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Namdini Gold Project Preliminary Feasibility Study Technical Report

Ghana, West Africa

Technical Report compiled under NI 43-101

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Forward-looking statements and information includes statements concerning, among other things, cost reporting in the 2018 Namdini PFS, production, cost and capital expenditure guidance; development plans for processing resources; the generation of cashflow; matters relating to proposed exploration and expansion; communications with local stakeholders and community relations; negotiation and completion of transactions; commodity prices; Mineral Resources, Ore Reserves, realization of Ore Reserves, existence or realization of Mineral Resource estimates; the development approach, the timing and amount of future production; timing of studies, announcements and analyses, the timing of construction and development of proposed additional mines and process facilities; capital and operating expenditures; economic conditions; availability of project financing on terms reasonably acceptable to Cardinal; exploration plans and any and all other timing, exploration, development, operational, financial, budgetary, economic, legal, social, regulatory and political matters that may influence or be influenced by future events or conditions. Actual results may vary from such forward-looking information for a variety of reasons, including but not limited to risks and uncertainties disclosed in other Cardinal filings at www.sedar.com.

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1.0 SUMMARY

In this Section, Figure numbers and Table numbers refer to those in the main body of this Report.

1.1 Introduction

The principal activity of the Company (and its subsidiaries) is gold exploration and development in Ghana. The Company holds interests in five tenements prospective for gold mineralization in Ghana in two NE-SW trending Paleo-Proterozoic granite-greenstone belts: the Bolgatanga Project and the Namdini Gold Project ("Namdini Gold Project" or "Namdini"), which are, respectively, located within the Nangodi and Bole-Bolgatanga Greenstone Belts in northeast Ghana, and the Subranum Project, which is located within the Sefwi Greenstone Belt in southwest Ghana.

The main focus of activity is the Namdini Gold Project where a Mineral Resource at a 0.5 g/t Au cut-off grade has been established comprising: an Indicated Mineral Resource of 180 Mt grading 1.1 g/t Au for 6.5 Moz Au, and an Inferred Mineral Resource of 13 Mt grading 1.2 g/t Au for 0.5 Moz Au. Figure 1 shows the location of the Namdini Gold Project and the Company's other properties in Ghana.

This Report presents the result of the Preliminary Feasibility Study ("PFS") carried out by Cardinal on the Namdini Gold Project.

1.2 PFS parameters and material assumptions

The PFS capital cost estimates were completed to an accuracy of +30/-20% for the 9.5 Mtpa option and undertaken based on open pit mining from the existing March 2018 Mineral Resources.

Metallurgical testwork carried out to date indicates that gold can be satisfactorily recovered from Namdini ore using conventional flotation, regrind and carbon-in-leach (CIL) of the flotation concentrate. The testwork is considered sufficient to determine that the Namdini Mineral Resource represents a deposit with potential economic extraction. Estimation of capital costs was prepared by Lycopodium for the process plant and associated infrastructure.

The proposed plant incorporates primary crushing, grinding and re-crush (SABC), gravity, flotation, concentrate regrind and CIL gold extraction. Three production throughputs were assessed by Cardinal, namely 9.5, 7.0 and 4.5 Mtpa. The 9.5 and 4.5 Mtpa throughput options were factored from the 7.0 Mtpa option (+20/-15% accuracy) and are therefore lower in accuracy at +30/-20%. A contingency factor of 5% was added to the 9.5 and 4.5 Mtpa options over and above the project contingency.

Golder Associates Pty Ltd ("Golder") provided open pit mine engineering services. The work comprised collation of input parameters, open pit optimization studies, pit designs and detailed NPV optimized mine schedules. A series of shells from the open pit optimizations was selected and used to generate engineering pit designs that included a Starter Pit and Life of Mine (LOM) stages for the LOM production schedule.

Golder estimated the Ore Reserve in accordance with the JORC Code (2012). The term 'Ore Reserve' is synonymous with the term 'Mineral Reserve' as used by Canadian National Instrument 43-101 'Standards of Disclosure for Mineral Projects' (NI 43-101, 2014) and conforms with CIM (2014). The JORC Code (2012) is defined as an 'acceptable foreign code' under NI 43-101. For the purposes of reporting under NI 43-101 a JORC Table 1 is not required, but it was provided as Appendix 1, in Cardinal (2018c).

This Ore Reserve estimate is based on the revised Mineral Resource model referred to in the press release by Cardinal Resources Limited to the ASX and TSX, dated March 5, 2018 titled 'Cardinal Upgrades Indicated Mineral Resource to 6.5 Moz' (Cardinal, 2018). Golder provided an estimate of mining costs, including drill and blast, haulage, rehabilitation and administration costs. Lycopodium provided processing cost estimates.

The Ore Reserve was estimated from the Mineral Resource after consideration of the level of confidence in the Mineral Resource and considering material and relevant modifying factors including mining, processing, infrastructure, environmental, legal, social and commercial factors. The Probable Ore Reserve estimate is based on Indicated Mineral Resources. No Inferred Mineral Resource was included in the Ore Reserve. The Ore Reserve represents the economically mineable part of the Indicated Mineral Resources. There is no Proved Ore Reserve since no Measured Mineral Resource has yet been defined. Mineral Resource estimates are reported inclusive of those Mineral Resources converted to Ore Reserves.

The proposed mine plan is technically achievable. All technical proposals made for the operational phase involve the application of conventional technology that is widely utilized in the gold industry. Financial modelling completed as part of the PFS show that the Project is economically viable under current assumptions. Modifying Factors (mining, processing, infrastructure, environmental, legal, social and commercial) were considered during the Ore Reserve estimation process. The PFS incorporates a number of factors and assumptions as outlined in the sections within the Report.

The financial model was completed as a real discounted model. A Life of Mine (LOM) financial analysis was performed using the discounted cash flow (DCF) method and varying real discount rates. The financial analysis was used to determine the potential economic return of the project over the LOM.

The preliminary schedule is shown in Section 26.4 (Table 154) and is subject to available funding, positive outcomes for the PFS and FS and favorable timelines for permitting.

The gold price assumptions used for the purposes of this PFS and the project gold price for the financial analysis are presented in Table 144.

1.3 Mining Licence

During the quarter ended 31 December 2017, a Large-Scale Mining Licence covering the Namdini Mining Lease was assigned to Cardinal Namdini Mining Limited (“Cardinal Namdini”), a wholly owned subsidiary of Cardinal, by the Minister of Lands and Natural Resources under the Ghanaian Minerals and Mining Act 2006 (Act 703). The Large-Scale Mining Licence, which covers 19.54 km^2 in the Dakoto area of the Talensi District Assembly in Upper East Region of Ghana evidenced by a Mining Lease, is for an initial period of fifteen (15) years and is renewable for up to a further thirty (30) years.

Savannah Mining Ghana Limited (“Savannah”) has completed an EIS (“Environmental Impact Statement”) scoping report for Namdini and has filed the EIS with the Environmental Protection Agency (“EPA”). In accordance with EPA Regulations 15(1b) and (1c) of the Environmental Assessment Regulations, 1999 (LI 1652) and Ghana’s Environmental Impact Assessment (EIA) Procedures, the Environmental Protection Agency (EPA) issued a public notification on the proposed Namdini Gold Mining Project. Cardinal will submit to the EPA and Minerals Commission an updated EIS for the selected project scale option envisioned for the FS prior to commencement thereof.

1.4 Namdini Mineral Resources

Independent mining industry consultant, MPR Geological Consultants Pty Ltd (“MPR”) was commissioned by Cardinal to estimate the Mineral Resources of the Namdini deposit. The Mineral Resource estimate was reported in accordance with the JORC Code (2012). The JORC Table 1 was provided as Appendix 1 of the Technical Summary provided to the ASX and TSX in Cardinal (2018c). The Mineral Resource estimate, summarized in Table 81 and Table 82, reports the Mineral Resources by category and material type (weathering) above a 0.5 g/t gold cut-off grade. The classification categories of Inferred and Indicated Mineral Resources under the JORC Code (2012) are equivalent to the CIM categories of the same name (CIM, 2014).

1.5 Geology

The Namdini gold deposit is a large, structurally controlled, orogenic gold deposit within the Nangodi Greenstone Belt, with numerous features similar to deposits found elsewhere in late Proterozoic Birimian terranes of West Africa. To date the Namdini gold deposit has been delineated over a strike length of 1.15 km, up to 300 m wide and 700 m deep.

In 2016, geological consultants from Orefind Pty Ltd conducted an on-site study and developed a structural framework of the controls and geometry of gold mineralization comprising the Namdini deposit.

Orefind concluded that the rock types comprising the Namdini Gold Project included a steeply west dipping Birimian sequence of interbedded, foliated, metasedimentary and metavolcanic units which have been intruded by a medium-grained granitoid and diorite. The southern part of the Project is covered by flat-lying Voltaian Basin clastic sedimentary rocks that have been deposited unconformably on the Birimian sequence and postdate mineralization and the host sequence.

Underneath the weathering profile, the Birimian units include metasedimentary, metavolcanic, granitoid (tonalite) and diorite. The metasedimentary and volcaniclastic lithologies have been intensely altered with a resulting pyrite-carbonate-muscovite-chlorite-quartz assemblage. Alteration is most prevalent in the volcaniclastic units. Similarly, the tonalite is extensively altered and has been overprinted by silica-sericite-carbonate assemblages.

In all rock types, the mineralization is accompanied by visible disseminated sulfides of pyrite and very minor arsenopyrite in both the veins and wall rocks. In diamond drill core, the mineralized zones are visually distinctive due to the presence of millimetre to centimetre wide quartz-carbonate veins that are commonly folded and possess yellow-brown sericite-carbonate selvages. Rare visible gold occurs in strongly altered granite and is associated with sub-millimetre wide silica-sericite shears.

1.5.1 Drilling techniques

The input dataset used for the Namdini Mineral Resource estimate comprises a total of 167 HQ diamond core holes and 144 RC drill holes totalling 82,870 m.

Reverse circulation drilling of 140 (nominally 5¼ inch) or 125 mm diameter was usually 200 m or less in depth. All reverse circulation holes were downhole surveyed at 30 m intervals.

Diamond drilling was HQ in both weathered and fresh rock. All diamond holes were downhole surveyed at 30 m intervals. All HQ core was orientated.

1.5.2 Sampling

All reverse circulation samples were collected at the drill site over 1 m intervals and split using a multi-stage riffle splitter.

Diamond core was generally sawn in half; with half sent for assaying, and half retained in core trays for future reference. One metre samples were taken and submitted to an independent laboratory for assaying. At the laboratory, both core and reverse circulation samples followed a standard procedure of drying, crushing and grinding. The pulverized samples were thoroughly mixed on a rolling mat (‘carpet rolled’) and then 200 g of sub-sample was collected. Internal laboratory checks required at least 90% of the pulp passing 75 microns. A 50 g charge was produced for subsequent fire assay analysis.

Very good recovery of both core and reverse circulation samples were recorded and considered to be representative of the mineralization defined by the drilling.

1.5.3 Sample analytical methods

Cardinal used two laboratories for its geological sample submissions: SGS Ouagadougou Laboratory in Burkina Faso, and SGS Tarkwa Laboratory in Ghana. The independent SGS commercial geochemical analytical laboratories are officially recognized by the South African National Accreditation System (SANAS) for meeting the requirements of the ISO/IEC 17025 standard for specific registered tests for the Minerals Industry.

As part of the Cardinal QAQC, a suite of internationally accredited and certified reference material (standards) and locally sourced blanks were included in the sample submission sequence. The standards cover gold grade ranges expected at Namdini. Interlaboratory umpire analyses were also conducted.

Certified reference material (blanks and standards) were submitted into the sample stream at a rate of 1 in 22 samples. Duplicate samples of reverse circulation chips were taken at a rate of 1 in 22.

No employee, officer, director, or associate of Cardinal carried out any sample preparation on samples from the Namdini Gold Project exploration programme.

Drill core was transported from the drill site by a Cardinal vehicle to the secure core yard facility at the Bolgatanga Field Exploration Office only.

All samples collected for assaying are retained in a locked, secure storage facility until they are collected and transported by the SGS laboratory personnel. Retained drill core is securely stored in the core storage facility and pulps and coarse rejects returned from the laboratories are securely stored in the exploration core logging area and at a nearby secure location in Bolgatanga, Ghana.

1.5.4 Estimation methodology

MPR estimated recoverable resources for Namdini using Multiple Indicator Kriging (“MIK”) with block support adjustment, a method that has been demonstrated to provide reliable estimates of recoverable open pit resources in gold deposits of diverse geological styles. The Mineral Resource was estimated by MIK using GS3M resource modelling software developed by FSSI Consultants (Australia) Pty Ltd.

Estimation was constrained within a mineralization envelope (wireframe) based on geological logging and grade thresholds. The three-main host lithologies are granite, metavolcanics and diorite. Where geological contacts were not clearly controlling the distribution of mineralization, a grade cut-off of approximately 0.1 g/t Au was used to construct Mineral Resource boundaries.

The domain trends north-northeast over 1.3 km and dips approximately 60º to the west with an average horizontal width of approximately 350 m. The Mineral Resource can reasonably be expected to provide appropriately reliable estimates of potential mining outcomes at the assumed selectivity, without application of additional mining dilution or mining recovery factors. Validation of the MIK model was undertaken visually and statistically and reviewed independently.

Parent block dimensions of 12.5 mE by 25 mN by 5 m RL were used for estimation. All sample assays were composited to 2 m prior to estimation.

1.5.5 Classification

The Namdini Mineral Resource has been classified into the Indicated and Inferred categories, in accordance with the JORC Code (2012) and the CIM Standards (CIM, 2014). A range of criteria were considered in determining this classification including geological and grade continuity, data quality and drill hole spacing.

Resource model blocks have been classified as Indicated or Inferred on the basis of search passes and a wire-frame outlining more closely drilled portions of the mineralization. Blocks within the classification wire-frame informed by all search passes were classified as Indicated. Blocks outside the classification wire-frame and estimated by iteration 1 are classified as Indicated. All remaining blocks estimated by iterations 2 and 3 were assigned to the Inferred category.

The three progressively more relaxed search criteria used for MIK estimation are presented in Table 68. The search ellipsoids were aligned with the general mineralization orientation.

