haulage Cost							In PCAF calculation USD/BCM is divided by Block S.G. to determine cost/tonne.
South - Strike	USD/BCM	0.817	0.342	0.912	4.769	4.408	
Middle - Strike	USD/BCM	0.000	0.399	0.000	0.000	4.636	
North - Strike	USD/BCM	0.760	0.399	0.855	4.883	4.636	
Incremental Cost/9 m bench	USD/t	0.040	0.040	0.040	0.040	0.040	Included in MCAF since the adjustment is the same for Ore and Waste
Reference Elevation	mRl	418.5	418.5	418.5	409.5	409.5	MCAF calculation - Inc. Cost * (Ref. Elev - Block Elev.)/9 m Bench Height
Incremental Ore Mining Costs							
Grade Control	USD/t-milled	0.180	0.180	0.180	0.180	0.180	
ROM & Stk. Rehandle	USD/t-milled	0.665	0.665	0.665	0.000	0.000	Assumed 100% rehandle. SW and SE rehandle in ore premium.
Ore Premium	USD/t-milled	0.450	0.400	0.450	1.000	1.000	Lower productivity, haul road const./maint., other operational issues.
Total Non-Variable Cost	USD/t-milled	1.295	1.245	1.295	1.180	1.180	Base Incremental Mining Cost used in PCAF calculation.

Phase 3 Parameters Continued

Parameter	Units	Phase 3 KE	KW	GW	SW	SE	Notes
Processing Costs							Variable cost by Rock Type in PCAF
Laterite Process Cost	USD/t-milled	14.14	14.14	14.14	14.14	14.14	per Email from MTB 16 Jan. 2015
Oxide Process Cost	USD/t-milled	14.14	14.14	14.14	14.14	14.14	
Transition Process Cost	USD/t-milled	17.08	17.08	17.08	17.08	17.08	Based on 50/50 split ox/fresh properties
Fresh Process Cost	USD/t-milled	20.02	20.02	20.02	20.02	20.02	
G&A Costs							
G&A Total Annual Cost	USD/yr	7,954,974	7,954,974	7,954,974	7,954,974	7,954,974	per email from MTB Jan. 20 ,2015
G&A Unit Cost	USD/t-milled	7.955	7.955	7.955	7.955	7.955	per email from MTB Jan. 20 ,2015
Unit Sustaining Capital Cost							
Mining	USD/t-mined	0.050	0.050	0.050	0.050	0.050	Included in MCAF
Plant, TSF	USD/t-milled	0.540	0.540	0.540	0.540	0.540	per email from MTB Jan. 20 ,2015
Closure	USD/t-milled	0.500	0.500	0.500	0.500	0.500	per email from MTB Jan. 20 ,2015
Total Included in PCAF	USD/t-milled	1.040	1.040	1.040	1.040	1.040	
Process Recoveries							
Gold Recovery	%	% = 98.6 - 9.5 * {ln(Au + 1)} / Au, Au = Head Grade g/t					Phase 3 recovery formula from Dec. 17, 2014 report by Brittan. Same equation applied to all deposits and rock types.
Treatment Charges & Refining Charges (TCs/RCs)							
Gold Refining Cost - Doŕe	USD/t. oz	6.50	6.50	6.50	6.50	6.50	per 14 Nov. email from Will. Metalor quote at 50 kg and USD 1,500/oz Au price. Included in Refining cost based on USD 1,500/oz Au price.
Physical Parameters							
Minimum Mining Width - Between Phases	m	75	75	75	75	75	
Minimum Mining Width - Pit Bottoms	m	20	20	20	20	20	
Ore Bench Height	m	9/4.5,3	9/4.5,3	9/4.5,3	9/4.5,3	9/4.5,3	4.5 m benches above Fresh rock - per discussions Jan. 14, 2015
Waste Bench Height	m	9.0	9.0	9.0	9.0	9.0	
Target Vertical Advance	m	72	72	54	54	54	
KW Flood Plain Offset	m	N/A	50	N/A	N/A	N/A	
Overall Pit Slopes							
Slope Sector Code 11: KE Above Fresh	degrees	30.0					per WLR OSA provided 16 Jan. 2015
Slope Sector Code 12: KE Fresh West Wall	degrees	38.0					
Slope Sector Code 13: KE Fresh East Wall	degrees	36.0					
Slope Sector Code 21: KW Above Fresh	degrees		30.0				
Slope Sector Code 22: KW Fresh West Wall	degrees		45.0				
Slope Sector Code 23: KW Fresh East Wall	degrees		36.0				
Slope Sector Code 31: GW Above Fresh	degrees			33.0			
Slope Sector Code 41: SW Above Fresh	degrees				32.0		
Slope Sector Code 51: SE Above Fresh	degrees					32.0	

Phase 3 Parameters Continued

Parameter	Units	Phase 3 KE	KW	GW	SW	SE	Notes
Scheduling Parameters							
Phase 3 Production Schedule Dilution & Loss							
Average Mineable Width	m	9.30	8.70	6.00	7.80	5.90	From CSA estimate.
Excavator Tolerance	m	0.40	0.40	0.40	0.40	0.40	
Physical Dilution	%	9%	9%	13%	10%	14%	
Grade Control Error	%	1%	1%	1%	1%	1%	
Total Mining Dilution	%	10%	10%	14%	11%	15%	Applied at zero grade in schedule.
Mining Losses	%	5%	5%	5%	5%	5%	Applied to ounces in schedule not tonnes.
Phase 3 Shell Generation Dilution & Loss							
Average Vein Width	m	10.00	6.00	3.00	10.00	5.00	From CSA estimate. Confirmed by Hum geology staff.
Dilution Rind Width	m	0.60	0.60	0.60	0.60	0.60	
Mining Dilution Whittle	%	112%	120%	140%	112%	124%	
Mining Dilution	%	12%	20%	40%	12%	24%	Applied at zero grade in schedule.
Mining Losses	%	5%	5%	5%	5%	5%	Applied to ounces in schedule not tonnes.
		Year	Qtr.	Month	Qtr. 1		
Process Plant Capacity	kt/period	1,000.00	250.00	83.33	174.99		
Fresh Rock Limit	kt/period	500.00	125.00	41.67	87.51		
Oxide/Fresh Rock Blend	ratio	1.00					
Mill Ramp-up Schedule							per MTB meeting Nov. 20
Month 1 - % of full production	%	50.0%					
Month 2 - % of full production	%	75.0%					
Month 3 - % of full production	%	85.0%					
Month 4 - % of full production	%	100.0%					
Process Plant Capacity - Year 1 w/ Ramp-up	Mt/year	925.000					
Construction Material - TSF, Plant Yr. -1	BCM	127,688					PH 3 per discussions week of Jan. 11, 2015. TSF Cost in Initial & Sustain. capital.
Haul Road Construction Cost	USD/km	96,000					BCM 2014 bid rate.
Diesel Price	USD/l	0.723	0.723	0.723	0.723	0.723	PH 3 per Jan. 15, 2015 emails - MTB, Marcel.

4.3.2 Shell and Phase Selection

A skin analysis of the pit shells generated at the various gold prices was used to select the ultimate pit for each of the deposits. All of the shells were evaluated at a USD$1,250/oz gold price. This analysis considered the change in strip ratio, contained ounces, per tonne margin and total cash flow. The selected shells were generated at gold prices from USD$1,063/oz to USD$1,125/oz which reduces the risk in a decreasing price environment. Table 4.3 contains the tonnes, grade, and contained ounces for the selected pit shells. Details on the shell selection are contained in Appendix 4 in the file: YNF_Ph3_Ult_Pit_Select_V4_NL.xlsx.

Each of the deposits was evaluated separately to determine the internal phases. Determination of these considered minimum mining width. Due to the sizes of the pits no internal phases were selected for GW, SW, and SE. However, the multiple pits at each deposit were assigned phase numbers sequentially from south to north, since the access for these is from the south. For KE and KW the evaluation considered the shells generated with and without sulphide material and directional shells, south to north and north to south respectively. These results were then further evaluated and adjusted in GEMS® based on operational considerations to arrive at the phases used for scheduling. Part of the adjustments in GEMS® involved the removal of benches at the bottom of the pit. These phases are also the basis of the on-going pit designs. The resulting ultimate pits and phases are contained in Appendix 4 in the file: PH3_Phases_Feb2015.zip.

4.4 Contained Mineral Resource Estimates

In this report the term “Ore” is used in the colloquial sense for ease of use as the standards for public reporting of Resources and Reserves have not been met due to the level of detail prior to completion of detailed mine design and the final optimization study.

4.5 Mine Production Schedule

4.5.1 Mining Overview

The proposed development plan for Yanfolila initially starts with a mineral sizer for oxide ore to be followed in later years by a second (hard rock) crushing line, at an annual total throughput rate of 1 Mt of ore.

The mine production schedule runs over seven years and includes a six month preproduction period, commencing six month prior to commissioning of the processing plant.

4.5.2 Scheduling Parameters

The base operating and scheduling parameters used to develop the open pit sequence plans are summarized below in Table 4.7.

Table 4.7 Base Operating and Scheduling Parameters

Parameters	Unit	Value
Annual target ore production rates:		
Y1 - Y7	Mt	1
Average Daily milling rates:		
Y1 - Y7	t	3,000
Operating hours per shift	h	11
Operating shifts per day	shifts/day	2
Operating days per week	d/wk	7
Scheduled operating days per year	d/a	365

4.5.3 Mine Production Scheduling

Whittle's™ multi-mine program was used to develop the Phase 1 and Phase 2 schedules. This experience provided insights on the driving factors in the schedule. There were several constraints placed on the Phase 3 schedule which could not be accurately reflected in Whittle™ so a schedule was developed in Excel. To start, a value/t-milled was developed for each phase to establish their ranking and potential mining sequence. This valuation took into consideration dilution, loss and strip ratios as well as the costs and revenue. The resulting ranking was then used in combination with the other scheduling constrains to generate the production schedule. This methodology is appropriate given the level of the study and that shells are the basis. Once the detailed designs are completed a detailed schedule by period will be developed. This detailed scheduling would also include development of the mine advance maps by period.

The schedule was done on a quarterly basis using the following:

- Six months for preproduction.
- Minimizing the preproduction tonnage.
- Balancing the mine production.
- Delaying mining in GW, SW, and SE as long as possible.
- A three month mill ramp-up at the start of year 1 (Table 4.2).
- A maximum blend oxide/sulphide blend of 50/50.
- Delaying significant sulphide production as long as possible.
- Vertical rate of advance (VRA) was limited to 72 m per year in a phase.
- Stockpiling was incorporated into the schedule so in some periods the mined tonnes and grade do not match the milled tonnes and grade.

Table 4.1 shows the annualized quarterly production schedule. The VRA limit is reached in all the pits and is one of the main limits on production. Production from GW starts in quarter 12 in order to provide oxide to meet the blend limits rather than reducing mill throughput.

For the financial model, preproduction and years 1 and 2 were then reported on a monthly basis by splitting the quarters. The financial model schedule and the detailed mine and mill schedules are contained in Appendix 4: YNF_PH3_Base_Case_Sched_Financial_9Feb15.xlsx and YNF_PH3_Base_Case_Sched_Detail_9Feb15.xlsx, respectively.

Table 4.8 Shell Selection

Deposit	Waste kt	Milled kt	Total kt	Strip Ratio	Au Grade g/t	Au Cont. oz
Revenue Factor 1 Shells (USD 1,250/oz Au) - In Situ Tonnes & Grade						
Komana East	45,280	2,970	48,250	15.2	3.27	311,826
Komana West	28,538	2,367	30,905	12.1	2.98	227,093
Guirin West	3,159	344	3,502	9.2	2.61	28,868
Sanioumale West	7,309	1,106	8,415	6.6	2.29	81,298
Sanioumale East	4,742	412	5,155	11.5	3.40	45,136
Total	89,028	7,199	96,227	12.4	3.00	694,222
Selected Shells - In Situ Tonnes & Grade						
KE - USD 1,125/oz RF 0.90	30,602	2,185	32,787	14.0	3.47	243,799
KW - USD 1,125/oz RF 0.90	25,587	2,177	27,763	11.8	3.06	214,203
GW - USD 1,063/oz RF 0.85	2,676	292	2,968	9.2	2.74	25,704
SW - USD 1,125/oz RF 0.90	6,304	1,003	7,307	6.3	2.35	75,859
SE - USD 1,125/oz RF 0.90	4,319	388	4,708	11.1	3.46	43,260
Total	69,488	6,045	75,533	11.5	3.10	602,825
Difference (Selected - RF 1.0) - In Situ Tonnes & Grade						
KE - USD 1,125/oz RF 0.90	-14,678	-785	-15,463	-1.2	0.21	-68,027
KW - USD 1,125/oz RF 0.90	-2,951	-191	-3,142	-0.3	0.08	-12,891
GW - USD 1,063/oz RF 0.85	-483	-52	-535	0.0	0.13	-3,164
SW - USD 1,125/oz RF 0.90	-1,005	-102	-1,108	-0.3	0.06	-5,439
SE - USD 1,125/oz RF 0.90	-423	-24	-447	-0.4	0.06	-1,876
Total	-19,540	-1,154	-20,694	-0.9	0.10	-91,397

4.6 Mine Equipment

4.6.1 Equipment Selection and Operating Parameters

Conventional drill and blast and load and haul open pit mining equipment, operating on 9 m and 4.5 m benches will be used to extract ore and waste rock for the Project.

The production schedule is calling for an average LOM production of 400 to 450 kbcm per month and this was used as the basis for determining equipment requirements. All mining services, including preproduction stripping, blasting, and maintenance and repair activities are contracted out. Mine operations are scheduled for two 10-hour shifts per day, 7 days per week, for a total of 365 days per year. A provision was made for an overall LOM mechanical availability of 85% and an overall LOM utilisation of 85%. Three crews, rotating between day, night and RDO shifts, will provide continuous operator and maintenance labour coverage for the mine.

4.6.2 Primary Mine Equipment

The selected primary mining fleet includes the following:

- Blasthole drills capable of 115 to 165-mm-diameter holes
- Grade control drills capable of 140 to 155-mm-diameter holes
- 125 tonne hydraulic backhoe excavator, with 8 m^3 bucket
- CAT 988 type (6 m^3) front-end loader for stockpile re-handle
- 90-tonne off-highway haul trucks
- CAT D9 type dozers
- CAT 834 type rubber-tired dozer
- CAT 14M type motor graders
- 50,000-liter water trucks

The laterite crust will require paddock blasting. The saprolites will require drill and blast from about 20 m below surface, using a powder factor of about 0.4 kg/bcm. Sulphide rock will require drill and blast, using a powder factor between 0.65 and 0.8 kg/bcm. Blastholes will be 9 m deep with 1 m subdrill. Drill patterns are in the range of 4.5m by 4.5m, with all blasts using a bulk emulsion. Average blasthole drill productivities for ore and waste are estimated at 25 to 35 m/hr in the saprolites and around 15 to 20 m/hr in the fresh rock.

The primary loading units will be 125 tonne type hydraulic excavators. The excavators have 8-m^3 buckets and are capable of loading a 90 tonne haul trucks in seven passes. Average excavator productivity is estimated at about 350 to 425 bcm per hour in oxide material and 250 to 350 bcm per hour in sulphide material.

Ore and waste haulage will be handled by 90 tonne off-highway trucks, which is a commonly used truck in the region with a few hundred units being in operation. Ore will be delivered to the ROM pad, just south of the plant, to feed the mineral sizer (and later on to the hard rock crusher), and waste will be deposited on the various waste dumps, located close to the pits. A relatively small amount of waste (approximately 200 kbcm) will be hauled to the tailings starter dam during preproduction.

Haulage profiles were measured from representative benches in each phase for each time period over the life of the project, including preproduction stripping. Cycle times and productivities were then calculated using OEM speed tables adjusted for a 2% rolling resistance, acceleration and deceleration adjustments, speed caps on roads with 10% gradients, and a maximum speed of 55 km/h on level segments.

Support services provided by the remaining primary equipment fleet (dozers, motor graders, and water trucks) include:

- Excavator area cleanup
- Haul road, pit bench, and waste dump maintenance
- Dust suppression
- New road/ramp development
- Cleaning areas for blasthole drilling
- Constructing and maintaining ditches and sumps
- Supplying crushed rock for blasthole stemming and road surfacing

Much of the dozing work will be performed by D9-class units. This work includes cleanup in the loading areas and maintenance of the waste dumps.

Road maintenance will be performed by a 14M class motor graders. Up to two 50,000-liter water trucks will be needed for dust suppression in the dry-season conditions at Yanfolila.

4.6.3 Auxiliary Equipment

A portable crushing and screening plant with a rated capacity of about 250 t/h will produce crushed rock for blasthole stemming and road sheeting. Other auxiliary equipment will be used for miscellaneous earthworks and construction around the mine and waste dumps, cleaning out minor ditches and sumps, equipment assembly and maintenance, fueling, and lubrication services.

4.7 Mine Workforce

Mine workforce requirements were estimated on the basis of working two 10-hour shifts per day, 7 days per week, 52 weeks per year. A standard, three-crew rotating work schedule for craft labour and front-line supervision will be used for around-the-clock coverage.

Expatriate personnel will be hired to help establish safe and efficient mine operations and maintenance systems, and to train Malians. As the skill levels of national workers increase, expatriates will be phased out with the eventual goal of minimal expatriate staffing.

Mine personnel requirements are based on the Owner managing the mining contractor and explosives supplier and performing all geological and survey functions.

4.7.1 Organization

The Mine Manager will be responsible for all mining operations, technical services, and safety/environment/training in the mine area. The Mine Manager will also be in charge of managing the mining contractor. The Mine Production Superintendent will be responsible for the day to day management of the contractor in the field. The Contractor's Project Manager will report directly to the Mine Manager. The mining organization chart is illustrated below in Figure 4.2.

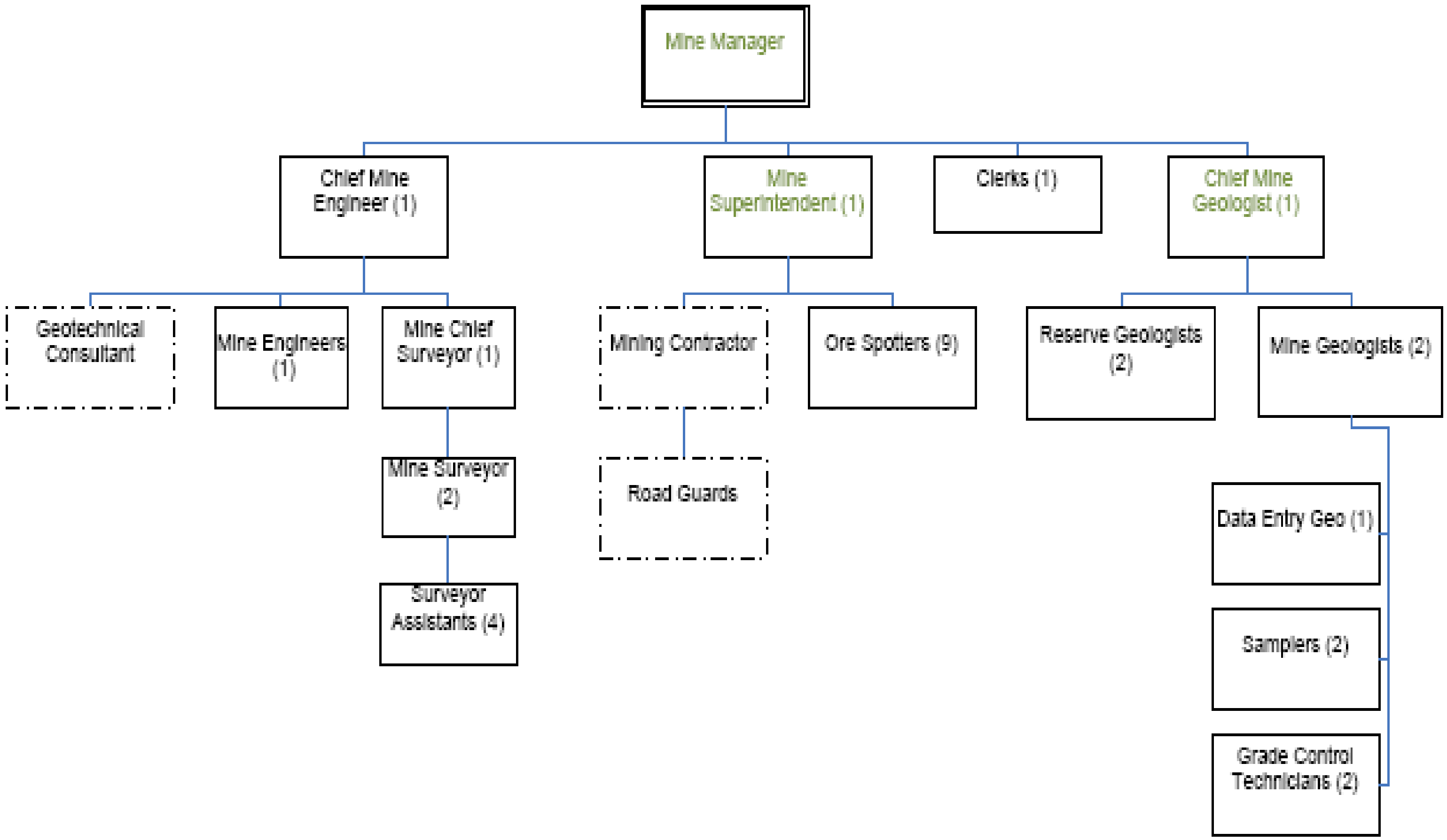

Figure 4.2 Mine Operating Organization Chart

5.0 METALLURGY AND PROCESSING

5.1 Introduction

The Yanfolila deposits host mainly saprolite and fresh sulphide ores. Saprock ore, which forms the transition zone between the saprolites and the fresh sulphide ore, constitutes only about 10% of the resource. Various testwork campaigns were conducted on saprolite, transition, and fresh samples from the Yanfolila deposits in order to understand and classify the comminution, rheology, and metallurgical behaviour of these ores. The previous testwork is summarised and described in Section 5.2. The most recently completed testwork campaign was conducted by GF through SGS South Africa during the DRS completed in 2013.

Interpretation of the results of the comminution, rheology, and metallurgical testwork was carried out by SENET, HUM's own consultants and independent consultants to develop an appropriate process flowsheet suitable for the exploitation of the deposits. Although the development of the process was primarily based on meeting the comminution and metallurgical properties of the Yanfolila ores, some aspects of the process plant design were duplicated from the Taparko Gold Plant, currently operating in Burkina Faso. The Taparko plant was engineered by SENET, therefore HUM decided to use Taparko as a model in order to reduce the amount and cost of engineering design.

The results of the comminution testwork indicated that the Yanfolila saprolite ores are very soft and friable. The saprolite and softer saprock ores are therefore suitable for processing through a mineral sizer, which also assists with the physical handling of the clay material. The fresh ores are significantly more competent and will require a three-stage hard rock crushing circuit for size reduction, to maintain throughput and grind criteria.

The testwork indicated that both the saprolite and fresh ores contain significant gravity recoverable gold. Intensive cyanidation of the gravity concentrate showed satisfactory extraction of the contained gold. Depending on the source of the ore and the head grade, cyanidation testwork on gravity tails indicated a recovery of up to 95% in 24 hours of leaching using conventional cyanidation for both the saprolite and fresh ores.

Detoxification testwork conducted on leach tailings using the typical INCO-type cyanide detoxification process showed that residual cyanide of 5 ppm WAD can be achieved within one hour of laboratory detoxification.

The initial flowsheet for Yanfolila was developed to process only the saprolite and softer saprock ore at a nominal rate of 1 Mtpa. The design, however, makes provision for the expansion of the process plant to treat up to 100% fresh ore, when required by the mine production schedule. The initial phase of the process plant development includes a single-stage soft rock crushing plant feeding directly into the milling circuit. The milling circuit is a single-stage ball mill in closed circuit with classifying hydrocyclones.

The recovery plant consists of a gravity circuit and a CIL train. Gravity gold recovery is achieved by the use of a centrifugal concentrator with the gravity concentrate being processed through an intensive cyanidation leach reactor. The pregnant solution from the intensive cyanidation is treated by a dedicated electrowinning cell. The gravity tails are returned to the milling circuit and processed through leach/CIL. Gold adsorbed on the activated carbon is recovered by means of Pressure ZADRA elution technology. Acid wash and carbon regeneration treatment are used to maintain the activity of carbon.

In summary, the process plant consists of the following sections:

- Crushing
- Milling
- Gravity Separation and Concentrate Leach
- Carbon in Leach
- Detox and Tailings
- Tailings Distribution and Reclaim Water Return
- Acid Wash
- Elution
- Electrowinning
- Carbon Regeneration
- Reagents
- Water and Air Services

5.2 Metallurgical Testwork

5.2.1 Introduction

A number of testwork campaigns have been conducted over several years on samples from the Project area. Details of these campaigns may be found in the relevant testwork reports, as well as being summarized in previous studies.[1] Summaries of the historical testwork have also been prepared by Edward Musonda[2] and Thomas Hayward.[3] Follow-up testwork is in progress at SGS Africa (Pty) Ltd in Johannesburg.

This section of the study traces the evolution of the testwork, and highlights the key metallurgical factors and conclusions which have a bearing on Project economics.

5.2.2 Previous Testwork Campaigns

The previous campaigns are briefly summarized below. They follow the expected sequence of project phases. These generally start out testing gold extraction amenability to assist an exploration program, then develop successively through phases of determining maximum gold extraction as deposits are further explored and drill core becomes available for testing. This phase is then typically followed by economically-driven optimization of test conditions to provide data for plant design purposes.

[1] Senet (Pty) Ltd, *Yanfolila De-Risking Study*, 15th November, 2013.
[2] Musonda, E., *Yanfolila Metallurgical Test Work History*, 17th November, 2012.
[3] Hayward, T., *Summary Report on a Review of Previous Metallurgical Test Work for the Yanfolila Gold Project, Mali*, 13th August, 2014.

5.2.2.1 Phases 1 and 2 (2007 NS 2008)

"BLEG" bottle roll cyanide leach testing was carried out on rotary core (RC) drill chip rejects to assess ore grades and cyanide extraction amenability, essentially to assist the Glencar Mining exploration program.[4] The tests were carried out by ABILAB Burkina Faso on KW oxide, transition, and fresh ore samples. The objective was to determine the cyanide leachable gold content of the ore and the residual tails gold values.

The impact of coarse gold on assay values and gold extraction was a notable feature of the test results. This has ongoing sampling and assay ramifications.

5.2.2.2 Phase 3 (2010)

With the Project now under GF aegis, scouting tests were carried out by McClelland Laboratories, Inc (McClelland) in the USA on seven assay reject composite samples from the KE deposit.[5] Cyanidation, CIL, and gravity concentration tests were performed. Since these scouting tests were carried out on assay rejects and were designed to provide a general view of the amenability of the samples to gold extraction, testwork conditions were aimed at maximizing extraction and not at optimizing test conditions. Consequently, excess cyanide concentrations and long residence times were used at a fixed grind p80 of 75 µm. The results demonstrated that gold extractions between 90% and 98% could be expected.

5.2.2.3 Phase 4 (2011)

Phase 4 represented an extension of the gold extraction amenability testwork on drill composite samples from KE as well as the KW and SE deposits. The KE and SE samples were divided between oxide and transition ore, while the KW samples included also fresh composites. These tests were again carried out by McClelland and covered cyanidation, CIL, and gravity concentration.[6] As before, maximum extraction was sought, and so test conditions included excess cyanide concentrations, long (72h) leach residence time, a grind p80 of 75 µm, and a slurry of 40% solids. Gold extraction results were again satisfactory. The CIL extractions, for example, were approximately 96% for all tests.

5.2.2.4 Phase 5 (2012)

Four drill core composites were tested at McClelland, with samples of KE fresh, Kabaya South oxide, SW oxide and Gonka fresh ores.[7] Having established that high gold extractions were possible, the Phase 5 tests investigated variation in the grind and cyanide concentration as a start to process optimization. Cyanidation leach extraction results were in accord with the findings of earlier testwork. Gravity recoverable gold was measured on the samples tested, and 48h leach extractions confirmed in the 90%+ range.

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4 Glencar Mining plc, *Komana West Ore Preliminary Metallurgical Test*, (2007).

5 McClelland Laboratories, Inc., *Gravity Concentration and Cyanidation Testing – Komana East Assay Reject Composites*, October 25, 2010.

6 McClelland Laboratories, Inc., *Whole Ore Milling/Cyanidation and Gravity Cyanidation Testing - Yanfolila Drill Hole Composites*, May 2, 2011.

7 McClelland Laboratories, Inc., *Milling/Cyanidation, Gravity Concentration and Flotation Testing – Yanfolila Drill Core Composites*, February 19, 2013.

5.2.2.5 Phase 6 (2013)

The Phase 6 testwork program was carried out at SGS South Africa (Pty) Ltd in Johannesburg on KW and KE samples and their blends.[8] With the knowledge gained from the previous scoping studies that high gold extractions could be anticipated, the Phase 6 testing focused more on generating design parameters for the process plant. The program included a suite of comminution tests, as well as some mineralogical studies. Parameters investigated included grind, gravity concentration, oxygen uptake, cyanidation and CIL, cyanide destruction, rheology, and settling tests. Samples of site water were also used in place of tap water, but showed no apparent effect on the gold extraction.

Much of the testwork was conducted using conditions which focused on the then-proposed design criteria – gravity concentration, CIL, a leach grind p80 of 106 µm, and 24h of leach residence time. The cyanide concentration was varied. While the grind is equivalent in principle to the Design Criteria,[9] the test leach residence time is longer than that in the proposed plant. The SGS tests were also carried out at a density equivalent to 45% solids, as compared with the 40% solids in the Design Criteria. Forty percent solids has been stipulated for the current SGS testwork program. The higher dilution associated with the lower percent solids may assist the leach kinetics, but will normally result in higher cyanide consumption in a laboratory test. Leach/CIL tests were run on whole ore as well as gravity tails (including gravity middlings and concentrate intensive leach tails).

Results were in keeping with those of the previous programs, though now with test parameters more closely aligned with plant design. Some variability in results is still apparent after the Phase 6 study.

5.2.2.6 Phase 7 (2013/14)

Additional gravity concentration testwork was carried out by Gravity Concentrators Africa on a 50/50 blend of KE North and South ores.[10] The samples were scrubbed following which the -2mm fraction was tested using different Knelson gravity concentration techniques. This program appeared to represent an attempt to pre-concentrate the gold from coarse material, thereby potentially reducing processing costs. The results, however, did not appear to warrant follow-up.

5.2.2.7 Phase 8 (2015)

Testwork currently in progress at SGS in Johannesburg was designed to fill testwork gaps and to generate data for samples of KE, KW, SE, SW, and GW ores. At the time the testwork was proposed, the Project concept was based on oxide ore processing only. The samples collected for testing therefore consist mainly of oxide drill core, with some transition material included.

Modification of certain SGS testwork procedures has been requested to provide better control over cyanide concentration, activated carbon preparation, and residue washing. This represents an effort to refine the consistency of data and to reduce variability.

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8 SGS South Africa (Pty) Ltd, *Report No: Met 12/483 Rev 1, Project A99*, 25th September, 2013.

9 It should be noted that the batch grind used in laboratory tests will not necessarily generate the same particle size distribution as continuous milling and cycloning in a plant.

10 Gravity Concentrators Africa, *Gravity Amenability Testwork on a Sample from Glencar Mali*, January, 2014.

The latest revision to the current testwork program has resulted in the testwork matrix for this phase of work as shown in Table 5.1.

Table 5.1 SGS Phase 8 Metallurgical Testwork

<table>
<tr><th>Deposit</th><th>KE</th><th>KW</th><th>SE</th><th>SW</th><th>GW</th></tr>
<tr><td>Head Assays</td><td rowspan="3">3 Samples (KEXN, KEXS & KEXT)</td><td rowspan="3">4 Samples (KWVN, KWVS, KWBS & KWXT)</td><td rowspan="7">3 Samples (SEVN, SEVC & SEVS)</td><td rowspan="7">3 Samples (SWSN, SWSC & SWSS)</td><td rowspan="7">2 Samples (GWSN & GWSS)</td></tr>
<tr><td>Assay by Size</td></tr>
<tr><td>Bond Ball Mill Work Index (BBWi)</td></tr>
<tr><td>Viscosity</td><td colspan="2" rowspan="4">not required</td></tr>
<tr><td>Oxygen Uptake</td></tr>
<tr><td>Gravity Gold Recovery & Intensive Cyanidation</td></tr>
<tr><td>Cyanidation</td></tr>
<tr><td>24h Leach in preparation for Carbon Adsorption test</td><td>1 Composite of KEXN / KEXS / KEXT</td><td>1 Composite of KWVN / KWVS / KWBS / KWXT</td><td>1 Composite of SEVN / SEVC / SEVS</td><td>1 Composite of SWSN / SWSC / SWSS</td><td>1 Composite of GWSN / GWSS</td></tr>
<tr><td>Carbon Adsorption</td><td>1 Composite of KEXN / KEXS / KEXT</td><td>1 Composite of KWVN / KWVS / KWBS / KWXT</td><td>1 Composite of SEVN / SEVC / SEVS</td><td>1 Composite of SWSN / SWSC / SWSS</td><td>1 Composite of GWSN / GWSS</td></tr>
<tr><td>Mineralogical - Head XRD, Clay Analysis, QEMSCAN BMA</td><td colspan="2">not required</td><td>3 Samples (SEVN, SEVC & SEVS)</td><td>3 Samples (SWSN, SWSC & SWSS)</td><td>2 Samples (GWSN & GWSS)</td></tr>
<tr><td>24h cyanidation test, preparation for Cyanide Destruction</td><td>1 Composite of KEXN / KEXS / KEXT</td><td>1 Composite of KWVN / KWVS / KWBS / KWXT</td><td>1 Composite of SEVN / SEVC / SEVS</td><td>1 Composite of SWSN / SWSC / SWSS</td><td>1 Composite of GWSN / GWSS</td></tr>
<tr><td>Cyanide Destruction</td><td>1 Composite of KEXN / KEXS / KEXT</td><td>1 Composite of KWVN / KWVS / KWBS / KWXT</td><td>1 Composite of SEVN / SEVC / SEVS</td><td>1 Composite of SWSN / SWSC / SWSS</td><td>1 Composite of GWSN / GWSS</td></tr>
</table>

The revisions proposed have provided for additional 24h leach and CIL testing of the KE and KW oxide ores under more controlled conditions, while trimming the large number of CIL tests that had been planned on the SE and SW, and GW samples. The Komana ores dominate the ore blend in the first few years of the Project.

An earlier proposal to prepare higher-arsenic KW fresh samples for tails disposal arsenic deportment testing has been reviewed. The original proposal called for bulk leaching of the samples to generate the required tails solution and solids. A revision has been recommended to convert the bulk cyanide leach to a controlled CIL test – the presence of activated carbon better represents the plant design and could have an impact on the residual arsenic deportment in tails disposed of in the TSF. The CIL tests will add to the data base on the gold extractions from the KW samples.

A proposal has also been requested for testing of Gonka fresh ore samples to improve the information on the metallurgical response of this ore to proposed process design conditions. Gonka fresh ore can potentially add a high-grade component to plant feed blends. Samples have been collected for this work, but the testing is in abeyance at the present time.

5.2.3 Process Design

The available testwork data have been used to address process design requirements and key inputs to the project financial evaluation.

5.2.3.1 Gold Recovery

Gold recoveries were evaluated primarily from the 2013 SGS testwork along with some input from the previous McClelland programs.

In determining the gold recovery, the relationship between recovery and costs needs to be borne in mind. Although relatively high gold extractions have been reported in all testwork campaigns, some of these were achieved with high reagent levels, fine grinds and long leach times, all reflecting higher cost operating conditions not necessarily conducive to optimized economics. Variability in test results, both with recoveries and reagent consumptions, is also a hallmark of such testing. In order to address this issue and reduce the number of degrees of freedom for Project financial evaluation purposes, the recoveries were normalized to a single recovery model. This approach then relies on cost-related variables (plant design, reagent consumptions, residence times, grind, etc) to be adequately addressed so as to achieve the projected recoveries. As an illustration, in Figure 5.1, KW fresh ore is shown to require higher cyanide concentrations (i.e. higher reagent cost) in order to achieve the target gold extraction.

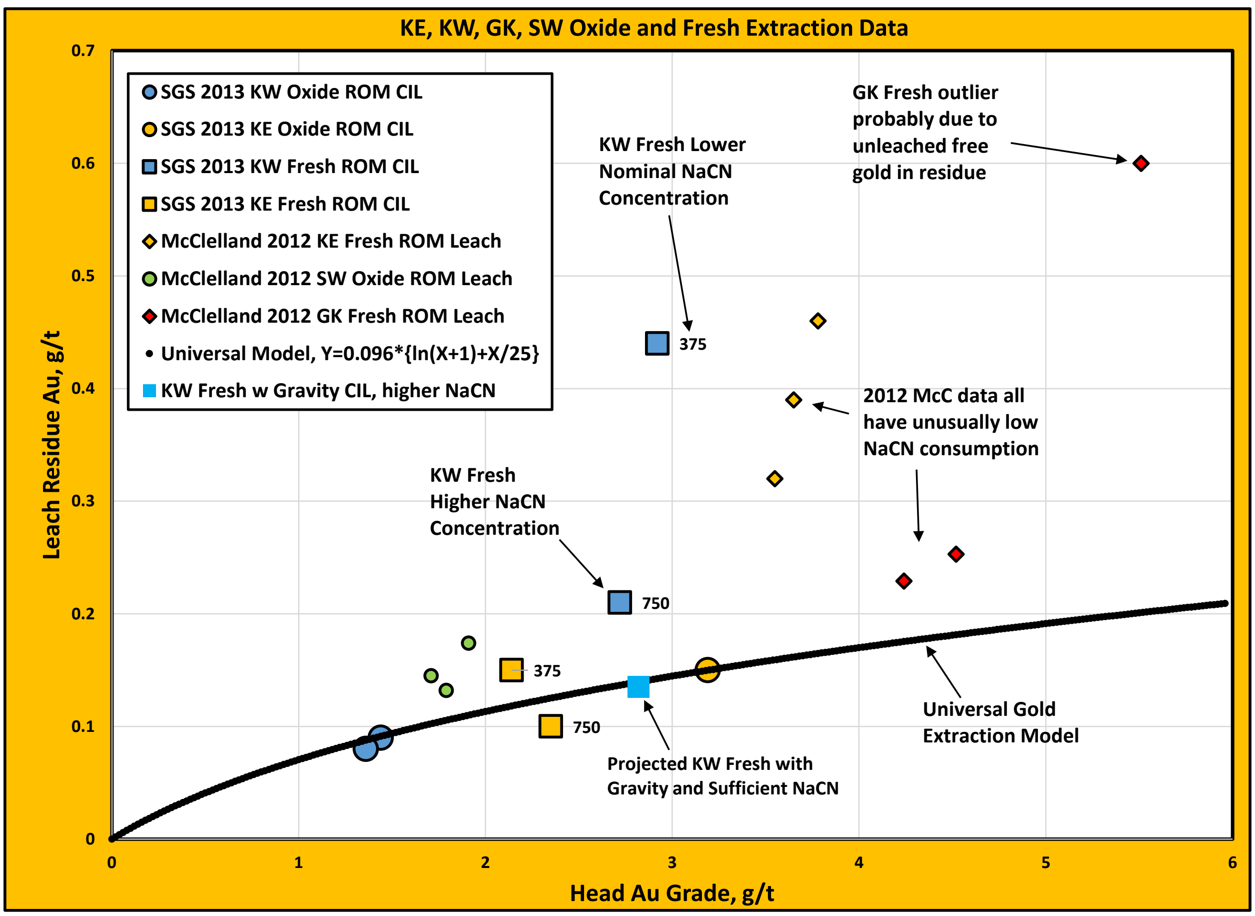


Figure 5.1 Cyanidation Leach Residue as a Function of Head Grade

The metallurgical recovery model for all ore types derived from the testwork data has been presented in a previous report. [11] The model recovery is a function only of the ore head grade and is as follows:

Au Recovery, % = 98.6 – 9.5*{ln[Au + 1]}/[Au]

where **[Au]** = ore head grade, g/t

Achieving the recoveries projected by this model is dependent on having appropriate plant design parameters which adequately reflect testwork conditions. The plant, when in operation, has also of course to be efficiently run. The recoveries allow for the design solution loss of 0.01 ppm Au.

This emphasizes that capital and operating cost estimates should support the design conditions required to achieve the recoveries – as noted above, the recoveries and costs are not independent of each other.

5.2.3.2 Operating Consumables

The primary economic operating consumable costs that can be estimated from the available testwork data are:

- Power for crushing and grinding
- Wear materials for crushing and grinding
- Leach/CIL cyanide and lime consumptions

Given the relatively high cost of electric power at the Project site, as well as the delivered cost of the cyanide and lime, these parameters have particular economic relevance in operating cost terms.

The principal test parameters dictating engineering power and wear material calculations are summarized in Table 5.2.[12]

[11] Brittan, M.I., *Yanfoliloa Gold Project, Mali, Metallurgical Recoveries*, December 17, 2014.
[12] Orway Mineral Consultants, *Yanfolila, Comminution Circuit Definitive Study Options*, 5th September, 2013.

Table 5.2 Comminution Parameters[13]

Parameters	Units	KME0001	KME0002	KME0002A	KME0003	KME0004	KME0005	KME0006A	KME0006B
		Laterite	Oxide	Fresh Mafic Intrusive	Fresh Siltstone	Fresh Mafic Intrusive	Oxide	Fresh Sandstone Siltstone	Fresh Basalt
CWi	kWh/t	-	-	15.3	13.7	17.16	3.82	10.51	16.82
SG		-	-	2.81	2.84	2.82	2.23	2.95	2.93
UCS									
Average	MPa	-	-	172.5	96.9	213.2	9.8	162.1	189.5
Range	MPa	-	-	90.2 – 290.7	28.0 – 186.3	116.1 – 332.0	6.9 – 12.7	104.7 – 310.7	159.1 – 223.1
Abrasion Index	g	-	-	0.2565	0.1053	0.4828	0.0738	0.5207	0.5257
Rod Mill Work Index	kWh/t	5.4	5.9	18.5	14.4	16.3	7.0	15.1	18.4
Ball Mill Work Index	kWh/t	2.1	3.2	17.8	13.9	15.7	9.0	16.5	15.3
SG		2.17	2.23	2.80	2.77	2.75	2.47	2.91	2.87
SMC Testwork									
A		89.4	69.7	100	92.6	100	68	97.2	95
b		2.41	4.21	0.23	0.33	0.26	1.95	0.34	0.22
Axb		215.5	293.4	23.0	30.6	26.0	132.6	33.0	20.9
Ta		2.57	3.40	0.21	0.28	0.24	7.6	0.29	0.19
Mia	kWh/t	5.3	4.1	30.8	24.2	27.8	7.6	22.6	32.0
Mih	kWh/t	2.7	2.0	25.8	19.0	22.6	4.4	17.6	27.2
Mic	kWh/t	1.4	1.0	13.3	9.8	11.7	2.3	9.1	14.1
JK Appearance Functions									
A		-	-	65.9	46.7	-	-	-	56.2
b		-	-	0.36	0.76	-	-	-	0.47
Axb		-	-	23.7	35.5	-	-	-	26.4
Ta		-	-	0.13	0.17	-	-	-	0.11
SG		-	-	2.80	2.81	-	-	-	2.93

3

The 2013 SGS data used for estimating the cyanide and lime consumptions are summarized in Table 5.3.

[13] Ball Mill Work Index values summarized in this table determined from 2013 SGS Test Work are currently being reviewed.

Table 5.3 SGS 2013 Testwork Data, 45% Solids, p80 106 µm

Report Page Number	Deposit	Ore Type	Leach or CIL Feed	NaCN Nominal ppm	NaCN Add Start ppm	NaCN 1 ppm	NaCN 2 ppm	NaCN 3	NaCN 4	NaCN Final ppm	NaCN Ave 1-24 ppm	Time h	Gold Assays Head Grade Au Assay	Gold Assays Head Grade Au Calc	Gold Assays Residue g/t	"Consumption" NaCN kg/t	"Consumption" CaO kg/t	Added NaCN kg/t
709	KW	Fresh	ROM	750	628		380		464	561	468	24	2.72	2.72	0.21	0.16	0.72	1.02
713	KE	Fresh	ROM	750	626		426		609	540	525	24	2.38	2.35	0.1	0.31	0.3	1.15
717	Comp	Fresh	ROM	750	626		397		580	435	471	24	1.62	1.78	0.11	0.44	0.43	1.18
703	KE	Oxide	ROM	500	648		232		299	290	274	24	3.03	3.19	0.15	0.86	1.24	1.67
699	KW	Oxide	ROM	500	417		360		490	464	438	24	1.44	1.44	0.09	0.07	0.53	0.77
705	Comp	Oxide	ROM	500	417		220		348	406	325	24	1.88	1.99	0.31	0.1	0.83	0.83
691	KE	Fresh	ROM	375	313		203		290	215	236	24	2.17	2.14	0.15	0.24	0.44	0.58
687	KW	Fresh	ROM	375	313		165		313	164	214	24	2.72	2.92	0.44	0.31	0.74	0.63
695	Comp	Fresh	ROM	375	313		189		319	305	271	24	1.62	1.74	0.23	0.23	0.24	0.6
679	KW	Oxide	ROM	375	313		189		305	247	247	24	1.44	1.36	0.08	0.21	0.91	0.6
605	Comp	Oxide	Gr Tl+Mid	150	233	145	278			316	246	24	1.47	1.58	0.14	0.43	0.79	0.81
607	Comp	Oxide	Gr Tl+Mid	250	443	218	444			319	327	24	1.47	1.42	0.12	0.42	0.78	0.78
609	Comp	Oxide	Gr Tl+Mid	500	764		1015			772	894	24	1.47	1.59	0.11	0.61	0.74	1.56
611	Comp	Oxide	Gr Tl+Mid	750	1160		1317			1094	1,206	24	1.47	1.96	0.12	0.92	0.72	2.26
613	Comp	Trans	Gr Tl+Mid	150	233	154	85			131	123	24	2.37	2.41	0.47	0.28	1.24	
615	Comp	Trans	Gr Tl+Mid	250	389		233			246	240	24	2.37	2.61	0.3	0.36	1.24	
617	Comp	Trans	Gr Tl+Mid	500	777		317			523	420	24	2.37	2.55	0.23	0.53	1.14	
619	Comp	Trans	Gr Tl+Mid	750	1157		464			623	544	24	2.37	2.55	0.13	0.65	1.12	
621	Comp	Fresh	Gr Tl+Mid	150	237	290	160			261	237	24	1.27	1.32	0.193	0.2	0.4	
623	Comp	Fresh	Gr Tl+Mid	250	386		276			499	388	24	1.28	1.3	0.18	0.24	0.39	
625	Comp	Fresh	Gr Tl+Mid	500	777		566		876	682	708	24	1.28	1.22	0.15	0.29	0.39	
627	Comp	Fresh	Gr Tl+Mid	750	1161		943		1172	1099	1,071	24	1.28	1.35	0.183	0.36	0.29	
629	Comp	Fresh	Gr Tl+Mid	750	1164		870		1102	992	988	24	1.28	1.44	0.24	0.34	0.32	
631	KW	Fresh	Gr Tl+Mid	200	167	87	58	210	225	160	148	24	1.08	1.13	0.3	0.22	0.51	0.42
633	KW	Fresh	Grav Tls	200	250	102	109	189	145	102	129	24	0.59	0.65	0.12	0.17	0.36	0.3
681	KW	Oxide	Gr Tl+Mid	375	313		189		319	186	231	24	0.42	0.47	0.08	0.28	0.7	0.6
683	Comp	Oxide	Gr Tl+Mid	375	313		203		313	232	249		0.69	0.73	0.17	0.21	1	0.58
685	KW	Trans	Gr Tl+Mid	375	313		151		262	232	215	24	0.88	0.86	0.19	0.41	2.97	
689	KW	Fresh	Gr Tl+Mid	375	313		160		334	174	223	24	1.07	1.04	0.21	0.33	0.79	0.63
693	KE	Fresh	Gr Tl+Mid	375	313		232		290	218	247	24	0.8	0.72	0.12	0.2	0.41	0.55
697	Comp	Fresh	Gr Tl+Mid	375	313		209		278	238	242	24	1.27	1.29	0.21	0.21	0.44	
701	KW	Oxide	Gr Tl+Mid	500	417		290		406	406	367	24	0.42	0.4	0.06	0.23	0.36	0.69
707	Comp	Oxide	Gr Tl+Mid	500	417		244		363	464	357	24	0.69	0.75	0.21	0.15	0.53	0.83
711	KE	Fresh	Gr Tl+Mid	750	626		380		609	603	531	24	1.07	1.19	0.17	0.3	0.55	1.22
715	KE	Fresh	Gr Tl+Mid	750	626		467		661	580	569	24	0.8	0.77	0.13	0.22	0.41	1.11
719	Comp	Fresh	Grav Tls	750	626		428		627	603	553	24	1.27	1.39	0.16	0.23	0.31	
721	KW	Oxide	Gr Tl+Mid	200	166	73	139	189	171	136	142	24	0.42	0.38	0.06	0.17	0.39	0.43
723	KW	Oxide	Grav Tls	200	167	87	116	123	109	80	103	24	0.46	0.38	0.05	0.41	0.74	0.53
725	KW	Oxide	Gr Tl+Mid	250	206	102	116	254	241	206	184	24	0.42	0.39	0.06	0.24	0.39	0.59
727	KW	Oxide	Gr Tl+Mid	300	248	102	248	107	249	209	183	24	0.42	0.4	0.05	0.24	0.34	0.61
729	KW	Fresh	Gr Tl+Mid	200	167	73	109	167	168	116	127	24	1.07	1.01	0.3	0.33	0.69	0.52
731	KW	Fresh	Gr Tl+Mid	250	206	73	181	218	218	180	174	24	1.07	1.12	0.18	0.33	0.79	0.63
733	KW	Fresh	Gr Tl+Mid	300	246	73	138	218	348	197	195	24	1.07	1	0.11	0.5	0.69	0.85
735	KE	Fresh	Gr Tl+Mid	200	164	131	119	174	194	168	157	24	0.8	0.73	0.14	0.18	0.33	0.45
737	KE	Fresh	Gr Tl+Mid	250	205	145	203	181	226	174	186	24	0.8	0.73	0.12	0.21	0.43	0.51
739	KE	Fresh	Gr Tl+Mid	300	246	203	109	225	203	241	196	24	0.8	0.78	0.14	0.3	0.33	0.68
513	Comp	Oxide	ROM	1507								24	1.88	2.77	0.09	0.7	1.28	1.89
515	Comp	Oxide	ROM	2072								30	1.88	1.52	0.1	0.87	1.28	2.49
517	Comp	Oxide	ROM	2053								36	1.88	1.92	0.07	1.05	1.46	2.49
519	Comp	Oxide	ROM	2051								48	1.88	1.64	0.08	1.02	1.45	2.48
547	Comp	Trans	ROM	1561								24	4.04	4.11	0.12	0.93	1.6	
549	Comp	Trans	ROM	2271								30	4.04	3.25	0.12	1.19	2.25	
551	Comp	Trans	ROM	2226								36	4.04	4.31	0.12	1.3	2.42	
553	Comp	Trans	ROM	2218								48	4.04	3.53	0.14	1.42	2.41	
581	Comp	Fresh	ROM	1546								24	1.62	1.59	0.11	0.63	0.62	
583	Comp	Fresh	ROM	1965								30	1.62	1.72	0.11	0.95	0.62	
585	Comp	Fresh	ROM	2154								36	1.62	2.48	0.13	1.12	0.63	
587	Comp	Fresh	ROM	2164								48	1.62	1.8	0.1	1.14	0.78	

The estimated CIL reagent consumptions from the tests with conditions most closely allied to the design criteria are given in Table 5.4.

Table 5.4 CIL Reagent Consumptions

	Consumptions	
Ore Type	NaCN kg/t	CaO kg/t
Oxide	0.75	0.85
Fresh	0.6	0.55

For design purposes, lime consumption can be considered reasonably directly translatable from batch laboratory testing to continuous commercial-scale plant operation. On the other hand, there are no firm principles for scaling up cyanide consumption from batch laboratory scale to continuous plant, particularly where cyanide in process tails is destroyed. When reporting cyanide consumption, laboratories traditionally take credit for residual cyanide remaining at the end of a cyanide leach or CIL test, deducting this from the total cyanide added in the test. In reality, allowing for the difference in conditions between bench scale and large tonnage operations, the true cyanide consumption is likely to be somewhere in between the laboratory-reported consumption value and the total addition, probably closer to the latter.[14]

5.2.4 Ongoing Testwork Requirements

5.2.4.1 Variability

The data used to establish the gold recovery model for the Project summarized in Figure 5.1 show test leach residues plotted as a function of the ore head grade. The variability in the data apparent from Figure 5.1 is not unusual at this stage of a project, but is exacerbated in Yanfolila's case by the following factors:

- The free gold (nugget effect) in Yanfolila ores results in variability due to complications with sampling and assays;
- There is a large number of permutations of orebodies and ore types – while the different ore types can be linked by the Universal Gold Extraction Model, their cost inputs (grind and reagent functionalities) can vary significantly;
- As a generality, there is always a reliance on drill samples to represent large tonnages of ore;
- High gold extractions can result in low residue grades, where sampling and assay uncertainty can have a large influence on assay measurement accuracy;
- Especially in cases where ores consume higher than average cyanide, standard laboratory procedures for pH and cyanide control during leaching tests may not be sufficiently rigorous to avoid cyanide depletion, or to yield meaningful cyanide consumption data. This confounds the financial trade-off between leach efficiency and

[14] Brittan, M.I., and Plenge, G., *Estimating Process Design Gold Extraction, Leach Residence Time and Cyanide Consumption for a High Cyanide-Consuming Gold Ore*, Minerals and Metallurgical Processing Journal, SME, in press, 2015.

cyanide consumption, and complicates the standardization or normalization of recoveries and costs;

- If leach tests are used instead of CIL tests, the results can be distorted by a preg-borrowing effect.[15] This effect has been noted before in West African ores. This may lead to apparently slow leaching kinetics, and requires controlled washing of residues, along with wash solution assays, to ensure that calculated head grades and extractions are not understated; and

- Test grind size distributions are determined from grind calibration curves and may not be strictly as specified.

An effort has been made in ongoing testwork programs to tighten laboratory procedures in order to reduce the variability of results. Nonetheless, given the large number of deposits and ore types potentially involved in this Project, more variability sample testwork would not be misplaced in future programs.

5.2.4.2 Optimization

Changing requirements dictated by an available plant design or mill can influence plant design optimization, as can variability of metallurgical response among different ores from different sources, though ostensibly of the same ore type. The latter effect is illustrated in Figure 5.2 where the final test residue gold values for KW and KE fresh ores are plotted as a function total cyanide addition.

[15] Brittan, M.I., *Kinetic and equilibrium effects in gold ore processing*, Minerals & Metallurgical Processing, 25 (3), August 2008, pp 117–122.

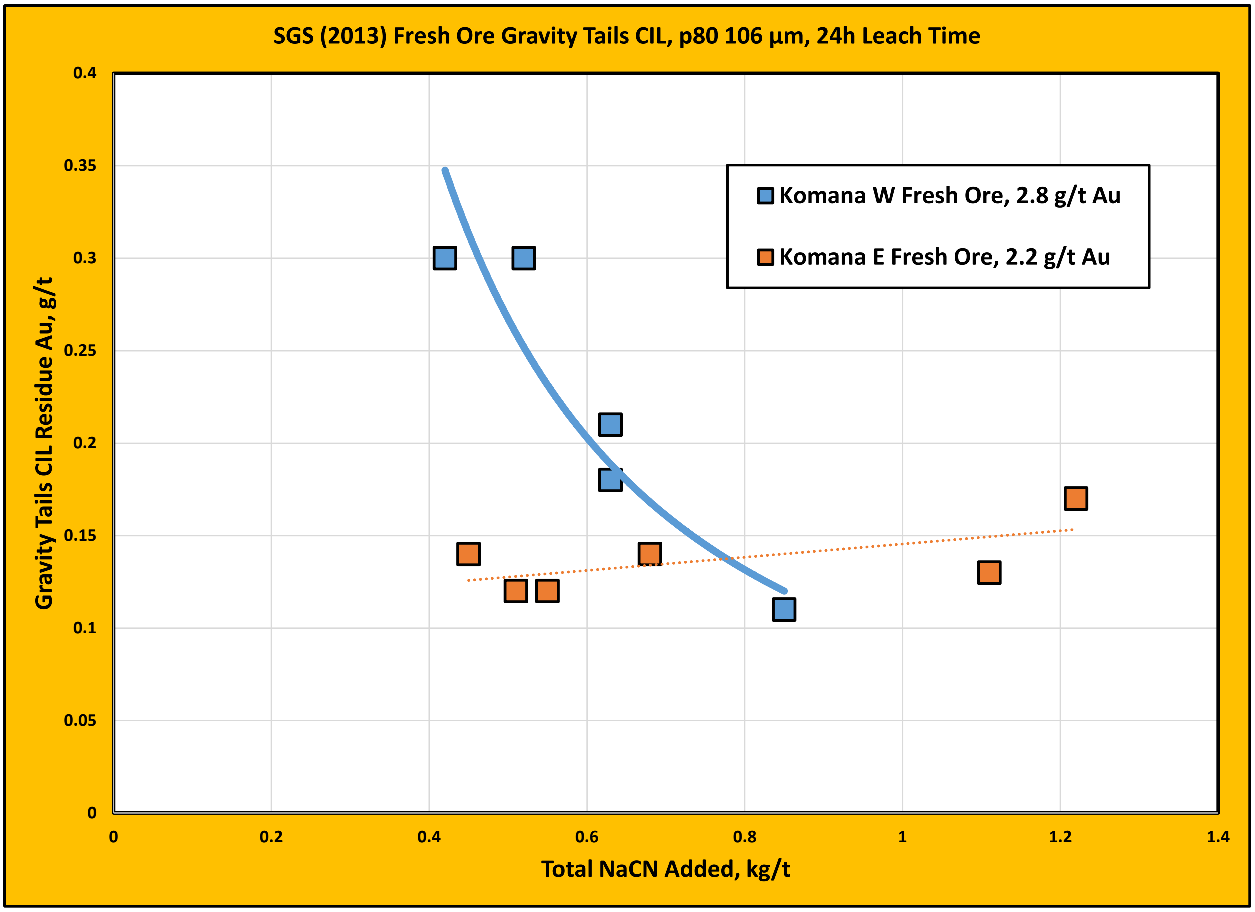


Figure 5.2 Effect of Cyanide on Gravity Tails CIL Au Residues

What Figure 5.2 shows is that the KW ore gold recovery is sensitive to laboratory cyanide addition, at least up to 0.8 kg/t, whereas KE ore recovery is independent of the cyanide addition, certainly above 0.4 kg/t. Depending on the mine schedule as well as confirmation with further testwork, such effects may facilitate optimization of the cyanide concentration as a means of reducing cost.

While the design grind p80 of 106 µm was established at a point when oxide ore was essentially considered the sole ore feed, the testwork data had demonstrated that some ore types were sensitive to grind, as illustrated below in Figure 5.3.

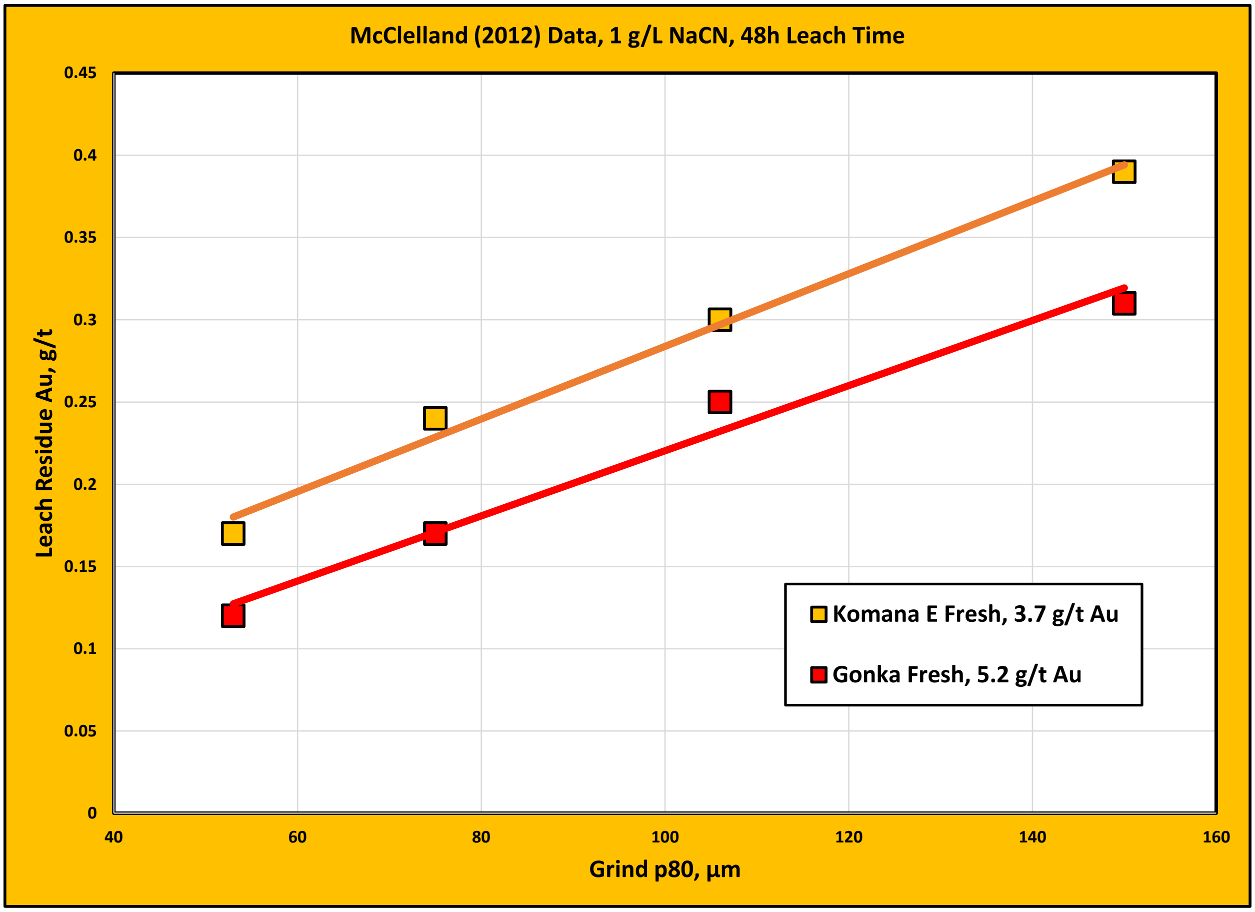


Figure 5.3 Effect of Grind on Cyanide Leach Residue Au

These McClelland tests were conducted under conditions to maximize gold extraction (high cyanide concentration and long leach time) so are not truly representative of an optimized situation. Nonetheless, they illustrate that the additional gold extracted by grinding to a p80 of 75 µm instead of the design 106 µm is worth about $2.70/t at a gold price of $1,250/oz. This again may allow some further grind optimization given sufficient data on the mill, power costs, mine schedule, and other factors.

In any project, there is always more testwork that could be done to keep refining the information for design purposes. From a practical perspective, the appropriate approach at any given time in a project's life is to ensure as far as possible that recovery and cost estimates used for financial evaluation allow for the assessed reliability of the available testwork data.

5.2.5 Alternative Processing Scenarios

During the Phase 5 testwork period, a request was received to send samples to Gekko Systems Pty Ltd in Australia for gravity and flotation testing using the Gekko Python concept.[16] The Python is a skid-mounted concentrator generally incorporating fine crushing, gravity, and flotation modules as may be required. The intention was to determine if this offered a more

[16] Gekko Systems Pty Ltd, *TO926 – Interim Testwork Update – 5 Reports on Python Amenability for Gonka Fresh, Kabaya South Oxide, Komana East Fresh, Komana East Oxide and Sanioumale West Oxide*, 18th August, 2012.

economic alternative to conventional CIL processing, or whether on-site concentration of ore from Yanfolila satellite orebodies using a skid-mounted plant would be a feasible alternative to trucking of whole ore to a central processing plant. The ores tested were Gonka fresh, Kabaya South oxide, KE fresh, KE oxide and SW oxide. The concentrate mass, however, proved too high and the overall gold recovery to concentrate too low to render this a scenario worth pursuing.

Heap leaching was also considered as an alternative for low grade or satellite ore processing, but was similarly found to be financially less attractive than the CIL mill option.

The key issues involved in these alternative processing scenarios were summarized in a 2012 note.[17]

5.3 Flowsheet Development

5.3.1 Introduction

The Yanfolila process plant was developed from the interpretation of the results of various testwork completed on mainly oxide and transitional ores. This section relates to the process development of only the oxide ore. Fresh ore was initially excluded from the development of the process plant, and work to evaluate the processing of the fresh ore was only considered later during the development of the process plant.

The Yanfolila process plant was developed to meet the processing requirements of the oxide ore as interpreted from the results of the various testwork programmes conducted by SGS and McClelland. Although the plant was designed to meet the metallurgical requirements of the oxide ore (as interpreted from the results of the testwork programmes), some aspects of the plant were duplicated from the Taparko gold plant. This was a Project decision taken in order to minimise the engineering requirements, project implementation schedule, and the capital cost for the plant.

The Yanfolila process plant is based on a conventional gold cyanidation leach process capable of treating one million tonnes per year. The plant is designed to process primarily the oxide material, with up to 50% transitional and fresh material in the plant feed.

The process flow is based on soft rock crushing, single-stage ball milling, gravity recovery with intensive cyanidation of the gravity concentrate, six-stage CIL with a single-stage pre-leach of the gravity tails, and cyanide detoxification using the INCO process. The plant design includes a Pressure ZADRA elution circuit with acid wash and carbon regeneration. The detailed process design criteria (PDC) are included in Appendix 6.

The proposed expansion of the Yanfolila process plant will be required to maintain the plant throughput at one million tonnes per year when processing fresh ore. The expansion of the process plant will include the installation of a three-stage hard rock crushing to allow the plant to process fresh ore.

[17] Brittan, M.I., *Process Alternatives Review*, Gold Fields note, December 5, 2012.

5.3.2 *Crushing and Milling*

The Yanfolila deposit hosts significant quantities of saprolitic ores which are generally very soft and friable. The deposit also hosts a minor fraction, about 10%, of saprock material. Metallurgical testwork on saprolitic ore samples showed the saprolite ore to be very soft and suitable for processing using the soft rock crushing method. The comminution properties of the saprolites and saprock are shown in Table 5.5 and 5.6, respectively. The comminution circuit was sized to process saprolite and saprock material at a nominal rate of one million tonnes per year.

Table 5.5 Comminution Properties of Saprolitic Ores

Parameter	Unit	Nominal	Design
Unconfined Compressive Strength	MPa	6.60	100
Bond Crusher Work Index	kWh/t	3.88	3.88
Rod Mill Work Index	kWh/t	6.70	8.71
Ball Mill Work Index	kWh/t	7.40	9.62
Abrasion Index		0.041	0.050
JK Tech Parameters			
Design A		66.0	66.0
Design b		2.46	2.5
Design Axb		162.4	162.4
Design T_a		1.06	1.06

Table 5.6 Comminution Properties of Saprock Ores

Parameter	Unit	Nominal	Design
Unconfined Compressive Strength	MPa	6.60	100
Bond Crusher Work Index	kWh/t	3.88	3.88
Rod Mill Work Index	kWh/t	11.55	13.55
Ball Mill Work Index	kWh/t	11.70	13.55
Abrasion Index		0.201	0.201
JK Tech Parameters			
Design A		66.00	66.00
Design b		2.46	2.46
Design Axb		162	162
Design T_a		1.06	1.06

5.3.2.1 Crushing

A single-stage tooth roll crusher was selected to process the saprolitic material. The crusher was designed to give a final product with F100 of 250 mm and F80 of 18 mm for direct feed to the mill. The sizing of the crusher took into account the need to process some softer saprock material through the soft rock crushing. The fresh ore was excluded from the initial process flow development; therefore, the initial crushing circuit was developed with only soft rock crushing. Crushing of fresh ore is covered under process plant expansion.

5.3.2.2 Milling

Leach optimisation showed that an optimum grind of P80 of 106 µm was required for the Yanfolila saprolitic and saprock ores. The plant was designed with a single mill in a closed circuit with a single hydrocyclone cluster. The milling circuit design parameters are shown in Table 5.7.

Table 5.7 Milling Design Parameters

Parameter	Unit	Design
Number of Ball Mills		1
Wet or Dry Milling		Wet
Open or Closed Circuit		Closed Circuit
Overflow or Grate		Overflow
Product Size Control $D_{50,60,70,80,90,92,95}$		D_{80}
Feed Size (F_{80})	mm	18
Product Size (P_{80})	µm	106
Ball Material (High Cr, Cast or Forged)		Cast
Mill Discharge Screen Type (Trommel/Vibrating)		Trommel
Trommel Screen Aperture	mm	17 × 13
Trommel Screen Material		Polyurethane Panel

The milling circuit was sized to be able to process 100% saprock ore at a nominal processing rate of one million tonnes per year. This in effect makes the current design slightly oversized for the processing of 100% saprolite ore.

5.3.3 Gold Recovery

The results of the gold recovery testwork indicated that the Yanfolila ore contains a significant portion of gravity recoverable gold. Testwork results indicated a final gravity recovery of up to 64% of contained gold. In addition, the testwork showed that the recovered gravity concentrate could be processed through an intensive cyanidation process. The results of intensive cyanidation testwork indicated that more than 91% of the gold could be recovered into solution by intensive cyanidation in 24 hours of leaching.

Results of testwork on the gravity tails showed a good recovery of gold using a conventional cyanide leach process. CIL testwork showed recoveries above 90% for all the Yanfolila ores.

The overall plant recovery was modelled to give a relationship between the recovery and head grade. The following model was produced by the client's metallurgical consultant, Michael Brittan:

Au Recovery, % = 98.6%-9.5*{ln[Au +1]}/[Au],

where Au is the gold head grade in g/t.