MPR considers the estimation technique and parameters appropriate for this style of mineralization.

Based on the information presented in this PFS, the Ore Reserve estimation process has converted 73% of the Indicated Mineral Resources to Probable Ore Reserves.

1.6 Mining

The mine design and Ore Reserve estimate is based on the revised Mineral Resource model referred to in the press release by Cardinal Resources Limited to the ASX and TSX, dated 5 March 2018 (Cardinal, 2018a).

Trial open pit optimizations were run in Whittle 4X at a US$1,300/oz gold price (which was the appropriate gold price at the time of the optimization runs) to define the base of potentially economic material. Four cut back pits were then selected and full mine designs applied.

The Ore Reserve reported in the Preliminary Feasibility Study is a sub-set of the Indicated Mineral Resource which can be extracted from the mine and processed with an economically acceptable outcome.

Mining of the Namdini project has been assumed to be medium-scale using conventional open pit mining equipment. The mining process will include drill and blast as well as conventional load and haul operations. There is expected to be a very limited amount of free-dig material with the majority of material assumed to require drilling and blasting.

Mining will be carried out using staged cut backs with four identified stages incorporated within the LOM final pit. The mining schedule incorporates movement of ore and waste on 10 m mining benches, by year for each of the four mining stages.

Except for the initial plant commissioning, Oxide ore will be stockpiled temporarily and batch-fed into the process plant when suitable volumes are available, ensuring that no more than 10% of the plant available time is used to process Oxide in any one year. Waste rock will be stockpiled separately on the western side of the pit.

Metallurgical work carried out to date indicates that gold can be satisfactorily recovered from Namdini ore using conventional flotation, regrind and carbon-in-leach (CIL) cyanidation techniques. The work is considered sufficient to determine that the Namdini Mineral Resource represents a deposit with potential for economic extraction.

1.6.1 Mining factors

The *in situ* deposit Mineral Resource Model is the basis for the mining model used for the Starter Pit and Life of Mine (LOM) pit planning and assessment reporting. The resource model has cell dimensions of 12.5 m (east) by 25 m (north) by 5 m (elevation). The MIK adjustment assumes a moderately selective mining unit (SMU) of 10 m × 5 m × 2.5 m, which has been applied to Namdini's relatively low-grade, large-tonnage, disseminated deposit.

Mining will consist of a conventional hydraulic shovel operation typically using 400 t class excavators in a face-shovel configuration and 150 t class (Cat 785 or similar) rigid body dump trucks hauling on designed access roads. An auxiliary mining fleet of dozers, graders, water carts and utility vehicles will support the mining operation. The appropriately-sized equipment is of medium scale and is less amenable to selective mining. With 60 m minimum mining width as noted, selective mining practices are limited for development of this orebody.

Mining is proposed on 5 m flitches in the ore, within 10 m benches. The base case optimization was determined as part of the PFS and was run using Indicated Mineral Resources only. There is currently no Measured Mineral Resource within the Namdini resource model.

A gold cut-off grade of 0.5 g/t Au was applied to the mineralized material. Process costs and mining costs were supplied by independent consultants and compared with similar gold projects. Gold grades were supplied with the model as estimated proportional grades using the MIK recoverable resource estimation technique.

For purposes of selecting the optimum Whittle pit for mine design purposes, Golder estimated a mining cost of US$3.50 per tonne of rock mined (see Section 16.14) based on experience with similar mining operations in the region, which includes grade control sampling, laboratory assay, analysis and supervision costs. The input process and G&A cost for the baseline 7.0 Mtpa option was estimated at US$14.50/t milled plus an additional US$1.50/t allowance for stockpile reclaim. All tonnes were assumed to be on a dry basis.

Once the optimum selected Whittle pit was selected and mine design completed, a detailed mining movement schedule was supplied to two prospective mining contract companies to assist with the provision of a detailed mining cost estimate. Quotations were provided by both companies which supported an all-in contract mining cost used in this PFS. Further discussions and negotiations will continue with suitable mining contractors prior to any award of the mining contract.

Metallurgical test work was used to estimate the recoverable fraction from the Oxide, Transition and Fresh ore components, with gold grade and proportion of the block at varying MIK cut-off points coded in the block model.

Using the identified marginal Cut-off Grade, the proportion of ore per parcel and gold grade above the Cut-off Grade were included within the mining model to allow export of the parcelled (ore + waste) blocks to the pit optimiser for open pit optimization.

No consideration has been made for underground extensions of the operation in this PFS. A minimum mining width of 60 m was assumed. Mining dilution and recovery are addressed in the modelling method (MIK with variance adjustment) and the utilization of flitch mining. No Inferred Mineral Resources have been included for the PFS within the LOM planning. Mining Infrastructure requirements were assumed to be provided by the selected mining contractor with the mining performed on an outsourced basis.

Grade control will be based on sampling from RC drilling spaced at approximately 10 mE by 15 mN with samples taken at 1.5 metre intervals downhole. All grade control sampling assays are assumed to be determined by fire assay on the mine site. Standard QAQC protocols will be applied comprising 1 in every 10 samples.

1.6.2 Geotechnical parameters

In support of the mine design, Golder carried out a study of existing geotechnical information, reviewed information on mineral resource estimates, conducted a detailed pit geotechnical drilling campaign supervised by a site visit by a senior Golder engineer and gathered detailed rotary core logging data from selected drill locations within the Namdini project area.

The Life of Mine pit design considers slope performance based on models developed from laboratory results of sampled drill core. The results present feasibility-level slope designs based on data collected in the field, plus data and reports made available by Cardinal.

Based on geotechnical and hydrogeological considerations from site investigations at the project area, the design sectors were designated around Namdini Pit.

Inter-ramps (bench stacks) in slightly weathered to fresh rock should consist of four benches. These are to be separated by 25 m ramps or geotechnical berms (this means that a 25 m geotechnical berm should be included after every 80 m of fresh rock benches). The design table includes an alternative berm width of 5 m, along with the corresponding inter-ramp angle.

Golder recommended that at the beginning of excavation the narrower width be used for benches in SOX, MOX and TRANS materials in temporary walls. Should this geometry perform well then it could be applied to the final walls as well. Should it prove inadequate or problematic, the wider 6-m berms could be used for the final walls in SOX, MOX and TRANS materials.

1.6.3 Pit optimizations

Pit optimizations were completed using the Lerchs-Grossman (LG) algorithm in Whittle 4X to calculate the optimal pit at specified input parameters that were determined prior to the study. A wireframe pit shell for each gold price considered was the resultant output. One of these was selected as the base for the final LOM pit design. A smaller pit approximately 1 Moz was chosen for the Starter Pit to maximize discounted cash flow and minimize time for capital payback.

1.6.4 Mine scheduling

Mine scheduling was used to maximize value through deferring of larger strip ratio cut-backs until later in the mine life. The maximum value pit was selected using a discounted average Net Present Value and determined to align with a Revenue Factor (“RF”) shell of approximately $1,105/oz using estimated LOM input prices and costs. Pit shells were converted into engineering designs prior to export of the contained resource model for scheduling purposes.

A commercial linear programming software package (Minemax Scheduler) was used to model the mining sequence, the processing plant and different ore feeds to maximize NPV for the nominated parameters and constraints. Major constraints included mill throughput, mining limits and oxide feed proportion. The material selection to satisfy processing requirements was based on cut-off grade, mineable ore, processing and selling costs.

The mine scheduling programme includes revenue and cost information. The scheduling software assesses the value generated by each block to determine whether the block is fed directly to the plant, stockpiled or treated as waste. Further financial analysis to determine more realistic absolute financial indicators and sensitivity analysis are performed separately, using the tonnes and grades extracted from the schedule.

The mine design of the Namdini Gold Project consists of a series of nested conventional pit layouts with orebody access provided by a series of ramps. The orebody can be considered a layered sequence consisting of strongly oxidized, moderately oxidized, transition, and fresh mineralized zones.

High level mine production schedules were evaluated for the three scenarios considered (9.5, 7.0 and 4.5 Mtpa mill throughputs) using a Starter Pit with subsequent pushbacks to the final LOM pit extent.

The schedules allowed an initial ramp up for the process plant in each case before full process plant production was assumed. In order to gain maximum value from the 9.5 Mtpa option, an estimated total peak rock movement of some 30 Mtpa is required in year 7 of the schedule, whereas the 7.0 Mtpa option indicated a total peak required movement of some 17 Mtpa. The 4.5 Mtpa option saw a peak total required rock movement of some 15 Mtpa.

1.6.5 Mine design criteria

The mine design criteria were developed to allow for development and assessment of designs to provide plant feed rates of 9.5, 7.0 and 4.5 Mtpa.

For this mining study, the maximum mining movement has allowed for a strip ratio of up to 2:1 in order that the initial optimizations are not ‘mining-limited’.

For the conceptual pit design, two geotechnical domains namely Zone 1 – Slightly and Moderately Oxidized Weathering Domain and Zone 2 – Transitional and Fresh Weathering Domain, were used to define pit bench heights, berm widths and slope angles.

Pit design criteria were based on Golder's geotechnical recommendations with the deposit broadly broken up into weathered (Oxide), partially weathered (Transition) and Fresh domains, with two distinct domains on the hangingwall and footwall sides of the ore zone (bearing 295°). Refer to Table 96 for the geotechnical configurations used for the mine pit design criteria.

For practical pit design purposes, the berm widths were rationalized to an 8 m wide berm to avoid having multiple berm widths required on the same mining bench. Analysis of the block model indicated that the semi-weathered (Transition) material reaches a maximum depth of 160 m RL. Thus, it was deemed prudent to maintain single benches with 6 m berm widths above this level and adopt double-benching (20 m) with 8 m berms below it. Adoption of the 6 m berm in both the Oxide and Transition zones adds a level of increased safety and ease of management in the weathered part of the deposit. Detailed geotechnical zones were then flagged into the mining resource model with which to guide the pit design angles.

The pit was designed with four stages. The initial stage (Starter Pit) provides early access to the higher-grade ore near the surface. The second stage is largely an expansion of the initial stage targeting the ore to a greater depth. The stage designs were created for optimal ore delivery from the first two stages, due to their low strip ratio and waste rock movement. The third and fourth stages contain a greater proportion of waste rock. A minimum mining width of 60 m was established between the stages.

The pit designs have targeted the maximum discounted value pit shell at a US$1,300/oz gold price (note that the US$1,300/oz gold price was applicable at the time of the Whittle optimizations performed in Q2 2018). Pit optimization using Whittle software was used to identify the optimum pit shell with the Inferred Resource material considered as waste rock. The identified pit was then considered for practical staging to minimize waste movement and improve the cashflow for the project. The analysis allowed the selection of four stages with the initial stage targeting a relatively higher-grade area of ore near surface. Access was allowed to the first three stages by a ramp from the northern edge of the pit as the volume of waste rock in the first three stages is considered modest. The final fourth stage has a main access ramp on the western side of the pit to provide a shorter haul to the waste rock dump, given that the final stage has a higher strip ratio than the preceding three stages. Having the primary access on the western side of the pit reduces waste rock haulage costs and thus improves the overall value.

Given limited opportunity outside the starter pit to target higher grade zones, stage design was largely focused on targeting maximum value change points within practical mining constraint limits, such as the minimum mining width for the pushbacks. The first Stage is a relatively small 'mini-pit' on the northeast side of the deposit. The first stage contains an estimated 19.9 Mt of Fresh ore with an additional 4.0 Mt of Oxide and Transition ore. This will be stockpiled and processed in campaigns such that a maximum of 10% of available processing time is used for treating the Oxide and Transition ore in any annual period. The remaining three pit Stages follow a traditional pit expansion with the pits pushing out towards the dip of the ore and the pit deepening with each stage.

The indicative production schedules are outlined in Table 152.

1.6.6 Mining cost

The PFS assumes the mining contractor will bear the total mining capital cost under an outsourced mining arrangement with the costs recovered by the mining contractor on a cost per tonne mined basis.

Mining costs were solicited from two of the largest in-country mining contractors. The estimated base mining cost has an applied incremental cost with depth, to account for increased haulage costs and the depth of mining increases in line with standard mining cost principles.

All costs have been determined on a US dollar basis.

1.6.7 Cut-off grade

An estimated marginal cut-off grade was established at 0.5 g/t using an assumed long-term gold price of US$1,300/ounce. The provided Mineral Resource model was validated and used to develop a mining model, as the basis for the Life of Mine ("LOM") plan and economic assessment.

Gold royalties were assumed at 5% of gold price, with payable gold estimated at 99.8% of doré exported. The net gold price was thus $39.63/g. The input processing cost provided in May 2018 was $14.49/t plus an additional $1.50/t allowed for stockpile reclaim giving a total of $15.99/t of mill feed (as dry tonnes). Thus, the marginal cut-off grade ("COG") was estimated as: process cost/(net gold price * process recovery) giving 0.5 g/t (to one significant figure).

Using this marginal COG, the proportion of ore and the gold grade above the COG, were defined in the mining model and the parcelled proportions of ore, above cut-off within the blocks were exported for open pit optimization.

The 0.5 g/t Au cut-off approximates an operational parameter that the Company believes to be applicable. This is in accordance with the guidelines of Reasonable Prospects for Eventual Economic Extraction ("RPEEE") in the CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014) and the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code, 2012).

1.6.8 Ore Reserve

Ore Reserves were estimated for the Namdini Gold Project as part of this PFS by Golder. The total Probable Ore Reserve is estimated at 129.6 Mt at 1.14 g/t Au with a contained gold content of 4,760 koz.

The Ore Reserve for the Project is reported according to the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC Code, 2012) and CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014). The Mineral Resource was converted by applying Modifying Factors. The Probable Ore Reserve estimate is based on the Mineral Resource classified as Indicated. Table 83 presents a summary of the Ore Reserves on a 100% Project basis at a US$1,300/oz gold price.

1.7 Mineral processing and metallurgical testing

1.7.1 Introduction

The PFS phase of metallurgical testwork focused on development of the flowsheet as presented in Cardinal's PEA (Golder, 2018). The flowsheet is described as a conventional primary crush, SAG/Ball mill and re-crush ("SABC"), gravity, flotation, regrind and carbon-in-leach process. Metallurgical results proved to be consistent and strongly supported the flowsheet as described above.

All Fresh rock metallurgical testwork for the PFS was carried out by ALS Laboratory ("ALS") in Perth, Australia. This work followed on from the PEA metallurgical testwork completed by Suntech Geomet Laboratories ("SGL") in Johannesburg, South Africa.

Further Oxide (weathered rock) metallurgical testwork was not necessary. The Oxide PEA results were carried into the PFS.

The PFS Fresh rock was metallurgically tested in four parts:

- Starter Pit and Flotation specific testwork (to test for initial ore metallurgical response):
  - Mineralogy and gold deportment
  - Gravity recoverable gold
  - Flotation
  - Cyanide leach testwork of bulk rougher flotation concentrate at various regrind sizes.
- Life of Mine ("LOM") testwork (to test for ore metallurgical response for the entire mine life):
  - Mineralogy and gold deportment
  - Gravity recoverable gold
  - Flotation
  - Cyanide leach testwork of bulk rougher flotation concentrate at various regrind sizes.
- Variability testwork (to test for combinations of gold and varying sulfur grades):
  - Mineralogy and gold deportment
  - Gravity recoverable gold
  - Flotation
  - Cyanide leach testwork of bulk rougher flotation concentrate at various regrind sizes.
- Comminution testwork:
  - Semi-autogenous mill comminution ("SMC") variability testwork using HQ core samples
  - Julius Kruttschnitt Drop Weight Test ("JKDWT") testwork using PQ core samples.

The ALS test scope focused on optimization of the flotation and cyanide leach response across a range of representative Master and Variability Composites formed from selected drill core samples.

Metallurgical flowsheet development testwork has been supported based on a wide range of testwork, including:

- Comprehensive Master and Variability Composite head assay analysis, XRD and QEMSCAN mineralogy, and diagnostic leach characterisation.
- Coupled gravity recovery and flotation optimization across a range of primary grind sizes and including assessment of alternative reagent regimes and conditions.
- Flotation Variability testing based on the optimal regime derived from the Master Composite tests.
- Comparative whole ore leach testing over a range of grind sizes, including: direct cyanidation ("DCN"), carbon-in-leach ("CIL"), lead nitrate dosing, air and oxygen sparging.
- Flotation concentrate leaching based on bulk flotation concentrates formed from Master Composites, Life of Mine ("LOM") and Variability composites. The test scope covered a wide range of regrind sizes and leach methodologies including lead nitrate assisted leaching.

- Comminution characterisation, including: JKDWT, Uniaxial Compressive Strength ("UCS"), Bond suite and SMC style testwork applied to HQ drill core variability lithological composites and PQ drill core lithological composites.

1.7.2 Results

Key findings from the testwork were:

Comminution:

- The JK Drop Weight Test showed A*b values ranged between 30 and 42 for the PQ samples.
- The SMC Variability Test showed derived A*b values between 30 and 47 for the HQ samples.
- The BBWi test indicated a range between 14.7 and 19.7 with an average of 16.9 kwh/t.
- Bond Abrasion Index ranged between 0.03 and 0.29, with the Granite ore type having the highest values.

Mineralogy:

- Native gold is the predominant gold bearing mineral with very low silver content (<2 ppm Ag average).
- Pyrite is the dominant sulfide mineral in the composites where the majority of the gold is associated.
- P_{80} of the pyrite ranged from 82 µm to 93 µm.
- Pyrite is classified as 'well-liberated' from the host rock minerals with close to 85% liberation.
- Free gold particles were detected during a thorough optical search (using a binocular stereo-microscope) of the unmounted gravity concentrates. These gold grains are approximately 200 µm in diameter and rounded in appearance.
- Separate testing of the mineralization confirms that it is not preg-robbing.

Flotation:

- High gold recoveries averaging 95% to concentrate for the majority of composites.
- Fast flotation kinetics observed with majority of the gold recovered in less than 5 minutes.
- Addition of a co-collector blended with base case (PAX) collector did not show a definite benefit to gold recovery.
- Upfront gravity gold recovery improved total gold recovery and mostly eliminated variability in the flotation tails grade.
- Gravity and flotation resulted in average tail grades ranging from of 0.05 to 0.14 g/t.
- Flotation mass recovery to concentrate averaged 7% w/w; concentrate grade based on the Starter Pit composite averaged approximately 30 g/t without gravity recovery and approximately 20 g/t with prior gravity recovery.
- All gravity gold, flotation, regrind and leach results were analysed to produce regression recovery curves for the starter pit and for the Life of Mine samples. These curves were then applied to the varying head grades of the mine schedule to yield an overall recovery result which for the starter pit was 86% and LOM (including the Starter Pit) was 84%. These overall recoveries include Oxide ore, which were applied at an average of 90% based on previous testwork. These were achieved at a grind P_{98} of 15 µm.

Leach:

- Concentrate leach feed grades based on Starter Pit and separate Life of Mine lithology composites ranged between 10 g/t and 27.5 g/t, ranging between 15 and 17 g/t on average.
- Leach residues ranged from 1.94 g/t for the Starter Pit composite to 2.28 g/t for the separate LOM lithology composites at a grind P_{98} of 15 µm. This range in leach residue grade is equivalent to 0.14-0.16 g/t on a whole ore basis, assuming an average flotation concentrate mass recovery of 7%.

Comminution and metallurgical testwork has provided preliminary information about the physical characteristics and metallurgical response of the three Namdini lithologies.

The processing route for the Namdini ores would be a conventional: crush, primary grind, sulfide flotation followed by regrind and CIL cyanidation of the flotation concentrate.

Orway Mineral Consultants (OMC) has utilized the comminution results for comminution circuit selection and mill sizing. A primary crushing and SABC comminution circuit (open circuit SAG mill with recycle pebble crushing followed by closed circuit ball mill/hydrocyclones) was selected by OMC based on the available comminution parameters.

1.7.3 Metallurgical testing conclusion

Key results supporting selection of the process flowsheet from the metallurgical testing are as follows:

Comminution:

- From UCS testing, the Namdini ore is amenable to conventional jaw or gyratory crushing
- SAG Mill Comminution (SMC) testing supports configuration of the comminution circuit based on the proposed SABC flowsheet, incorporating SAG milling and recycle crushing of SAG mill scats, coupled with Ball milling to a finished primary grind (P_{80}) size of 106 µm.

Gravity:

- Gravity testwork proved the requirement of an upfront gravity recovery process. Prior gravity treatment lifted base-line recovery, mitigating lower flotation recovery of free gold. The inclusion of a gravity circuit is justified based on recoveries achieved for the Starter Pit composite.

Flotation:

- Metallurgical response was very consistent for all lithologies
- Gold and sulfur flotation kinetics were rapid with high recovery (> 90%) achieved after five minutes with an industry standard reagent regime
- Variable grind and flotation testing confirmed a primary grind and flotation P_{80} of 106 µm suitable for on-going development. However, scope for increasing the primary grind to P_{80} of 150 µm without compromising flotation recovery is a possibility
- Site water analysis indicated low concentrations of sulfate and multi-valent cations, also indicating little impact on flotation. Subsequent testing in site water confirmed no measurable impact on flotation kinetics or recovery.

Leach:

- Leach kinetics were rapid with extraction plateauing after 24-36 hours retention. Consistent with the current flowsheet, CIL leaching and recovery are supported. However, an opportunity exists to reduce the process design leach residence time allowed which is currently 72 hours
- Leach residue grade is predicated primarily on grind size and leach feed grade; separate lithology residues tended toward a relatively constant (terminal) grade at low leach feed grade
- Leach residue grade reduces with decreasing regrind size
- Starter Pit composite reported 87.9% extraction at a calculated head of 1.96 g/t and a regrind P_{98} of 15 µm. Gold recovery based on the average Starter Pit head grade of 1.31 g/t is calculated at 85.1% based on a concentrate regrind P_{98} of 15 µm. A combination of oxide and fresh material yields an overall recovery of for the starter pit of 86%
- Gold recovery based on LOM blend proportions: 60% Metavolcanics, 30% Granite and 10% Diorite and LOM head grading 1.15 g/t is calculated at 84% based on a fine concentrate regrind P_{98} of 15 µm.

Flowsheet development:

- Metallurgical testwork carried out to date indicates that the Namdini project can utilise a standard gold recovery process plant design with no innovative technology required
- The metallurgical process uses well-tested technology for all unit operations
- No deleterious elements were identified in the testwork that could affect the saleability or price of the gold doré produced
- Namdini will produce readily saleable gold doré which will be exported for refining
- Gold is recovered using primary crushing, SAG + ball mill grinding with re-crush (“SABC”), gravity recovery (Knelson Concentrator), flotation, concentrate regrind circuit and a CIL circuit
- A grind P_{80} of 106 µm was utilized for the primary grind design of the PFS assessment
- Gravity recovery has been incorporated given the presence of gravity recoverable gold
- Laboratory flotation testwork indicated fast sulfide flotation kinetics; the circuit comprises six stages of rougher flotation
- The flotation concentrate is reground and subjected to pre-aeration before CIL
- Gold recovery will be by a conventional CIL with elution circuit, electrowinning and gold smelting to recover gold from the loaded carbon to produce doré
- Industry typical design parameters were assumed for this study where testwork was not completed
- Detailed metallurgical testwork is continuing for the Namdini project under the direction of Cardinal to support completion of the FS.

1.8 Process Plant

Annual nominal throughput processing options of 9.5, 7.0 and 4.5 Mtpa were investigated as part of the PFS. Note that all options were designed to meet the International Cyanide Management Code for the manufacture, transport, and use of cyanide in the production of gold.

Assessment of the comminution circuit identified upper and lower throughput limits as follows:

- 9.5 Mtpa as the largest throughput that could be achieved with dual pinion mill drives
- 7.0 Mtpa throughput that could be accommodated with dual pinion mill drives
- 4.5 Mtpa as the largest throughput that could be accommodated by a jaw crusher.

1.8.1 Flowsheet

The process plant design incorporates the following unit process operations:

- Single stage primary crushing with a gyratory crusher to produce a P_{80} of 150 mm
- Crushed ore feeding a coarse ore stockpile (12 hours live) with ore reclaim via two apron feeders
- Two stage SAG/Ball mill grinding in closed circuit with cyclones to produce a grind P_{80} of 106 µm. This includes recrushing of pebbles from the SAG mill
- Gravity recovery including a scalping screen, a single 70-inch centrifugal concentrator and a CS4000 intensive leach reactor
- Rougher flotation to produce a gold-rich sulfide concentrate
- High intensity regrind of the flotation concentrate followed by thickening to minimize carbon-in-leach (CIL) tankage and reduce overall reagent consumption
- Commercially recognised HIG mill technology is utilized to regrind flotation concentrate
- Thickening of the flotation tails for water recovery prior to disposal in a separate non-cyanide tailings storage facility (TSF)
- A concentrate CIL circuit incorporating one pre-leach tank and seven CIL tanks for gold and silver adsorption
- A 3.5 tonne split AARL elution circuit, electrowinning and smelting to recover gold and silver and produce doré
- CIL tailings treatment incorporating cyanide destruction by sulfur dioxide and oxygen
- Concentrate CIL tailings disposal in a lined tailings storage facility.

Figure 130 indicates the selected PFS flowsheet for the Namdini project.

1.9 Infrastructure

1.9.1 Roads and power

Lycopodium has completed PFS level analysis covering all related aspects of the infrastructure requirement including power, water, road access and waste management.

The site will be accessed by road from the west with a new, approximately 25 km, gravel road linking the site to the existing national road N10 between Pwalagu and Winkogo. The N10 provides good access to the major cities and ports in southern Ghana and no upgrades of the N10 will be required. The site access road will follow a similar route to the proposed new power line for the existing substation north of Pwalagu.

Infrastructure will include the following dedicated elements:

- Unsealed road
- HV power line
- Water supply line from the White Volta River.

The site is located approximately 20 km outside Bolgatanga and 180 km from Tamale. Serviced camp style accommodation will also be integrated in the proximity of the operation. A shuttle bus service will operate from Bolgatanga to and from site as required.

Cardinal Resources has sufficient area on its leases to cater for its planned land requirements.

This study assumed that a new, approximately 30 km dual high voltage transmission power line will be constructed.

Power supply to the Process Plant includes the modifications necessary in the electricity grid connection, and associated GRIDCo Substations as well as the 161 kV high voltage power line to the Process Plant.

The Ghana Grid Company Ltd (GRIDCo) currently supplies a 161 kV high voltage power line from Tamale Substation to the Bolgatanga Substation. The connection point for the Namdini Gold Project will be near Pwalagu and will traverse a corridor to a new GRIDCo Substation close to the Namdini Mine.

The GRIDCo Substation will transform power at 11 kV to a plant feeder circuit breaker terminal in the Namdini Mine Substation at the plant site which will then be distributed mine-wide, including the accommodation and other site infrastructure facilities.

1.9.2 Site facilities and layout

The location of the plant, pit and waste dump is shown in Figure 132.

1.9.3 Hydrogeology and hydrology

A hydrogeological fieldwork programme was undertaken comprising a hydro-census of surrounding properties to identify groundwater users. Groundwater exploration drilling of five pairs of boreholes converted to deep and shallow monitoring wells was completed. Characterization of groundwater quality by sampling and laboratory analysis, groundwater monitoring and hydraulic testing was completed. Development of a conceptual model for assessment of pit inflows, potential impacts on mine dewatering on local, plus regional groundwater and surface water systems, has been completed in support of the mine design.

A hydrology programme including the development of a stormwater plan and overall site water balance was also completed. Hydrological design criteria are being developed, largely based on International Finance Corporation requirements.

1.9.4 Geotechnical investigation

A geotechnical investigation of the proposed site facilities is summarized in Section 18.4.

1.9.5 Water supply

A river abstraction system will be installed to provide any shortfall in process water requirements during the operation. An abstraction tower will be constructed on the northern bank of the White Volta River approximately 8.5 km to the west of the process plant. This will comprise submersible pumps situated within an intake tower located within a trench excavated into the northern bank of the White Volta River. A water storage facility will store 30 days' supply of process water to account for periods during which pumping from the river is not permitted. The facility will comprise a lined 'turkey's nest' pond located directly to the north of the Process Plant.

A pipe branch from the main raw water pipeline will supply the potable water treatment plant located at the camp that will purify the water after which it will be reticulated across the site.