This recovery includes both gravity and CIL recoveries and depends considerably on achieving consistently good gravity recoveries. The model was used to calculate the expected overall plant recoveries for the specified head grade. Note that this recovery is independent of the solution and other process losses.

5.3.4 Gravity Recovery

The process plant design includes a centrifugal gravity recovery circuit that recovers gold from the cyclone underflow stream. A portion of the cyclone underflow is screened through a 2 mm scalping screen and diluted to 50% m/m solids for feed to the centrifugal concentrator. The plant was designed on the basis of an average of 40% gravity recovery. The parameters used for the design of the gravity circuit are shown in Table 5.8.

Table 5.8 Gravity Recovery Design

Parameter	Unit	Nominal
Type of Gravity Unit		Centrifugal Fluidised
Aperture Size to Concentrator Feed	mm	2
Number of Gravity Units		1.0
Concentrator Feed Slurry Solids Concentration	% m/m	50 %
Gravity Recovery (% of Head Grade)	%	40 %
Gravity Concentration Ratio		2 000
Concentrate SG	t/m^3	4.00
Concentrate Mass as % of Feed to Concentrate	%	0.06 %
Gravity Concentrate Treatment Type		Intensive Cyanidation

5.3.5 Carbon in Leach (CIL)

The CIL plant tank farm was based on the Taparko plant. The parameters used for the leach/CIL design are shown in Table 5.9.

Table 5.9 Leach/CIL Design Parameters

Parameter	Unit	Nominal	Design
Leach Solids Feed	% m/m	38.9 %	
Leach Feed Au Grade (no gravity) – Design	g/t	2.89	3.47
CIL Solid Tails Au Grade	g/t	0.17	0.19
CIL Solution Tails	ppm	0.015	
Total Leaching Residence Time	h	24	
Number of Pre-Leach Tanks		1.0	
Number of CIL Stages/Tanks		6.0	
Total Number of Tanks		7.0	
Available Tank Volume (per tank)	m^3	830	
Carbon Concentration	g/L	10	
Gold to Carbon Loading Ratio – Design		1 200	
Barren Carbon Loading (Au)	g/t	100.0	
Time Required to Move Loaded Carbon	H	3.5	

The oxygen uptake results showed that the ore displays normal oxygen consumption. Therefore, the CIL plant was designed with normal aeration without the need for sparging or the addition of oxygen into the leach/CIL tanks. The CIL oxygen demand and aeration parameters used for the design are shown in Table 5.10 below.

Table 5.10 CIL Oxygen Demand and Aeration

Parameter	Unit	Nominal
Oxygen Demand	mg/L/min	0.028
Type of Oxygen Addition		Aeration
Air Requirements	Nm^3/1 000 m^3 Tank Volume	72

5.3.6 Acid, Elution, Electrowinning, and Carbon Regeneration

The Yanfolila elution process is based on the Pressure ZADRA process. The Taparko elution circuit was retained for Yanfolila, however, all the necessary checks were carried out to ensure that the circuit would be suitable and sufficiently sized to meet the production requirements for the Yanfolila plant.

The details of the design of acid wash, elution, and regeneration are shown in Table 5.11, Table 5.12, and Table 5.13, respectively.

5.3.6.1 Acid Wash, Elution, and Regeneration

The elution circuit is based on a 4 t elution batch. Acid wash is sized to accommodate a 4 t batch of loaded carbon. Regeneration is designed on the basis of regenerating 100% of the eluted carbon.

Table 5.11 Acid Wash Design

Parameter	Unit	Value
Batch Size	t (carbon)	4
Design and Material of Construction	-	60° FRP Cone with Peripheral Overflow Weir
Acid Wash Temperature	°C	Ambient
Acid Wash Flow rate	BV/h	2
Acid Wash Mix Tank Volume (minimum)	BV	1
Acid Wash Time	H	1
Acid Wash Solution Concentration	% w/v HCl	3 %
Acid Wash Rinse Water Volume	BV	4

Table 5.12 Elution Design

Parameter	Unit	Nominal	Design
Method	-	Pressure Zadra	
Batch Size	t (carbon)	4	
Maximum Eluted Carbon Grade	g Au/t	100	
Recovered Gold per Elution	kg Au	4.8	9.6
Elution Schedule	No./month	27	28
Column Loading	H	0.5	
Column Heat Up	H	2	
Elution and Electrowinning	H	14	
Column Cool down	H	2	
Column Emptying	H	1	
Total Duration	H	19.5	
Elution Column Material of Construction	-	304 L SS	
Operating Temperature	°C	120	
Operating Pressure	kPa (g)	350	
Elution Flow Rate	BV/h	2	

Table 5.13 Regeneration Design

Parameter	Unit	Value
Batch Size	t (carbon)	4
Fraction of Batches Regenerated	%	100
Kiln Type	-	Indirect Heated Horizontal Rotating Tube
Heating Method	-	Diesel Fired
Regeneration Temperature	°C	700
Regeneration Time at Temperature	min	15
Kiln Running Time	h/batch	20
Screen Type	-	Single Deck Horizontal Vibrating
Deck Material	-	Stainless Steel Wedge wire
Aperture	mm	1 slotted

5.3.6.2 Electrowinning

The electrowinning circuit was based on the Taparko circuit design and layout. An additional cell was included to treat electrolyte from the gravity concentrate intensive cyanidation process. The electrowinning circuit sizing was checked to ensure that the production requirements for the Yanfolila plant are met. Review of the circuit is, however, necessary in line with the higher nominal head grades expected from the new mining schedule. The electrowinning design details are shown in Table 5.14.

Table 5.14 Electrowinning Design

Parameter	Unit	Nominal	Design
Cell Type	-	Atmospheric, Sludging	
Anodes/Cathodes Per Cell	No.	9/8	
Electrolyte Temperature	°C	< 90	
Flow Rate Per Cell	m^3/h	< 15	
Cell Operating Configuration	-	Parallel	
Cells In Use (CIL Gold)	No.	2	1
Cells In Use (Gravity Intensive Cyanidation)	No.	1	
Total Cells In Use	No.	3	2
Type of Cathode	-	Stainless Steel Knit Mesh – Reusable	
Cathode Dimension	-	700 mm × 700 mm × 40 mm	
Type of Anode	-	Stainless Steel	
Rectifier Rating	V / A	0 - 12 / 0 – 1 200	
Ventilation Flow Rate	$Nm^3/min/cell$	15	
Cathode Sludge Filter Type	-	40 L Batch Stainless Steel Pressure Pot Filter	

5.3.6.3 Cyanide Detoxification

The Yanfolila process plant design includes an INCO-type cyanide detoxification process. The cyanide detoxification process was not part of the Taparko plant design, and was designed and sized from first principles. The detoxification process was designed to give 5 ppm of residual WAD cyanide in the final tailings. The cyanide detoxification process design is a single-stage process using sodium metabisulphite and air to achieve cyanide destruction. The design details for the cyanide detoxification process are given in Table 5.15.

Table 5.15 Design Details for Cyanide Detoxification

Parameter	Unit	Design/Value
Detox Method	-	Air/SO_2 with Copper Catalyst
Detox Volumetric Feed Design Margin	%	22 %
Reactor Residence Time (minimum)	H	1
Aeration Hold-Up Factor	% v/v	15 %
Slurry pH	-	8 -10
WAD Cyanide in Detox Feed Slurry Solution	ppm CN_{WAD}	125
WAD Cyanide in Final Tails Slurry Solution	ppm CN_{WAD}	5
Expected Cu^{2+} in Detox Feed	ppm Cu	2.5
Excess Cu^{2+} Required	mg/L	10
SMBS to CN_{WAD} Ratio	g/g	6
Molar Ratio CaO : SMBS	-	1
Molar Ratio Oxygen : CN_{WAD} removed	-	1

5.3.6.4 Water, Air Services, and Reagents

The Yanfolila plant design includes provision of services as well as facilities for the make-up and distribution of reagents. Raw water is supplied either from the Sankarani River or pit dewatering sources, which include dewatering boreholes. Clean raw water is collected in the clean water pond and is pumped to the plant raw and process water tanks on demand. Dirty water from pit dewatering is mainly used as plant process water, whereas the clean water is primarily used for plant raw water and potable requirements.

The plant design includes fire water provision and reticulation. The fire water is stored in the raw water tank. A minimum reserve volume for fire water is allowed for by using raised suctions for raw water distribution.

Potable water distribution is included for plant safety showers and offices. The potable water for safety showers is distributed from a header tank placed at the high point of the plant structure. A hydrosphere is used to maintain the pressure of the potable water supply to the ablution blocks and offices. The design details used for water services are shown in 5.16.

Table 5.16 Water Services Design

Parameter	Unit	Design/Value
Raw Water		
Raw Water Storage Method	-	Tank
Tank Size	m^3	600
Firefighting Water		
Raw Water Storage Method	-	Lowest Section of Raw Water Tank
Firefighting Water Flow Rate	m^3/h	228
Fire Water Storage Required	min	90
Process Water		
Process Water Storage Method	-	Tank
Tank Size	m^3	600

5.3.6.5 Air Services and Plant Diesel

Compressed air is provided for leaching, cyanide detoxification, general air purposes, and instrument air. The air to CIL and detoxification are supplied by low pressure and high flow compressors, while the general purpose and instrument air is supplied by high pressure and low flow compressors. This separation of the high pressure and low flow from low pressure and high flow duties allows optimisation of the compressor duties and therefore optimisation of the energy required to supply air.

Plant diesel is pumped from the mine reservoir into the plant header tank. The header tank supplies diesel by gravity to the regeneration kiln, the gold room, and elution. The details used to size the plant air equipment are shown in Table 5.17.

Table 5.17 Plant Air Services Design Details

Parameter	Unit	Design
Low Pressure Compressed Air		
Delivery Pressure	kPa	250
Low Pressure Compressor Operating Configuration	-	Duty / Installed Standby, Lead-Lag
High Pressure Compressed Air		
Delivery Pressure	kPa	750
High Pressure Compressor Operating Configuration	-	Duty / Installed Standby, Lead-Lag
Instrument Air		
Air Quality - Residual Particle Size	µm	0.1
Air Quality - Residual Oil Concentration	ppm	0.01
Air Filter Operating Configuration	-	Duty / Installed Standby
Air Dryer Pressure Dew point	ºC	3

5.3.7 Reagents

Various reagents are required for different process sections. Make-up and distribution areas are provided for each of the reagents to supply adequate amounts of reagents to the different sections when required. Details for the design of reagents make-up and distribution areas are shown in Table 5.18 to Table 5.23.

Table 5.18 Sodium Cyanide Make-up and Distribution

Parameter	Unit	Design
CIL Cyanide Addition (as 100 % NaCN)	kg/t	0.75
Fresh Eluant Make-up Frequency	-	After 2 Elutions
Cyanide per Eluant Batch (as 100 % NaCN)	kg/batch	160
Intensive Cyanidation Addition (as 100 % NaCN)	kg/batch	144
Delivery Packaging	-	Bulk Box
Package Size	kg	1 000
Physical Form	-	Briquettes
Quality (minimum)	w/w % NaCN	98 %
Solids Specific Gravity	t/m^3	1.80
Solution Make-Up Strength	% w/w	25 %
Solution Specific Gravity	t/m^3	1.13
Storage Residence Time	d	2
Dosing Method	-	Ring Main
Ring Main Flow Factor (minimum)	× Demand	40

Table 5.19 Caustic Soda Make-up and Distribution

Parameter	Unit	Design
Fresh Eluant Make-up Frequency	-	After 2 Elutions
Caustic Soda Addition per Eluant Batch (as 100 % NaOH)	kg	240
Acid Wash Neutralisation Frequency	-	After 4 Acid Washes
Caustic Soda Addition per Acid Wash Neutralisation (as 100 % NaOH)	kg	267
Intensive Cyanidation Addition (as 100 % NaOH)	kg/batch	1
Delivery Packaging	-	Bags

Package Size	kg	25
Physical Form	-	Pearls/Flakes
Quality (minimum)	w/w % NaOH	98 %
Solids Specific Gravity	t/m³	2.13
Solution Make Up Strength	w/w % NaOH	20 %
Solution Specific Gravity	t/m³	1.24
Storage Residence Time	d	3

Table 5.20 Acid and Lime Make-up and Distribution

Parameter	Unit	Design
Hydrochloric Acid		
Fresh Acid Make-up Frequency	-	After 4 Acid Washes
Acid Requirement per Make-up (as 100 % HCl)	Kg	240
Delivery Method	-	1 000 L IBC (or 200 L drums)
Delivery Size	Kg	1 150 (or 230)
Physical Form	-	Solution
Quality (minimum)	w/w % HCl	33 %
Solution Specific Gravity	t/m³	1.15
Lime Make-Up and Dosing		
CIL Lime Addition (100 % CaO)	kg/t	0.85
Detox Lime Addition (100 % CaO)	kg/t	0.49
Total Addition (as 100 % CaO)	kg/t	1.34
Quality (minimum)	% CaO	74.5 %
Total Hydrated Lime Consumption	kg/t	1.80
Delivery Packaging	-	Bulk Bags
Package Size	Kg	1 000

Physical Form	-	Powder
Solids Specific Gravity	t/m^3	2.25
Lime Slurry Solids Content	% w/w	20 %
Lime Slurry Specific Gravity	t/m^3	1.12
Make-Up Schedule	batch/d	2
Dosing Tank Capacity	d	0.5
Lime Dosing Method	-	Modulating 3-Way Dart Valve
Ring Main Flow Factor (minimum)	× Demand	25

Table 5.21 SMBS and Copper Sulphate Make-up and Distribution

Parameter	Unit	Design
SMBS		
Detox Addition	g/g CN_{WAD}	6.00
Consumption (as 100 % $Na_2S_2O_5$)	kg/t	1.51
Quality (minimum)	% $Na_2S_2O_5$	97 %
Delivery Packaging	-	Bulk Bags
Package Size	Kg	1 000
Physical Form	-	Granular Powder
Solids Specific Gravity	t/m^3	1.40
Solution Make-Up Strength	% w/w	25 %
Solution Specific Gravity	t/m^3	1.08
Make-Up Schedule	batch/d	1
Copper Sulphate		
Detox Feed Solution Copper Concentration Required (min.)	g Cu/m^3	10
Consumption (as 100 % $CuSO_4$)	kg/t	0.07
Quality (minimum)	%	98 %

Parameter	Unit	Design
	$CuSO_4.5H_2O$	
Delivery Packaging	-	Bags
Package Size	kg	25
Physical Form	-	Crystals
Solids Specific Gravity	t/m³	2.28
Solution Make-Up Strength	w/w % $CuSO_4$	15 %
Solution Specific Gravity	t/m³	1.21
Make-Up Schedule	batch/d	5

Table 5.22 Lead Nitrate, Hydrogen Peroxide, and Flocculant

Parameter	Unit	Design
Lead Nitrate		
Intensive Cyanidation Addition (as 100 % $PbNO_3$)	kg/batch	1.4
Quality (minimum)	% $PbNO_3$	98
Delivery Packaging	-	Bags
Package Size	Kg	25
Physical Form	-	Crystals
Hydrogen Peroxide		
Intensive Cyanidation Addition (as 100 % H_2O_2)	kg/t	TBC
Quality (minimum)	% H_2O_2	70 %
Delivery Packaging	-	IBC/200 L Drum
Package Size	Kg	1 000
Physical Form	-	Liquid
Flocculant		

Intensive Cyanidation Addition	kg/t	TBC
Delivery Packaging	-	Bags
Package Size	Kg	25
Physical Form	-	Powder
Make-up method	-	Manual

Table 5.23 Activated Carbon and Smelting Fluxes

Parameter	Unit	Design
Activated Carbon		
Carbon Specification	type	Coconut
Carbon Particle Size P_{100}	mm	3.4
Carbon Particle Size P_0	mm	1.4
Dry Carbon Specific Gravity	t/m^3	0.80
Wet Carbon Specific Gravity	t/m^3	1.37
Wet Carbon Bulk Density	t/m^3	0.50
Carbon Losses	g/t	50
Smelt Flux Reagents-Delivery Packaging/Sizes		
Borax	- / kg	Bags / 25
Silica	- / kg	Bags / 50
Sodium Carbonate	- / kg	Bags / 50
Potassium Nitrate	- / kg	Bags / 50
Fluorspar	- / kg	Bags / 50
Manganese Dioxide	- / kg	Bags / 25

5.3.7.1 Process Plant Expansion to Process Fresh Ore

Expansion of the process plant is required to maintain the plant throughput at 1 Mtpa when processing fresh ore. The fresh ore is much harder than the saprolite and saprock ores. A hard rock crushing circuit is required to produce crusher product suitable for the ball milling operation.

The harder fresh ore also requires a much higher energy input into the milling circuit in order to maintain the design throughput at the same grind as for the saprolite ores. The leach/CIL circuit also requires expansion in order to accommodate the higher residence time required for the fresh ore to maintain optimal gold dissolution. The comminution parameters for the proposed expansion of the crushing and milling circuits are detailed in Table 5.24.

Table 5.24 Fresh Ore Comminution Parameters

Parameter	Unit	Nominal	Design
Unconfined Compressive Strength	MPa	196.00	196
Bond Crusher Work Index	kWh/t	16.90	16.60
Rod Mill Work Index	kWh/t	16.40	18.40
Ball Mill Work Index	kWh/t	16.00	17.50
Abrasion Index		0.360	0.522
JK Tech Parameters			
Design A		97.5	100.0
Design b		0.26	0.2
Design Axb		25.4	23.0
Design T_a		0.24	0.22

The proposed expansions to the comminution circuits showing the proposed configuration is shown in Figure 5.4.

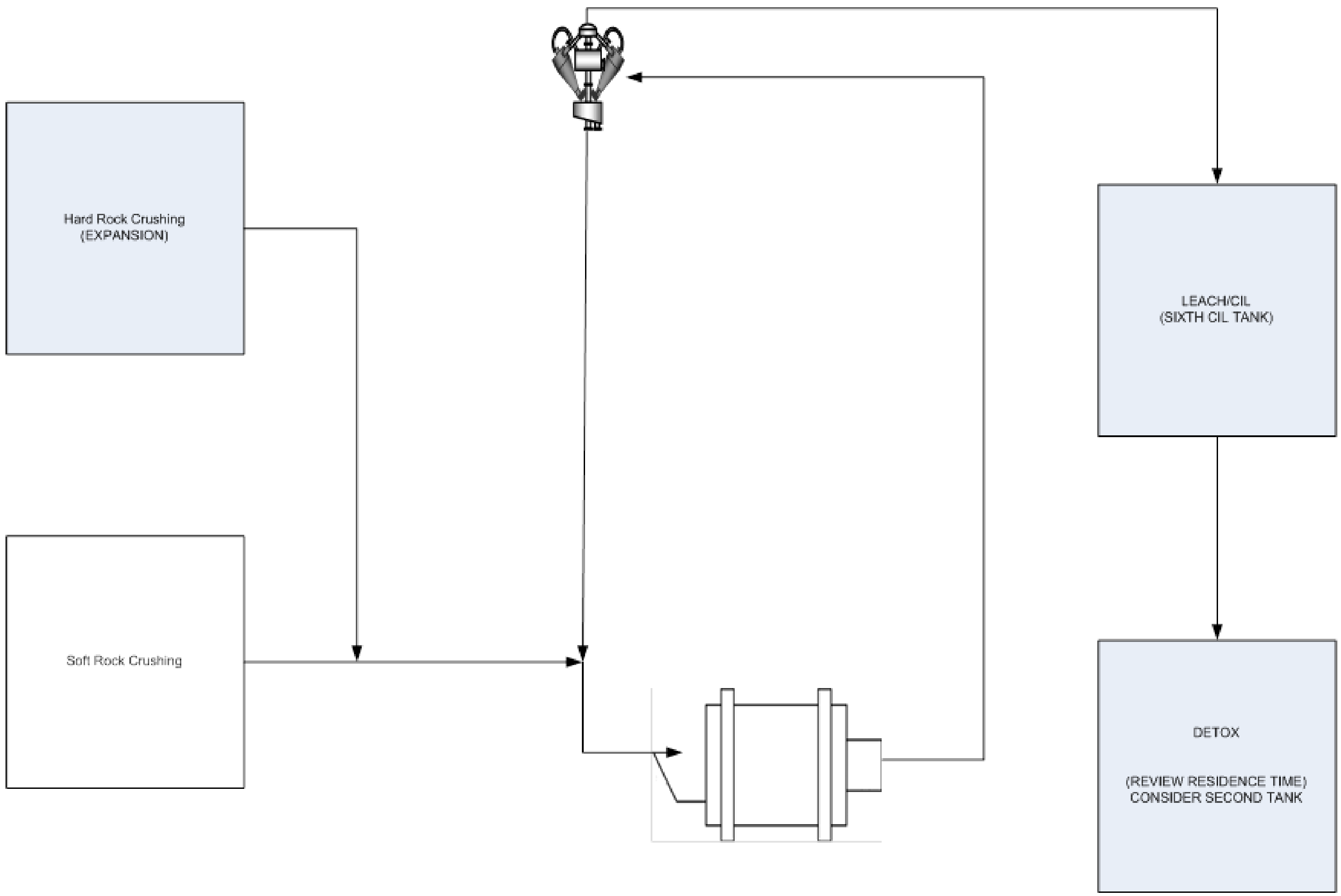


Figure 5.4 Proposed Process Plant Expansion Showing Additional Crushing and Mill

5.4 Process Flow

SENET described the process flow in detail in Appendix 6. The general process flow is listed below:

- Mineral Sizing and Crushing
- Milling
- Gravity Concentration and Intense Cyanidation
- Carbon in Leach
- Electrowinning
- Doré Production
- Cyanide Detoxification and Tailings Storage
- Reagents
- Process Control Philosophy

5.5 Process Control Philosophy

5.5.1 PLC and SCADA

The plant is relatively small and can operate with a low level of automation. Plant drives and valves critical to production are operable remotely and automatically from the control room via a Supervisory Control and Data Acquisition (SCADA) system. The SCADA system operates the control loops and is utilised for overall plant control.

The plant is controlled by Programmable Logic Controllers (PLCs) housed in the various Motor Control Centres (MCCs). A smoke and/or thermal fire detection system that is connected to a fire alarm siren system is included in each MCC Building.

Field instruments are wired back to input/output cards in the PLC rack. PLCs and instruments are supplied from an uninterruptible power supply (UPS), which provides approximately 20 minutes backup in the event of a complete power failure. This permits an orderly shutdown of the control system.

Each drive has the following inputs to the PLC:

- MCC Healthy
- Field Start
- Stops Healthy
- Drive Running

Each drive, with the exception of spillage pumps, has a Run Command output from the PLC.

The operator uses the SCADA system situated in the central control room to observe and operate the plant. The plant areas are presented in graphic form on individual screens.

Each SCADA screen displays all the drives and instrumentation in that area, together with the status of the drives and the current value of the instruments. Alarms are generated and displayed in a dedicated portion of the screen.

Drives can be individually started from the SCADA system and all interlocking between drives is carried out from the PLC.

The SCADA operator can place each drive into field control. The drive interlocks can be disabled and the unit run in "maintenance mode" from the field stop/start station. Once the drive is placed back in "SCADA mode", the interlocks are automatically re-enabled, and thereafter the drives have to be started in the correct sequence. The operators are required to walk through the plant before start-up to make sure that it is safe to start any drive.

All proportional-integral-derivative (PID) control loops are monitored from the SCADA system and controlled by the PLC. All analogue values are logged into a historical file and the data stored for one month.

All drives have a start and lockable stop button station at the motor. The stop button acts as an emergency stop regardless of the control mode adopted.

External field circuit breakers are provided for the mills and crushers. The duty electrician locks the drive out in the MCC and the operators lock out at the field breaker. This prevents unauthorised persons from entering the MCC and allows operators to safely lock out an item at the field breaker without calling out the electrician who holds the MCC key after hours for minor stoppages, i.e. not for reline maintenance. The field breaker shall not be used as an emergency stop because the breaker cannot be safely used to disconnect the power supply under load.

Equipment that is not required or is impractical to be operated remotely from the SCADA system is only operable in field mode. This includes the following:

- All cranes and hoists
- All auxiliary drives at crushers, apron feeders and mills, e.g. lubrication systems
- Mobile high pressure wash pumps, e.g. screen wash pump and cathode wash pump
- Regeneration kiln and kiln screw feeder
- Compressors, because they have their own auto-start load management system, and the air dryer
- Calcine ovens and smelting equipment in the gold room
- Portable acid offload pump

The crusher, apron feeder, and ball mill auxiliaries, compressors, air dryer, and calcining oven have running feedback to the PLC/SCADA system to indicate when they are running.

The balance of the process control details are provided in Appendix 6 (SENET Report).

6.0 TAILINGS STORAGE FACILITY (TSF)

6.1 TSF Introduction

Conventional storage of tailings is typically in surface retention structures that use embankments to store both tailings and water (from mining and surface run on) with the goal of reclaiming water from the facility for mine operations. These dams are raised in intervals over time to increase the available storage capacity for both tailings and water over the life of the project. The materials utilized in the construction of tailings dams range from tailings to waste rock. There are three principal types of embankment raises used in the construction of tailings facilities; these include upstream, downstream, and centerline raises.

Ausenco was first tasked with determining the costs associated with the development of the TSF over the LOM for the three different types of embankment raises along with their major advantages and disadvantages, including potential risks and recommendations (January 2015). The outcome of this study identified that the centerline raise was the preferred option, even though it had a slightly higher sustaining capital cost than the upstream raise. However, the centerline raise is less susceptible to failure if the facility is not properly managed both from construction to water management. In addition, wider beaches are required for upstream raises to ensure that the phreatic surface does not encroach on the embankment, possibly triggering a dam failure, which requires a slightly higher embankment for the upstream option.

Based on the current mine plan provided by HUM, the TSF storage requirements are approximately seven million tonnes of tailings over the life of the Project. In previous studies a small basin located northeast of the process plant was identified for the TSF. Based on a review of the topography and proposed infrastructure by Ausenco, there is no other suitable site close to the plant (refer to Figure 6.1). In addition, Ausenco also reviewed the previous work completed for the Project and in combination with our site visit in December 2014, determined there was sufficient information to develop the optimization study for the TSF without any additional field or laboratory geotechnical or hydrological/hydrogeological work.

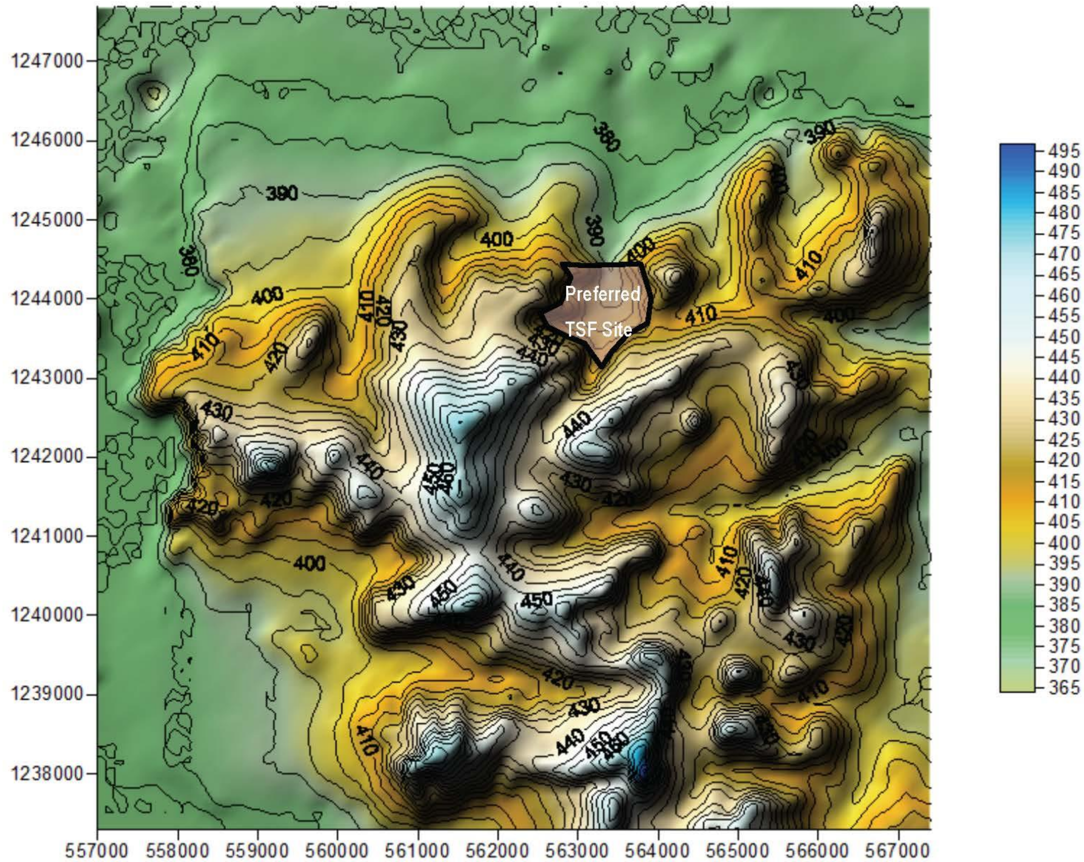


Figure 6.1 Preferred Tailings Storage Facility Site

6.2 Design Criteria

The design of the TSF was based on the following design criteria listed in Table 6.1.

Table 6.1 Design Criteria

Description	Unit	Value	Source
Life of Facility	Yrs	7	Hummingbird
Tailings Production Rate	Mtpa	1	Hummingbird
Specific Gravity	unitless	2.8	Hummingbird
***In-situ* Tailings Density**	t/m^3	1.35 (dry)	Hummingbird
Freeboard	m	1.0	Ausenco
Operational Reclaim Water	m^3	341,000 (max)	Schlumberger
Stormwater Retention Event	Yr - 24 hr	(392,400 m^3)	Ausenco
Spillway Design Event	Yr – 24 hr	PMF	Ausenco
Perimeter Diversion Canal	Yr – 24 hr	25	Ausenco
Embankment Stability Factor of Safety	unitless	Static ≥1.5 Pseudostatic ≥1	Ausenco
Perimeter Diversion Canal	Yr – 24 hr	25	Ausenco

The following additional design requirements/assumptions were applied to the TSF design:

- Phasing the capital cost where possible;
- The operations reclaim pond's maximum storage capacity is maintained below 341,839 cubic meters;
- The main TSF basin is unlined; seepage water quality will meet discharge standards or is captured by the seepage intercept drain and pumped back into the facility;
- The location of open pits, mine waste stockpiles, process plant; and
- The proximity to sensitive environmental and social factors, such as villages, rivers, and farm lands.

6.2.1 TSF Stage Capacity Requirements

The TSF stage capacity requirements for LOM are present in Table 6.2 below.

Table 6.2 TSF Storage Capacity Requirements

Phase	Tailings Storage (m³)	Operation Pond Storage (m³)	100 yr Storm Event Storage (m³)	Total Storage Requirement (m³)
0	740,741	341,839	392,417	1,474,993
1	1,481,481	341,839	392,417	2,215,733
2	2,222,222	341,839	392,417	2,956,474
3	2,962,962	341,839	392,417	3,697,215
4	3,703,703	341,839	392,417	4,437,956
5	4,444,444	341,839	392,417	5,178,696
6	5,185,185	341,839	392,417	5,919,437

6.3 Site Characteristics

The TSF and supporting infrastructure cover an area of 180 Ha with the embankment running east-west direction.

The TSF valley is generally flat and sparsely vegetated with large areas of savanna comprising grasslands, woodland, and brush land. Informal agriculture and artisan mining occurs within the footprint of the TSF.

6.3.1 Local Geology

Plateaus of ferruginous duricurst dissected by the broad TSF valley floor characterize the geomorphology within the TSF. Within the site area, these features are softened although the hardpan ferruginous crust is evident over much of the area and less in the valley bottom. The soil structure is common for a mature lateritic weathered soil profile. The TSF overlying surface layers are comprised of ferruginous zone, which consist of duricurst and/or loose gravel on the upper slopes of the valley and thick layers (over 15 m) of mottled silt and sandy clays located along the valley floor.

6.3.2 Hydrogeology

The area is located within the Soudan-Sahel region of West Africa. The climate is subtropical and characterized by distinct seasons. The rainy season is from June to October and the dry season from November to May. The annual rainfall is around 1,300 mm. March to May is generally the hottest period and December to February is the coolest.

While a large portion of the rainfall will evaporate, surface flows within the TSF flow towards the north to the Sankarini River, which flows into the Selinque Dam.

The groundwater within the lateritic soils below the TSF is variable and dependent on the parent rock and weathering. The geotechnical investigation by Project Management International indicated that the overall permeability within the TSF is low with pockets of higher permeability due to lens of gravelly material.

6.3.3 Seismology

The entire region of southern Mali is a low seismic risk area based on the 2007 seismic risk map published by the United Nations (OCHA). The USGS seismic hazard map for West Africa indicates that the seismic acceleration for the Project area is 0.2 m/s^2, which is considered very low.

6.3.4 Geotechnical Investigation

Project Management International performed an extensive field and laboratory program in the footprint of the TSF in 2013 to support the development of the TSF. The program consisted of:

- 40 cone penetration tests;
- 11 bore holes; and
- 29 core holes.

The program was used to identify subsurface conditions (geotechnical and hydrogeological) below the TSF footprint along with identifying borrow materials for construction of the TSF. The results of the investigation showed that the topsoil ranged from 0 to 15 cm across the tailings facility. Along the tailings embankment the slopes above the base of the valley are weak ferricrete over 10 m deep that are underlain by clay. Heading down the slope towards the valley floor, the conditions are variable with duricurst absent in some areas. Along the base of the valley, the duricurst is either thin or not encountered. Over the valley floor, gravelly clays are encountered to depths around 12 m becoming mottled clay that extend to depths of 30 m where weathered mudstone is encountered. The laboratory program was utilized to develop borrow areas within the TSF footprint and for modeling the stability of the embankment and seepage from the TSF.

6.4 Tailings Storage Facility Design

The proposed TSF is an unlined valley fill with a starter embankment and subsequent centerline raises to contain the tailings through the LOM and closure. In addition, the facility will be surrounded by a perimeter diversion canal which will convey surface runoff from storm events away from the TSF. A perimeter access road will be located next to the canal to provide operational and closure access to this facility. A security fence will be installed around the facility to preclude access by animals and humans for safety reasons. Tailings will be discharged from the west embankment crest, as well as from a portion of the east embankment crest to develop beaches that push free water (decant) towards the back of the facility. A floating barge will be located towards the back of the faciity. Barge mounted reclaim water pumps will recycle water back to the process plant. When required to prevent build-up of water above the operational design criteria for the facility, excess water will be returned to the environment, after any required treatment to satisfy environmental requirements.

6.4.1 Tailings Storage Facility – Phase 0

The layout of the Phase 0 TSF is shown in Figure 6.2. The TSF is a valley impoundment embankment utilizing waste rock to build the outer shell to a crest elevation of 371.6 masl that provides storage for both tailings and water for the first year of operation. The upstream face of the embankment will consist of a filter (to prevent migration of the low permeability liner if there is a leak), a low permeability soil liner, and a geomembrane to minimize seepage through the embankment.

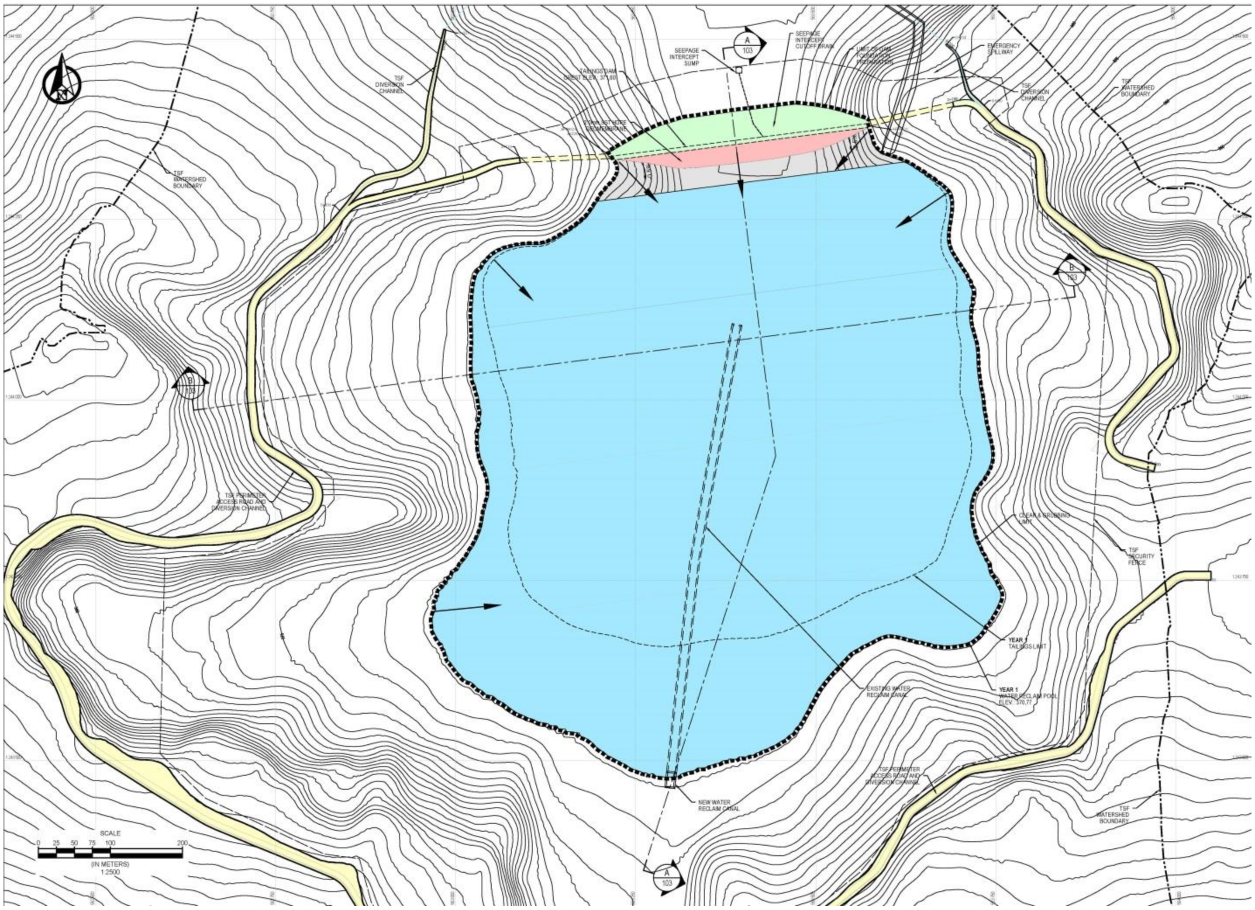


Figure 6.2 Phase 0 Tailings Storage Facility Configuration

The key construction works associated with Phase 1 are as follows:

- Clearing and grubbing of the Phase 0 TSF footprint;
- Removal of topsoil over the TSF footprint (150 mm) to a stockpile;
- One meter over excavation of embankment foundation;
- Excavation of a seepage intercept trench beneath the embankment and sump and the installation of drainage gravel, piping, concrete sump, pump, and discharge line back to the TSF;
- Compacted waste rock fill for the embankment from a suitable open pit source;

- Placement of the filter, low permeability soil, and geomembrane liner along the upstream face;
- Excavation of the perimeter storm water diversion canal and the construction of the perimeter road;
- Construction of the emergency spillway located on the eastern side of the TSF;
- Installation of the perimeter 1.5 m high five strand barb wire fence with three gates around the TSF; and
- Instrumentation to monitor the performance of the dam and telemetry within the pond to monitor the TSF pond level.

6.4.2 Tailings Storage Facility – Phases 1 through 6

The tailings embankment will be raised yearly to accommodate tailings over the seven year LOM, the last raise to be completed at the end of year six. The raises will be centerline and are typically one to two meters high. The liner system will continue up through the raises to provide a barrier system against seepage. The plan view of the Phase 6 TSF (ultimate raise) is shown in Figure 6.3 and the longitudinal cross section in Figure 6.4.

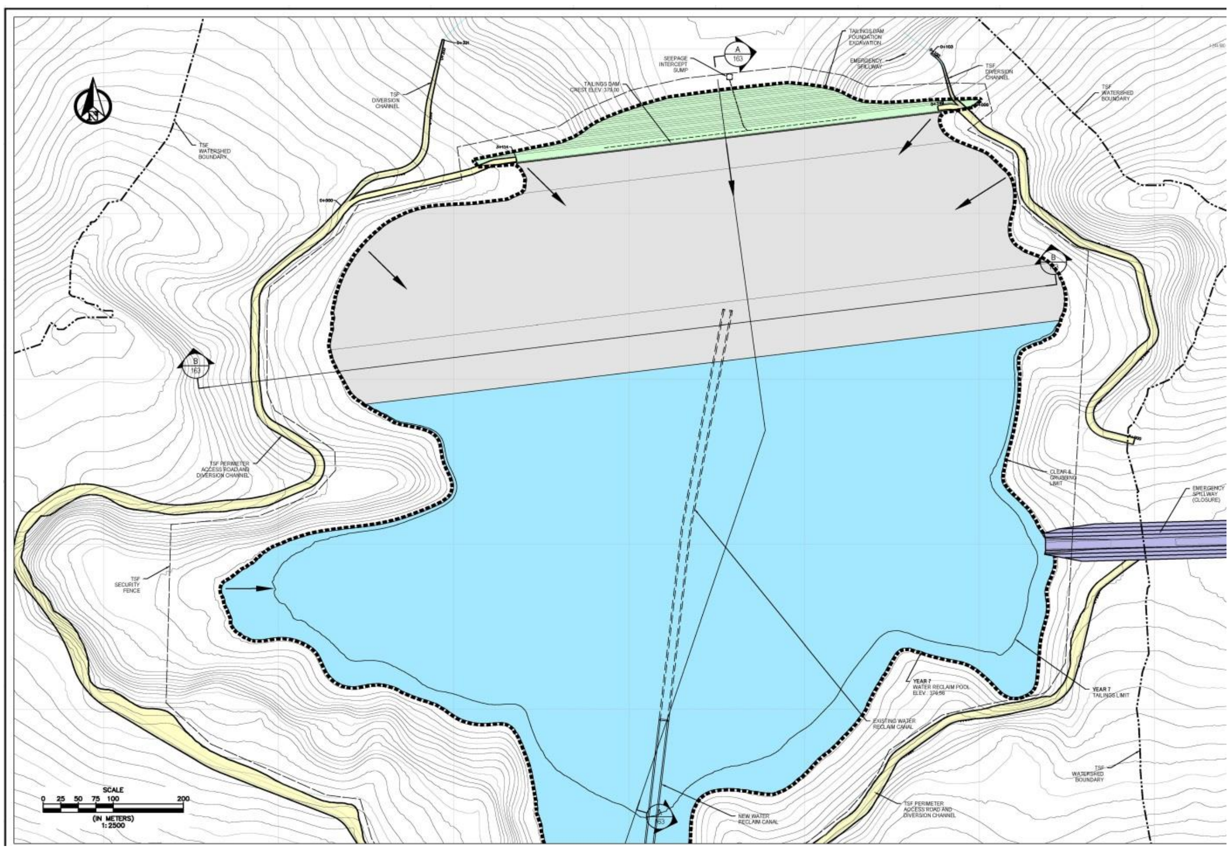


Figure 6.3 Phase 6 (Ultimate) Tailings Storage Facility Configuration

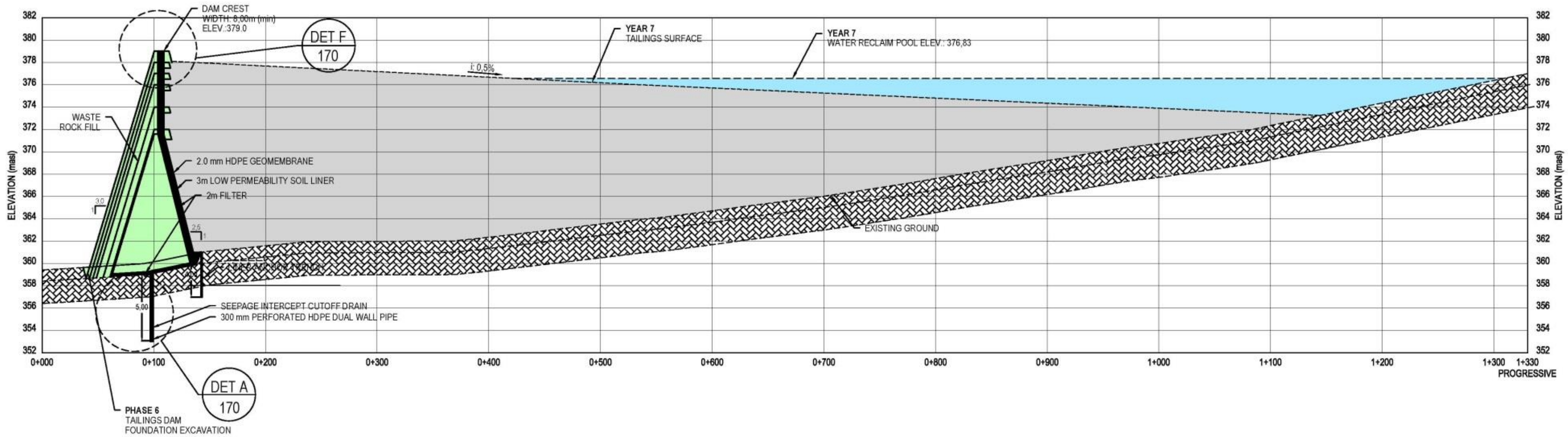


Figure 6.4 Phase 2 Centerline Raise

The key construction works associated with Phases 1 through 6 are as follows:

- Clearing and grubbing of the TSF footprint expansion;
- Removal of topsoil over the footprint expansion to stockpile;
- One meter excavation of the embankment foundation expansion;
- Compacted waste rock fill for the embankment raises from a suitable open pit source;
- Placement of the filter, low permeability soil, and geomembrane liner along the upstream face of the raises; and
- Construction of the emergency spillway located on the northeast side of the TSF for Phase 1 through 5 and construction of the closure spillway located on the east side of the TSF for Phase 6.

6.4.3 Tailings Deposition and Operational Methodology

Tailings will be discharged from multiple spigot points along the crest of the embankment to form a beach against the dam. In addition, tailings will also be deposited from the west and east sides of the facility to maximize tailings storage to embankment height. The growing beaches will push free water toward the back of the facility and the reclaim barge, and away from the embankment. Figure 6.5 illustrates this concept.

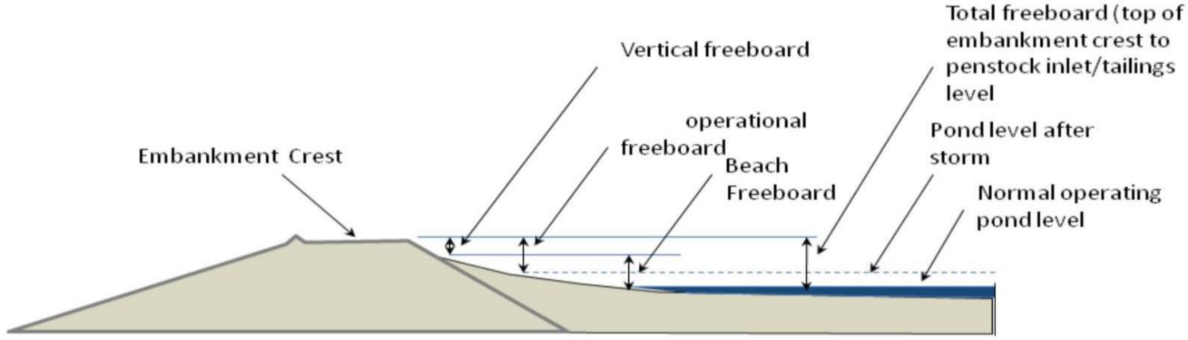


Figure 6.5 Tailings Beach Formation

6.5 Tailings Storage Facility Water Management

Water has been the underlying cause of virtually all tailings storage facility failures. The management of water within tailings facilities involves managing:

- Supernatant slurry water from tailings;
- Direct rainfall onto the TSF;
- Surface run on from storm events;
- Seepage into the subgrade; and
- Utilization of reclaim water for operations.

Diligent water management is one of the most important aspects to prevent water management issues during operations and closure. According to the most recent water balance model prepared by Schlumberger Water Services (SWS), water continues to accumulate within the TSF over the LOM due to precipitation events smaller than the 100 year 24 hour event. Therefore, it requires either a larger facility to contain the excess water or the annual release of water to maintain a pond level that will allow storm events up to and including the 100 year 24 hour storm event, to be captured within the limits of the TSF without discharging into the environment.

Water collected within the TSF is pumped back to the process plant as process make-up water. The maximum pond level within the TSF will be maintained below 341,839 m^3 to enable containment of the 100 year 24 hour design storm event, should it occur. Larger storm events will discharge through the emergency spillway. During operations and closure the maximum pool level, including allowance for the 100 year 24 hour design storm event, shall remain one meter below the crest of the embankment.

7.0 INFRASTRUCTURE

7.1 Introduction

The Project area covers a greenfield site without any existing infrastructure except for the Komana Camp. Several existing laterite roads provide access to local villages scattered around the area. The proposed infrastructure will support the process plant and associated construction operations.

The following infrastructure is required to develop the Project:

- Site access roads
- Expanded camp facilities
- Mine office building
- Warehouse
- Assay laboratory
- Change house block (including security office, change house, laundry, first-aid room, and dining/conference room)
- Security and access control
- Reagents storage sheds
- Power plant
- Fuel depot
- Airfield
- Water supply storage, purification, and distribution
- Sewage treatment and disposal
- Site electrical distribution

These facilities are described in more detail in the following sections.

7.2 Site Layout

The Project team developed the site layout taking into consideration the process plant facility, accommodation camp, relevant access roads, airfield, TSF, mining areas, including haul roads, pits, and WRDs, and other related infrastructure requirements. The overall plant site plan is shown below in Figure 7.1.

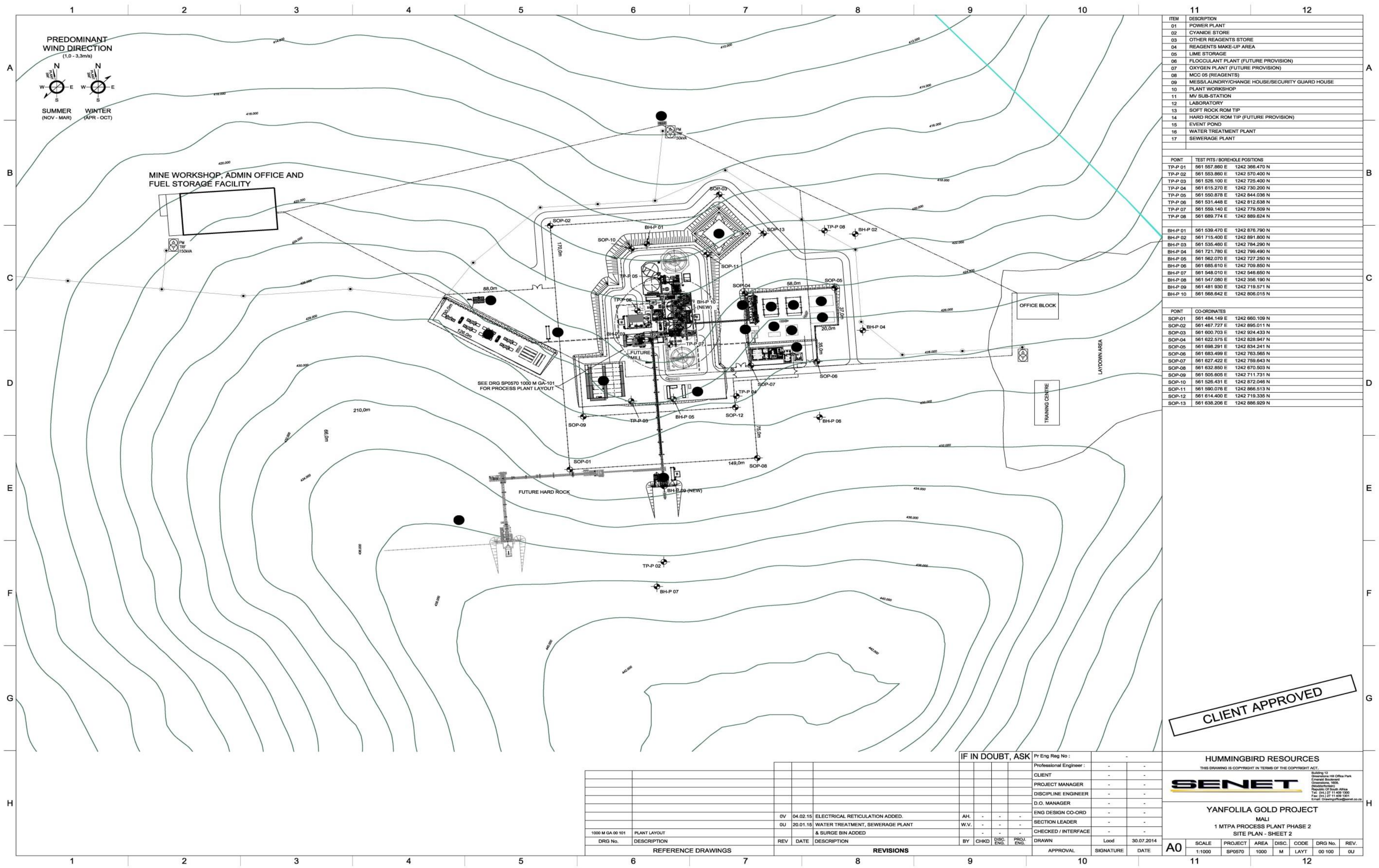


Figure 7.1 Plant Site Plan

SENET split the process plant layout into a high security area containing milling, CIL tanks, gold room, and assay laboratory, and a medium security area containing plant access and change house, dining facilities, first-aid room, reagents store, and reagents make-up area. Two layers of security fencing will enclose the high security area: an inner layer of high security fencing surrounded by a 15 m exclusion zone and an outer layer of medium security fencing.

Access roads to and from the processing plant will be stripped of organic material and compacted. Drainage ditches and culverts will be placed in accordance with the site drainage requirements.

All in-plant roads will be constructed as part of an engineered fill terrace. The in-plant roads will connect plant infrastructure buildings, such as the change house, security area, and reagent storage/handling areas. Roads will provide access to the main plant infrastructure, such as the mineral sizer, mill, CIL, gold room areas, and the power generating plant.

7.3 Site Buildings

There are two distinct building philosophies envisioned for the plant: steel portal frame buildings and conventional block buildings.

Portal frame buildings consist of steel members assembled in portal frames with square fluted steel sheeting serving as both side cladding and roof sheeting. The steel frames are founded on shallow reinforced concrete spread footing foundations with a mesh-reinforced concrete surface bed where the thickness of the floor varies depending on the loads associated with the use of the building. The choice of portal frame construction was based on its ability to offer vast open and internal column-free spaces necessary for typical storage and workshop areas.

All inhabited buildings, such as the administration offices and change house, will consist of conventional block buildings. SENET selected conventional block building construction instead of prefabricated modular buildings due to the superior durability of block buildings, their reduced logistics costs, and the fact that conventional block building construction can be executed by local Malian contractors.

The following buildings are included within the processing plant:

- Change house block (including security office, change house, laundry, first-aid, and dining/conference room)
- Gatehouse
- Cyanide store
- Reagents store
- Assay laboratory
- Control room building

7.3.1 Change House Block (322 m^2)

The change house block is a conventional brick and mortar building. The change house block building will be constructed to the east of the process plant and be located at the access point to the plant within the medium security area.

Included within the change house building is the security office, change house, laundry, first-aid room, and dining/conference room. A layout of the change house is illustrated in Figure 7.2 below.

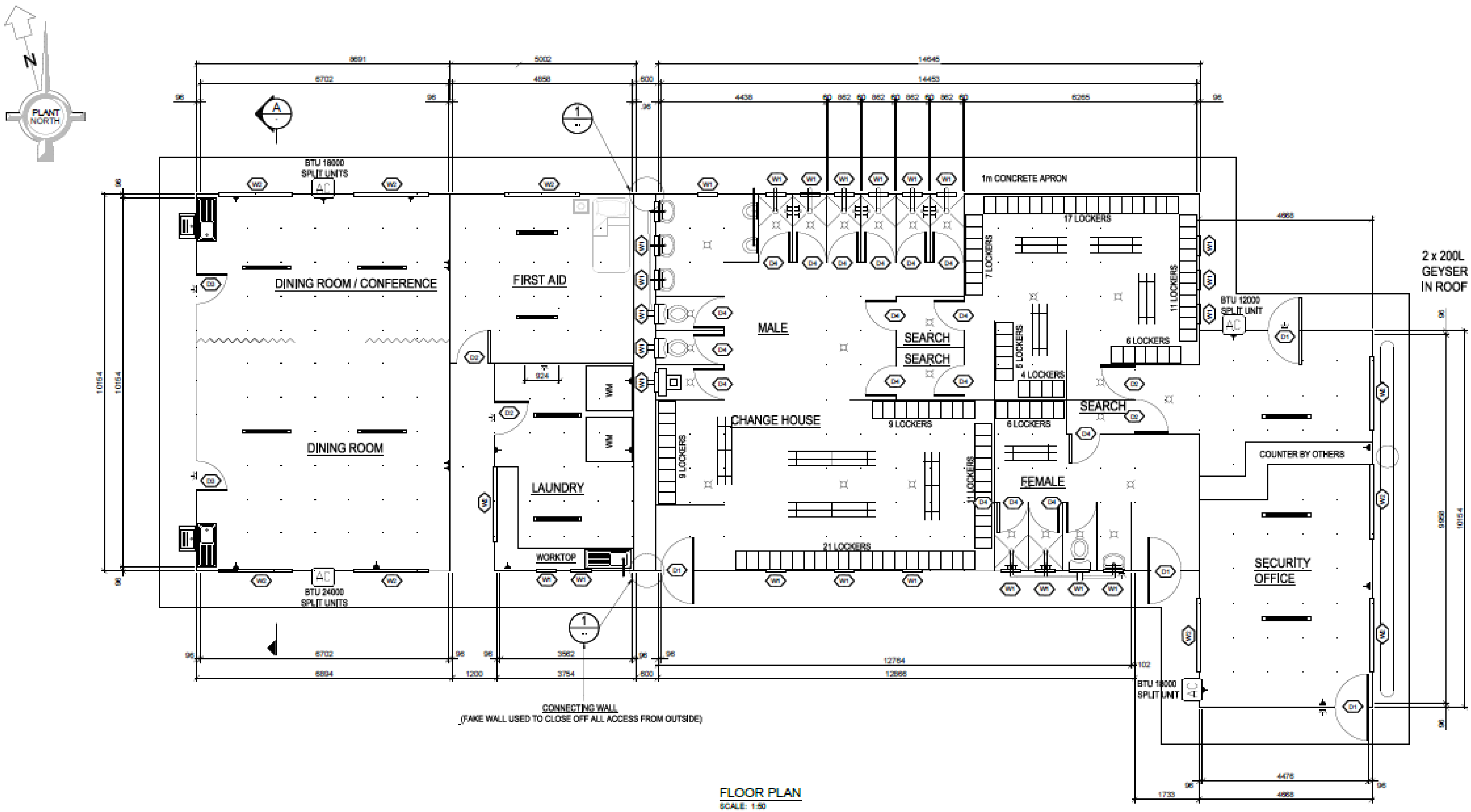


Figure 7.2 Change House Block

The change house consists of a male 'clean' change area consisting of lockers and benches provided for the process plant staff to store their clothing when they arrive for their shift. The male staff will then move through a search zone into the 'dirty' change area where they will change into their work clothes and proceed into the medium security area on the way to the process plant. At the end of shift, staff will return to the 'dirty' change area, remove dirty work clothes, which will be washed in the adjacent laundry, shower and then proceed through the search area to the 'clean' change area where they will retrieve their clothing.

The change house has been sized for 25 shift workers passing through the change house per shift, with a total of three shifts per day and one shift is on vacation leave. Therefore, 50 double lockers (a total of 100) have been provided in both the 'dirty' and 'clean' change areas to ensure one locker per shift worker. The change house will have a male and a female change area.

A fully equipped laundry room will be provided adjacent to the change house.

A first-aid room will be provided for the treatment of minor injuries sustained within the process plant, or as a holding room before evacuation from the process plant site.

A dining room/conference room designed to accommodate 25 shift workers will be provided. A section of the room can be partitioned off to form a conference/ training room.

7.3.2 Main Entrance/Gatehouse Building (28 m^2)

The main gate to the plant will enable control of vehicle and pedestrian access to the high security area of the plant via interlocked gates. The gatehouse will be a conventional brick and mortar building with a toilet and wash hand basin. The gatehouse layout is shown in Figure 7.3.

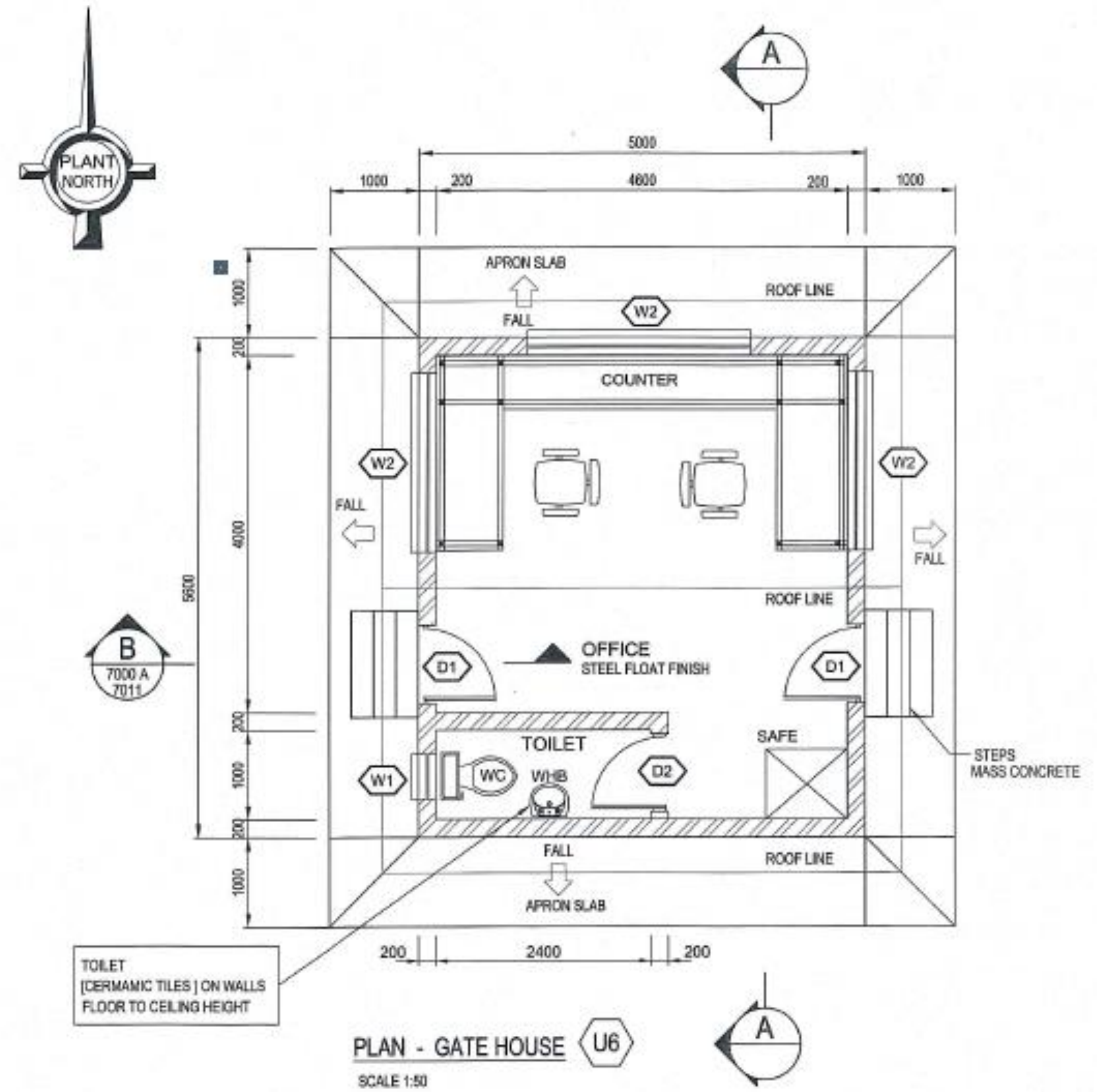


Figure 7.3 Gatehouse Layout

7.3.3 Process Plant Workshops

The process plant workshops will be located inside the high security fencing zone and will be located southwest of the process plant. There will be three modified 12 m containers: one for mechanics and welders; one for electricians and instrument technicians; and one for storage of short term plant consumables.

The bulk of non-reagent process consumables will be stored at the main warehouse located adjacent to the administration office building. Process plant spare parts will also be stored in the main warehosue, or in the outdoor fenced laydown area, and protected accordingly.

Much of plant maintenance occurs at the equipment location, or on a nearby open area on the concrete floor. Substantial maintenance will also occur at remote pump, generator, or pipe locations, for which a service truck with crane has been provided. Seldom does equipment get removed to a central shop area for repair. In the occasions it is, the modified container based shops are deemed adequate.

Furniture and shelving will be provided within the container conversions as applicable. An allowance to provide individual tools for each maintenance person, and equipment for the shops has been provided in the capital cost estimate.

No ablution facilities have been provided within the plant workshop area, as the change house facilities are nearby.

7.3.4 Assay Laboratory (260 m^2)

The assay laboratory will consist of a prefabricated structural steel building (10 m × 6 m) with roof sheeting. Two 40 foot container conversions will form the sides of the building and one 20 foot container conversion will form the front of the building. Figure 7.4 shows the laboratory layout. The overall dimensions for the concrete slab of the laboratory will be 20 m × 13 m, and steel columns will be supported on reinforced concrete foundations.

The laboratory will be located within the high security area south of the CIL area.

The laboratory will perform daily analyses of mining and process samples. The building layout is based on the requirements specified by SGS to be fit for purpose for all laboratory testing required in the plant. SGS will be responsible for providing all the laboratory equipment required as part of their contract.

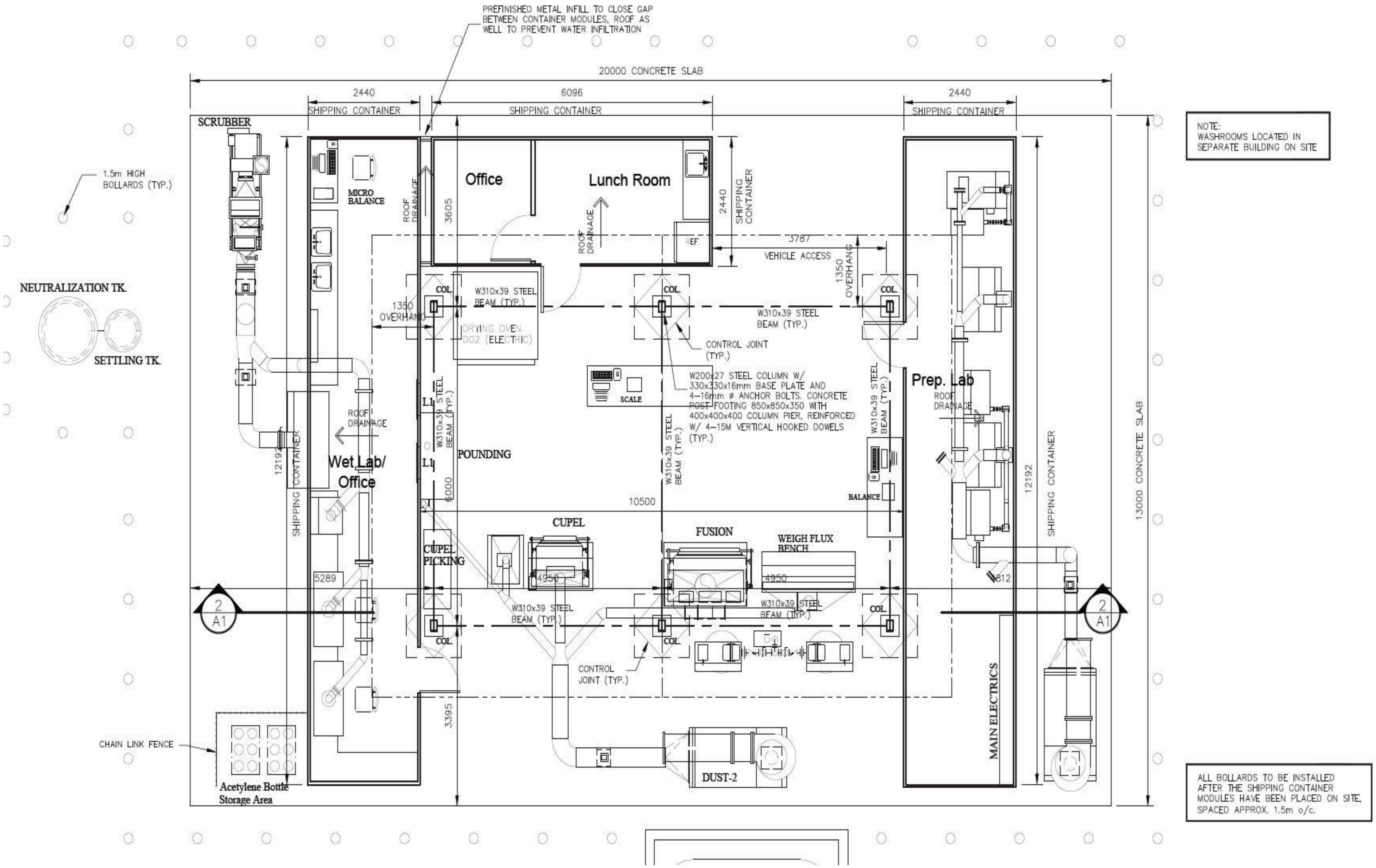


Figure 7.4 Assay Laboratory Plan Layout

7.3.5 Reagent Storage

Reagent consumptions for the Yanfolila plant are provided in Table 7.2. A dedicated store will be provided for cyanide and an additional store will be provided for the balance of the reagents. Lime, sodium metabisulphite, and activated carbon will be stored outside on a concrete slab and covered by a tarp. Due to the remote location of the Project, covered warehouse facilities to store three months' reagents stockholding will be provided. The store and slab sizing are also shown in Table7.2.