A vendor packaged modular potable water treatment plant including filtration, ultraviolet sterilization and chlorination will be installed at the accommodation camp with the treated water reticulated to the site buildings, ablutions, safety showers and other potable water outlets.

1.9.6 Tailings Storage Facilities

1.9.6.1 Tailings testing

Tailings were subject to physical testing. Results indicate that flotation tails will have a rapid rate of supernatant release of 46% of contained water excluding rainfall. CIL tails would be similar but at a slower rate. Ultimate settled density (air dried) was 1.47 t/m^3 for CIL tails and 1.67 t/m^3 for flotation tails.

Geochemical testing indicated the following:

- The flotation tailings samples recorded negative net acid producing potential (NAPP) values and weakly alkaline net acid generating (NAG) pH values. Therefore, the diorite and metavolcanic flotation tailings are classified as Acid Consuming (AC) and the granite rougher tailings as Non-Acid Forming (NAF).
- The CIL tailings sample recorded a positive NAPP and a low NAG pH, resulting in a classification of Potentially Acid Forming (PAF).
- On the basis of the multi-element results, both the Flotation and CIL TSF's will be designed to prevent the loss of solids. The Flotation TSF will require a basic cover system on closure. The cover system for the CIL TSF will be driven by the need to control potential acid generation by precluding oxygen and water ingress to limit on-going oxidation of the tailings and seepage.
- Based on supernatant analysis, the flotation tailings facility will require a compacted soil liner to limit seepage. In addition, the facility will have an under-drainage system to limit the hydraulic head acting on the soil liner. The CIL tailings facility will require a robust engineered liner system, likely comprising of a compacted soil liner with overlying HDPE liner and underdrainage system.

1.9.7 Tailings Storage Facility design

1.9.7.1 Flotation TSF

The Flotation TSF will be constructed as a side valley-type storage facility to the southwest of the open pit. The facility will be constructed as two cells with zoned earth fill perimeter embankments and will be lined with a low permeability compacted soil liner. The total basin area will be 311 Ha and is designed to accommodate 120 Mt of tailings. The TSF embankments will be constructed in stages to suit storage requirements with Stage 1 constructed initially to provide capacity for the first 12 months of operation and subsequent stages constructed using downstream, modified centerline and upstream raise construction methods.

The TSF basin area will be cleared, grubbed and stripped of topsoil. A 300 mm depth compacted soil liner will be constructed over the entire TSF basin area as either reworked *in situ* material (assumed 70%) or imported Zone A (30%) material.

The TSF design incorporates an underdrainage system comprising a network of branch and collector drains in each cell. The underdrainage system drains by gravity to a collection sump located at the lowest point in each cell.

Supernatant water will be removed from the TSF via a submersible pump (designed by others) mounted in a decant tower. Temporary decants will be provided to suit the tailings deposition schedule in each cell. The final decants will be located along the divider embankment between the two cells.

1.9.7.2 CIL TSF

The CIL TSF will be constructed as a paddock-type storage facility to the south of the open pit. The facility will be constructed as a single cell with zoned earthfill perimeter embankments and will be lined with compacted soil liner overlain by a synthetic HDPE geomembrane. The total basin area will be approximately 45 Ha and is designed to accommodate 9.6 Mt of tailings. The TSF embankments will be constructed in stages to suit storage requirements with Stage 1 constructed initially to provide capacity for the first 12 months of operation and subsequent stages constructed using downstream raise construction methods to a final elevation of RL266.0 m (all throughput options). Staged embankment crest elevations will vary between throughput options.

The TSF basin area will be cleared, grubbed and topsoil stripped, and a 200 mm depth compacted soil liner will be constructed over the entire TSF basin area as either re-worked *in situ* material (assumed 30%) or imported Zone A (70%) material. This will be overlain by a 1.5 mm thick smooth HDPE geomembrane liner.

The TSF design incorporates an underdrainage system comprising a network of branch and collector drains. The underdrainage system drains by gravity to two collection sumps located at the lowest points in the cell at the southeast and southwest corners.

Supernatant water will be removed from the TSF via a submersible pump (designed by others) mounted in a decant tower located along the western embankment of the facility.

In order to mitigate seepage losses through the basin area, minimize the phreatic surface in the embankments, and increase the settled density of the deposited tailings, a number of seepage control and underdrainage collection features have been integrated into the design of each facility. The seepage control and underdrainage collection systems will consist of the following components:

- Cut-off trench
- Low permeability soil liner
- Synthetic HDPE geomembrane
- Basin underdrainage collection system
- Underdrainage collection sump
- Leak collection system
- Upstream toe drain.

Each cell of the Flotation TSF will operate with a series of three decant towers which will be constructed, operated and subsequently decommissioned to suit the staged development of the facility and of the tailings beaches in each cell. The CIL TSF will operate with a single decant tower throughout the life of the facility.

The decant towers will be raised as required with each embankment lift and will consist of the following components:

- An access causeway constructed of local coarse gravel material.
- A slotted concrete decant tower consisting of 1.8 m square slotted precast concrete sections surrounded by clean waste rock with a minimum size of 100 mm.

- A submersible pump with float control switches mounted on a lifting hoist.
- The decant pump in each tower will be raised on a regular basis to ensure that no tailings enters the pump intake.
- The tailings storage facilities have been designed to completely contain storm events during operation up to and including an annual exceedance probability (AEP) of 1 in 1,000 (Flotation TSF Cell 2) or 1 in 10,000 (Flotation TSF Cell 1 and CIL TSF) on top of the predicted maximum pond level under average climatic conditions, without the emergency spillways operating. Consequently, exceeding the storm storage capacity of the facilities at any stage of operation is unlikely. Regardless, in the event that the storage capacity of a facility is exceeded, water which cannot be stored within the facility will discharge via an engineered spillway.

1.10 Operating costs

The purpose of this operating cost estimate is to provide substantiated costs which can be utilized for a preliminary assessment of the viability of the Namdini Gold Project. The operating costs have been developed by:

- Lycopodium – Processing and General and Administration costs
- Golder – Mining costs
- Cardinal – Owners costs.

Operating costs have been determined for a mine that operates 24 hours per day, 365 days per year. The operating estimate is considered to have an accuracy of ±25%, is presented in United States dollars (US$) and is based on prices obtained during the first quarter of 2018 (1Q18). Study currency exchange rates were confirmed by Cardinal Resources.

The 9.5 Mtpa and 4.5 Mtpa options were factored from the 7.0 Mtpa option.

The operating costs have been compiled from a variety of sources, including the following:

- The LOM design mass recovery to flotation concentrate of 7.5% (this is based on recent testwork showing good gold recovery to concentrate at this mass pull)
- Flotation reagent consumption based on recent prefeasibility optimization testwork
- Leaching reagent consumption based on industry norms in anticipation of final testwork results
- Calculated reagent usage regimes for cyanide detoxification prior to testwork
- Modelling by OMC for crushing and grinding energy and consumables based on the final comminution testwork
- Typical industry data from equipment vendors
- Budget pricing or Lycopodium's database of prices for consumables
- Lycopodium's database of costs for similar sized operations
- Additional operating costs added by Cardinal to allow for the finer grind results
- Mining costs solicited from two of the largest in-country mining contractors. The estimated base mining cost has an applied incremental cost with depth, to account for increased haulage costs and the depth of mining increases in line with standard mining cost principles.

Operating Costs per tonne of ore processed (129.6 Mt of ore) are tabulated in (Table 141).

Owners Costs are tabulated in (Table 142).

Sustaining Capital Costs provided by consultants and Cardinal were compiled from a variety of sources and compared against existing and planned operations elsewhere in Ghana.

Sustaining Capital Costs which include rehabilitation and mine closure are tabulated in (Table 143).

1.11 Capital costs

The mining establishment cost was provided by in-country mining contractors. The process plant and infrastructure costs were estimated by Lycopodium. The costs for the TSF were provided by Knight Piésold. The capital costs include owner's project cost and contingency as calculated by Lycopodium.

The PFS capital cost estimate was completed to an accuracy of +30 %/-20 % for the 9.5 Mtpa option and was undertaken based on open pit mining from the existing March 2018 Mineral Resource. The proposed plant comprises primary crushing, SAG + ball grinding with re-crush (SABC), gravity recovery, flotation, concentrate regrind and CIL recovery. Three production throughputs were assessed by Cardinal, namely 9.5, 7.0 and 4.5 Mtpa. The 9.5 and 4.5 Mtpa throughput options were factored from the 7.0 Mtpa option (+20/-15% accuracy) and are therefore lower in accuracy at +30/-20%. A contingency factor of 5% was added to the 9.5 and 4.5 Mtpa options over and above the project contingency.

The factored estimates were established by assessing the correlation between cost and the process design criteria with factors being determined by discipline for all areas of the estimate.

Capital Costs are summarized below from Table 150.

Unit	9.5 Mtpa	7.0 Mtpa	4.5 Mtpa
US$ (M)	414	348	300

The capital cost in all three throughput scenarios reduced compared to those in the PEA. The main contributors to the reduction were:

- Flotation recovery mass pull was reduced from 15% to 7.5%, based on flotation testwork optimization. This reduction effectively reduced this section of the processing plant by 50% in terms of duty.
- Optimized plant layout, which reduced the plant footprint.
- Optimized steel structures, which ensured that all structures are fit for purpose.
- More accurately designed plant based off engineered quantities, which allowed a reduction in the growth allowance and contingency.

Overall plant layout and equipment sizing was prepared with sufficient detail to permit an assessment of the engineering quantities for the majority of the facilities for earthworks, concrete, steelwork and mechanical items. The layouts enabled preliminary estimates of quantities to be taken for all areas and for interconnecting items such as pipe racks.

Unit rates for labour and materials were derived from responses to BQRs sent to fabricators and contractors experienced in the scale and type of work in the region.

Budget pricing for equipment was obtained from reputable suppliers with the exception of low value items which were costed from Lycopodium's database of recent project costs.

For the accommodation camp, offices, workshops and similar items, appropriate budget pricing was obtained from reputable suppliers of similar prefabricated designs.

Knight Piésold provided the design and quantities of the following infrastructure items that were subsequently costed by Lycopodium.

The capital cost estimate includes:

- Direct costs of the Project development
- Indirect costs associated with the design, construction and commissioning of the new facilities
- Owner's cost associated with the management of the Project from design, engineering and construction up to the handover to operations and Project close-out
- Insurance and operating spares, first fills
- Costs associated with operational readiness and pre-production operations
- Growth allowance on quantity, pricing and unit rates variance
- Contingency on project scope definition and risks.

The material quantities and unit cost estimates were developed from engineering drawings, estimates and calculations at the level required for PFS and validated against estimates from similar sized projects.

Cardinal allowed for additional capital costs for a finer grind. These were factored costs obtained from Lycopodium.

1.12 Environmental

NEMAS Consult has undertaken a site reconnaissance visit and completed the scoping stage of the process in accordance with the Ghanaian Environmental Protection Agency procedures for the EIA. The NEMAS Scoping Study included preliminary field surveys, literature reviews and examination of appropriate legal and regularity frameworks.

In compliance with the above regulations, the Namdini Gold Project was registered with the Ghana EPA for environmental permitting. The EPA in response to the registration application by the Proponent in a letter dated 23 November 2016, indicated that the project which falls under Schedule 2 makes mandatory a full-scale Environmental Impact Assessment ("EIA") study and submission of Environmental Impact Statement ("EIS") to the EPA.

In compliance with directives by the EPA, a scoping report was prepared and submitted to the Agency on 22 June 2017, which also set out the Terms of Reference ("ToR") for the EIA and EIS (the "ESIA") study. The Scoping Study report highlighted the following issues among others: Project Description, Environmental and Social baseline conditions (mostly from secondary sources) and key environmental and social issues of impact and some preliminary proposed mitigation measures. The Scoping Study report also captured the various national and internal laws, policies and guidelines that shall be triggered. Additionally, the concerns of some key stakeholders consulted were captured in the report as consulted. Other key stakeholders that need to be consulted were also identified.

On receiving the Scoping Report the EPA posted a Scoping Report Notification on page 24 of the August 18, 2017 edition of the Ghanaian Times (a government-owned daily newspaper with a wide national circulation) requesting persons who have an interest, concern or special knowledge relating to the potential environmental effect of the proposed undertaking to contact or submit such concerns, etc., before the Environmental Impact Statement notification, to the Executive Director at its National Office in Accra and/or the Regional Director at its Regional office in Bolgatanga or the Managing Director of the proponent's company in Bolgatanga. The EPA also provided copies of the Scoping Report to the Talensi District Assembly in Tongo and to its Regional Office in Bolgatanga.

NEMAS are in the process of a detailed Environmental Impact Study which will be submitted to the Ghanaian EPA for approval.

1.13 Social

The PFS Environmental study was progressed by NEMAS, including active engagement of local and state regulatory bodies.

Cardinal has an excellent relationship with neighbouring stakeholders, including engagement with the local stakeholders. Granted mining leases cover all of the proposed mining and processing assets. There are no title claims pending.

Expatriate and skilled Ghanaians from outside the local community will be accommodated in a single status camp on site. An allowance for an accommodation camp to house up to 200 people has been made in the capital cost estimate.

The local workforce will be bussed from the neighbouring population centers.

Compensation agreements are being negotiated for the proposed mining operation.

1.14 Economic evaluation

Key economic statistics for the comparison of the 9.5 Mtpa, 7.0 Mtpa and 4.5 Mtpa option are included in Table 150.

The Starter Pit key estimated production comparison results are presented in Table 151. The Starter Pit includes the first 2.5 years of operation (24 Mt at 1.31 g/t for 1.06 Moz at 0.5 g/t cut off).

The LOM key estimated production comparison results for the 3 throughput options are presented in Table 152.

Figure 137 to Figure 140 illustrate the 9.5 Mtpa option Pre-Tax and Post-Tax sensitivities at a gold price of US$1,250/oz.

Based upon Life of Mine production and cost parameters, the Post-Tax NPV sensitivities are shown in Table 149 for the 9.5 Mtpa option.

The results of a study on higher throughput options is provided in Section 24.1.

A staged funding approach for the on-going development of the Namdini project is discussed in Section 22.6.

1.15 Next stages

Based on the positive PFS outcome, the Cardinal Board has approved the immediate progression to a Feasibility Study ("FS") of the Project to further define and support the case for full project funding and development.