Table 7.1 Reagents Store Sizing

Reagents	Consumption		Storage (month)	Storage (t)	Three months' storage				Proposed Length (m)	Package Size	Notes
	kg/t	t/month			Pallet (No.)	Required Area (m^2)	Width (m)	Calculated Length (m)			
CYANIDE STORE											
Cyanide	0.880	73	3	220		132				1 t box, 1.2 m^2 per box	Stack boxes 2 high
Driveway						60					Driveway down the side of the building
Store Dimensions						192	12.00	16.00	16.00		Bulk bag can be stored in a portion of the driveway for first fills/or stacked 3 high
REAGENTS STORE											
Caustic	0.062	5	3	16	16	19				25 kg bags	Pallet stacked 1 high, 40 bags/pallet
Hydrochloric Acid	0.061	5	3	15	14	10				1 m^3 isotainer (1.16 t/m^3; 1.44 m^2)	Pallet stacked 2 high
Copper Sulphate	0.068	6	3	17	17	20				25 kg bags	Pallet stacked 1 high, 40 bags/pallet
Flocculant	-	-	12	0.025	1	1				25 kg bags	Pallet stacked 1 high, 40 bags/pallet
Borax	0.0018	0.15	12	1.777	2	2				25 kg bags	Pallet stacked 1 high, 40 bags/pallet
Silica	0.0009	0.07	12	0.888	1	1				25 kg bags	Pallet stacked 1 high, 40 bags/pallet
Sodium Carbonate	0.0009	0.07	12	0.888	1	1				25 kg bags	Pallet stacked 1 high, 40 bags/pallet
Sodium/ Potassium Nitrate	0.0009	0.08	12	0.900	1	1				25 kg bags	Pallet stacked 1 high, 40 bags/pallet
Fluorspar	0.0001	0.01	12	0.108	1	1				25 kg bags	Pallet stacked 1 high, 40 bags/pallet
Manganese Dioxide	0.0000	0.00	12	0.048	1	1				25 kg bags	Pallet stacked 1 high, 40 bags/pallet

Reagents	Consumption		Storage (month)	Storage (t)	Three months' storage				Proposed Length (m)	Package Size	Notes
	kg/t	t/month			Pallet (No.)	Required Area (m^2)	Width (m)	Calculated Length (m)			
Lead Nitrate	-	-	12	0.025	1	1				25 kg bags	Pallet stacked 1 high, 40 bags/pallet
Peroxide	-	-	12	6	1	3				1 000 L IBC	Stacked 2 high, 1 000 L IBC
Driveway						60					Driveway down the side of the building (4 m × 15 m)
Aisles 1						28					Aisles (4 m × 28 m)
Aisles 2						28					Aisles (4 m × 28 m)
Store Dimensions						179	12.0	14.9	16.0		Bulk bags can be stored in a portion of the driveway for first fills/or stacked 3 high
LIME SLAB											
Lime	1.950	163	3	488		195				1 t bags. 1.2 m^2 per bag	Stack bulk bags 3 high; cover by a tarpaulin
Apron (1 m)						60					
Slab Dimensions						255	12.50	20.40	20.00		
SODIUM METABISULPHITE SLAB											
Sodium Metabisulphite	0.900	75.0	3	225		90				1 t bags	Stack bulk bags 3 high; cover by a tarpaulin
Apron (1 m)						60					
Slab Dimensions						150	10.00	15.00	20.00		
ACTIVATED CARBON SLAB											
Activated Carbon	0.050	4.2	3	13	13	6				500 kg bags (1.44 m^2)	Stack boxes 3 high
Slab Dimensions						6	4.00	1.50	4.00		

7.3.5.1 Cyanide Store (192 m^2)

The cyanide store will be a structural steel building with roof sheeting and cladded sides, including a roller shutter door, with internal dimensions of 16 m × 12 m. A 4 m wide driveway has been allowed for forklift access and the stacking of bulk bags two high has been considered. There is a possibility of increasing the stack height to three high for the initial order, if required.

The cyanide store will be located in the medium security area east of the process plant, alongside the reagents make-up area.

7.3.5.2 Reagents Store (192 m^2)

The reagents store will be identical to the cyanide store with internal dimensions of 16 m × 12 m and will be located alongside the cyanide store. The reagents building will store the balance of the reagents as shown in Table 7.2 above. A 4 m wide driveway has been allowed for forklift access and, in addition, two aisles of 4 m width have been allowed between the different reagents.

7.3.5.3 Lime Slab (250 m^2)

A 200 mm thick concrete slab of dimensions 20 m × 12.5 m has been allowed for the storage of lime bulk bags. The lime bulk bags will be stacked three high and covered by a tarpaulin. The lime slab will be located within the medium security area alongside the reagents store.

7.3.5.4 Sodium Metabisulphite Slab (200 m^2)

A 200 mm thick concrete slab of dimensions 20 m × 10 m has been allowed for the storage of sodium metabisulphite bulk bags. The sodium metabisulphite bulk bags will be stacked three high and covered by a tarp. The sodium metabisulphite slab will be located within the medium security area alongside the reagents store.

7.3.5.5 Activated Carbon Slab (200 m^2)

A 200 mm thick concrete slab of dimensions 4 m × 1.6 m has been allowed for the storage of activated carbon. The activated carbon bulk bags will be stacked three high and covered by a tarp. The activated carbon slab will be located within the medium security area alongside the reagents store.

7.3.5.6 Control Room Building (36 m^2)

The Control Room Building will be a 12.2 m × 3 m modular prefabricated building located on a structural steel platform above the CIL tanks. The control room building will include a control/SCADA office, shift boss office, and a titration and testing room.

7.4 Utilities

7.4.1 Process Water Supply, Storage, and Distribution

Three types of water will be needed for the Project. They include: process water, raw water/fresh water, and potable water.

Process water is supplied to the Yanfolila plant from the sources in the order of priority shown below in order to minimize the use of additional non-contact resources:

- TSF reclaim water
- KW, KE, and GW pit dewatering sumps (potentially higher TDS)
- KW pit mine dewatering boreholes (low TDS)
- Sankarani River

Refer to layout drawing SP0570 5500 P LAYT 00 101 Rev 0C provided in Appendix 6. A summary of what has been allowed for each source is given in the following sections.

7.4.1.1 TSF Reclaim Water

Reclaim water is pumped, using barge mounted vertical turbine pumps capable of 200 m^3/h, from an excavated canal within the TSF to the process water tank (84-TKFO-02, refer to pipe P3 on the layout drawing). The piping (2,300 m of 250 OD HDPE piping) is run along a causeway.

The TSF reclaim water pipeline is assumed to follow natural ground level; no allowance has been made for trenching or sleepers.

7.4.1.2 Pit Dewatering

Water from pit dewatering from the KW pit is collected in the pit dewatering transfer pond. The water is then pumped, after suspended solids have settled, using a skid-mounted diesel-driven pump capable of 200 m^3/h, from the pit dewatering transfer pond to the process water tank (84-TKFO-02, refer to pipe P6 on the layout drawing). The piping (4,800 m of 250 OD HDPE piping) is run overland. A complete standby pump is included.

The pit dewatering transfer pond will be sized to allow a capacity of 5,000 m^3 and will be lined using 1.5 mm HDPE.

7.4.1.3 Clean Water Sump

Water from the pit boreholes is pumped, using a skid-mounted diesel-driven pump capable of 200 m^3/h, from the clean water sump to the clean water pond (83-POND-01, refer to pipe P2 on the layout drawing). The piping (1,800 m of 250 OD HDPE piping) is run overland. A complete standby pump is included.

An allowance has been made for a reinforced concrete sump of dimensions 4 m × 4 m × 3.5 m deep, with a capacity of 50 m^3.

Clean water is pumped, using an electrically driven submersible pump capable of 200 m^3/h, from the clean water pond to the raw water tank (84-TKFO-01, refer to pipe P5 on the layout drawing). The piping (2,100 m of 250 OD HDPE piping) is run overland.

The clean water pond will be sized to allow a capacity of 10,000 m^3 and will be lined using 1.5 mm HDPE.

7.4.1.4 Sankarani River Take-off

Raw water from the Sankarani River is pumped, using a skid-mounted diesel-driven pump capable of 200 m^3/h, from the river to the 50 m^3 clean water sump. The piping (750 m of 250 OD HDPE piping) is run overland. A standby pump is not included. The capability to extract water from the river is intended for short term use at plant commissioning, before TSF reclaim water is available (2-3 months), and in an emergency such as extended drought conditions.

7.4.1.5 Potable Water Distribution

The potable water treatment facility for the plant will be located inside the high security area and will consist of a containerised plant. The treated potable water will then feed to the safety showers distribution network, as well as to a potable water header for distribution to remaining project manned facilitie,s including the change house, laboratory, gold room, administration offices, power plant, fuel depot, and mining contractor's facilities.

Potable water supply to the Komana camp is excluded, as it has its own potable water supply from well boreholes. The water supply to the camp meets the Yanfolila Environmental Design Criteria, with the exception of coliforms for which treatment will be provided.

7.4.1.6 Process Plant Site Drainage

There are no permanent creeks or rivers on the site; however, during periods of rain, there will be some surface run-off, open channel flow, and standing water in low areas. Nominal grading and ditching will be adequate to maintain a well-drained site. V-drains will be installed where required to ensure proper drainage and to maintain good road conditions. Provision has also been made to collect all storm water run-off from the plant and reagents storage terrace, which will be discharged to an HDPE-lined emergency pond to prevent any contact water from being discharged into the environment. The emergency pond has a capacity of 1,500 m^3 and is located on the northeast side of the plant terrace.

7.4.2 Fire Protection

SENET's specifications describe the fire detection and protection system. This will include items such as the following:

- Fire main reticulation on site complete with isolation valves
- Installation of fire hydrants in accordance with SANS 1128-1 and SANS 1128-2
- Installation of fire hose reels
- Provision of fire pumps and tanks
- Fire detection in control room and MCCs
- Manual alarm call points
- Portable fire extinguishers

Each area will be fitted with detectors and a control/alarm station linked to a master fire zone control panel located in the control room. The system shall be designed to detect the presence of fire, smoke, or ionised gas, and to initiate alarms, trips, and the firefighting medium release for extinguishing in accordance with the duty description.

7.4.3 Sanitary Sewage

A sewage treatment plant will be located north of the plant infrastructure terrace, and has been sized for a total of 75 persons per day (3 shifts × 25 persons); 200 L per person per day has been assumed. The sewage treatment plant will be a complete system designed by the selected vendor.

A sewage reticulation system from the change house ablution, guard house ablution, and other project-wide facilities with the exception of the camp, will tie into this sewage treatment plant. Gravity sewer pipes will consist of Class 51 uPVC, 110 mm and 160 mm pipes, complete with manholes at bends and a minimum spacing of 50 m.

Sewage from the gold room ablution will discharge into a conventional septic tank located at the west side of the plant terrace, but within the high security area.

7.5 Power

The Project requires 2.7 MW of base level power to operate. HUM has solicited and received tenders from multiple prospective power plant providers. The selected candidate plans to generate power on site with a nominal 6 MW plant and sell the power to HUM on a kWh basis. Diesel engines will turn the power generators.

7.6 Diesel Fuel

A local supplier will provide an estimated 14,000,000 litres per year diesel fuel for the Project. The supplier will store diesel on site and invoice the project based on litres discharged at the pump. A minimum of 180,000 litres of diesel will be stored on site in a tank farm owned and operated by the diesel fuel supplier.

The fuel depot will supply fuel to the contracted mining equipment fleet, power plant, process plant, and HUM's mobile equipment and light vehicles.

7.7 Property Security

The process plant will be surrounded by a 2.4 m high security fence with 0.6 m high flat razor wire extending above the top of fence to prevent unauthorised entry. The high security fence is surrounded by a 15 m exclusion zone, which is in turn enclosed by a 1.8 m medium security fence.

Access to the process plant will be by means of a main access gate manned 24 hours per day by security guards. Restricted access to the premises will be by means of an access control system. Provision for additional emergency access gates will be provided, however, these will be locked at all times.

Additional fencing for further safety and security will be provided within the gold room area, transformers, and substations.

Gold will be transported by the offtake refiner from the gold room to the landing strip, with suitable security arrangements in place. Though variable based on production and security considerations, nominally gold will be transported on a biweekly frequency.

Furthermore, the plant will be fitted with CCTV cameras installed at strategic locations to provide for monitored surveillance. The cameras will be integrated with the plant's overall network including internet access to these cameras. Views from the cameras will be fed to a central security control room situated in the security/access room of the gold room.

8.0 ENVIRONMENTAL AND SOCIAL

8.1 Introduction

This section provides an overview of the HUM activities, completed and planned, related to environmental and social (E&S) work on the Project. The format of the document is as follows:

- Brief review of the E&S work undertaken pre-HUM acquisition of the Project, and the GAP analysis undertaken by HUM as part of due diligence and the acquisition process.
- Description of the E&S work programme undertaken by HUM to help fill these gaps and inform design and management of E&S risks as the Project moves through feasibility phase into construction.
- Description of health and safety monitoring and community development programs for 2014.
- Details the expected work programme for 2015 and the construction phase of the Project.

8.2 Review of Exsisting E&S Information

This section chronologically covers E&S work undertaken since 2010. Specifically:

- Scoping stage work: Environmental Resource Management (ERM) 2010 – 2011 [Appendix 7: ERM, 2011]
- Feasibility stage work: SWS: 2012 – 2013 [Appendix 7 DRS, 2013]; GF internal 2012 – 2013 [Appendix 7: Gold Fields 2014].
- Environmental and Social Impact Assessment (ESIA): ESDCO: 2012 – 2013.

The section concludes with a brief overview of HUM analysis of this work and gaps that remain(ed).

8.2.1 Pre-ESIA Studies

GFs' subsidiary, Glencar Mining, contracted ERM to conduct a number of phases of early stage E&S work. These are summarised in Table 8.1 below:

Table 8.1 Environmental and Social Work Stages

Phase	Work Area	Deliverable(s)	Date
1	Risk Assessment	Preliminary Environmental and Social	2010
2	Water Study	Komana High Level Water Supply and	October 2010
		Yanfolila Project Water Study Scoping	August 2011
3	Sensitivities Mapping	Komana GIS Sensitivities Report	December 2011
		GIS database	
4	Scoping Study	ESIA Scoping Report	December 2011
		Regulatory Scoping	
		Environmental baseline	
		Social baseline	
		Stakeholder Engagement Plan	
		Impact assessment tables	
		Mitigation tables	
		Waste Management Plan	
		Resettlement Policy Framework	

Extensive scoping stage public consultation was undertaken by ERM and local partners ESDCO during 2011. The reach of this is summarised in the table below.

Table 8.2 Scoping Stage Stakeholder Engagement and Consultation

	Number	% of Study Area Total [18]
Public meeting participants	1,030	9
Focus group participants	374	-
Households surveyed	85	5

Key concerns raised by stakeholders during consultation meetings included the following:

- Potential loss of livelihoods from the Project land use (crop fields, artisanal mine sites, orchards, and market gardens) without prior compensation.
- Potential loss of farmland and grazing areas.
- Encroachment of the Project on forest areas and resources (decrease of available firewood, fruit, natural medicines, timber, etc).
- Water pollution and environmental degradation.

(18) Based on an estimated 2009 population estimate of 11,722

- Negative impacts on traditional customs and social norms.

The Scoping Study did not find any fatal flaws to the Project as described, however it did identify a number of key environmental and social risks that would need further investigation to better understand the risk and plan appropriate mitigation measures (ERM, 2011). These are detailed below:

Table 8.3 Key Environmental and Social Risks

Environment	Social
Contamination of surface water, soil and	Economic (and physical) displacement
Increased pressure on surface and	Increased pressure on social
Potential environmental contamination	Sustainability of local economy
Remobilisation of pollutants from	Social cohesion, sense of place, cultural
Destruction of habitat and increased	Increased communicable diseases and
Air quality impacts on human and	Traffic relate accidents
Noise impacts to human and fauna	Traffic related accidents
Increased stress on existing traffic	Land ownership
Decommissioning	

8.2.2 De-Risking Study (DRS)

As part of the DRS conducted by GF for the assessment of Project feasibility, SWS was contracted to carry out further work on hydrological aspects of the Project. Significant hydrological drilling and testwork was undertaken:

- 23 long term monitoring holes (subjected to longer duration pumping tests) drilled across the Komana area.

- 11 falling head test holes across Komana deposits.

Deliverables included work on mine-site water balance, assessment of TSF design with specific reference to seepage for the Epoch TMF design, and environmental considerations of dewatering and water quality.

Details of the results of this programme can be found in Appendix 7.

A programme of waste characterisation was conducted internally by GF. This built on the initial ERM work that suggested acid rock drainage (ARD) was unlikely to be a significant issue on the Project. GF conducted geochemical waste rock characterisation on 35 samples from KW and Gonka. In general the results show that these samples were non-acid generating. Oxide and transition zone samples contain low sulphide sulphur contents and therefore are at low risk of acid generation. Fresh zone samples locally contain more sulphides, but also contain carbonate materials (dolomite and calcite) which have a high neutralisation potential. Short term leach tests showed that arsenic (As) is mobilised at KW and that As concentrations in the leachate exceed drinking water standards (WHO and USEPA guidelines). However, for IFC effluent guidelines, As concentrations met the standards except for ore samples. A programme of tailings characterisation and field kinetic test was recommended to further understand ARD ML at the site.

Field kinetic tests were set up in September 2013 with samples from lithologies across the KW deposit (deemed to be most at risk, based on earlier results). This field programme was started, however no geochemical work was undertaken on samples, and no leachate samples were analysed as per the GF internal memo instructions (Appendix 7). Following review of the programme by SWS, HUM has re-started this field programme and is also now undertaking work on tailings characterisation.

8.2.3 ESIA and Environmental Permit

GF, through its subsidiary Glencar Mining, commissioned ESDCO (a Malian based environmental and social consultant) to undertake an Environmental and Social Impact Assessment of the Project to meet local regulatory requirements (ESDCO, 2013). Work was undertaken throughout 2012 and early 2013 to compile data and complete the impact assessment. This built on scoping stage work by ERM (2011) described above.

The ESIA was mainly desk-based, with fieldwork centred on socio-economic baseline and land-use, limited flora and fauna studies, and integration of hydrological data. Work was also undertaken to complete a "Cadre Politique de Reinstallation" (ESDCO, 2013) (Resettlement Policy Framework) as well as various environmental and social management plans and other technical studies. Extensive stakeholder engagement and consultation was undertaken as part of the ESIA process and it is clear that stakeholders have a good understanding of the Project and the predicted impacts and mitigation measures. Some gaps have been identified in the ESIA when compared with international requirements mainly related to the absence of quantitative data for baseline and impact assessment in several key areas.

The ESIA contains a number of commitments, with more made in the ESMP. These commitments cover a number of topics but can be summarised into several key areas:

- Environmental monitoring.
- Protection of natural resources (forest, water, soil).
- Waste management for the camp and mine.
- Livelihood development and social investment.
- Project closure and rehabilitation.

HUM has analysed these commitments and compiled them in a Commitments Register that will be used to guide and track E&S actions and budget requirements.

Based on the ESIA described above, an environmental permit was awarded to Glencar Mining on 29th April 2013. This permit stipulates that the Project must commence within three years of award, otherwise a new ESIA must be completed. Alongside commitments made within the ESIA and accompanying ESMP, a list of specific conditions are attached to the permit (Appendix 7 – Environmental Permit).

A copy of these commitments can be found in Appendix 7 – Commitment Register.

8.2.4 Gap Analysis and Risk Assessment

In July 2014, on acquisition of the Project by HUM, an internal review was undertaken by HUM and external advisors (ERM) on environmental and social work completed to date. A process of risk assessment was undertaken to identify gaps and prioritise a work programme to fulfil obligations for international good practice, future regulatory compliance, and internal HUM policy commitments. Priority gaps and an explanation are provided:

- **Management of social issues**: some work had been completed giving a qualitative understanding of baseline conditions, but site specific data and detailed plans to address these issues, given the aggressive HUM project development schedule, was required. These issues include:
  - Artisanal and small-scale mining.
  - Stakeholder engagement.
  - Livelihoods and development projects.
  - Land acquisition.
  - Potential resettlement of Komana Bozodaga due to proximity to KW pit.
- **Air quality**
  - No quantitative baseline data had been collected.
  - Permit requirement to set up monitoring program.
  - No modelling or impact assessment to inform design and management of impact.
- **Noise**
  - No quantitative baseline data had been collected.
  - Permit requirement to set up monitoring program.
  - No modelling or impact assessment to inform design and management of impact.
- **Hydrology**
  - Limited water quality baseline data.
  - Potential As issue in natural baseline data, particularly in KW area (concerns about previous sampling methodologies).
  - No hydro-geological modelling work on Sanioumale area.
- **Biodiversity**
  - Patchy baseline data with no clear indication of methodology.
  - Results not placed in local, national, or international context for risk management.
- **Waste rock and tailings characterisation**
  - Field kinetic testwork not completed.
  - Tailings characterisation not included in DRS (results received subsequently).

A proposal was received from ERM to undertake all work, although on review it was decided to pursue individual work packages with smaller specialist consultancies and undertake management plan work in-house to a large extent. Following review of a number of proposals across work programmes, fieldwork started in September 2014 and continues through into early 2015.

8.3 E&S Studies and HUM Work Programme 2014

This section provides an overview of the work programme for 2014-early 2015, including details of consultants and their scopes of work and preliminary findings.

8.3.1 *Social Management*

HUM contracted RePlan to assist in the development of an operational ASM strategy, paying particular attention to the wider social sphere including community development and land acquisition. This looked to build on the work undertaken by GF. Previously GF undertook considerable work on ASM management (see Appendix 7 for example), instigated an ASM monitoring program, and invested into community development programmes around education, health and livelihoods development.

RePlan conducted an eight day field visit to the Project in November 2014. The visit encompassed numerous key informant interviews at the Project site and in Bamako with national level stakeholders, an assessment of work completed to date on ASM, work with the SHEC team on-site to help develop operational management and mitigation measures including project zoning, and the preparation of a presentation summarising a suggested way forward. This presentation included operational controls, advice on resourcing requirements, and a clearly articulated work programme focussing on:

- Stakeholder engagement;
- Formalisation of existing ASM management measures;
- Planning of the land acquisition process; and
- Advice related to linking community development and alternative livelihoods programs into Project risk management.

This presentation is attached in Appendix 7. Following the visit and presentation, the HUM SHEC team has worked to implement a number of these recommendations. The following work has been commenced/completed:

- Updated stakeholder engagement plan and development of new village consultation committees and a permit level consultation group for facilitating engagement work – see Appendix 7 – SEP, 2014. Specific reference has been made to inclusiveness and representation on each of these committees including land owners, key ASM participants, women, and youth representatives;
- Census on existing ASM operations across the permit area, updating previous census undertaken in 2012. Additional criteria related to age of pits, ownership of pits, and ownership of land have been added to help focus in forthcoming management measures (ongoing);
- Draft Land Acquisition and ASM strategy (ongoing) – see Appendix 7; and
- Livelihoods calendar – see Appendix 7.

Many of these tasks prepare the ground for major deliverables and work programmes in early 2015. These will include:

- Formalised village level development agreements (similar to Newmont Ahafo agreements[19] - Appendix 7) between the Project and affected communities detailing ASM management, land acquisition process, stakeholder engagement and grievance management protocols, community development / expected mitigation measures;
- Development of Land Acquisition and Compensation Plan (LACP) and Livelihood Restoration Plan (LRP), ready for land acquisition process prior to Project development. This is likely to be undertaken by national consultants ESDCO, with additional oversight provided by RePlan.

Figure 8.1 Community Meeting in Sanioumale as Part of Revised SEP, November 2014

8.3.2 Biodiversity

A team of experts led by Sally Johnson (Fairfields Consultancy) was contracted to undertake a Rapid Biodiversity Assessment (RBA) and provide suggested mitigation measures based on fieldwork. This work is designed to build on biodiversity studies undertaken by ERM (2011) and ESDCO (2012) and better understand the Project biodiversity risk profile.

8.3.2.1 Flora

The flora team was headed by Dr William Hawthorne (Oxford University) with Mathieue Gueye (IFAN, Senegal) in collaboration with two Malian botanists, undertaking fieldwork for 15 days. Analytical work, including GIS mapping and plant use assessment, is provided by Dr Hawthorne and Cicely Marshall (Oxford University). A field survey was conducted 16-29 October 2014 following the Rapid Botanic Survey (RBS) protocol, which included the recording of local names

[19] Available to download at: http://www.sdsg.org/wp-content/uploads/2013/04/Newmont-Ahafo-Development-Agreement.pdf [Accessed 13/11/2014 17:10]

and uses of plants. Forty-four sample areas were assessed. 394 species were identified from these samples, and some 80 species have been identified to genus level only. Of the 44 samples, 36 samples were conducted with local informants who gave local name and usage for the species.

- Five vegetation classes were recognised: *Garcinia livingstonei-Afzelia* gallery and fringing forest; A*nogeissus-Bridelia* lowland woodland; *Guiera-Acacia macrostachya* upland savanna; *Xeroderris-Ficus thonningii* woodland; and *Pericopsis-Pteleopsis* woodland. These vegetation classes were mapped where possible;

- No 'critical habitat' IFC PS 6 (2012) was identified in the area based on plant species or vegetation types;

- Almost all the species sampled were Green Star species, (widespread across Africa), or at least very common in West Africa, and if they were to be formally assessed by IUCN, the majority would almost certainly rank as "Least Concern";

- Across all the samples, with all the species taken together, a "Global Heat Index" of 50 is calculated from the balance of species of different stars. Fifty is a low GHI score and typical of any cold-spot region in the tropics, of no particular biodiversity conservation merit;

- Three globally very rare species ('Black Star species') were found: *Pavetta cinereifolia, Cyphostemma descoingsii, Cissus gambiana;* however, the habitat of the concession cannot be considered significant for the species as it is so common in the area.

- The highest ('hottest') GHIs occur in the wooded savanna, Class 4 vegetation both of the Xeroderris-Ficus thonningii woodland type[20.] This should be considered of medium sensitivity as it harbours the highest concentration of globally rare species, and a high number of the most important species to local communities.

- Even though there is not a high proportion of endemic or globally rare species, all natural (non-agricultural) vegetation in the area woodland does have significant ecosystem service (for local communities) and ecological value (for fauna). A very high proportion of the wild flora is locally recognised and valued. More species were used medicinally (244 species) than for any other category of use.

- Further period of sampling in the wet season would likely reveal additional species, which may or may not be of conservation concern, and would also improve the resolution and characterisation of the ferricrete vegetation, and the riverine vegetation, which were under-sampled.

The draft report is available in Appendix 7.

8.3.2.2 Fauna

The fauna team was led by Okapi, with Dr Francis Lauginie heading the large mammal team and Malian specialists working on small mammals, birds, reptiles, and amphibians. The seven

[20] This is pending clarification from Will Hawthorne *et al*.

person team undertook fieldwork from 22nd October to 4th November on the Project site. Their report found:

- The area (particularly the two most northern communes [Yallankoro Soloba and Séré Moussa ani Samou] where the Project is located) appear to be impoverished both in terms of diversity of mammal and bird species and density of individual species. This is likely to be the result of uncontrolled hunting together with artisanal mining, pastoral, and other agricultural activities;

- No terrestrial biodiversity features triggered 'critical habitat'. However that stretch of the Sankarani *could be* 'critical habitat', as it supports an extraordinary diversity of fish species (> 90) including several endemic species and forms part of a Ramsar site on the Guinean side of the river. The Hippopotamus *Hippopotamus amphibius* (VU) is also present in this stretch of the river, (in a zone approximately 20 km of the mining permit), where a female recently caused some damages to crops. It is also possible that the African manatee, *Trichechus senegalensis* (VU), is present near the Sélingué dam;

- It is possible that the critically endangered species *Mecistops cataphractus* Slender-snouted crocodile is present in the Project area. It is unclear from the report where this species was located and this needs to be clarified immediately. This species is incredibly shy and susceptible to human disturbance. It occupies gallery forests and other forested wetland habitats by rivers;

- There are a number of other threatened reptiles may be present in the project area including the West Africa dwarf crocodile (VU) *Osteolaemus tetraspis* which is thought to occur in the small Kolé / Lambidoula gallery forest of the future tailings area; and

- The only other species that is of some conservation concern is the Waterbuck *Kobus ellipsiprymnus defassa (*NT) and it has not been seen in the past three years and probably has disappeared from the two northernmost communes, and the *Profelis aurata* (NT) the Golden Cat.

- The wildlife status is somewhat better in the Djallon Foula commune (Southern section of Komana Exploitation Permit) where the vegetation cover is better preserved and supports more mammals such as the Bohor Reedbucks, the Kob and the Waterbuck, Western hartebeest and Roan Antelopes.

- Although the fauna report mentions Colobus polykomos vellerosus, as possibly once having been present, Sally Johnson reports never having heard of this species[21].

The draft report is available in Appendix 7.

[21] "I think they are referring to Geoffrey's black and white colobus Colobus vellerosus but it might be the King Colobus Colobus polykomos both (VU). I am not convinced that either of these species have ever occurred in Mali." (Sally Johnson, 2015)

Figure 8.2 Okapi Team Conducting Key Informant Interviews - RBA October 2014

8.3.2.3 Summary of Biodiversity Risk Profile

The Project is not located in an area of global terrestrial biodiversity value. The area is not a Biodiversity hotspot, a Global 200 ecoregion, an Endemic Bird Area, an Important Bird Area or an Alliance for Zero Extinction site. There are no Key Biodiversity Areas (KBAs) within the Project area of influence. However, it is located in the Upper Niger freshwater Ecoregion of the world (see Figure 8.2). The Upper Niger and its tributaries (including the Sankarani) define this ecoregion. The Upper Niger is the crucial source of floodwaters that support the rich inner delta downstream as well as providing important flow for the Lower Niger. The Upper Niger ecoregion is host to about 150 fish (including 8 endemics).

A review of the Integrated Biodiversity Assessment Tool (IBAT) tool for the Project area was undertaken. A total number of 19 globally Threatened (CR, EN, VU), Near Threatened (NT) and Data Deficient (DD) species were found in the polygon in which the Project area occurs. This list (see Table 8.4 below) is derived directly from the species distribution maps produced as part of each individual Red List assessment. These identify possible occurrences of species and not actual presence. None of these species were directly or indirectly observed in the Project area, although the leopard and possibly the manatee are still thought be present in the wider area.

Table 8.4 Globally Threatened, Near Threatened, Near Threatened, and Data Deficient Species In the Project Area

Taxonomic group	Species	IUCN Red List
Birds	*Circaetus beaudouini Beaudouin's* Snake-eagle	VU
Birds	*Circus macrourus* Pallid Harrier	NT
Birds	*Gallinago media* Great Snipe	NT
Birds	*Gyps africanus* White-backed Vulture	EN
Birds	*Limosa limosa* Black-tailed Godwit	NT
Birds	*Necrosyrtes monachus* Hooded Vulture	EN
Birds	*Neophron percnopterus* Egyptian Vulture	EN
Birds	*Neotis denhami* Denham's Bustard	NT
Birds	*Polemaetus bellicosus* Martial Eagle	NT
Birds	*Rynchops flavirostris* African Skimmer	Nt
Birds	*erathopius ecaudatus* Bateleur	NT
Fish	*Elops senegalensis*	DD
Mammals	*Eidolon helvum* Straw-coloured Fruit Bat	NT
Mammals	*Hipposideros jonesi* Jones' Roundleaf Bat	NT
Mammals	*Myopterus daubentonii Daubenton's* Free-tailed *Bat*	DD
Mammals	*Panthera leo* Lion	VU
Mammals	*Panthera pardus* Leopard	NT
Mammals	*Scotoecus albofuscus* Light-winged Lesser House Bat	DD
Mammals	*Trichechus senegalensis* West African Manatee	VU

Both reports suggest that impacts are unlikely to result in the significant loss of important biodiversity features.

Johnson (2015) provides more details in Apppendix 7.

8.3.3 Air Quality

SRK (Durban) have been contracted to conduct baseline air quality monitoring and undertake an impact assessment. This will provide key input into resettlement planning (if needed), operational design, and mitigation measures, regulatory required monitoring program.

Specifics of each element are given below:

- **Baseline monitoring**
    - Procurement and installation of equipment, setting up a laboratory at site.

- Develop air quality monitoring plan.
- Monthly ambient monitoring of dust, specifically dust fallout, PM10 and PM2.5 for a period of 12 months.
- Quarterly passive air quality monitoring for gases (sulphur dioxide and nitrogen dioxide).
- Monthly update of databases.
- Annual report.

All equipment has been procured and is on site at Komana Camp. Between the 16th and 18th December 2014, HUM SHEC team installed eight dust fallout monitoring stations across the Project area following the SRK air quality monitoring protocol for Yanfolila (Appendix 7). The first samples will be collected 30 days later.

Table 8.5 Air Quality Monitoring Locations

Field ID	Village/Area	X	Y
GDM01	Komana	558834	1240341
GDM02	Leba Bozodaga	557818	1243696
GDM03	Sanioumale	567667	1255760
GDM04	Leba	558256	1245312
GDM05	Komana Bozodaga	557939	1240981
GDM06	Kenieba	561702	1245701
GDM07	Bougoudale	560229	1245068
GDM08	Camp	558912	1243061

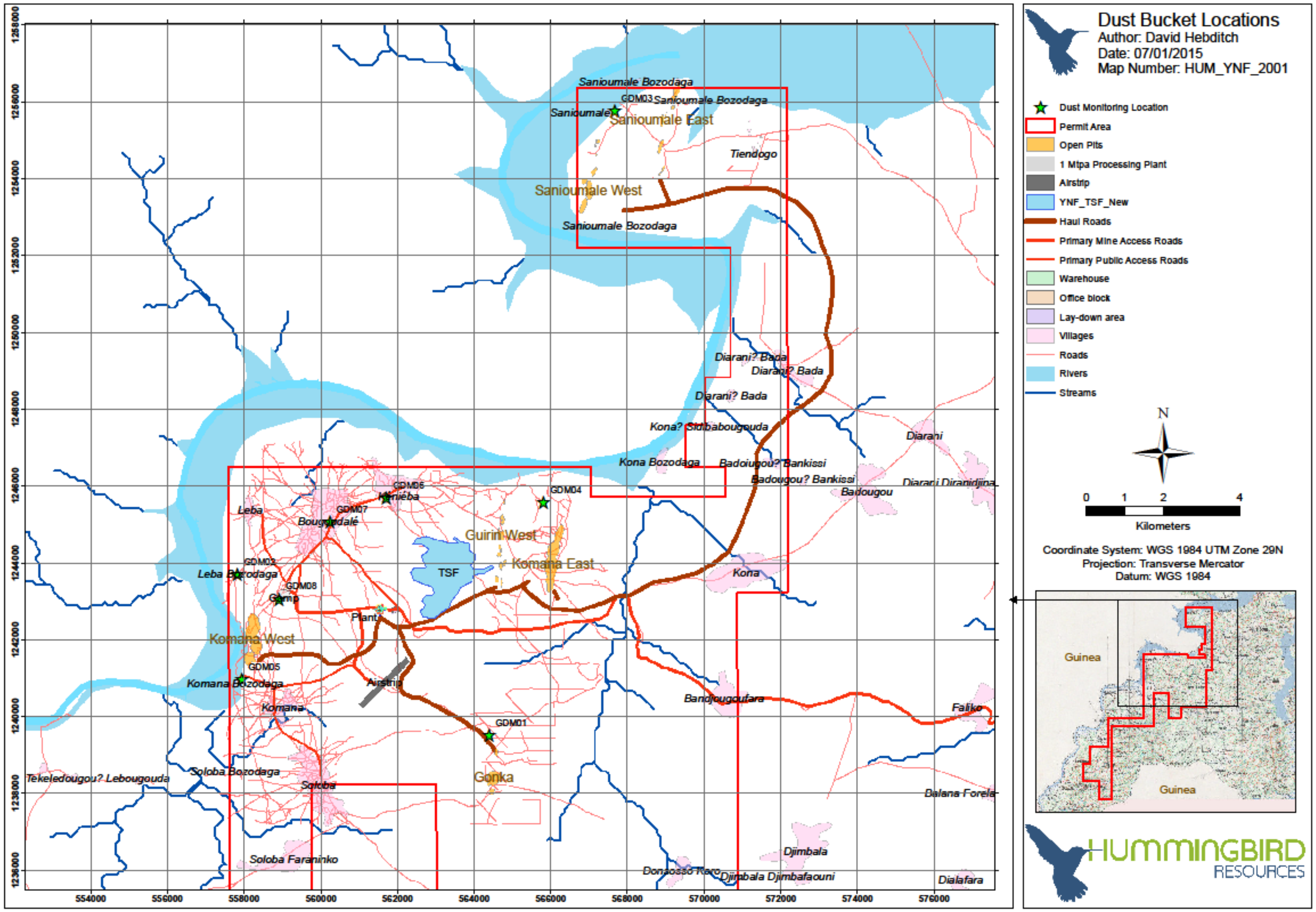


Figure 8.3 Air Quality Dust Bucket Locations

- **Impact assessment**
  - Review of project documentation.
  - Preparation of air emissions inventory.
  - Setup and undertake atmospheric dispersion modelling for dust (PM10, PM2.5 and dust fallout) and gases.
  - Preparation of impact assessment and recommendation of management and mitigation measures.

The SRK team completed their site visit 26th – 30th January 2015. Working in close partnership with HUM SHEC team, SRK installed the remaining monitoring equipment, on-site laboratory for dust fallout measurement, and undertook full training of HUM staff.

A number of Standard Operating Procedures (SOPs) for laboratory operation and sampling have been developed by SRK.

Figure 8.4 Dust Fallout Bucket at Bougoudale

8.3.4 Noise Baseline

Mackenzie-Hoy Consulting (South Africa) were contracted to undertake noise baseline monitoring, impact assessment, and development of a noise management plan / mitigation measures.

Terry Mackenzie-Hoy completed an eight day site visit conducting baseline measurements across the Komana and Sanioumale area. Details of locations of the noise testing and duration of testwork are given in Table 8.6 below.

Table 8.6 Noise Testing Locations and Testwork Durations

Location ID	Name	Date	Test	UTM 29N Co-ordinates	
				x	Y
MR01	Leba	06/11/2014	24 hours	557776	1243713
MR02	Komana	07/11/2014	24 hours	557776	1240934
MR03	Leba Village	07/11/2014	24 hours	558179	1245173
MR04	Bougoudale	10/11/2014	24 hours	559595	1244819
MR05	Kenieba	10/11/2014	24 hours	562340	1246138
MR06	Komana	10/11/2014	24 hours	559226	1240678
MR07	Kona	11/11/2014	1 hour	570198	1243701
MR08	Sanioumale	12/11/2014	1 hour	567134	1255438
MR09	Soloba	13/11/2014	1 hour	560447	1237598

Results show that the average day and night time noise levels in key receptors (communities) are often above WHO standards for residential setting. The table below provides some further detail.

Table 8.7 Results of Day and Night Noise Levels in Key Communities

Location	Name	Date	Test	Daytime	Night time
MR01	Leba	06/11/2014	24 hours	56	55
MR02	Komana	07/11/2014	24 hours	60.5	52
MR03	Leba Village	07/11/2014	24 hours	67.6	67
MR04	Bougoudale	10/11/2014	24 hours	66.8	56
MR05	Kenieba	10/11/2014	24 hours	63.4	50
MR06	Komana	10/11/2014	24 hours	78.5	41.2
MR07	Kona	11/11/2014	1 hour	54.2	
MR08	Sanioumale	12/11/2014	1 hour	54.4	
MR09	Soloba	13/11/2014	1 hour	61.2	
WHO Noise Level Guidelines					
				Daytime	Nighttime
Residential, institutional, educational				55	45
Industrial, commercial				70	70

Using SoundPlan © software, and in-house assumptions (see report in Appendix 7 for more detail) Machoy engineers produced noise contour maps for each pit, and infrastructure locations. An example of this is shown in Figure 8.5. As well as the high baseline noise conditions encountered on site, there is high noise attenuation due to local conditions. Concerns over noise impact on the population of Komana Bozodaga (located to SSW of KW pit) would appear to be allayed through this work. However there were also some early concerns regarding SE pit and the potential impact it may have on two small Bozo communities nearby. Modelling work by Mackenzie Hoy following the draft report submission suggests measures such as noise berms will mitigate impact.

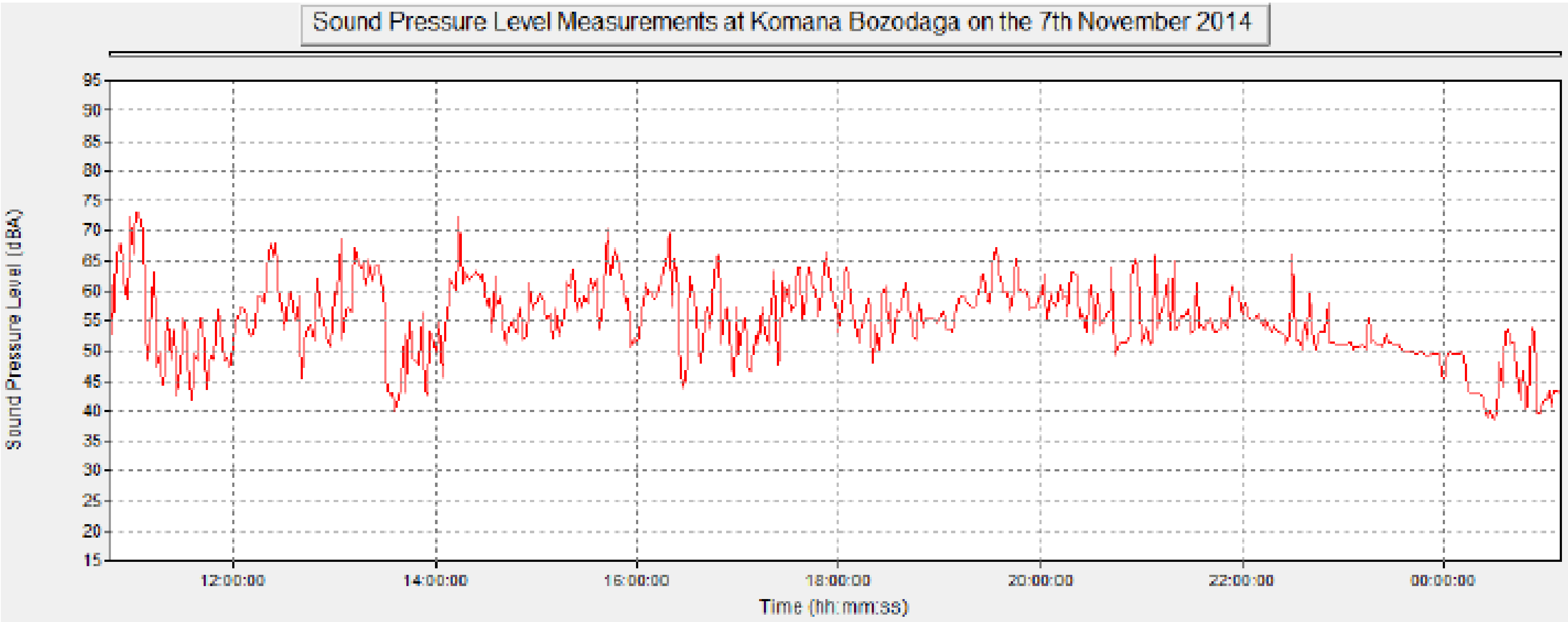


Figure 8.5 Sound Pressure Level Profile Measured at MR2 (Komana Bozodaga) on 7th November

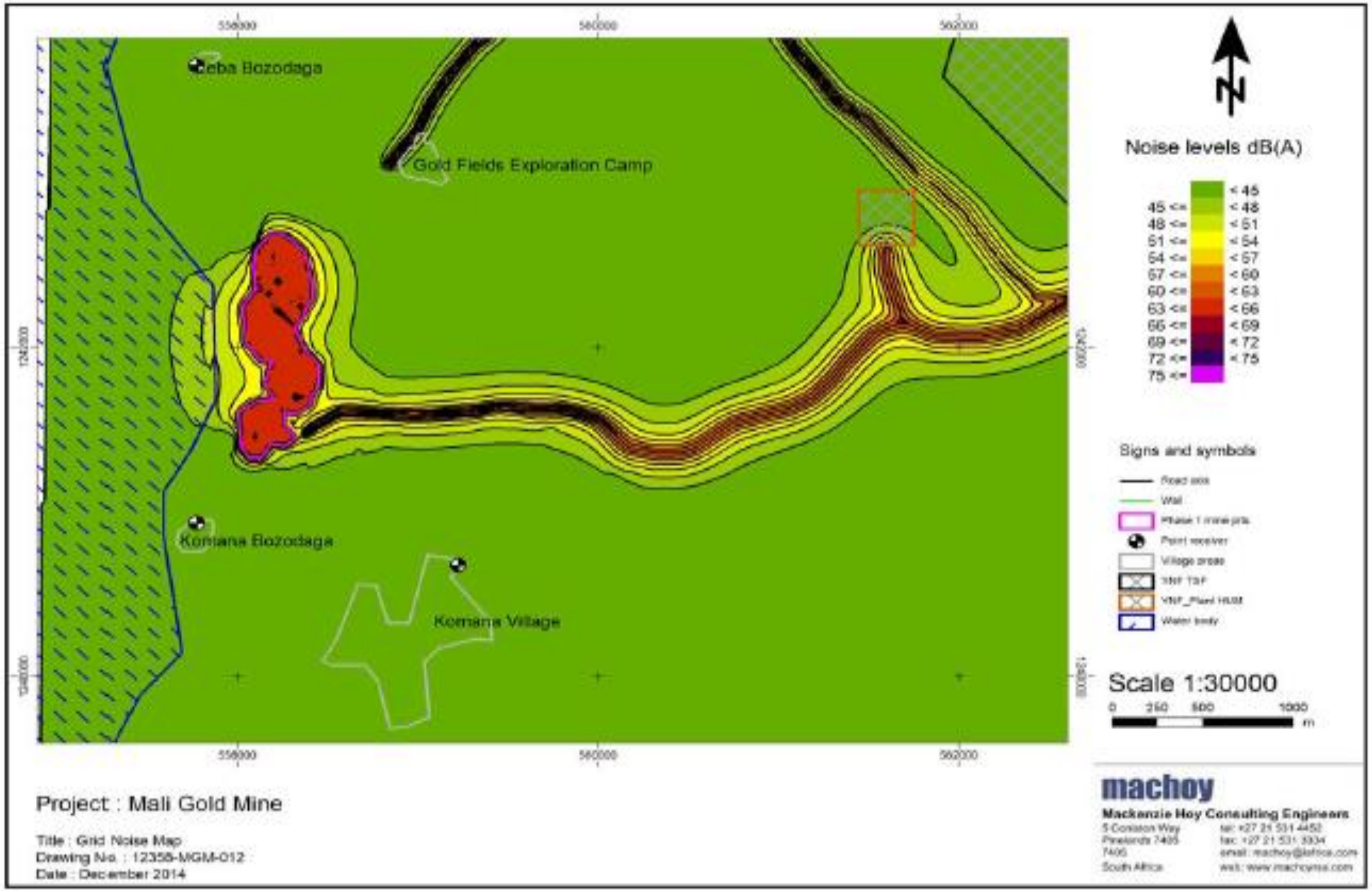


Figure 8.6 Noise Contours from Komana West Pit in Vicinity of Komana, Komana Bozodaga, Leba Bozodaga, and Camp Site

8.3.5 *Hydrology*

Hydrological studies have been completed on the Project by both ERM and SWS as described above. However hydro-geological work was not completed on the Sanioumale area, and baseline water quality data (groundwater, surface water, community wells) is considered patchy with potential issues of elevated baseline Arsenic (above IFC and WHO drinking water guidelines) in the KW area and a lack of field water quality data. Some concerns exist over the As results previously received for two reasons: the level of confidence in the laboratory used, and potentially flawed sampling methodologies. To counter this, HUM SOPs for water sampling were applied (reviewed by SWS and AMEC at Dugbe) and SGS Envilab in Bamako was contracted to provide services.

To that end a significant field programme was started in September 2014:

- Hydro-geological work programme in Sanioumale (designed by SWS, with field supervision of drilling and test work provided by GCS).
    - Drilling of eight long term baseline monitoring holes :
        - Average depth 60 m
        - Cased with plain and slotted 5” PVC
        - Gravel pack
        - Lockable top caps(average 60 m depth, cased with perforated and plain PVC, gravel pack, and lockable top-caps), subjected to 24 hour constant discharge test
    - Drilling of 12 falling head test holes at varying depths across the two deposits.
    - Excavation of four shallow pits to test permeability of laterite layers.

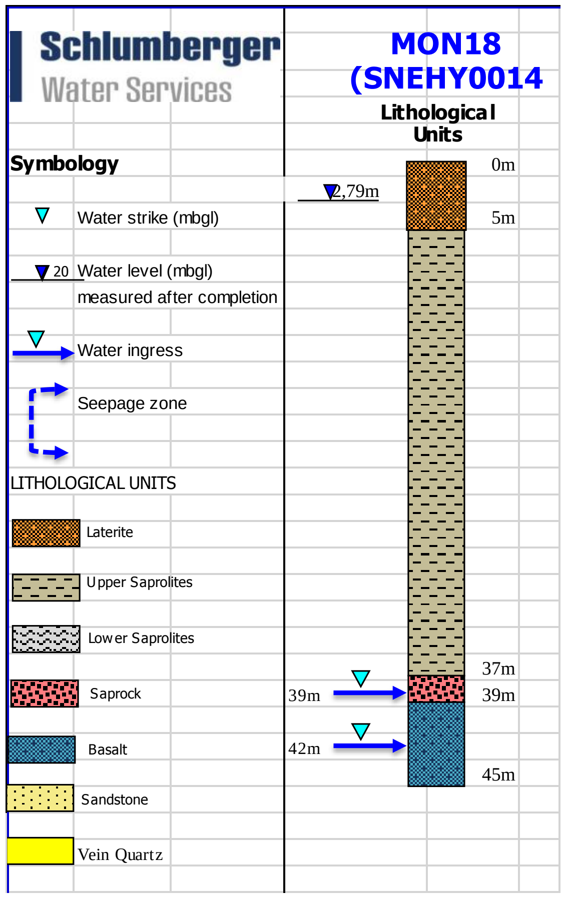


Figure 8.7 Examples of Log of Monitoring Borehole at Sanioumale

- Field water quality monitoring program.
  - Weekly groundwater level monitoring of all drilled boreholes (no. 50).
  - Monthly groundwater level monitoring of all community wells (no. 116).
  - Monthly field water quality testing of groundwater, surface water and community wells.
- Quarterly laboratory analysis of groundwater, surface water and community wells.

Lab results have been received from the October round of sampling. These confirm earlier lab results indicating relatively high levels of arsenic in baseline groundwater conditions around the KW deposit and along strike. Appendix 7 provides lab results from SGS.

8.3.6 Waste Rock Characterisation

Based on previous waste rock characterisation studies (ERM, 2012), the main parameter of interest is Arsenic, which occurs naturally at elevated concentrations in groundwater in the KW area, and is shown to be mobilised from transitional and fresh ore/waste from samples from this deposit. In some instances short term leach test results have previously shown leachate with As levels above IFC effluent guidelines. A need to better understand behaviour of arsenic under site-specific climatic conditions (hot and dry with seasonal rains) has been identified. SWS provided instruction to HUM in a memo (Appendix 7) on re-starting the field kinetic tests at Komana originally set up by GF.

Actions to restart these tests included:

- Existing leachate sample collected, analysed for field parameters, and sent to SGS Bamako for laboratory analysis.
- Drill core from each barrel was emptied and then a representative (quartered and coned) 5 kg sample was taken from each and sent to Maxxam Analytics in Canada for Acid Base Accounting (ABA,) ICPMS and Acqua Regis testing, Sulphate and Total Carbon analysis, and Rietveld XRF analysis.
- Drill core was crushed by hand into more representative sizing and then placed back into barrels. All barrels were thoroughly cleaned beforehand.
- Leachate samples now collected on a weekly basis when available due to rainfall, analysed for field parameters and samples sent to SGS Bamako for laboratory analysis.

Table 8.8 Barrel Details For Onsite Leaching Tests

Sample ID	HUM ID	Hole ID	From	To	Interval	Estimate Weight	Zone	Observations	Date set up
1	HUM-ARD-001	KGT0030	38.5	49.8	11.3	67.34	RSL	Central	03/10/2013
2	HUM-ARD-002	KGT0030	56.6	64	7.4	26.22	RSSR(Tans)		03/10/2013
3	HUM-ARD-003	KGT0030A	90	111	21	168.24	Fresh	Central	22/09/2013
4	HUM-ARD-004	KGT0034	19	32.5	13.5	76.68	RSSR(Tans)	Unique sample in ICFP	02/10/2013
5	HUM-ARD-005	KGT0028	10	33.7	23.7	84.56	RSU	Outer part of pit for reference	04/10/2013
6	HUM-ARD-006	KGT0032	7.6	22.6	15	49.1	RSL	Shallow above mineralisation	03/10/2013
7	HUM-ARD-007	KGT0003	5	28	23	94.86	RSL	In high as zone	02/10/2013
8	HUM-ARD-008	KGT0036	29	45.5	16.5	73.76	RSSR(Tans)	South pit	02/10/2013
9	HUM-ARD-009	KGT0039	56.8	64.4	7.6	31.7	RSSR(Tans)	Far south pit	03/10/2013
10	HUM-ARD-010	KGT0030A	136.7	158	21.3	166.36	Fresh	Central zone	22/09/2013
11	HUM-ARD-011	KGT0032	128	150	22	168.3	Fresh	below mineralisation	22/09/2013

Figure 8.8 Leachate Barrels at Komana Camp

Initial results show that As is mobilised in some samples from the field kinetic tests (HUM-ARD-002; 007; 010; 011), but to date laboratory results have shown that As levels are all well below IFC effluent guidelines (highest result HUM-ARD-007 at 0.024 mg/l) although above IFC and WHO drinking water standards. Results to date can be found in Appendix 7.

Geochemical tailings characterisation work is currently being undertaken through SGS South Africa. 20 kg samples of tailings derived from the processing of oxide and fresh material from KW will be analyzed for the deportment of arsenic in the tailings.

8.4 Health and Safety

Since acquisition of the Project, HUM has maintained the Project zero LTI rate since recording started in October 2012. The existing health and safety management system at Yanfolila is currently being reviewed and updated as the Project enters construction. In particular the Project will bolster the SHEC department with the hire of a Health and Safety Manager and Officer once finance is received.

Health and safety monitoring and reporting statistics are shown in the table below.

Table 8.9 Komana Camp Injury Frequency (December 2014)

Period	Man	LTIFR	MTIFR	MIFR	TIFR
Month to Date	21954	0.0	0.0	0.0	**0.0**
Quarter to	75586	0.0	0.0	0.0	**0.0**
Year to Date	309891	0.0	0.00	9,68	**9,68**
Project to	**1132709**	**0.0**	**1.77**	**3.53**	**5.30**

Note: - LTIFR: Lost Time Injury Frequency Rate, MTIFR: Medically Treated Injury Frequency Rate, MIFR: Minor Injury Frequency Rate. Frequency rates are reported per 1 million hours.

The onsite clinic treatment figures for 2014 are given in the table below.

Table 8.10 Onsite Clinic Treatments for 2014

Month	Malaria	ENT	Gastro-Intestinal	Diarrhea	Low pulmonary Infection	High pulmonary Infection	Musculoskeletal	Urinary Infection	Opthalmology	Typhoid	Cardiovascular	Neurology	Psychiatry	Minor surgery	Spot accident	Domestic accident	Injury on duty	HIV	OI	Other
Jan-14	7	15	9	0	0	1	12	2	2	0	2	0	0	0	0	0	0	0	0	20
Feb-14	0	6	3	0	0	2	1	0	0	0	3	0	0	0	0	0	0	0	0	14
Mar-14	1	4	4	0	0	1	0	0	1	0	0	1	1	0	0	0	0	0	0	10
Apr-14	2	4	3	0	1	0	0	1	1	0	2	4	0	0	0	0	0	0	0	15
May-14	1	1	1	0	0	1	1	0	2	0	0	2	1	0	0	0	0	0	0	15
Jun-14	4	9	6	0	0	0	4	1	0	0	2	1	0	0	0	0	0	0	0	16
Jul-14	6	8	6	1	0	3	1	0	0	0	1	0	1	1	0	0	1	0	0	20
Aug-14	7	10	1	0	1	1	0	0	2	1	2	0	0	2	0	0	1	0	0	27
Sep-14	16	7	5	0	0	1	4	1	3	0	0	1	0	0	0	0	0	0	0	41
Oct-14	5	4	3	1	0	1	3	0	2	0	8	0	0	1	0	0	0	0	0	10
Nov-14	5	0	7	0	0	0	2	0	0	0	2	1	1	0	0	0	0	0	0	24
Dec-14	5	4	5	0	0	0	6	1	0	0	0	0	1	0	0	0	0	0	0	7
Quarter 1 2014	8	25	16	0	0	4	13	2	3	0	5	1	1	0	0	0	0	0	0	44
Quarter 2 2014	7	14	10	0	1	1	5	2	3	0	4	7	1	0	0	0	0	0	0	46
Quarter 3 2014	29	25	12	1	1	5	5	1	5	1	3	1	1	3	0	0	2	0	0	88
Quarter 4 2014	15	8	15	1	0	1	11	1	2	0	10	1	2	1	0	0	0	0	0	41
Year to date 2014	59	72	53	2	2	11	34	6	13	1	22	10	5	4	0	0	2	0	0	219

Table 8.11 Malaria Incidence Statistics for 2014

Month	Rainfall (mm)	Total			KC residents		KC non-resident	
		Cases	Incidence rate	Days lost	Cases	Incidence rate	Cases	Incidence rate
Jan-14	4.1	16	0.1301	26	2	0.0488	14	0.1707
Feb-14	0	2	0.0163	2	1	0.0244	1	0.0122
Mar-14	0	2	0.0741	3	0	0.0000	2	0.2000
Apr-14	8.1	2	0.0870	0	1	0.0769	1	0.1000
May-14	267.7	7	0.2593	9	2	0.1538	5	0.3571
Jun-14	131.3	6	0.1875	6	1	0.0769	5	0.2632
Jul-14	182.6	11	0.2619	7	1	0.0455	10	0.5000
Aug-14	246.9	7	0.0614	13	1	0.0323	6	0.0723
Sep-14	348.7	16	0.1019	26	2	0.0645	14	0.1111
Oct-14	58.89	8	0.0769	7	5	0.1613	3	0.0411
Nov-14	0.25	5	0.0758	8	3	0.1500	2	0.0435
Dec-14	0	5	0.0847	3	1	0.0667	4	0.0909
Total / average	**1248.54**	**87**	**0.1181**	**110**	**20**	**0.0751**	**67**	**0.1635**

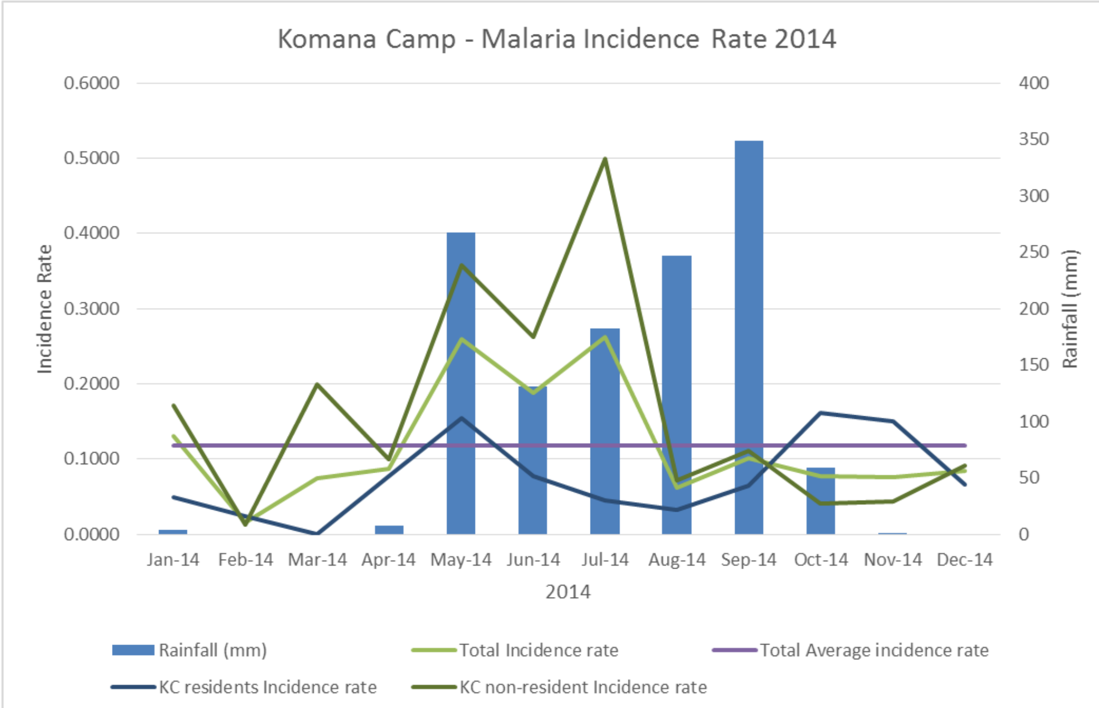


Figure 8.9 Komana Camp Malaria Incidence Rate 2014

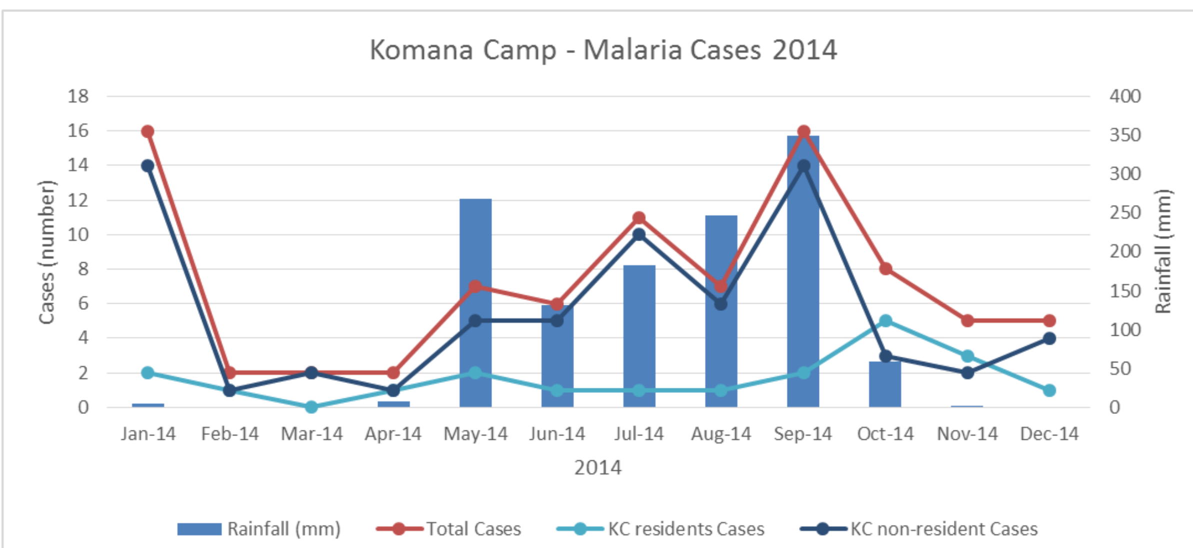


Figure 8.10 Komana Camp Malaria Cases 2014

Despite current efforts at malaria prevention and awareness, there is a relatively high baseline malaria incidence present in the area. As the Project moves into construction, this presents an operational risk. To that end HUM has contacted a number of healthcare professionals experienced in malaria assessment and setting up of Integrated Control Programmes. A proposal has been received from Mark Divall at SHAPE Consulting to undertake a full assessment at site and design such a program.

This assessment is scheduled to be undertaken prior to construction.

8.5 Community Development

Upon acquisition of the project, HUM undertook a review of the implemented, current, and planned community development initiatives committed to by previous Project proponents.

During the optimisation study period, HUM has committed to continuing on-going support to communities, whilst also reviewing the expected efficacy of initiatives planned by GF. Since July 2014 the Project has:

- Continued on-going support to payment of teachers and nurses salaries (18pax) paid monthly across the Project area.
- Continued on-going monitoring and support to market garden projects in collaboration with the Agricultural Department of Yanfolila.
- Increased local procurement (especially through catering contractors ATS) and assisted to identify and fulfil opportunities for local food suppliers (meat, fruit, vegetables).

Two major projects were committed to, but not implemented by GF. These were:

- Local brickmaking for construction.
- Rice irrigation project (alternative livelihoods for ASM).

A third project, drilling of wells for drinking water, was started by GF but never completed due to concerns with arsenic in drinking water.

Since acquiring the Project, HUM has been investigating methods to ensure that local employment is placed firmly in the centre of all community development activities. Contact has been made with South African based M&M Initiatives to investigate non-verbal, psycho-metric testing for screening of potential workers.

Each of these projects is discussed further below.

8.5.1 Brick Making

The brickmaking project was piloted by GF, however HUM review has found a number of shortcomings:

- Poor quality of bricks made using laterite.
- Lack of agreement on pricing of bricks.
- Reluctance of local communities to truly own this project.

There are locally made bricks that are of better quality, available through established channels. As such HUM will not be supporting the GF project, and will rather ensure that local procurement of building materials is maximised wherever possible.

8.5.2 Rice Irrigation

The rice irrigation project has been proposed as a central pillar of alternative livelihoods development for those displaced from ASM activities in the Project area, especially since it has the ability to provide alternative and sustainable employment to large numbers of people. Rice irrigation holds great potential given the water resources available from the Sankarani River, as well as the local population's historical experience of rice production.

Figure 8.11 Rice Irrigation Project Near Yanfolila Undertaken by Helvetas, October 2014

HUM has begun making initial investigations into this Project, primarily though visits to other irrigation projects within the region. Based on the advice of RePlan, and HUM observations of other development projects, the first stage in implementing any feasibility study for this project will be extensive stakeholder engagement and helping to set up community led institutions to ensure such a project is truly owned by the intended beneficiaries. Steps have been made with recent stakeholder engagement work. This will continue into 2015.

8.5.3 Boreholes

GF / Glencar started a project of drilling boreholes and installing handpumps across the Project area in June 2013. A total of seven including Gendarmerie and Mayor's Office boreholes were set to be drilled. However only two in Gendarmerie and Mayor's Office were drilled and the Project was stopped in July 2013 due to concerns over baseline As levels in groundwater and budgetary constraints.

HUM has investigated the As issue, and latest monitoring results do indeed show elevated As levels in community groundwater (0.010 – 0.018mg/l) in Soloba (main village of the Commune and seat of the Mayor). Alternative plans were made to drill the remaining two boreholes elsewhere namely in Bougoudale and Kenieba.

However this Project is currently on hold due to financial constraints.

8.5.4 Psycho-Metric Testing

HUM has been in discussion with M&M Initiatives (South Africa) to pilot and then hopefully implement two types of test designed to assess competency and potential of individuals for future employment. Details of each test are given below and in Appendix 7.

- Learning Potential Computerised Adaptive Test (LPCAT)
    - Developed specifically for South African conditions to address challenge of assessment in multicultural and multilingual societies.
    - Used to identify present and potential future level of reasoning ability.
    - Does not depend on language proficiency or prior learning opportunities – makes use of non-verbal figural pattern solution.
    - Generic reasoning skills involved in solving these questions are for example identification, comparison, patterns, and relations - elements that can be extrapolated in various applied fields.
- *Career Preference Test (CPT)*
    - Measures career-related interest with a focus on career related fields, activities, and environments.
    - Allows for not only a measure of the level of interest in each of the sub-dimensions, but also for the relative level of interest across pertinent dimensions.
    - Following two rounds of questions posed from all 34 the sub-dimensions, the top 6 field, top 6 activity, and top 4 environment sub-dimensions are then combined to provide a final profile.
    - This profile indicates the relative ratings across the broad dimensions providing unique and valuable information for guiding individuals towards career-related decision making.

An alternative to M&M Initiatives is currently being sought for comparative purposes[22].

8.6 Input into Design and Contractor Management

8.6.1 ESIA, ESMP, and Conditions of ESIA Approval and Commitments Register Derived from These

The Commitments Register has been developed to enable the commitments made in the ESIA and the ESMP which has been approved, and the conditions of approval. The Register will be updated as required as mitigation measures are identified during the additional and ongoing studies (e.g. air quality, noise, ARD, biodiversity, ecosystem services, etc). The Register is

[22] http://www.leadthefield.co.za/index.php/lab/introduction-to-lab.html

used to inform responsibilities, budgeting, and design of the Project, and will continue to be updated as necessary throughout life of mine with progress to date.

8.6.2 Environmental Design Criteria Document

An environmental design criteria document was compiled based on national and international standards as well as available site information from baseline studies. The data may be updated as more site specific baseline information becomes available from ongoing monitoring, particularly with relation to water quality, air quality, and noise. This document has been made available to the Project manager for use by engineers during detailed design, and by contractors, as it sets out the minimum environmental standards that the Project must achieve through design or performance. See Appendix 7.

8.6.3 Tender Process Questionnaire

A questionnaire was developed to enable an assessment of potential contractor's awareness and understanding of SHEC issues, as well as performance to date, particularly with regard to Health and Safety metrics and environmental site management. The responses can be used in the adjudication process alongside other technical and economic factors. See Appendix 7.

8.6.4 Contract Clauses

Generic example SHEC contract clauses have been provided to the Project Managers for insertion into contracts to ensure that contractors are aware of their responsibilities with regard to SHEC performance and the implementation of design elements designed to safeguard the environment and our proximate communities. See Appendix 7.

8.6.5 Staffing of SHEC Department

The Yanfolila SHEC department currently consists of the Yanfolila SHEC Manager, who oversees a Community Liaison Officer, and Environment Officer, and their assistants. This will be augmented once finance is received. Expected new hires include a Health and Safety Manager, Health and Safety Officer, and Community Relations / Livelihoods Office.

8.6.6 Environmental Management System and a Suite of Plans as Follows

An ESMS is currently being developed for the Project that is in line with ISO14001. This includes a suite of management plans given below.