Feasibility Study budget for Namdini Gold Project is summarised below from Table 153.

Item	Cost (US$ k)
Feasibility Study Value Engineering	**101**
Feasibility Study	**1,408**
Detailed design and long lead equipment procurement	**5,732**
Namdini drilling	**1,121**
Namdini geophysics	**13**
Total	**8,375**

2.0 INTRODUCTION

2.1 Scope of work

This report was prepared by Golder Associates Pty Ltd ("Golder") at the request of Cardinal Resources Limited ("Cardinal"). The purpose of this report is to provide Cardinal with an independent NI 43-101 compliant, Independent Technical Report and Pre-feasibility Study on the Namdini Gold Project in Ghana, Africa.

Cardinal is a gold exploration and development company based in Western Australian and has been a reporting issuer on the Australian Stock Exchange ("ASX") as ASX:CDV since August 2011 and on the Toronto Stock Exchange ("TSX") as TSX:CDV since July 2017. Cardinal's key assets are located in Ghana and include the Namdini, Bolgatanga, and Subranum Projects.

This Preliminary Feasibility Study ("Pre-feasibility Study" or "PFS") has defined the range of criteria under which the Namdini Gold Project in northeast Ghana may be considered potentially economic so that further development of the Project can be planned. The PFS was commissioned to evaluate the optimal extraction and processing rates for the project.

The Namdini Gold Project is based on development of a gold deposit in northern Ghana northern Ghana approximately 50 km south-east of the regional center of Bolgatanga, and close to the southern border of Burkina Faso. The Project area is located approximately 6 km south-east of the operating Shaanxi Mining Company Limited's (Shaanxi) underground gold mine.

This report was produced for Public Reporting under Canadian National Instrument ("NI") 43-101 in Canada (NI 43-101, 2014). It was prepared in accordance with the requirements of:

- disclosure and reporting requirements of the Toronto Stock Exchange ("TSX") as stipulated in TSX (2010);
- Canadian National Instrument 43-101, 'Standards of Disclosure for Mineral Projects', Form 43-101F1 and Companion Policy 43-101CP (NI 43-101, 2014); and
- Canadian Institute of Mining, Metallurgy and Petroleum Definition Standards (CIM, 2014).

The results of the PFS exclude Inferred Mineral Resources. Inferred Mineral Resources are considered too speculative geologically to have economic considerations applied to them that would enable them to be categorized as Ore Reserves, so any Inferred Resources within the pit shell are considered 'waste' material in this PFS.

Golder estimated the Ore Reserve in accordance with the JORC Code (2012). The term 'Ore Reserve' is synonymous with the term 'Mineral Reserve' as used by Canadian National Instrument 43-101 'Standards of Disclosure for Mineral Projects' (NI 43-101, 2014). The JORC Code (2012) is defined as an 'acceptable foreign code' under NI 43-101.

This PFS is preliminary in nature and there is no certainty that the conclusions of this PFS will be realized.

Inferred Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Ore Reserves. Inferred Mineral Resources have not been used in determining the potential viability of the Namdini Gold Project.

2.2 Namdini Gold Project concept

The Namdini Gold Project ("the Project" or "Namdini") is located in the Upper East Region of Ghana approximately 50 km south-east of the regional center of Bolgatanga.

The Project is conceived as a large-scale open pit, mined for gold by conventional drill and blast, dug by face-shovel configured excavators feeding 150 t trucks. The ore will be fed to a large conventional CIL process plant with a sulfide flotation circuit to enhance gold recovery. Gold bullion will be produced on site for sale into the international market.

The Namdini Gold Project will operate in a safe, responsible and technically efficient way to add benefits to all stakeholders including Ghana, the owners, shareholders, employees, and local communities.

2.3 Statement of independence

Golder is an independent consulting company contracted by Cardinal to carry out this PFS. Neither Golder, nor the authors of this report, has or has had previously, any material interest in Cardinal or the mineral properties in which Cardinal has an interest. Golder's relationship with Cardinal is solely one of professional association between client and independent consultant.

This report was prepared in return for professional fees based upon agreed commercial rates and the payment of these fees is not contingent on the results of this report. No member or employee of Golder is, or is intended to be, a director, officer or other direct employee of Cardinal.

In the preparation of this Independent Technical Report Golder has used information provided by Cardinal and other experts. Golder has verified this information, making due enquiry of all material issues that are required in order to comply with NI 43-101 requirements.

There is an on-going consultancy agreement between Golder and Cardinal regarding Golder conducting further work for Cardinal as this project progresses to the Pre-feasibility and Feasibility Study stages.

The positive result of this PFS is such that it is likely that the Namdini Gold Project will continue to progress towards development of a large open pit mine and process plant, on-going drilling and assessment, while meeting all required permits and approvals, to add significant value to the Upper East Region of Ghana.

2.4 Risks and forward-looking statements

The business of mining and mineral exploration, development and production by its nature has significant operational risks. The business depends upon, amongst other things, successful prospecting programmes and competent management. Profitability and asset values can be affected by unforeseen changes in operating circumstances and by technical issues.

Factors such as political and industrial disruption, currency fluctuation and interest rates could have an impact on the proposed project's future operations, and potential revenue streams can also be affected by these factors. The majority of these factors are, and will be, beyond the control of Cardinal or any other operating entity.

This Independent Technical Report contains forward-looking statements. These forward-looking statements are based on the opinions and estimates of Cardinal, Golder and other specialist consultants at the date the statements were made. The statements are subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those anticipated in the forward-looking statements. Factors that could cause such differences include changes in world gold markets, equity markets, costs and supply of materials relevant to the projects, and changes to regulations affecting them.

Although Golder believes the expectations reflected in its forward-looking statements to be reasonable, Golder does not guarantee future results, levels of activity, performance or achievements.

2.5 Use of the term 'ore' in this PFS

The Canadian National Instrument Companion Policy 43-101 (Section 2.3) indicates that in the context of Mineral Resource estimates, the term 'ore' implies technical feasibility and economic viability that should only be attributed to 'Ore Reserves'. In compliance with Section 2.3 of the Companion Policy, the term ore is not used in the Mineral Resource context of this PFS.

The term ore is used in the mining and processing sections of this PFS in a generic way to describe the 'mineable' part of the resource estimate that will be extracted from the mine and fed to the process plant. Where appropriate this is referred to as the 'Ore Reserve' after investigation and application of all relevant Modifying Factors as discussed in Section 15.0, in conformance with the definitions in CIM (2014).

Golder estimated the Ore Reserve in accordance with the JORC Code (2012). The term 'Ore Reserve' is synonymous with the term 'Mineral Reserve' as used by Canadian National Instrument 43-101 'Standards of Disclosure for Mineral Projects' (NI 43-101, 2014). The JORC Code (2012) is defined as an 'acceptable foreign code' under NI 43-101.

3.0 RELIANCE ON OTHER EXPERTS

3.1 Introduction

This report was prepared by Golder for Cardinal as a PFS for Public Reporting. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to Golder at the time of preparation of this report
- Assumptions, conditions, and qualifications discussed in this report
- Data, reports, and other information supplied by Cardinal and other third parties, as documented and referenced in this PFS Report.

For the purpose of this report Golder has relied on ownership information and other local knowledge provided by Cardinal. Golder has no independent information regarding property title or mineral rights for the Namdini Gold Project and expresses no opinion as to the ownership status of the property.

Except for the purposes legislated under Canadian or other securities laws, any use of this report by any third party is at that party's sole risk.

3.2 The study team

The major components of this PFS comprise: resource modelling based on available data, preliminary mine design, metallurgical testwork, preliminary process design and process plant cost estimation, environmental assessment, preliminary financial analysis and other supporting studies on geology, hydrogeology, hydrology, rock mechanics for pit slope design and geotechnical engineering. This work tested the merits of proceeding towards a Pre-feasibility Study and determined criteria to be evaluated in such a study.

In addition, the PFS is based on the specialist consultant studies identified in Table 1 and information from studies conducted on the behalf of the project owner, by independent specialist consultants.

Table 1:Components for the Namdini PFS

PFS Component	Specialist Study	Consulting Company
Mineral Resource	Mineral Resource estimation	MPR Geological Consultants Pty Ltd
Ore Reserve Mine Design	Pit optimization Mine design Mine scheduling	Golder Associates Pty Ltd
Process Design	Oversee laboratory metallurgical recovery programme and estimates. Process plant design Project infrastructure requirements Capital and operating cost estimates	Lycopodium Ltd
Tailings Storage Facility	TSF design and costing	Knight Piésold Pty Ltd
Environment	Environmental assessment	Nemas Consult Ltd
Other Information	Hydrology Hydrogeology	Golder Associates Africa (Pty) Ltd
Other Information	Rock mechanics Geotechnical engineering	Golder Associates Ghana Limited

Cardinal commenced its PFS in October 2017 to further advance the Namdini Gold Project. This consisted of an Owner's Team and continuation with previously selected and newly selected consultants to assist with the phased development of the Namdini Gold Project. The consultants and their roles are tabulated below (Table 2).

Table 2: Study team

Company	Role
Golder Associates Pty Ltd	Study Managers. Mine planning and Whittle Optimization. Pit design and mine scheduling. Geotechnical, Hydrology and Hydrogeology engineering. Responsible for the compilation of the NI 43-101 reports
Lycopodium Limited	Process plant and associated infrastructure design. Capital and Operating cost estimation and input into the NI 43-101 report.
Orway Minerals Consultants	Comminution data analysis Crushing and grinding circuit option study
ALS Laboratory (Perth)	Metallurgical testwork to support the process design and criteria
Knight Piésold Consulting	Tailings Storage Facility and associated infrastructure design
Independent Metallurgical Operations Pty Ltd ("IMO")	Metallurgical testwork analysis and process flowsheet development
MPR Geological Consultants Pty Ltd	Mineral Resource Modelling of the Namdini Deposit
Orefind Pty Ltd	Geology and deposit structural genesis
Intermine Engineering Consultants Pty Ltd	Mine schedule optimization
NEMAS Consult Pty Ltd	Environmental Impact Assessment Study

3.3 Sources of information

Sources used in the compilation of this PFS Report include:

- Namdini Gold Project Mineral Resource Estimate Study Report. Prepared for Cardinal Resources Limited by ERGM Consulting Pty Ltd (Gossage, 2017), 17 February for which the estimate was released publicly to the ASX in Cardinal (2016).
- Technical Report on the Namdini Gold Project, Ghana, West Africa, April 5, by Roscoe Postle Associates Inc (RPA, 2017) describing the Mineral Resource estimated at April 5 and released publicly to the TSX under NI 43-101.
- Technical reports on Mineral Resource estimation for the Namdini Gold Project, Ghana by MPR Geological Consultants Pty Ltd, released publicly to the TSX under NI 43-101 (MPR, 2017; 2018).
- The previous Preliminary Economic Assessment on the Namdini Gold Project dated 5 February 2018 released publicly to the TSX under NI 43-101 (Golder, 2018).
- Report sections by Lycopodium (2018) and Knight Piésold (2018) describing the metallurgy, process plant, plant design and costings see respectively Sections 12.2,17.0,18.0 and 21.0.
- Report on the environmental assessment by Nemas Consult Ltd, Ghana (NEMAS, 2018) see Section 20.0.

- Report on the hydrology studies (Golder Africa, 2018a) by Johan Jordaan and Trevor Coleman.
- Report on the hydrogeological studies (Golder Africa, 2018b) by Jennifer Pretorius and Alan Puhalovich.
- Report on the geotechnical (rock mechanics) studies supporting the preliminary Namdini Pit design (Golder Ghana, 2018), by Reginald Hammah and Timothy Martin.

Golder has reviewed the independent technical reports and is reasonably assured, having made due enquiry, that these reports are based on accepted international industry practice and fairly represent the Namdini Gold Project.

3.4 Personal inspection

The following independent consultants visited the Project site in the course of developing their opinions:

- Nicolas Johnson of MPR Geological Consultants Pty Ltd (“MPR”) in Perth visited the Namdini Gold Project site on 11 to 14 January 2017.
- Dr Frank Anim of Nemas Consult Ltd (“NEMAS”) in Accra visited the site on 1 to 4 May 2017.
- Dr Reginald Hammah of Golder Associates Ghana Limited (“Golder Ghana”) in Accra visited the site on 29 to 30 May 2017.
- Priscilla Nettey, Brendan Atarigiya and Alex Amaniampong of Golder did geotechnical investigations on site from 15 August to 17 October 2017.
- Glenn Turnbull, Principal Mining Engineer of Golder Associates Pty Ltd (“Golder”) in Perth visited the Namdini Gold Project site from 11 to 15 December, 2017.
- Simon Smith, Civil Engineer of Knight Piésold Pty Ltd (“KP”) in Perth visited the Namdini Gold Project from 11 to 15 December, 2017.
- Jennifer Pretorius of Golder Associates Africa (Pty) Ltd (“Golder Africa”) in Johannesburg visited the Namdini Gold Project site in October and November 2017 with Izak Marais, and assisted by Benjamin Asiedu (Environmental specialist), Dorcas Adjei-Sasu (Water Resources Engineer) and Edna Addai (Civil Engineering Intern).
- Givarn Singh (Water Resources Engineer) visited site for one week from 3 September 2018.

3.5 QPs and experts relied upon

The Qualified Persons (“QPs”) identified as the authors responsible for this PFS Technical Report have specifically relied on other experts as defined in NI 43-101 (Section 3.0). These other experts and the Sections for which they take responsibility are identified in Table 3.

Table 3: Qualified Persons for this PFS

Company	Report section	PFS Component	Name	Role
MPR	Sections 7 to 12 and 14.1 to 14.13	The data and basis for the Mineral Resource estimate	Nicolas Johnson	QP
Golder	All other Sections	The costings and basis for the Ore Reserve estimate (the Modifying Factors)	Glenn Turnbull	QP
IMO	Sections 13 and 17.	Metallurgy, process design and process plant	Daryl Evans	QP

Other personnel involved in preparing this PFS included:

- Cardinal: Richard Bray; Bruce Lilford
- Golder: Dr Sia Khosrowshahi, Henry Dillon, Richard Goldsmith, David Reid, Alan Puhalovich
- Golder Africa: Trevor Coleman, Johan Jordaan, Jennifer Pretorius
- Golder Ghana: Dr Reginald Hammah, Timothy Martin
- NEMAS: Dr Frank Anim, Emmanuel Acquah, Dr Kwesi Boadi, A. Adu-Nyarko, Dr J. Adomako, Dr Samuel Nkumbaan, Dr Francis Nunoo.