Table 8.12 List of Management Plans

Management Plan	HUM Code	IFC
ESMS		**PS1**
Footprint Management Plan	YNF-SHEC-MP-001	PS1/3
Stakeholder Engagement	YNF-SHEC-MP-002	PS1
Emergency Preparedness and Repsonse	YNF-SHEC-MP-003	PS1
Conceptual Closure	YNF-SHEC-MP-004	PS1
ASM/Livelihood Restoration	YNF-SHEC-MP-005	PS1
Human Resources	YNF-SHEC-MP-006	PS2
Occupational Health and Safety	YNF-SHEC-MP-007	PS2
Air quality	YNF-SHEC-MP-008	PS3
Noise, Vibration and Blasting	YNF-SHEC-MP-009	PS3
Water resources/management	YNF-SHEC-MP-010	PS3
Waste	YNF-SHEC-MP-011	PS3
Hazardous Materials	YNF-SHEC-MP-012	PS3
Mine Waste	YNF-SHEC-MP-013	PS3
Transport Management Plan	YNF-SHEC-MP-014	PS3
Cyanide	YNF-SHEC-MP-015	PS3
Community Health, Safety, Security	YNF-SHEC-MP-016	PS4
Local Sustainable Development Plan	YNF-SHEC-MP-017	PS5
RAP /LACP	YNF-SHEC-MP-018	PS5
Biodiversity	YNF-SHEC-MP-019	PS6
Cultural Heritage	YNF-SHEC-MP-020	PS8

The draft framework ESMS is available at Appendix 7.

8.7 Conclusion

This short report has aimed to give an overview of work completed to date on the Yanfolila Project as well as work completed or currently on-going by HUM. It started with an overview of environmental and social work undertaken in the Scoping, ESIA and De-Risking Study Stage. It then described the internal GAP analysis undertaken by HUM upon acquisition of the Project. Finally it moved onto describe the work currently on-going to fill these gaps and manage environmental and social risks for the Project.

Work remaining from the current programme includes:

- Air quality impact assessment
- Development of the Yanfolila ESMS

9.0 INITIAL CAPITAL, OPERATING, SUSTAINING CAPITAL, AND WORKING CAPITAL COSTS

9.1 Initial Capital Cost Estimate

9.1.1 Summary

The capital cost estimate prepared for this Study assumes a greenfield gold project capable of processing a nominal 1 Mtpa of mineralized material.

The key objectives of the capital cost estimate are to:

- Support the economic evaluation of the Project;
- Support the identification and assessment of the processes and facilities that will provide the most favorable return on investment; and
- Provide guidance and direction for project financing and execution.

The total estimated initial cost to design, procure, construct, and commission the facilities described in this Study is $71.6 million. Table 9.1 summarizes the initial capital costs by major area.

9.1.2 Exclusions and Clarifications

The estimate is expressed in first quarter 2015 United States dollars and the following items are not included in the capital estimate:

- Sunk costs that are expected to be incurred prior to completion of a positive Study;
- Reclamation costs, which are included in the financial analysis;
- Working capital and sustaining capital are not included in the financial analysis; but which are both discussed in this section of the report;
- Interest and financing costs;
- Escalation beyond first quarter 2015; and
- Risk due to political upheaval, government policy changes, labor disputes, permitting delays, weather delays, or any other force majeure occurrences.

Table 9.1 Initial Capital Costs

Description		Estimate (000's)	Subtotal (000's)	Total (000's)
Direct Costs	General & Site Development	2,820		30,324
	Process	8,404		
	Tailings Management	491		
	Gold Recovery and Refining	1,293		
	Reagents	410		
	Services	2,013		
	Plant Infrastructure	220		
	Reagents Infrastructure	339		
	Laboratory Infrastructure	271		
	Other Infrastructure	1,846		
	Field Construction	10,567		
	Adjustments	1,650		
Contracted Indirect Costs	EPCM Services	6,737		10,174
	Comm Spares	69		
	Testwork Consultants	91		
	Vendor Representatives	473		
	First Fills	736		
	Freight	2,067		
Owner's Direct Costs	Preproduction Mine Development	7,749	18,073	25,751
	Mobile Equip & Lt Vehicles	439		
	Owners Camp Expansion	951		
	Office/Engineering Equipment, Software, Furniture	172		
	Employee Transporation	60		
	Warehouse Equipment	50		
	Communications	20		
	Airstrip	416		
	Access Roads	1,234		
	Mine Office	225		
	Laboratory (Upfront/Mob)	344		
	Warehouse/Vehicle Repair Shop	120		
	Security & Safety	49		
	Power Plant	590		
	Tailings Storage Facility	5,351		
	Waste Disposal Facility	44		
	Landfill	25		
	Environmental Control Dams	234		
Owner's Indirect Costs	Preproduction Employment & Training	1,943	7,678	
	Project & Construction Management	1,559		
	Operations Catering	810		
	Preproduction Power	541		
	Social Investment	191		
	Land Access & Compensation	456		
	Corporate Travel & Services	1,120		
	Environmental Monitoring	141		
	Security	303		
	Legal, Permits, & Fees	30		
	Communications Expenses	84		
	Insurance	400		
	Consultants	100		
Other	Contingency-Senet	3,413		5,326
	Contingency-Owner's Costs	1,914		
Total Initial Capital Cost (000's)		**$71,575**		**$71,575**

The currency exchange rates shown below in Table 9.2, effective December 31,2014, were utilized in completing the cost estimates throughout this section.

Table 9.2 Exchange Rates

From Currency	To Currency	Multiply By
USD	ZAR	11.6017
USD	AUD	1.2258
GBP	USD	1.5532
EUR	USD	1.2155
EUR	CFA	655.957

The estimate is based on the assumption that new equipment and materials will be purchased on a competitive basis and installation contracts will be awarded in defined packages for lump sum or unit rate contracts. Various sources for pricing were used, including commerical quotations, in-house historical data, published databases, factors, and estimators' judgment.

9.1.3 Contingency

A contingency of $5.3 million has been included in the initial capital cost. This contingency is based on the level of definition that was used to prepare the estimate.

Contingency is an allowance to cover unforeseeable costs that may arise during the project execution, which reside within the scope-of-work but cannot be explicitly defined or described at the time of the estimate, due to lack of information. However, it does not cover scope changes or project exclusions. For the purposes of the financial analysis, it is assumed that the contingency will be spent.

9.1.4 Accuracy

The capital cost estimate included in this Study has been developed to a level sufficient to assess/evaluate the project concept, various development options, and the potential overall project viability. After inclusion of the recommended contingency, the capital cost estimate is considered to have a level of accuracy in the range of minus10 percent plus 15 percent. This is based on the level of contingency applied, the confidence level of SENET, Ausenco, BWF, HUM, and MTB on the estimate accuracy, and an assessment comparing the Study estimate to standard accuracy levels on similar estimates.

9.2 Operating Cost Estimate

9.2.1 Summary

Operating costs have been estimated by main project areas, mining, processing, and general and administration (G&A). Table 9.3 shows the average LOM unit operating costs on a per tonne of ore feed to the process plant.

Initially (approximately the first two years of operation) the plant will be fed nearly 100% oxide material. Thereafter, the plant will be fed a blend of oxide and fresh material, culminating in a blend of 50% maximum fresh material.

Table 9.3 LOM Unit Operating Costs (Oxide)

Area	Cost/Tonne Mined (USD)	Cost/Tonne Ore (USD)
Mining	2.59	27.46
Process		14.02
G&A (Yr 1)		6.99
Total Operating Cost	**$ 2.59**	**$ 48.47**

Table 9.4 LOM Unit Operating Costs (Fresh)

Area	Cost/Tonne Mined (USD)	Cost/Tonne Ore (USD)
Mining	2.59	27.46
Process		18.89
G&A (Yr 1)		6.43
Total Operating Cost	**$ 2.59**	**$ 52.78**

Labor components for all three operating cost areas were developend from detailed organization charts and burdened all-inclusive compensation schedules complied by HUM's Malian HR specialist.

Contract mining rates were taken from two rounds of competitive tendering for the prospective mining contract.

Electrical power and diesel fuel costs were obtained from several rounds of recent competitive tendering and clarifications for the long term supply and operation of an onsite diesel generator power station and fuel stroage and dispensing depot.

Process consumables pricing was obtained from recent quotations by leading West African suppliers, having been provided consumable specifications and estimated consumption. Transport and handling costs to site have been included.

9.2.2 Mine Operating Costs

Mine operating costs consist of three components, as shown below in Table 9.5.

Table 9.5 Detailed Mine Operating Costs

Area	Cost/Tonne Mined (USD)	Cost/Tonne Ore (USD)
Contractor Fixed Fee	1.01	10.71
Contractor Variable Fee	1.43	15.16
Owner's Tech Support & Supervision	0.15	1.59
Total Operating Cost	**$ 2.59**	**$ 27.46**

9.2.3 Process Operating Costs

Detailed plant operating costs for oxide material are shown below in Table 9.6. Consumptions for consumables were determined from metallurgical testing. Electrical power consumption was calculated from a detailed electrical load list, using actual connected loads for selected equipment, and applying appropriate load, utilization, and availability factors to obtain total kWh per annum.

Table 9.6 Detailed Process Operating Costs for Oxide Materials

Opex Parameter	Units	***Value***
		Oxides
Tonnage Processed	tpa	1,000,000.00
Labour	USD/t	1.94
Power	USD/t	4.77
Consumables	USD/t	6.25
Maintenance Material	USD/t	0.39
Assay Costs	USD/t	0.67
Combined Plant Costs	**USD/t**	**$ 14.02**

Similar detailed process operating costs for fresh (sulphide) material are shown below in Table 9.7.

Table 9.7 Detailed Process Operating Costs for Fresh Materials

Opex Parameter	Units	*Value*
Tonnage Processed	tpa	1,000,000.00
Labour	USD/t	1.94
Power	USD/t	7.49
Consumables	USD/t	8.24
Maintenance Material	USD/t	0.55
Assay Costs	USD/t	0.67
Combined Plant Costs	**USD/t**	**$ 18.89**

9.2.4 G&A Operating Costs

As mentioned in the introductory discussion of operating costs in general, labor costs for G&A were developed from organization charts and a project specific burdened, all-inclusive compensation schedule.

Camp catering and maintenance costs were developed using detailed estimates of camp residents requiring daily accommodations and meals, and workers residing in local communities requiring mid-shift meals. Rates for full daily camp services and mid-shift (chop) meals were taken from the current camp caterer's (ATS) contract.

Environmental monitoring, social investment, and land access/compensation costs were developed by HUM in conjunction with outside specialty consultants in these areas. Estimates for land access/compensation and environmental monitoring were calculated from measured areas and ongoing existing monitoring, respectively.

Most of the other categories of overhead related costs were provided by HUM based on existing budgets and historical spending experience for Bamako, London, and Mali in general.

G&A costs are presented below in Tables 9.8 and 9.9. Years 2-7 differ from Year 1 only in the areas of environmental monitoring and land access/compensation.

Table 9.8 G&A Operating Costs – Year 1

General and Administration	Annual Cost (USD)
Personnel	1,656,734
Vehicle & Mobile Equipment Opex & Maintenance	226,570
Communication	90,000
Camp Operating & Catering	981,952
Health & Safety Supplies	14,715
Training	50,000
Insurance	300,000
Corporate Travel & Services	1,120,240
Environmental Monitoring	571,320
Security	818,780
Social Investment	376,000
Consultants	129,900
Computer Equipment/Software	33,595
Legal, Permits & Fees (Allowance)	20,000
Land Access & Compensation	384,200
Employee Transportation	213,756
G&A Total	**6,987,762**

Table 9.9 G&A Operating Costs – Years 2 - 7

General and Administration	Annual Cost (USD)
Personnel	1,656,734
Vehicle & Mobile Equipment Opex & Maintenance	226,570
Communication	90,000
Camp Operating & Catering	981,952
Health & Safety Supplies	14,715
Training	50,000
Insurance	300,000
Corporate Travel & Services	1,120,240
Environmental Monitoring	91,320
Security	818,780
Social Investment	376,000
Consultants	69,900
Computer Equipment/Software	33,595
Legal, Permits & Fees (Allowance)	20,000
Land Access & Compensation	361,600
Employee Transportation	213,756
G&A Total	**6,425,162**

9.3 Sustaining Capital Cost

The major contributors to sustaining capital (stay in business) cost are listed below in decreasing order of amount.

- Tailings Storage Facility expansions
- Haul road construction
- Installation of dewatering wells
- Light vehicle replacement near end of useful life
- Office and engineering equipment replacement

Though not a stay in business cost, an expansion capital cost for installing a hard rock crushing facility to process fresh material has been included in Year 2 of the below table.

Table 9.10 Sustaining Capital Cost Estimate

Sustaining Capital Estimate								
Description	**Year 1**	**Year 2**	**Year 3**	**Year 4**	**Year 5**	**Year 6**	**Year 7**	**TOTAL**
Mining								
Mine Planning Software	50,000							50,000
Geology Software								-
Whittle License								-
GIS/CAD Software								-
Computers, plotters, printers				16,200	16,200	21,600		54,000
Surveying Equipment					25,000			25,000
Miscellaneous	12,500	10,000	10,000	10,000	10,000	10,000	5,000	67,500
KW Dewatering Well Install								-
KW Dewatering Well Opex								-
KE Dewatering Well Install	382,904							382,904
GW Dewatering Well Install		255,269						255,269
TSF Overhaul Yr 2		39,028						39,028
TSF Overhaul Yr 4				34,149				34,149
TSF Overhaul Yr 6						24,392		24,392
SW/SE haul Road		1,118,400	1,118,400					2,236,800
Contractor Demob							264,970	264,970
								-
Subtotal Mining	**445,404**	**1,422,697**	**1,128,400**	**60,349**	**51,200**	**55,992**	**269,970**	**3,434,012**
Warehouse Equipment			10,000					10,000
Environmental Control Dams		116,994	116,994	58,497	58,497			350,982
Tailings Storage Facility	740,694	898,416	920,216	941,549	1,030,300	1,055,751		5,586,926
Landfill			24,970					24,970
Pickups		315,000	315,000	270,000				900,000
Office Equipment, Computers, Software, Furniture			100,000	85,000	52,000			237,000
Expansion Capital-Hard Rock Crusher Allowance		2,000,000						2,000,000
								-
Subtotal	**740,694**	**3,330,410**	**1,487,180**	**1,355,046**	**1,140,797**	**1,055,751**	**-**	**9,109,878**
Total Sustaining Capital	**1,186,098**	**4,753,107**	**2,615,580**	**1,415,395**	**1,191,997**	**1,111,743**	**269,970**	**12,543,890**

9.4 Working Capital

Working capital for the Project consists of the maximum cash required to pay for operating expenses until revenue exceeds costs. The Project's revenue in Year 1 totals $108,247,000 based on the initial financial model. The production ramp up schedule defines the early revenue generated based on the mine and mill production schedules, in which 50% production occurs in Month 1, 75% in Month 2, and 85% in Month 3 followed by 100% production the balance of the year.

Project costs consist of mining costs at $28.08 per tonne of ore, process operating costs of $14.14/tonne milled, and G&A costs of $8,492,000 per annum. To calculate weekly cash flow, mining and operating costs were assigned weekly payment starting in Week 2 of production, and G&A costs were assumed to be paid monthly starting in Week 5.

The basis of Project revenue consists of planned doré shipments commencing in Week 4 after gold inventory buildup in the carbon in leach circuit supports regular production. Metalor supplied their refinery terms and conditions which provided cost and payment terms for doré product used in working capital calculations. Based on the production schedule of 1 Mtpa at 2.65 g/t and a 94% recovery, 26 shipments per year will each contain approximately 96 kg (3,081 t-oz) of gold doré product and revenue is assumed to be $1,250/t-oz, 85% payable one week after shipment, the balance two weeks later once the final assays confirm the gold content of the shipment.

To calculate working capital, analysts compared cumulative costs with cumulative revenue to determine the maximum cash outlay for the Project prior to the time cumulative revenue exceeds cumulative expenses, which occurs in Week 4 for the Project. The cost at that time is the working capital: $1,218,822, which is included in the financial model in the first month, and recaptured at the end of the project to reflect the capture of initial gold inventory in the CIL circuit and the need to no longer replenish inventories of consumables.

10.0 ECONOMIC ANALYSIS

10.1 General Criteria

Endeavour Financial (Endeavour) developed an economic model for the Project based on the following main inputs shown in Table 10.1 below.

Table 10.1 Economic Model Inputs

Description	Values
Construction Period	12 months
Life of Mine (LOM)	6.5
LOM Ore (tonnes)	6,414,000
LOM Processing Plant Feed Grade	2.65 g/tAu
LOM Gold Production (kozs)	514
Average Gold Recovery	94.00%
Average Annual Gold Production (ozs)	73,373
Gold Price	$1,250/oz (flat)
Inflation/Currency Fluctuation	None
Leverage	100% Equity
Income Tax	35% min.; 0.75% of Revenues
Carry Forward Tax Losses	3 years
Withholding Tax	15%
Stamp Duties on Mineral Product Export	0.60%
Depreciation	Straight line
Value Added Tax (VAT)	15%
VAT Payment/Recovery	Included (to commence 3 years after start of production)
Government Royalty ("Special tax on Certain Products")	3%
LPMDO Royalty	1%
Transportation and Refining Charges	$4.807/oz

10.2 Production Summary

At the foundation of the economic model, data was drawn from the mine production and process production schedules, which were produced by MTB and are summarized in Table 10.2.

Table 10.2 Process Production Schedule

Year	Total Ore Processed	Total Gold Recovered
	K-Tonnes	k-ounces
1	923	87
2	1,000	87
3	995	71
4	998	84
5	1,002	79
6	1,003	74
7	493	32
LOM	**6,414**	**514**

10.3 Gross Income from Mining

A flat gold price of $1,250/oz has been used in the economic model.

10.4 Transportation

MTB provided a quote of $4.807/oz for transportation, insurance, and refining of the gold doré produced at Yanfolila.

10.5 Royalties

The royalty imposed by the Malian government named “Special tax on Certain Products” is calculated as three percent of gross revenue.

In addition, a net smelter royalty of one percent is paid to La Petite Mine d’Or (LPMDO). However the royalty shall only accrue in respect of any financial year when the Yanfolila Project shall make a profit. The royalty shall be paid to LPMDO in two tranches. Fifty percent of the estimated amount due to LPMDO is to be paid within 30 days of the financial year end while the remaining amount is to be paid within six months of the financial year end.

10.6 Operating Costs

Operating cost estimates were provided by MTB and served as input to the economic model. Over the mine life, mining cost average $2.53 per tonne mined and processing cost average $15.61 per tonne processed while G&A cost average $6.5M per annum.

10.7 Depreciation and Income Tax

Gross income from mining is reduced by total annual operating costs, leaving net profit before depreciation. In the absence of a tax review it was decided that most capex items will allow for an annual depreciation of 8, 10, or 12 years. For corresponding calculation of depreciation charges it was assumed that one third of capex items will allow for depreciation over 8 years, one third of capex items will allow for depreciation over 10 years and one third of capex items will allow for depreciation over 12 years.

Other deductions for income tax calculations include a recovery of historic spend of $75M over 4.5 years after commencement of production as well as interest charges, if any.

After deduction of depreciation, recovery of historic cost, carry forward losses, income tax was calculated on net profit before taxes using the rate of 35% subject to a minimum income tax payment of 0.75% of gross revenue. Tax losses can be carried forward for a duration of 3 years after which they expire. It was assumed that income tax payments are made on an annual basis in the first quarter immediately following the close of the financial year which is based on the calendar year.

Because depreciation is a non-cash expense, it is added back after determination of income tax liability for purposes of the cash flow estimate.

10.8 Initial Capital Costs

An Initial capital cost estimate of $71.6M (excluding working capital, financing charges, preproduction interest and taxes) was provided by MTB and served as input to the economic model.

10.9 Sustaining Capital Costs

Sustaining capital cost estimates over the life of the project were provided by MTB and served as input to the economic model. Sustaining capital cost average $1.9M per annum over the life of the Project.

10.10 Working Capital

A working capital requirement of $1.2M was calculated by MTB and served as input to the economic model.

10.11 Base Case Analysis

The results of this optimization study show payback to occur early in the mine life, approximately 2.6 years after start of production. The base case financial model was developed from information described in this section. Based upon this information, the Yanfolila Project is estimated to have an after-tax IRR of 35.1%. Assuming a discount rate of eight percent over an estimated mine life of 6.5 years, the after-tax NPV is estimated to be $72.4M. Base-case NPV's at various discount rates are presented in the Table 10.3.

Table 10.3 NPV At Various Discount Rates

Discount Rate	4%	6%	8%	10%	12%
NPV ($M)	$93.9	$82.6	$72.4	$63.3	$55.0

10.12 Base Case Sensitivity Analysis

Table 10.4 reflects the sensitivities for IRR and NPV in 5% increments of negative and positive deviation from the base case for gold price, operating cost, and initial capital costs.

Table 10.4 Sensitivity Analysis of IRR and NPV

Base Case Variance	-20%	-15%	-10%	-5%	Base	+5%	+10%	+15%	+20%
Gold Price $/oz	1,000	1,063	1,125	1,188	1,250	1,313	1,375	1,438	1,500
NPV 8% ($M)	10.4	26.3	42.1	56.8	72.4	87.6	102.9	117.9	132.9
IRR	11.90%	18.00%	23.90%	29.40%	35.10%	40.60%	46.00%	51.30%	56.60%
LOM Opex ($/t ore)	41	43.6	46.2	48.7	51.3	53.9	56.4	59	61.6
NPV 8% ($M)	106.6	98.2	89.8	80.9	72.4	63.7	54.9	46.7	37.8
IRR	47.10%	44.20%	41.20%	38.10%	35.10%	32.00%	28.70%	25.70%	22.30%
Capex ($M)	57.3	60.8	64.4	68	71.6	75.2	78.7	82.3	85.9
NPV 8% ($M)	83.6	80.8	78	75.2	72.4	69.6	66.8	64	61.1
IRR	44.70%	41.90%	39.50%	37.20%	35.10%	33.10%	31.30%	29.60%	28.00%

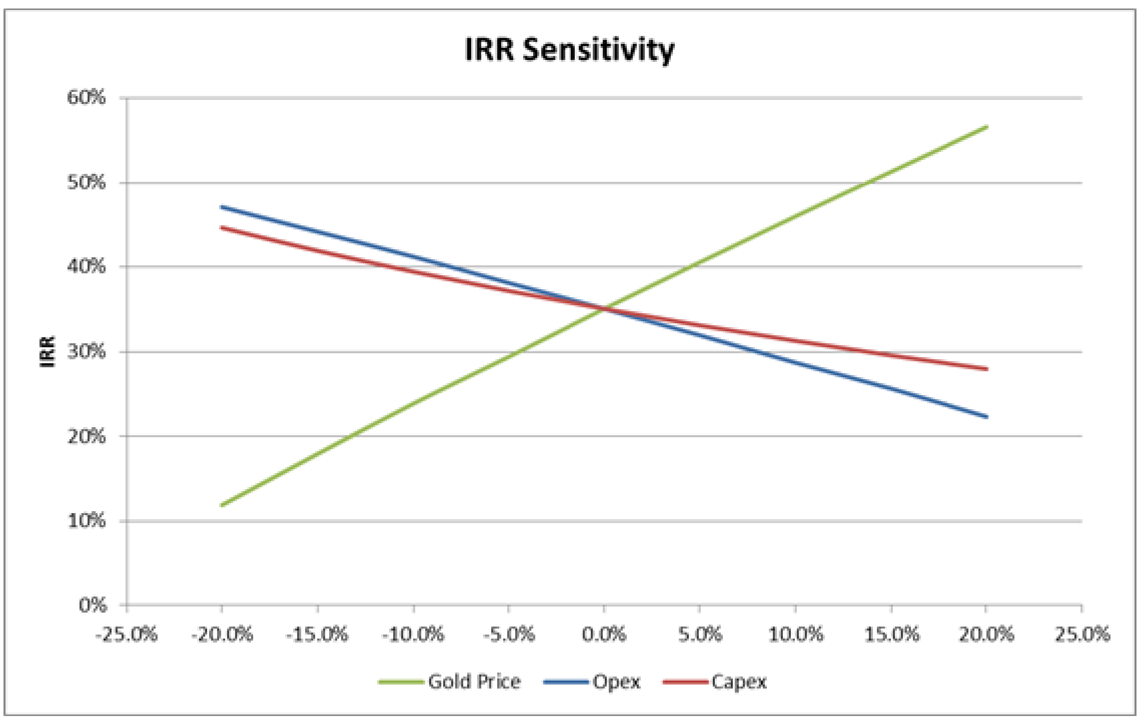


Figure 10.1 IRR Sensitivity

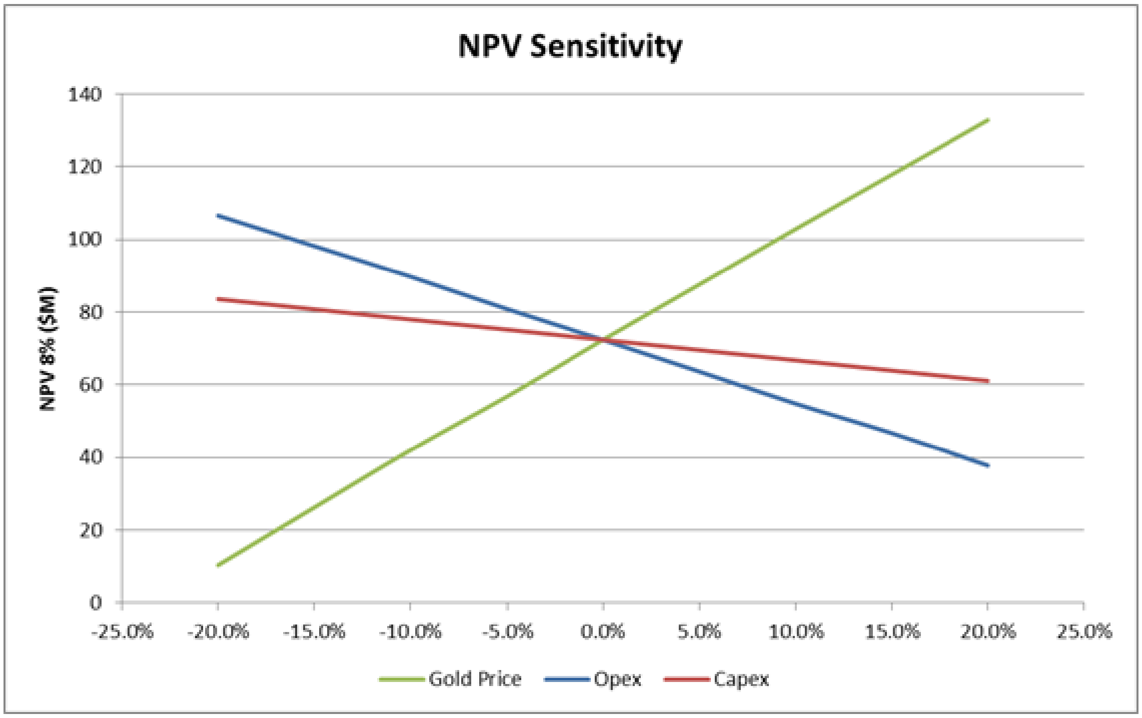


Figure 10.2 NPV Sensitivity

10.13 Economic Model

The cash flow model is illustrated in the Table10.5.

Table 10.5 Cash Flow Model

Yanfolila Gold Project - Cash Flow

Project Year			-1	-1	-1	-1	-1	-1	-1	-1	-1	-1	-1	-1	1
Calendar Year			2015	2015	2015	2015	2015	2015	2015	2015	2015	2015	2015	2015	2016
Period			1	2	3	4	5	6	7	8	9	10	11	12	1
Months in Period			1	1	1	1	1	1	1	1	1	1	1	1	1
Period Start Date	01-Jan-15		01-Jan-15	01-Feb-15	01-Mar-15	01-Apr-15	01-May-15	01-Jun-15	01-Jul-15	01-Aug-15	01-Sep-15	01-Oct-15	01-Nov-15	01-Dec-15	01-Jan-16
Period End Date			31-Jan-15	28-Feb-15	31-Mar-15	30-Apr-15	31-May-15	30-Jun-15	31-Jul-15	31-Aug-15	30-Sep-15	31-Oct-15	30-Nov-15	31-Dec-15	31-Jan-16
Construction Period	01-Jan-15	31-Mar-16	1	1	1	1	1	1	1	1	1	1	1	1	1
Operations Period	01-Apr-16	30-Sep-22	0	0	0	0	0	0	0	0	0	0	0	0	0
Mining															
Waste Mined	t	67,456,000	0	0	0	0	0	0	0	0	0	0	0	0	0
Ore Mined	t	6,298,000	0	0	0	0	0	0	0	0	0	0	0	0	0
Total Material Mined	t	73,754,000	0	0	0	0	0	0	0	0	0	0	0	0	0
Strip Ratio	w:o	10.71	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Au Grade Mined	g/t	2.64	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Au Cont. Mined	oz	534,540	0	0	0	0	0	0	0	0	0	0	0	0	0
Processing															
Ore Milled	t	6,414,000	0	0	0	0	0	0	0	0	0	0	0	0	0
Au Grade	g/t	2.65	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Au Cont.	oz	546,530	0	0	0	0	0	0	0	0	0	0	0	0	0
Recovery	%	93.98%	0	0	0	0	0	0	0	0	0	0	0	0	0
Au Recovered	oz	513,610	0	0	0	0	0	0	0	0	0	0	0	0	0
Price Assumptions															
SpotGold Price	US$/oz	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250
Income Calculation															
Gross Revenue	US$k	642,013	0	0	0	0	0	0	0	0	0	0	0	0	0
Transport & Insurance Costs	US$k	2,212	0	0	0	0	0	0	0	0	0	0	0	0	0
Refining Costs	US$k	257	0	0	0	0	0	0	0	0	0	0	0	0	0
Net Revenue	US$k	639,544	0	0	0	0	0	0	0	0	0	0	0	0	0
Mine Operating Cost	US$k	186,574	0	0	0	0	0	0	0	0	0	0	0	0	0
Processing Operating Cost	US$k	100,133	0	0	0	0	0	0	0	0	0	0	0	0	0
G&A Costs	US$k	42,327	0	0	0	0	0	0	0	0	0	0	0	0	0
Management Fee	US$k	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Rehabilitation	US$k	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total Operating Costs	US$k	329,034	0	0	0	0	0	0	0	0	0	0	0	0	0
Total Cash Cost per Ounce Recovered	US$/oz	641	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
Operating Profit	US$k	310,510	0	0	0	0	0	0	0	0	0	0	0	0	0
Government Royalty	US$k	23,112	0	0	0	0	0	0	0	0	0	0	0	0	0
LPMDO Royalty	US$k	5,532	0	0	0	0	0	0	0	0	0	0	0	0	0
Taurus Royalty	US$k	6,420	0	0	0	0	0	0	0	0	0	0	0	0	0
EBITDA	US$k	275,445	0	0	0	0	0	0	0	0	0	0	0	0	0
Interest Expense	US$k	13,782	0	0	0	0	0	60	0	0	0	0	0	1,408	0
Taxes	US$k	58,184	0	0	0	0	0	9	0	0	0	0	0	211	0
Depreciation	US$k	52,698	0	0	0	0	0	0	0	0	0	0	0	0	0
Recovery Historic Cost	US$k	75,000	0	0	0	0	0	0	0	0	0	0	0	0	0
Net Profit/Loss	US$k	75,781	0	0	0	0	0	-69	0	0	0	0	0	-1,619	0
Cashflow Calculation															
Net Profit	US$k	75,781	0	0	0	0	0	-69	0	0	0	0	0	-1,619	0
Add Interest, Depreciation and Recovered Cost	US$k	141,479	0	0	0	0	0	60	0	0	0	0	0	1,408	0
Change in Working Capital Account	US$k	2,676	0	0	0	0	0	0	0	0	0	0	0	0	0
Change in VAT Account	US$k	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Pre-production Capex	US$k	-86,479	0	0	0	-14,077	-3,436	-7,978	-3,937	-7,014	-10,407	-10,879	-6,513	-6,100	-4,295
Sustaining Capex	US$k	-12,544	0	0	0	0	0	0	0	0	0	0	0	0	0
Cashflow before Financing	US$k	120,914	0	0	0	-14,077	-3,436	-7,987	-3,937	-7,014	-10,407	-10,879	-6,513	-6,311	-4,295

Cash Flow Model (continued)