QP certificates for the Public Reporting of this PFS are provided at the end of this Report.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Location of the Namdini Gold Project

Namdini is approximately 50 km southeast of the regional town of Bolgatanga in northern Ghana and around 60 kilometres south of the Burkina Faso border (Figure 1). Namdini lies within the Nangodi Greenstone Belt, one of a series of southwest–northeast trending granite-greenstone belts that host significant gold mineralization in Ghana and Burkina Faso.

The Talensi district which was part of the Talensi-Nabdam district became a separate district in 2012, through Legislative Instrument LI 2110, with Tongo as the capital. The District, which is one of the thirteen Municipalities and Districts in the Upper East Region, is bordered to the north by the Bolgatanga Municipality, to the south by West and East Mamprusi districts (both in the Northern Region), to the west by Kassena-Nankana district, and to the east by the Bawku West and Nabdam districts. The District lies between latitude 10° 15' and 10° 60' North of the equator and longitude 0° 31' and 1° 05' West of the Greenwich meridian and covers a land area of 838.4 km^2 with a total population of 81,194 which constitute 7.8% of the regional population. It is made up of 96 towns and villages with a settlement pattern which is predominantly rural (NEMAS, 2017).

The District context of the Lease boundary is shown in Figure 2.

The project is roughly six kilometres southeast of the operating Shaanxi underground gold mine. Namdini lies within the Nangodi Greenstone Belt, one of a series of southwest-northeast trending granite-greenstone belts that host significant gold mineralization in Ghana and Burkina Faso.

The topographic relief within the project area is generally flat with gently undulating terrain, rising to the south where the area is overlain by sediments. Elevation varies from 175 to 250 m above sea level with average elevation at approximately 190 m. Physiography is primarily savanna grassland characterized by short scattered drought-resistant trees, scrub, and grass that is seasonally burned by bushfires or scorched by the sun during the long dry season.

Figure 1: Location of Namdini Gold Project in northern Ghana (source: Cardinal 2018)

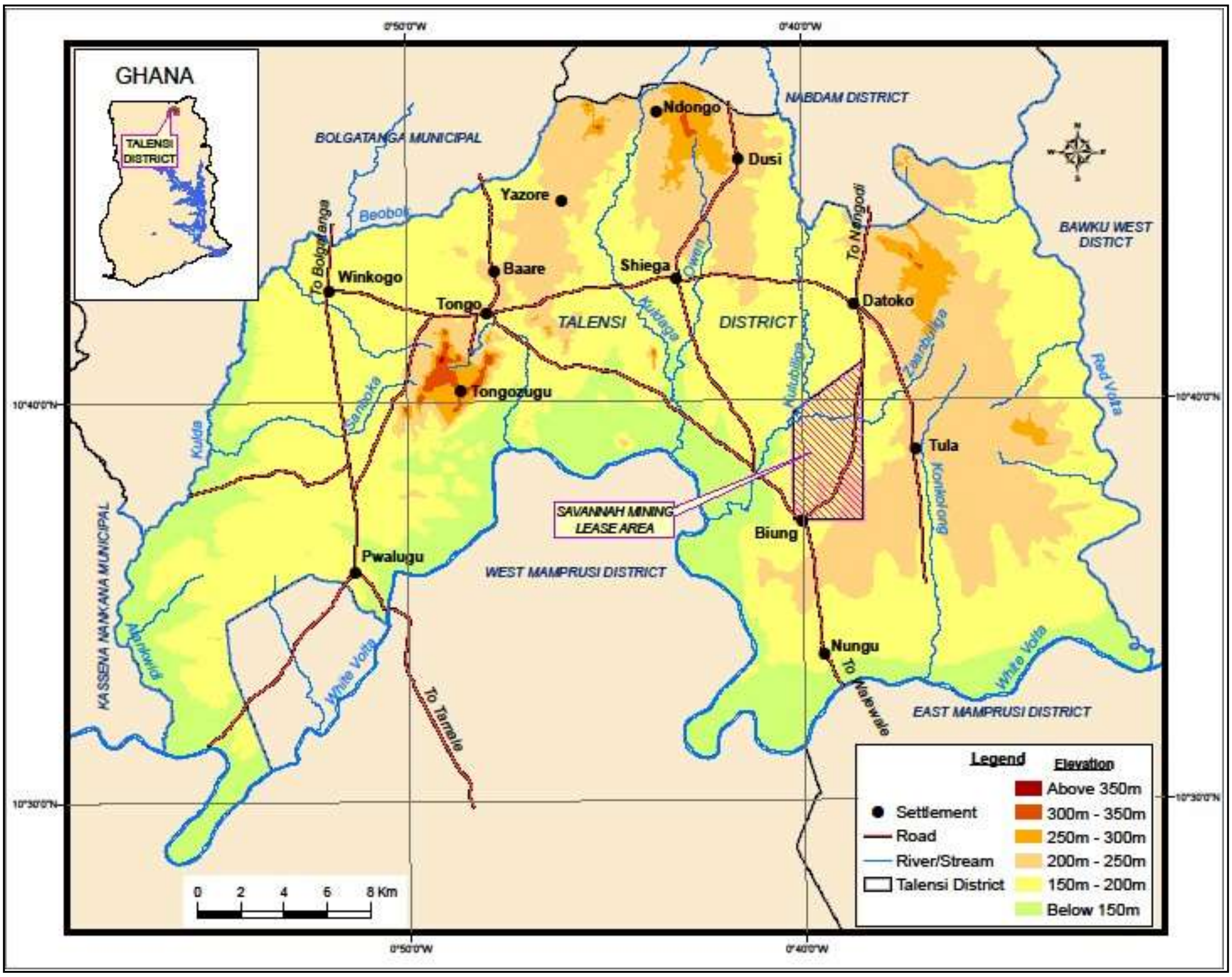


Figure 2: District map of Talensi showing the location of the Namdini Mining Lease (Source: NEMAS, 2017)

4.2 Tenement description

The description of mineral tenure and project ownership is from MPR (2018) based on Golder (2018) and previous work.

The application by Savannah for a Large-Scale Mining Licence over an area of approximately 19.54 km^2 in the Upper East Region of Ghana covering the Namdini Gold Project was granted by the Minister of Lands and Natural Resources of Ghana under the regulations of the Minerals and Mining Act of Ghana 2006 (Act 703).

During the December 2017 quarter, a Large-Scale Mining Licence covering the Namdini Mining Lease was assigned to Cardinal Namdini Mining Limited ("Cardinal Namdini"), a wholly owned subsidiary of Cardinal, by the Minister of Lands and Natural Resources

The Large-Scale Mining Licence, which covers 19.54 km^2 in the Dakoto area of the Talensi District Assembly in the Upper East Region of Ghana is for an initial period of 15 years and is renewable for a further thirty (30) years.

Table 4 lists the coordinate extents of the Namdini Gold Project lease. Approximate central coordinates of the deposit are 756400.0 m N, 1177050.0 m E in WGS (UTM) 84 Zone 30N projection or 10°38' 21" N Longitude and 0°39'.23" W Latitude.

Table 4: Coordinates of the Namdini Gold Project Lease

Corner	Longitude	Latitude
Top Left	10° 39' 42" N	0° 40' 15" W
Top Right	10° 40' 57" N	0° 38 30" W
Bottom Right	10° 37' 00" N	0° 38 30" W
Bottom Left	10° 36' 60" N	0° 40' 15" W

The lease covers approximately 19.54 km^2. Golder has sighted the stamped Mining Lease boundary survey document. Figure 3 shows the lease superimposed on the regional physiography image.

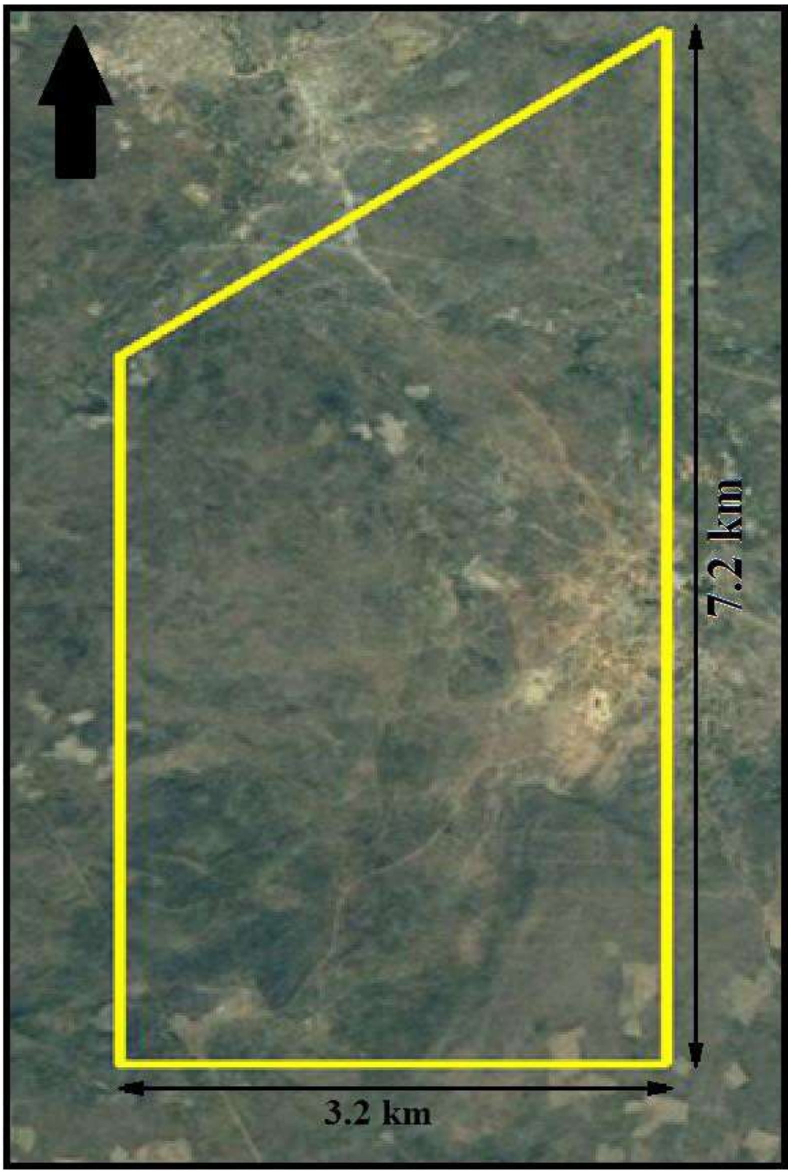


Figure 3: Namdini Gold Project lease boundary (source: Cardinal 2018)

4.3 Ownership

The following description of mineral tenure and property ownership for the Namdini deposit is derived from information in Golder (2018) and information supplied by Cardinal. Confirmation of the status of the mineral tenure for the Namdini deposit and the status of Cardinal's interest in the Namdini Gold Project is provided by Kuenyehia (2017).

During the December 2017 quarter, a Large-Scale Mining Licence covering the Namdini Mining Lease was assigned to Cardinal Namdini Mining Limited, a wholly owned subsidiary of Cardinal by the Minister of Lands and Natural Resources under the Ghanaian Minerals and Mining Act 2006 (Act 703). The Large-Scale Mining Licence, which covers 19.54 km^2 in the Dakoto area of the Talensi District Assembly in Upper East Region of Ghana evidenced by a Mining Lease, is for an initial period of 15 years and is renewable for up to a further 30 years.

Cardinal is obligated to abide by the rules of the Minerals and Mining Act 2006 (Act 703) over the area covered by its Mining Lease. The Minerals and Mining Act 2006 (Act 703) was an Act of Parliament to consolidate the law relating to minerals and mining. This Act provides rules relative to rights regarding minerals and mining operations in Ghana.

Cardinal completed and filed an EIS Scoping Study with the Environmental Protection Agency (EPA). In accordance with EPA Regulations 15(1b) and (1c) of the Environmental Assessment Regulations, 1999 (LI 1652) and Ghana's Environmental Impact Assessment Procedures, the EPA issued a public notification on the proposed Namdini Gold Project. Cardinal will submit to the Minerals Commission an updated EIS and an application for an Operating Permit for the project scale envisioned in this PFS.

The royalty rate for the Large-Scale Mining Licence is 4% of Net Smelter Return (NSR) for the first 50,000 ounces of gold produced from each Small-Scale Licence within the Large-Scale Mining Licence where production is undertaken and 2% NSR for production in excess of 50,000 ounces from each small-scale licence.

Cardinal is not aware of any specific environmental liabilities on the property. Cardinal has all required permits to conduct the proposed work on the property. The QPs and Cardinal are not aware of any other significant factors or risks that may affect access, title, or the right or ability to perform on-going work programs on the Namdini Gold Project.

4.4 Royalties, payments or encumbrances

An assumption of 5% was made to account for all royalties in the Mineral Resource estimation (Section 13.1), the Mining cost estimates (Section 16.0) and the Economic Analysis (Section 22.0).

4.5 Environmental studies

Savannah Mining Ghana Limited (Savannah) on behalf of Cardinal has completed an EIS Scoping Report for Namdini and has filed it with the Environmental Protection Agency (EPA). In accordance with EPA Regulations 15(1b) and (1c) of the Environmental Assessment Regulations, 1999 (LI 1652) and Ghana's Environmental Impact Assessment (EIA) Procedures, the Environmental Protection Agency (EPA) issued a public notification on the proposed Namdini Gold Mining Project. Cardinal will submit to the Minerals Commission an updated EIS and an application for an Operating Permit for the Project scale envisioned in this PFS.

The Qualified Persons signing this report are not aware of any specific environmental liabilities on the property. Further information regarding the proposed Environmental Impact Assessment study is provided in Section 20.0.

4.6 Permits and other factors

The Qualified Persons signing this report have been advised that Cardinal has all required permits to conduct the proposed work on the property.

The authors of this report (Cardinal, Golder and the other signatory QPs) are not aware of any specific environmental liabilities on the property. They consider that Cardinal has all required permits to conduct the proposed work on the property. They are not aware of any other significant factors and risks that may affect access, title, or the right or ability to perform on-going work programs on the Project.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Project layout

Numerous historical trenches and adits, as well as organized artisanal gold mining sites, are located throughout the property although only approximately 5% of the permit area has been affected by these activities. Artisanal miners extract gold from the saprolite horizon, but also sink shafts as deep as 20 m to recover gold from quartz veins.

The site layout is shown in Figure 4. The project consists of the following:

- Planned open pit
- Tailings Storage Facility ("TSF") located to the southwest of the open pit
- Waste rock dump located to the northwest of the pit
- Process Plant and Run Of Mine stockpile area ("ROM Pad") to the north of the pit.

The elevations on site are also shown in Figure 5. The topography on the site is generally flat except for the highland to the southeast of the open pit, which slopes steeply towards the proposed TSF.

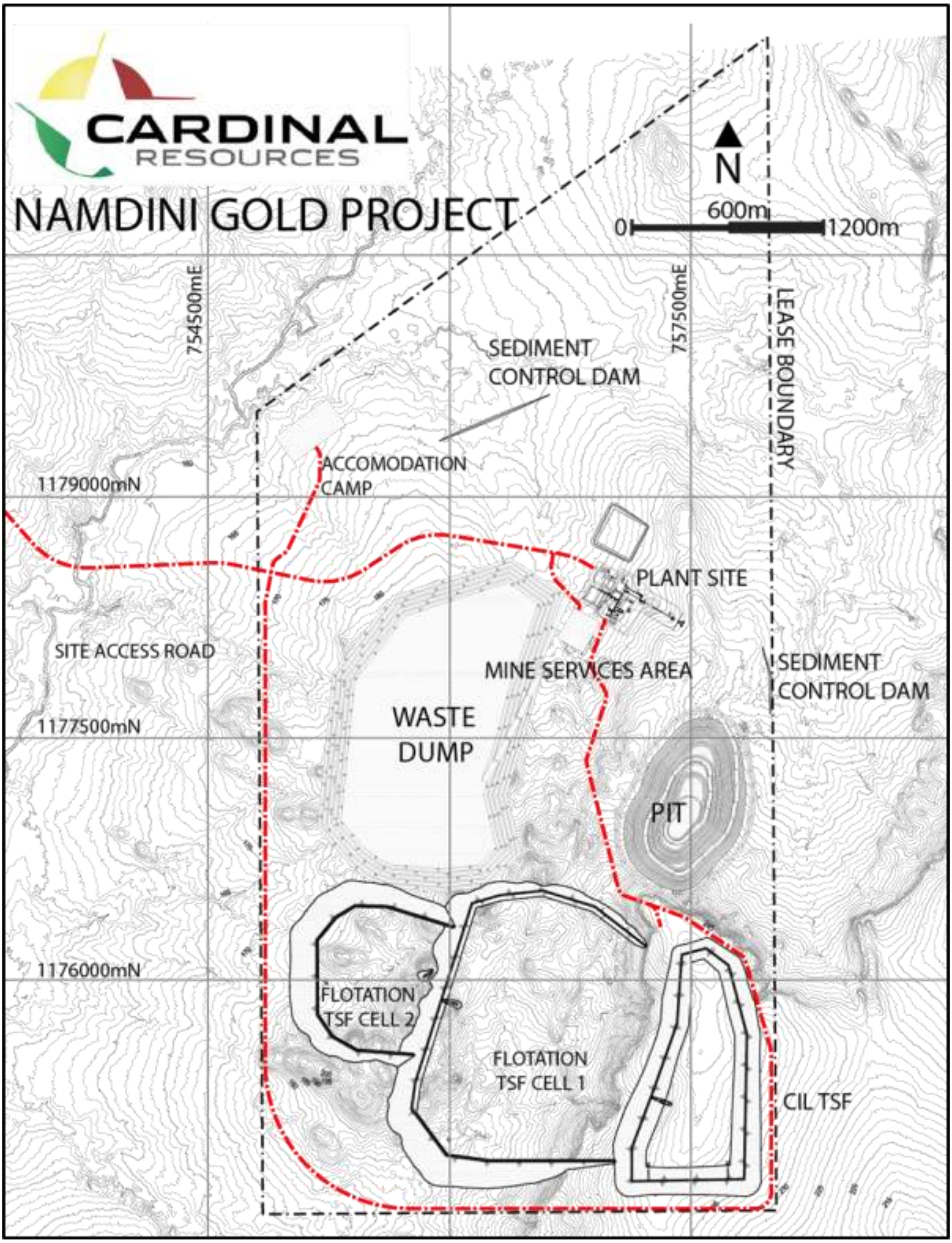


Figure 4: Namdini Gold Project infrastructure layout (source: Cardinal 2018)

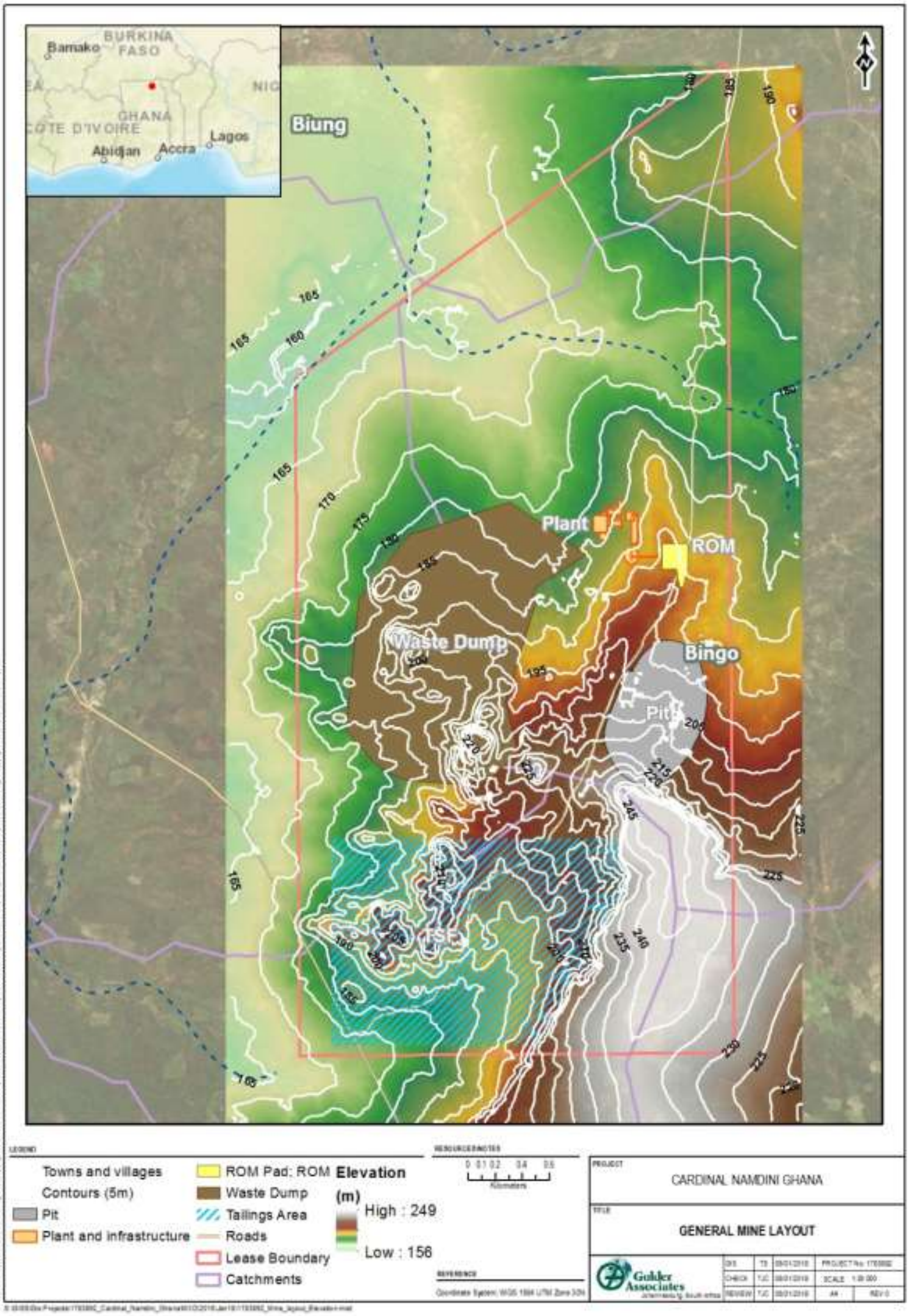


Figure 5: Project site layout and site topography (source: Golder 2018)

5.2 Access, infrastructure and population

The following descriptions of accessibility, local resources and infrastructure are derived from RPA (2017) and the other references cited in this section.

Namdini is approximately 50 km southeast of Bolgatanga, the capital of the Bolgatanga Municipal District, within the Talensi District in the Upper East Region of northern Ghana. This District is close to the southern border of Burkina Faso. The property is readily accessible from Bolgatanga along paved highway followed by 15 km of well-travelled gravel roads. Access during the rainy season is slower due to waterlogged roads, but the main access roads are passable all year round.

The nearest airport is at Tamale, 160 km south of Bolgatanga by paved road on National Highway N10. Tamale is serviced by daily scheduled commercial flights from the capital Accra. Travel time from Accra to the Namdini Gold Project is approximately four hours using a combination of air and road travel or 14 hours solely by road. Accra is serviced by direct flights to the United Kingdom, Europe, South Africa, the Middle East and USA.

The Project site is located approximately 6 km southeast of the operating Shaanxi Mining Company Limited's ("Shaanxi") underground gold mine (RPA, 2017).

For exploration and resource definition activities to date, personnel have generally commuted daily from Bolgatanga where Cardinal has an Exploration Office. Fuel supply for the drill rigs is provided by diesel tankers. Fresh water is taken from a bore located on the Namdini Gold Project site. Cardinal maintains trails on the Namdini Gold Project site to facilitate drilling and other exploration activities.

Evaluation of the project is at an early stage and infrastructure to support mining has not yet been established. For future development it may be necessary to build all-weather access roads, possibly bridges, power, water and other infrastructure.

Cardinal's surface rights cover areas sufficient for potential process plant sites, tailings storage areas, and waste disposal areas. The national Ghana power grid 161 kV above-ground transmission line runs approximately 30 km west of the Namdini Gold Project. The Namdini Gold Project is located approximately six kilometres southeast of the operating Shaanxi underground gold mine which is supplied by grid power.

Ghana has a long mining history and has experienced technical personnel including geologists and engineers. Exploration and mining supplies are readily available within Ghana. In 2002 the Upper East Ghana region had a total population of approximately 964,500. In 2012 Bolgatanga had a population of 66,685. There are two small settlements in the vicinity of the Namdini Gold Project which generally rely on subsistence farming, artisanal mining, and harvesting of wood. There is a significant local labour pool available for recruitment for any envisioned mining operation.

Numerous historical trenches and adits, as well as organized artisanal gold mining sites, are located throughout the property and approximately 5% of the permit area was affected by these activities. Artisanal miners extract gold from the saprolite horizon, but also sink shafts as deep as 20 m to recover gold from quartz veins. These artisanal workings can result in pits and subsidence which pose hazards for people and animals.

5.3 Physiography

5.3.1 Topography

The hydrological study summarized in Section 24.1 (Golder Africa, 2018a) noted that three broad physiographic regions can be distinguished in northern Ghana: Savannah high plains covering all of Upper East and Upper West regions and the western part of the Northern Region, Voltaian sedimentary basin covering most of the remainder of the Northern Region, and Scarps bordering the Voltaian basin within the Northern Region.

Topography of the Namdini area is generally flat to gently undulating (Figure 6) and rises to the south where the area is overlain by sediments. Elevation varies from 175 to 250 metres above sea level, averaging approximately 190 m.

The topography of the district is characterized by scattered rock-outcrops and undulating gentle slopes of the upland and lowland areas. A view of the Project site looking north is provided in Figure 7.

A site layout was developed based on supplied site topography and satellite images of the area. The topography of the site area is gently undulating with an average elevation of approximately 190 m and overall fall towards the west. A plateau is located directly to the south of the open pit which is bounded to the north and east by an escarpment with an average elevation of approximately 240 m. The site area is lightly to moderately vegetated with grass and small trees up to 3 to 4 m in height. The topography to the north and west of the pit footprint is relatively flat.

The site area consists of small mounds and outcrops and generally drains to the west. The highland to the southeast of the mining area slopes steeply towards the proposed TSF site.

Figure 6: Namdini Gold Project site showing terrain facing north in 2017 (source: Lycopodium)

Figure 7: View of Namdini Gold Project site facing north in 2017 (source: Lycopodium)

5.3.2 Drainage

There are four main sub-basins within the Volta catchment, consisting of the Black Volta, White Volta, Oti-Pendjari and Lower Volta rivers. The sub-basins, excluding the Lower Volta system, flow to the Volta Lake which was created by the construction of Aksombo Dam in 1964.

The White Volta River first flows south on entering Ghana, turns west to be joined by the Red Volta River, continues westwards through the Upper East Region and then turns south, where it is joined by several tributaries, including the Kulpawn/Sissili and Nasia Rivers.

The Project Area falls within the Volta River catchment and is located approximately 7 km north of the White Volta River. The main rivers that drain through the district are the White and Red Volta and their tributaries (Figure 8).