Yanfolila Gold Project - Cash Flow

Project Year			1	1	1	1	1	2	2	2	2	3	3	3	3	4
Calendar Year			2016	2016	2016	2016	2016	2017	2017	2017	2017	2018	2018	2018	2018	2019
Period			2	3	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1
Months in Period			1	1	3	3	3	3	3	3	3	3	3	3	3	3
Period Start Date	01-Jan-15		01-Feb-16	01-Mar-16	01-Apr-16	01-Jul-16	01-Oct-16	01-Jan-17	01-Apr-17	01-Jul-17	01-Oct-17	01-Jan-18	01-Apr-18	01-Jul-18	01-Oct-18	01-Jan-19
Period End Date			29-Feb-16	31-Mar-16	30-Jun-16	30-Sep-16	31-Dec-16	31-Mar-17	30-Jun-17	30-Sep-17	31-Dec-17	31-Mar-18	30-Jun-18	30-Sep-18	31-Dec-18	31-Mar-19
Construction Period	01-Jan-15	31-Mar-16	1	1	0	0	0	0	0	0	0	0	0	0	0	0
Operations Period	01-Apr-16	30-Sep-22	0	0	1	1	1	1	1	1	1	1	1	1	1	1
Mining																
Waste Mined	t	67,456,000	0	0	2,640,000	3,120,000	3,088,000	3,117,000	3,091,000	3,426,000	3,637,000	3,593,000	3,441,000	3,591,000	3,615,000	3,628,000
Ore Mined	t	6,298,000	0	0	175,000	199,000	250,000	249,000	250,000	284,000	179,000	250,000	229,000	239,000	248,000	250,000
Total Material Mined	t	73,754,000	0	0	2,815,000	3,319,000	3,338,000	3,366,000	3,341,000	3,710,000	3,816,000	3,843,000	3,670,000	3,830,000	3,863,000	3,878,000
Strip Ratio	w:o	10.71	0.00	0.00	15.09	15.68	12.35	12.52	12.36	12.06	20.32	14.37	15.03	15.03	14.58	14.51
Au Grade Mined	g/t	2.64	0.00	0.00	2.77	3.01	2.98	3.55	3.61	3.12	2.26	2.14	2.30	2.22	2.54	2.32
Au Cont. Mined	oz	534,540	0	0	15,560	19,240	23,980	28,410	29,050	28,500	12,980	17,220	16,940	17,070	20,240	18,670
Processing																
Ore Milled	t	6,414,000	0	0	175,000	249,000	250,000	249,000	250,000	250,000	250,000	250,000	247,000	250,000	248,000	250,000
Au Grade	g/t	2.65	0.00	0.00	2.77	3.05	2.98	3.55	3.61	3.31	2.38	2.14	2.33	2.24	2.54	2.32
Au Cont.	oz	546,530	0	0	15,560	24,400	23,980	28,410	29,040	26,590	19,160	17,220	18,510	18,020	20,240	18,680
Recovery	%	93.98%	0	0	1	1	1	1	1	1	1	1	1	1	1	1
Au Recovered	oz	513,610	0	0	14,630	22,980	22,590	26,850	27,470	25,100	17,950	16,100	17,340	16,870	19,010	17,500
Price Assumptions																
SpotGold Price	US$/oz	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250
Income Calculation																
Gross Revenue	US$k	642,013	0	0	18,288	28,725	28,238	33,563	34,338	31,375	22,438	20,125	21,675	21,088	23,763	21,875
Transport & Insurance Costs	US$k	2,212	0	0	63	99	97	116	118	108	77	69	75	73	82	75
Refining Costs	US$k	257	0	0	7	11	11	13	14	13	9	8	9	8	10	9
Net Revenue	US$k	639,544	0	0	18,217	28,615	28,129	33,433	34,205	31,254	22,351	20,048	21,592	21,006	23,671	21,791
Mine Operating Cost	US$k	186,574	0	0	6,285	7,427	7,436	7,502	7,468	8,337	8,438	9,320	8,773	8,697	8,800	9,471
Processing Operating Cost	US$k	100,133	0	0	2,458	3,523	3,510	3,523	3,634	3,662	3,805	3,663	4,021	4,094	4,086	4,053
G&A Costs	US$k	42,327	0	0	1,747	1,747	1,747	1,747	1,606	1,606	1,606	1,606	1,606	1,606	1,606	1,606
Management Fee	US$k	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Rehabilitation	US$k	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total Operating Costs	US$k	329,034	0	0	10,490	12,696	12,692	12,771	12,708	13,605	13,848	14,589	14,399	14,397	14,492	15,130
Total Cash Cost per Ounce Recovered	US$/oz	641	n/a	n/a	717	552	562	476	463	542	771	906	830	853	762	865
Operating Profit	US$k	310,510	0	0	7,727	15,918	15,437	20,662	21,498	17,649	8,503	5,458	7,192	6,609	9,180	6,661
Government Royalty	US$k	23,112	0	0	658	1,034	1,017	1,208	1,236	1,130	808	725	780	759	855	788
LPMDO Royalty	US$k	5,532	0	0	0	0	0	375	375	0	0	606	606	0	0	0
Taurus Royalty	US$k	6,420	0	0	183	287	282	336	343	314	224	201	217	211	238	219
EBITDA	US$k	275,445	0	0	6,886	14,597	14,138	18,744	19,543	16,206	7,471	3,926	5,589	5,639	8,086	5,655
Interest Expense	US$k	13,782	0	0	2,830	0	2,939	0	2,110	0	1,322	0	954	0	970	0
Taxes	US$k	58,184	0	0	688	329	769	3,958	654	353	544	12,364	500	356	507	1,023
Depreciation	US$k	52,698	0	0	1,839	1,847	1,854	1,862	1,870	1,900	1,931	1,961	1,992	2,008	2,025	2,042
Recovery Historic Cost	US$k	75,000	0	0	4,167	4,167	4,167	4,167	4,167	4,167	4,167	4,167	4,167	4,167	4,167	4,167
Net Profit/Loss	US$k	75,781	0	0	-2,637	8,255	4,409	8,757	10,743	9,786	-493	-14,566	-2,023	-892	418	-1,577
Cashflow Calculation																
Net Profit	US$k	75,781	0	0	-2,637	8,255	4,409	8,757	10,743	9,786	-493	-14,566	-2,023	-892	418	-1,577
Add Interest, Depreciation and Recovered Cost	US$k	141,479	0	0	8,835	6,013	8,960	6,029	8,146	6,067	7,420	6,128	7,112	6,175	7,162	6,209
Change in Working Capital Account	US$k	2,676	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Change in VAT Account	US$k	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Pre-production Capex	US$k	-86,479	-5,726	-4,898	-1,219	0	0	0	0	0	0	0	0	0	0	0
Sustaining Capex	US$k	-12,544	0	0	-297	-297	-297	-297	-1,188	-1,188	-1,188	-1,188	-654	-654	-654	-654
Cashflow before Financing	US$k	120,914	-5,726	-4,898	4,683	13,972	13,072	14,489	17,701	14,664	5,739	-9,626	4,435	4,630	6,925	3,978

Cash Flow Model (continued)

Yanfolila Gold Project - Cash Flow

Project Year			4	4	4	5	5	5	5	6	6	6	6	7	7	7
Calendar Year			2019	2019	2019	2020	2020	2020	2020	2021	2021	2021	2021	2022	2022	2022
Period			Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3
Months in Period			3	3	3	3	3	3	3	3	3	3	3	3	3	3
Period Start Date	01-Jan-15		01-Apr-19	01-Jul-19	01-Oct-19	01-Jan-20	01-Apr-20	01-Jul-20	01-Oct-20	01-Jan-21	01-Apr-21	01-Jul-21	01-Oct-21	01-Jan-22	01-Apr-22	01-Jul-22
Period End Date			30-Jun-19	30-Sep-19	31-Dec-19	31-Mar-20	30-Jun-20	30-Sep-20	31-Dec-20	31-Mar-21	30-Jun-21	30-Sep-21	31-Dec-21	31-Mar-22	30-Jun-22	30-Sep-22
Construction Period	01-Jan-15	31-Mar-16	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Operations Period	01-Apr-16	30-Sep-22	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mining																
Waste Mined	t	67,456,000	2,931,000	2,894,000	3,029,000	2,503,000	2,070,000	2,081,000	2,824,000	2,667,000	1,743,000	1,368,000	1,400,000	917,000	664,000	378,000
Ore Mined	t	6,298,000	249,000	249,000	250,000	250,000	250,000	251,000	250,000	251,000	251,000	250,000	251,000	251,000	250,000	243,000
Total Material Mined	t	73,754,000	3,180,000	3,143,000	3,279,000	2,753,000	2,320,000	2,332,000	3,074,000	2,918,000	1,994,000	1,618,000	1,651,000	1,168,000	914,000	621,000
Strip Ratio	w:o	10.71	11.77	11.62	12.12	10.01	8.28	8.29	11.30	10.63	6.94	5.47	5.58	3.65	2.66	1.56
Au Grade Mined	g/t	2.64	2.64	2.66	2.81	3.01	3.06	2.86	2.23	2.25	2.72	2.89	2.03	2.18	2.31	2.06
Au Cont. Mined	oz	534,540	21,130	21,260	22,600	24,150	24,560	23,080	17,910	18,180	21,960	23,240	16,410	17,590	18,530	16,080
Processing																
Ore Milled	t	6,414,000	249,000	249,000	250,000	250,000	250,000	251,000	250,000	251,000	251,000	250,000	251,000	251,000	250,000	243,000
Au Grade	g/t	2.65	2.64	2.66	2.81	3.01	3.06	2.86	2.23	2.25	2.72	2.89	2.03	2.18	2.31	2.06
Au Cont.	oz	546,530	21,130	21,260	22,600	24,150	24,560	23,090	17,920	18,180	21,960	23,240	16,410	17,600	18,530	16,090
Recovery	%	93.98%	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Au Recovered	oz	513,610	19,860	19,980	21,270	22,750	23,140	21,730	16,770	17,020	20,640	21,870	15,330	16,470	17,360	15,030
Price Assumptions																
SpotGold Price	US$/oz	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250
Income Calculation																
Gross Revenue	US$k	642,013	24,825	24,975	26,588	28,438	28,925	27,163	20,963	21,275	25,800	27,338	19,163	20,588	21,700	18,788
Transport & Insurance Costs	US$k	2,212	86	86	92	98	100	94	72	73	89	94	66	71	75	65
Refining Costs	US$k	257	10	10	11	11	12	11	8	9	10	11	8	8	9	8
Net Revenue	US$k	639,544	24,730	24,879	26,485	28,328	28,814	27,058	20,882	21,193	25,701	27,232	19,089	20,508	21,617	18,715
Mine Operating Cost	US$k	186,574	7,730	8,148	8,973	7,528	6,625	6,438	8,779	8,361	6,193	4,808	4,800	4,023	3,354	2,871
Processing Operating Cost	US$k	100,133	4,075	4,097	4,072	4,111	4,043	4,086	4,111	4,084	4,142	4,016	3,575	4,064	4,111	3,512
G&A Costs	US$k	42,327	1,606	1,606	1,606	1,606	1,606	1,606	1,606	1,606	1,606	1,606	1,606	1,606	1,606	1,606
Management Fee	US$k	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Rehabilitation	US$k	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Total Operating Costs	US$k	329,034	13,411	13,851	14,651	13,245	12,274	12,130	14,496	14,051	11,941	10,430	9,981	9,693	9,071	7,988
Total Cash Cost per Ounce Recovered	US$/oz	641	675	693	689	582	530	558	864	826	579	477	651	589	523	532
Operating Profit	US$k	310,510	11,318	11,028	11,834	15,083	16,540	14,928	6,386	7,142	13,760	16,802	9,108	10,815	12,545	10,727
Government Royalty	US$k	23,112	894	899	957	1,024	1,041	978	755	766	929	984	690	741	781	676
LPMDO Royalty	US$k	5,532	0	0	0	489	489	0	0	525	525	0	0	466	466	0
Taurus Royalty	US$k	6,420	248	250	266	284	289	272	210	213	258	273	192	206	217	188
EBITDA	US$k	275,445	10,176	9,879	10,611	13,285	14,720	13,678	5,421	5,638	12,047	15,545	8,226	9,402	11,081	9,863
Interest Expense	US$k	13,782	867	0	323	0	0	0	0	0	0	0	0	0	0	0
Taxes	US$k	58,184	469	349	418	2,749	320	314	358	9,523	307	276	253	11,779	237	208
Depreciation	US$k	52,698	2,059	2,068	2,077	2,085	2,094	2,102	2,110	2,117	2,125	2,132	2,139	2,146	2,154	2,157
Recovery Historic Cost	US$k	75,000	4,167	4,167	4,167	4,167	4,167	4,167	0	0	0	0	0	0	0	0
Net Profit/Loss	US$k	75,781	2,615	3,295	3,626	4,284	8,139	7,096	2,954	-6,002	9,615	13,137	5,834	-4,524	8,691	7,498
Cashflow Calculation																
Net Profit	US$k	75,781	2,615	3,295	3,626	4,284	8,139	7,096	2,954	-6,002	9,615	13,137	5,834	-4,524	8,691	7,498
Add Interest, Depreciation and Recovered Cost	US$k	141,479	7,093	6,235	6,567	6,252	6,261	6,269	2,110	2,117	2,125	2,132	2,139	2,146	2,154	2,157
Change in Working Capital Account	US$k	2,676	0	0	0	0	0	0	0	0	0	0	0	0	0	2,676
Change in VAT Account	US$k	0	-392	-10	-20	29	29	6	-44	10	44	32	22	4	34	29
Pre-production Capex	US$k	-86,479	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Sustaining Capex	US$k	-12,544	-354	-354	-354	-354	-298	-298	-298	-298	-278	-278	-278	-278	-135	-135
Cashflow before Financing	US$k	120,914	8,961	9,166	9,819	10,211	14,132	13,073	4,722	-4,173	11,506	15,023	7,718	-2,651	10,743	12,225

Cash Flow Model (continued)

Yanfolila Gold Project - Cash Flow

Project Year			7	8	8	8	8	9	9	9	9	10
Calendar Year			2022	2023	2023	2023	2023	2024	2024	2024	2024	2025
Period			Q4	Q1	Q2	Q3	Q4	Q1	Q2	Q3	Q4	Q1
Months in Period			3	3	3	3	3	3	3	3	3	3
Period Start Date	01-Jan-15		01-Oct-22	01-Jan-23	01-Apr-23	01-Jul-23	01-Oct-23	01-Jan-24	01-Apr-24	01-Jul-24	01-Oct-24	01-Jan-25
Period End Date			31-Dec-22	31-Mar-23	30-Jun-23	30-Sep-23	31-Dec-23	31-Mar-24	30-Jun-24	30-Sep-24	31-Dec-24	31-Mar-25
Construction Period	01-Jan-15	31-Mar-16	0	0	0	0	0	0	0	0	0	0
Operations Period	01-Apr-16	30-Sep-22	0	0	0	0	0	0	0	0	0	0
Mining												
Waste Mined	t	67,456,000	0	0	0	0	0	0	0	0	0	0
Ore Mined	t	6,298,000	0	0	0	0	0	0	0	0	0	0
Total Material Mined	t	73,754,000	0	0	0	0	0	0	0	0	0	0
Strip Ratio	w:o	10.71	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Au Grade Mined	g/t	2.64	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Au Cont. Mined	oz	534,540	0	0	0	0	0	0	0	0	0	0
Processing												
Ore Milled	t	6,414,000	0	0	0	0	0	0	0	0	0	0
Au Grade	g/t	2.65	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
Au Cont.	oz	546,530	0	0	0	0	0	0	0	0	0	0
Recovery	%	93.98%	0	0	0	0	0	0	0	0	0	0
Au Recovered	oz	513,610	0	0	0	0	0	0	0	0	0	0
Price Assumptions												
SpotGold Price	US$/oz	1,250	1,250	1,250	0	0	0	0	0	0	0	0
Income Calculation												
Gross Revenue	US$k	642,013	0	0	0	0	0	0	0	0	0	0
Transport & Insurance Costs	US$k	2,212	0	0	0	0	0	0	0	0	0	0
Refining Costs	US$k	257	0	0	0	0	0	0	0	0	0	0
Net Revenue	US$k	639,544	0	0	0	0	0	0	0	0	0	0
Mine Operating Cost	US$k	186,574	0	0	0	0	0	0	0	0	0	0
Processing Operating Cost	US$k	100,133	0	0	0	0	0	0	0	0	0	0
G&A Costs	US$k	42,327	0	0	0	0	0	0	0	0	0	0
Management Fee	US$k	0	0	0	0	0	0	0	0	0	0	0
Rehabilitation	US$k	0	0	0	0	0	0	0	0	0	0	0
Total Operating Costs	US$k	329,034	0	0	0	0	0	0	0	0	0	0
Total Cash Cost per Ounce Recovered	US$/oz	641	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a	n/a
Operating Profit	US$k	310,510	0	0	0	0	0	0	0	0	0	0
Government Royalty	US$k	23,112	0	0	0	0	0	0	0	0	0	0
LPMDO Royalty	US$k	5,532	0	304	304	0	0	0	0	0	0	0
Taurus Royalty	US$k	6,420	0	0	0	0	0	0	0	0	0	0
EBITDA	US$k	275,445	0	-304	-304	0	0	0	0	0	0	0
Interest Expense	US$k	13,782	0	0	0	0	0	0	0	0	0	0
Taxes	US$k	58,184	0	8,361	0	0	0	0	0	0	0	0
Depreciation	US$k	52,698	0	0	0	0	0	0	0	0	0	0
Recovery Historic Cost	US$k	75,000	0	0	0	0	0	0	0	0	0	0
Net Profit/Loss	US$k	75,781	0	-8,665	-304	0	0	0	0	0	0	0
Cashflow Calculation												
Net Profit	US$k	75,781	0	-8,665	-304	0	0	0	0	0	0	0
Add Interest, Depreciation and Recovered Cost	US$k	141,479	0	0	0	0	0	0	0	0	0	0
Change in Working Capital Account	US$k	2,676	0	0	0	0	0	0	0	0	0	0
Change in VAT Account	US$k	0	228	0	0	0	0	0	0	0	0	0
Pre-production Capex	US$k	-86,479	0	0	0	0	0	0	0	0	0	0
Sustaining Capex	US$k	-12,544	0	0	0	0	0	0	0	0	0	0
Cashflow before Financing	US$k	120,914	228	-8,665	-304	0	0	0	0	0	0	0

11.0 EXECUTION PLAN AND SCHEDULE

11.1 Execution Plan

11.1.1 Overview

SENET have generated a CAPEX and OPEX estimate for the Project, using a 1 Mtpa design template from a previous project as the basis for the estimate and design, whilst introducing the following new elements into the CAPEX estimate:

- Milling – a new mill to suit the Yanfolila requirements.
- Treatment of gravity concentrate – ILR and EWC.
- Cyanide detoxification facility with associated reagents.
- Compressed air plant upgrade.

The Project is located in the Republic of Mali where the works will be executed under a fast track regimen.

11.1.2 Occupational Health and Safety

SENET places safety, health and the environment at the forefront of all activities from inception through to final handover to the Client.

All protocols contained within the SENET HSE policies will be implemented, maintained and monitored throughout the life of the Project commensurate with construction regulatory appointments.

11.1.3 Process

The process deliverables will be adopted from a previous 1 Mtpa gold plant, designed and constructed by SENET and will be enhanced in line with the requirements dictated by the addition of the new elements as listed above.

All related process documentation, drawings and data will be amended, updated and released for design on approval from the Client.

11.1.4 Design and Engineering

The design template from a previous plant will be adopted for all the respective disciplines except as noted otherwise.

All designs will be reviewed and verified to confirm suitability and compliance with relevant codes of practice.

Existing drawings will be reviewed and revised to suit design requirements and will be issued as required for the respective actions.

Layout of the plant will be optimized to minimize the footprint and thus optimizing the plant terrace simultaneously to reduce fill volumes.

Geotechnical requirements, assessments and reports will be provided by the Client.

Ground improvements as recommended by the Geotechnical specialists will be incorporated into the SENET earthworks design.

Structures within the plant will be designed, manufactured and installed in accordance with the relevant SANS codes and in accordance with the SENET specifications.

Materials of construction will be adopted in line with previous designs but also in line with the process design criteria for new elements of the plant.

11.1.5 Project Personnel

Personnel assigned to the Project team in all disciplines will be qualified in their respective fields and all of who have numerous years of experience in the associated fields of mineral process plant design and construction.

An organogram is attached to illustrate the composition of the SENET team assigned to the project.

11.1.6 Procurement

Vendors who supplied equipment to the previous project will be approached via an RFQ to requote for like equipment.

Vendors for new elements required for the plant will be approached in three-fold with RFQ's, following which technical and commercial adjudications will be conducted for review and approval.

The procurement of long lead equipment items will be prioritized so as to avert any project delays.

Purchase orders on South African suppliers will be placed by SENET on behalf of the Client and funding of such orders will be by the Client (inclusive of 14% VAT) by placing reserve amounts into a joint control Project bank account prior to placement of orders. SENET will pay vendors/suppliers from the joint bank account on submission and approval of invoices.

Orders placed on foreign suppliers will be placed on such supplier directly by the Client, based on a purchase instruction from SENET issued to the Client by SENET's procurement department.

A reconciliation of procurements versus payments (including VAT payments and refunds) from the joint bank account will be submitted to the Client on a regular basis.

11.1.7 Expediting and Inspection

Inspection and expediting of local market procurements and suppliers will be carried out on a continual basis to ensure quality and integrity of manufacture in accordance with OEM specifications and project specific requirements.

Status reports pertaining to this aspect will be included with monthly project reports.

All the necessary data books pertaining to manufacture, material identification, welding and non-destructive testing as required will be collated and issued to the Client at the close-out stage of the project.

OEM equipment manuals will similarly be collated and issued to the Client prior to commencement of commissioning.

11.1.8 Project Control

SENET utilizes Microsoft Dynamics for cost reporting on EPCM and capital expenditure.

Cost reports will be provided on a monthly basis of actuals against budget in the relevant disciplines of the project, indicating also project savings to date as well as forecast costs to completion.

Payment certificates will be generated on a monthly basis against progress claimed for construction activities.

11.1.9 Logistics

The SENET selected freight company is well versed with the required networks and infrastructure in the region.

All routing options and surveys have been conducted from site to port so as to optimize the efficiency of forwarding of equipment to the Yanfolila site.

Comprehensive materials control procedures will be implemented to ensure that all material and equipment is logged and tracked door to door throughout the life of the project.

Reports will be generated on a continual basis for submission to Clients Engineer.

11.1.10 Construction

In country Contractors have essentially been selected for invitation to tender for the construction work, namely earthworks, civil works and structural/mechanical/plate work and piping.

The electrical and instrumentation installations will be contracted to a contractor from South Africa.

It is envisaged that three independent contractors will be adjudicated to the constructions works respectively for earthworks, civil works, and SMPP.

The infrastructure works is envisaged to be adjudicated in favor of two of the independent contractors.

The selected construction contractors will have their independent “stand-alone” construction infrastructure facilities on site including accommodation and messing of their employees.

SENET will provide ex-pat personnel for the supervision of the respective contractors consisting of a Construction Manager together with support personnel in terms of administration, health and safety and the required discipline supervision in the field.

The SENET personnel will make use of the accommodation and messing facilities provided by the Client.

SENET ex-pat personnel will be employed on a rotational basis of eight weeks on and two weeks off.

The SENET Construction Manager will report to the Client's Engineer and will provide the necessary reports to the Client and to SENET regarding progress and daily activities and interfacing associated with the construction works on a regular basis.

11.1.11 Commissioning

Following practical completion of the works, such works will undergo a pre-commissioning phase in preparation for cold commissioning to ensure the functionality of all equipment and control systems.

Following cold commissioning, SENET's commissioning team, under the supervision and guidance of a commissioning Manager, will proceed with hot commissioning of the works in conjunction with the Client and introduce ore into the plant with the objective of achieving "First Gold" on the scheduled target date. At the Hot commissioning phase, the Client will assume full responsibility for the operation of the works with assistance as required from the SENET commissioning team.

Subsequent to the achievement of First Gold, all process parameters will be optimized to satisfy plant performance in line with the process guarantees and to achieve nameplate capacity.

During the commissioning phases, the SENET commissioning team will work closely with the Client's operations personnel.

11.2 Execution Schedule

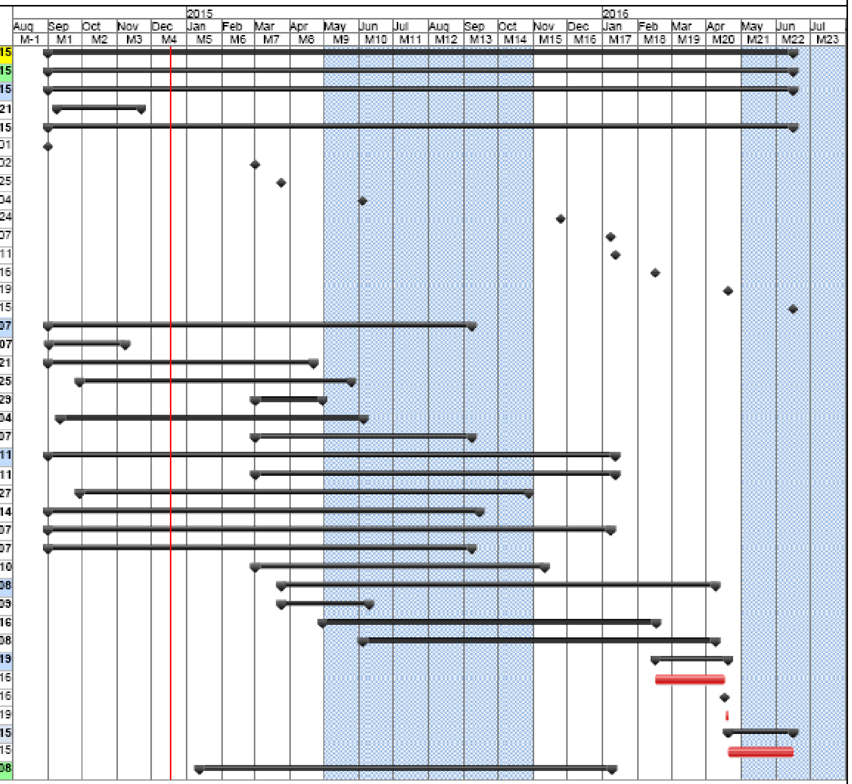
Table 11.1 illustrates the Project execution schedule.

Table 11.1 Project Execution Schedule

SENET

SP570 - Yanfolila Gold Project
Phase 2 & 3 Execution Schedule
Rev 06 - Forecast to complete

ID	Task Name	Duration	Start	Finish
1	**SP570 - Yanfolila Gold Project - Phase 2 & 3 Execution Schedule**	**438 days**	**2014/09/01**	**2016/06/15**
2	**SENET controllable**	**438 days**	**2014/09/01**	**2016/06/15**
3	**Key Project Milestones**	**438 days**	**2014/09/01**	**2016/06/15**
4	**Phase 1 Deliverables**	**54 days**	**2014/09/08**	**2014/11/21**
13	**Phase 2 & 3 Milestones**	**438 days**	**2014/09/01**	**2016/06/15**
14	Phase 2 & 3 Commence	0 days	2014/09/01	2014/09/01
15	Suspension on project lifted - Go ahead on placement of orders	0 days	2015/03/02	2015/03/02
16	Bulk earth works construction start	0 days	2015/03/25	2015/03/25
17	Start of CIL tank assembly	0 days	2015/06/04	2015/06/04
18	MMD Sizer tip station delivered to site	0 days	2015/11/24	2015/11/24
19	Ball Mill delivered to site	0 days	2016/01/07	2016/01/07
20	Medium voltage (Mv) Switch Gear delivered to site	0 days	2016/01/11	2016/01/11
21	Start of commissioning	0 days	2016/02/16	2016/02/16
22	1st Gold From Gravity Circuit	0 days	2016/04/19	2016/04/19
23	Finalize construction activities	0 days	2016/06/15	2016/06/15
24	**Design, Engineering & Drawing Office**	**250 days**	**2014/09/01**	**2015/09/07**
25	**Process Design**	**49 days**	**2014/09/02**	**2014/11/07**
31	**Structural / Mechanical & Plate work**	**155 days**	**2014/09/01**	**2015/04/21**
128	**Civil Foundations**	**156.25 days**	**2014/09/29**	**2015/05/25**
184	**Control & Instrumentation**	**40 days**	**2015/03/02**	**2015/04/29**
203	**Electrical**	**176 days**	**2014/09/12**	**2015/06/04**
211	**Piping**	**130 days**	**2015/03/02**	**2015/09/07**
214	**Procurement**	**330 days**	**2014/09/01**	**2016/01/11**
215	**Electrical Equipment**	**210 days**	**2015/03/02**	**2016/01/11**
361	**Piping Equipment**	**265 days**	**2014/09/29**	**2015/10/27**
428	**Structural Steel & Plate work**	**255 days**	**2014/09/01**	**2015/09/14**
453	**Mechanical Equipment**	**328 days**	**2014/09/01**	**2016/01/07**
791	**Civil Equipment And Packages**	**250 days**	**2014/09/01**	**2015/09/07**
954	**Control & Instrumentation Equipment**	**175 days**	**2015/03/02**	**2015/11/10**
1032	**Construction**	**304 days**	**2015/03/25**	**2016/04/08**
1033	**Bulk Earth Works**	**60 days**	**2015/03/25**	**2015/06/09**
1035	**Civil Foundations**	**235 days**	**2015/04/30**	**2016/02/16**
1082	**Structural, Mechanical, Piping Electrical & Instrumentation Installation**	**248.98 days**	**2015/06/04**	**2016/04/08**
1158	**Commissioning**	**52 days**	**2016/02/16**	**2016/04/19**
1159	Commissioning	10 wks	2016/02/16	2016/04/16
1160	Ore Introduction To Process Plant	0 days	2016/04/16	2016/04/16
1161	1st Gold From Gravity Circuit	2 days	2016/04/18	2016/04/19
1162	**Finalize construction activities**	**40 days**	**2016/04/20**	**2016/06/15**
1163	Finalize construction activities	2 mons	2016/04/20	2016/06/15
1164	**Hummingbird Resources controllable**	**288 days**	**2015/01/12**	**2016/01/08**



Project: SP570 - Yanfolila Gold Projec
Date: 2014/12/18

Task; Critical Task; Milestone; Summary; Rolled Up Task; Rolled Up Critical Task; Rolled Up Milestone; Rolled Up Progress; Split; External Tasks; Project Summary; Group By Summary; Inactive Task; Inactive Milestone; Inactive Summary; Manual Task; Duration-only; Manual Summary Rollup; Manual Summary; Start-only; Finish-only; Progress; Deadline

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12.0 PROJECT RISKS

Project team members assessed the Project risks in their area(s) of expertise.

This analysis presents Project risks, their impacts, and high/medium/low rankings for current risk severity and likelihood which will define the current risk. For each risk, the assessors developed potential mitigation factors, the residual risk severity, and likelihood and risk level. The definitions for risk severity and likelihood are shown below in Table 12.1.

Table 12.1 Definitions of Risk Severity and Likelihood

Severity	
High	Sustained production interruption or delay/reduction in cash flow, resulting in net negative cash flow available to service the debt.
Medium	Sustained substantial reduction in margin, resulting in reduced but adequate cash available to service the debt.
Low	Minor delay in production or reduction in cash flow, easily recoverable in the next operating month.

Likelihood	
High	Likely to occur and/or occur frequently in the initial five year mine plan.
Medium	Could occur sporadically, several times in the initial five year mine plan.
Low	Remote chance of occurrence. May occur sometime in initial five year mine plan.

The risk level is taken based on the severity and likelihood as shown in Table 12.2 below.

Table 12.2 Risk Level as a Function of Severity and Likelihood

		Likelihood		
Risk Level at Intersection of Severity and Likelihood		HIGH	MEDIUM	LOW
Severity	HIGH	HIGH	HIGH	MEDIUM
	MEDIUM	MEDIUM	MEDIUM	LOW
	LOW	LOW	LOW	LOW

Table 12.3 identifies the Project risks.

Table 12.3 Yanfolila Project Risks

Risk	Impact	Current Risk Severity	Current Risk Likelihood	Current Risk Risk Level	Potential Mitigation	Residual Risk Severity	Residual Risk Likelihood	Residual Risk Risk Level
Resource database sampling, data representivity and database integrity not representative.	Lower gold production than expected.	H	L	M	Update model with ongoing results.	H	L	M
Geologic model too simplified compared to actual deposit.	High levels of variability.	H	M	H	Ongoing evolution of XRF data, alteration and structure.	H	L	M
Grade model not accurate.	Mineralisation controls not identified, continuity not understood.	H	M	H	Selective close spaced drilling on SW, SE and GW.	H	L	M
Excessive dilution & ore loss	Deferred and/or lost revenue. Additional ore processing costs per ounce Au recovered.	M	H	M	improve grade control drilling program, including possible use of trenching. Close supervision of contractor.	M	M	M
Pit Flooding	Potential interruption of mining operations near pit bottoms from surface runoff accumulation and erosion of pit walls and haul roads.	M	M	M	In-pit sumps where space permits. Schedule stripping in dry season, dropping cutting below ore zones that will be mined in rainy season.	L	M	L
Geotechnical design failure.	Slope failures resulting in loss of internal ramps or covering targeted ore zones.	M	M	M	Diversion of surface runoff around open pits. Back slope benches in oxides to reduce crest erosion. Active dewatering program to reduce pore pressures.	L	M	L
Resource/Reserve Estimates too optimistic.	Shortfalls, if any, in ore tonnage and/or grade estimates could cause interruptions in mill feed and/or reductions in Au production. Deposits are presently drilled to only indicated resource classification.	M	M	M	In-fill drilling of 2- to 3-year forward outlook of planned extents to bring resources to measured classification.	L	M	L
Manning levels too low or inexperienced	Availability/recovery efficiency loss	M	M	M	Management adds manpower and/or experience.	L	L	L
Production ramp-up slower than projected.	Cash flow delay.	M	M	M	Provide adequate start-up resources, training and critical spares.	L	L	L
Active dewatering system unable to lower groundwater levels sufficiently to depressurize pit slopes and provide dry working conditions	Reduced slope angles impacting negatively on stripping ratios and project economics	H	L	M	Install horizontal drums at select locations.	M	L	L
Higher than expected volumes of water required to be extracted from pit sumps to maintain dry working conditions.	Wet operating conditions in pits impacting negatively on operations and resulting in increased operational costs.	M	M	M	Increase sump pump capacity. Ensure effective installation and operation of diversion channels.	L	L	L
HUM are required to clean up mine waste from ASM activities	Open pit mining will clear most excavations. Residual contamination from Hg use in villages	L	M	L	Education on better ASM program, assist organization of processes with a focus on H&S and Hg use. Check Mining Agreement for legacy/liability	L	L	L
Orpailleurs refuse to move from pit footprints or return to site after their removal and cause disruption to construction or operations	Construction delayed; operations stop / cannot start	H	M	H	Properly planned land acquisition programme timely investment in alternative livelihoods and other mitigation measures	H	L	M
Police or military get involved in removing orpailleurs, resulting in violence	work stoppages; social unrest; degrading relations with community. Negative publicity for HUM, impacting investors	M	M	M	Stakeholder engagement; formalized agreements; strengthening and support for local governance structures; clear policy and work with police	L	L	l

Yanfolila Project Risks, continued

		Current Risk				Residual Risk		
Risk	Impact	Severity	Likelihood	Risk Level	Potential Mitigation	Severity	Likelihood	Risk Level
Major reagent or fuel spill to soils or watercourse en route to site	Regulatory inspection, impacts to communities and ecosystems, ops closedown; social unrest; international scrutiny	H	M	H	Hazardous materials mgmt plan; Traffic management plan; reduced volumes to transport at once; community emergency preparedness; equipm transport / facilities with spill kits and training.	M	L	L
Major pollution incident affecting the Sankarani river (international border)	Government suspends operations. Removes environmental permit. Reputational damage to mine Loss of social license to operate	H	L	M	operational design; emergency preparedness and response plan; operational SOPs	M	L	L
Tailings pipeline failure	No possibility to store tails; operations stopped; environmental incident; impacts to farmland, ecosystems, groundwater, surface water regulatory investigation - breach of permit if no detection system installed	H	L	M	Design; inspections; leak detection system; regular maintenance; capture of spills; spill prevention and response	M	L	L
Significant numbers of the workforce become ill with malaria on a regular basis, impairing their ability to work	direct impact to operational workforce; work stoppages or slow down	M	H	M	Malaria prevention programme both in and out of the fence	M	L	L
Social obligations in ESIA not met	Grievances from local community; social unrest; Govt suspends permit?	H	M	H	Make resources and budget available to meet commitments	L	M	L

Five current risk assessments contained high risk levels, but in each case the residual risk, after potential mitigations, reduced to medium or low. The five high risk areas include the following risks:

- Geologic model too simplified compared to actual deposit; mitigation is the ongoing evolution of XRF data, alteration, and structure.
- Grade model not accurate; mitigation is selective close spaced drilling on SW, SE, and GW.
- Orpailleurs refuse to move from pit footprints or return after their removal and disrupt construction or operations; mitigation is a properly planned land acquisition program and timely investment in alternative livelihoods.
- Major reagent or fuel spill to soils or watercourse en route to site; mitigation is the hazardous material management plan, traffic management plan, reduced volumes transported at once, community emergency preparedness, mandated driver screening and training, pre-transport load inspections, and possibly piloted/escorted transit.
- Social obligations in EISA not met; mitigation is to make resources and budget available to meet commitments.

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14.0 APPENDICES

Appendix 1 **Resource**

Appendix 2 **Mine Geotechnical**

Appendix 3 **Hydrology/Hydrogeology**

Appendix 4 **Mining**

Appendix 5 **Metallurgy**

Appendix 6 **Process**

Appendix 7 **Environmental and Social**

Appendix 8 **Tailings Storage Facility**

Appendix 9 **Infrastructure**

Appendix 10 **Execution Plan and Schedule**