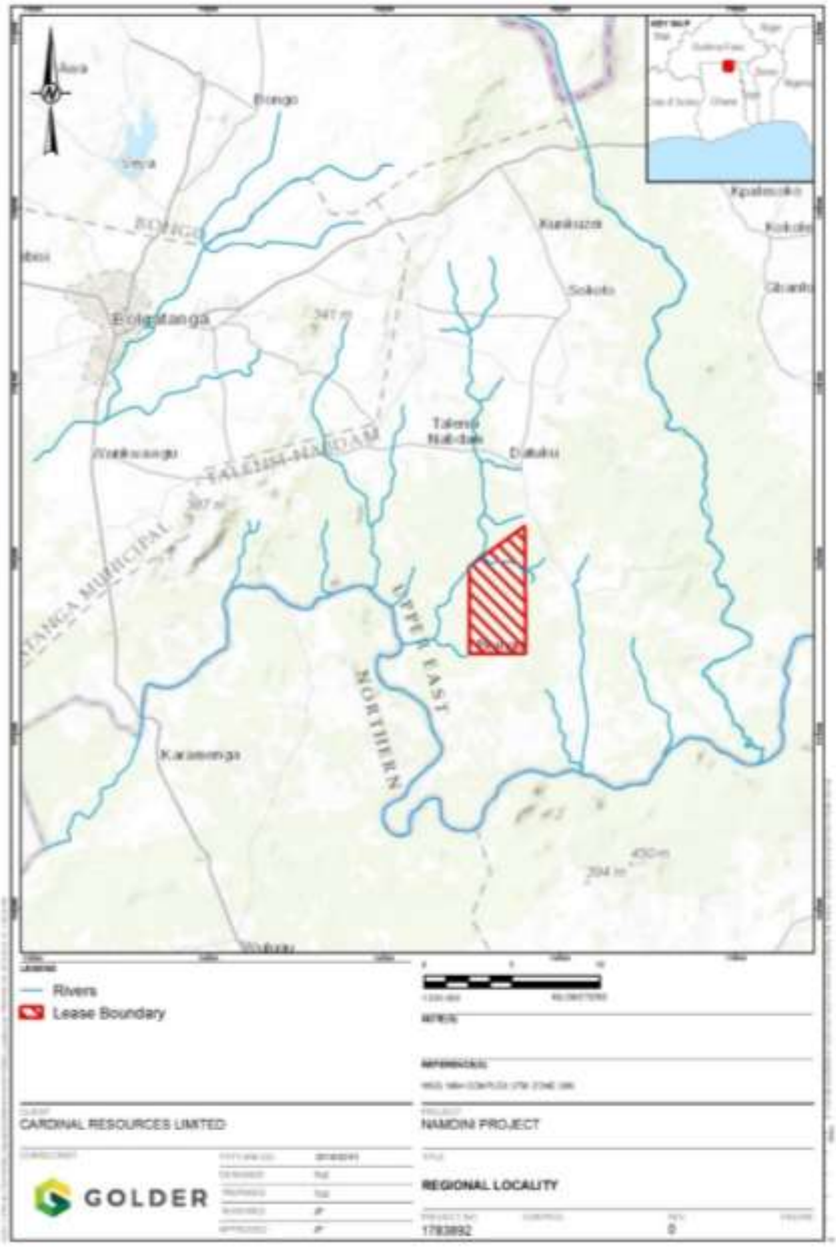


Figure 8: Drainage and rivers around the Namdini Gold Project site (source: Golder 2018)

The Project site drains to the west towards the White Volta River 6 km west of the site, which eventually forms part of the Volta River. The Red Volta is east of the Project site (Figure 9). A small non-perennial stream, the Zoan Buliga, passes the northern boundary of the site before joining the White Volta. This stream and the White Volta are being considered as a possible source of raw water to supply the mine.

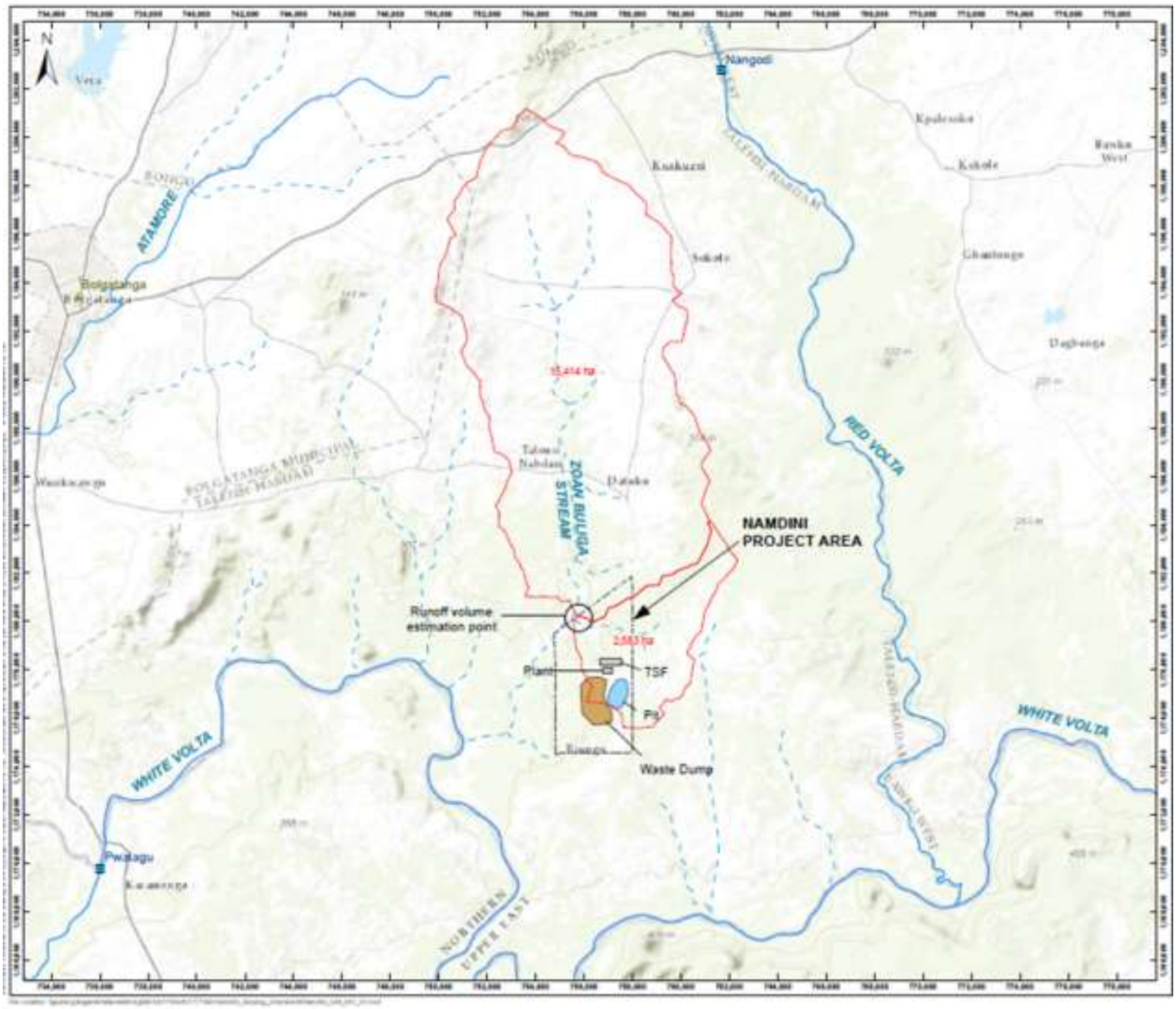


Figure 9: Location of the Namdini Gold Project Site in relation to rivers (source: Golder 2018)

5.3.3 Land use

The landscape has promoted small-scale agriculture in the district.

The water resources of the White Volta River Basin contribute substantially to the economic livelihood of the people living in the basin. Water is used for a variety of purposes in the domestic, agriculture and industrial sectors. Agriculture (including animal husbandry), fishery, hunting and forestry together constitute the main economic activity in the basin, particularly in the rural areas and provide occupation and employment for a vast majority of the people. Small-scale gold mining activities (galamsey) and stone quarrying are also common in some parts of the basin, particularly in the Upper East Region.

Land classified as cultivated is usually used for small-scale rain-fed agriculture in the form of compound or bush farming. Compound farms are located near the farmer's homes and crops grown usually include maize, vegetables and tobacco. In bush farms, which are generally located within 10 km of the community, a mixture of cereals/vegetables crops notably including maize, sorghum, millet, rice, yam, cassava and groundnuts are generally cultivated. Irrigated agriculture is also practiced in northern Ghana, though the irrigated land area only occupied about 30 km^2 as of 2000.

5.4 Vegetation

The area is primarily savannah grassland characterized by short scattered drought-resistant trees, scattered scrub, and grass. The most common trees are the Sheanut, Dawadawa, and Baobab (Section 20.2.1.2).

5.5 Climate

5.5.1 Annual range of climate variables

The hydrological study summarized in Section 24.1 (Golder Africa, 2018a) noted that average annual values for selected climate variables in Upper Eastern Region, as measured at the weather station in Navrongo, are:

- Rainfall (mm/y) 987
- Potential evap. (mm/y) 1,723
- Ave. temp. (°C) 28.9
- Min. temp. (°C) 19.3
- Max. temp. (°C) 39.3
- Min. rel. humidity (%) 40.3
- Max. rel. humidity (%) 68.8
- Sun hours/day (h) 7.8
- Wind speed (m/s) 0.91.

5.5.2 Temperature

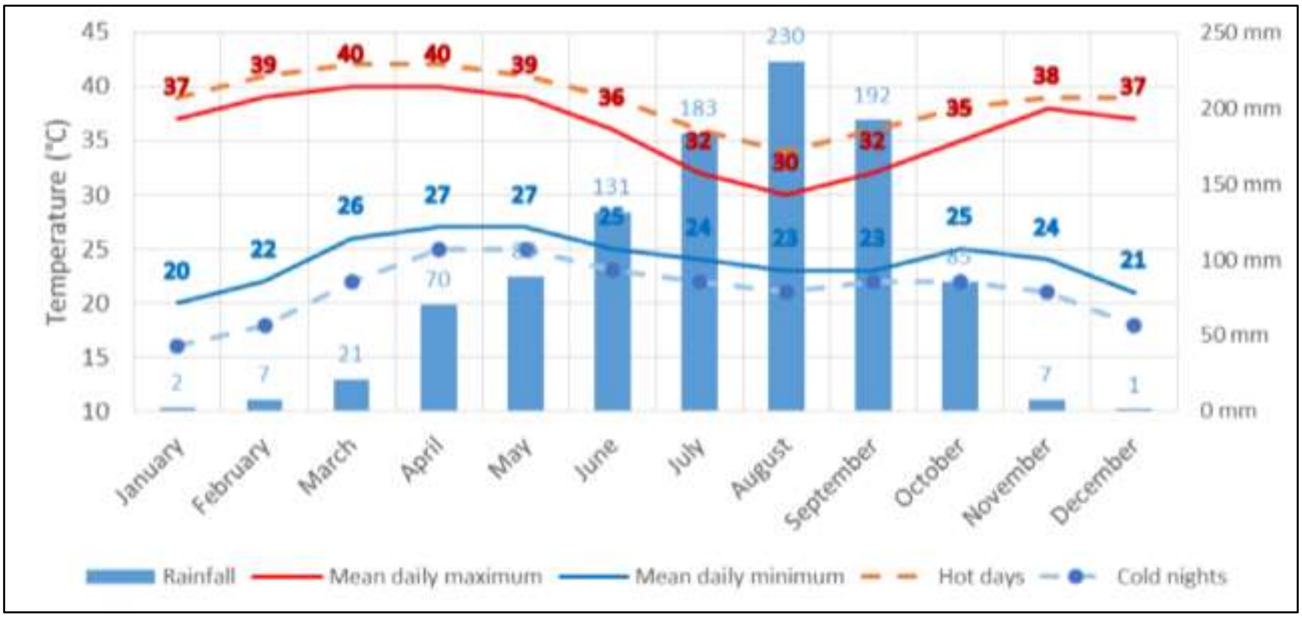
The mean annual temperature in Bolgatanga is 28.3°C. There is a long dry season from November to late February, characterized by cool, dry and dusty Harmattan winds. Temperatures during this period can be as low as 15°C at night and as high as 40°C during the day.

The rainy season is between May and October when the rainfall is erratic spatially and in duration. Mean annual rainfall varies between 800 and 1100 mm. Temperatures during this period can be as low as 20°C at night, but can reach more than 35°C during the day.

Humidity is very low during the dry season, but can be as high as 85% during the wet season.

Weather conditions have not significantly affected Cardinal's exploration activities, nor would they be expected to materially affect any potential mining operations.

A hydrological analysis of the Bolgatanga Region and the Namdini Gold Project Area was conducted by Golder Africa (2018a). The report used temperature data, compiled for stations within the Upper East Region of Ghana, to assess temperature ranges in the region. The data period covered by the observation stations ranged from 1954 to 2014. Monthly average minimum and maximum daily temperatures from the study are provided in Figure 10.



Also shows average monthly rainfall based on Global Precipitation Climatology Project data (1996-2015)

Figure 10: Average minimum and maximum daily temperatures for Bolgatanga (Meteoblue)

5.5.3 Rainfall

August is the wettest month of the year. Appreciable rainfall begins (on average) in April and finishes in October. Accordingly, the wet season typically lasts seven months out of each year, with the remaining five months experiencing low to negligible rainfall. The average annual rainfall for the project area is 1,069 mm and the average annual evaporation for the project area is 1,872 mm.

Daily rainfall series was also obtained from the Global Precipitation Climatology Project ("GPCP"). The GPCP provides daily precipitation estimates on a 1-degree grid over the globe from 1996 to 2015. The GPCP data gives an average annual rainfall of approximately 1020 mm. The average monthly rainfall for the data period is provided in Figure 10 (Golder, 2018a).

Rainfall data obtained was used to generate the 1, 2, 3, 6, 12 and 24 hour duration storms for the 1 in 2, 5, 10, 20, 50 and 100 Annual Exceedance Probability (AEP) events. The estimated mean depth-duration-frequency relationships are provided in Table 5 below.

Table 5: Mean depth-duration-frequency estimate (mm)

Storm Duration (hours)	Annual Exceedance Probability (1 in Year)					
	2	5	10	20	50	100
1	28	40	49	60	77	94
2	34	48	59	72	93	113
3	39	55	68	82	107	129
6	44	61	76	93	120	146
12	45	65	80	98	127	154
24	49	68	85	103	133	162

Average monthly means of daily Penman Reference evaporation rates were obtained from the meteorological station in Navrongo, Ghana for a period between 1961 and 1990. The mean annual evaporation is 2044 mm for the station (Golder, 2018a).

The monthly rainfall, evaporation values and average recurrence interval ("ARI") were summarized by Lycopodium (2018) for the purpose of designing the Project Infrastructure (Section 18.0) as shown in Table 6.

Table 6: Summary of rainfall and evaporation

Month	Pan Evaporation (mm)	Rainfall		
		100-yr ARI Wet (mm)	Average (mm)	100-yr ARI Dry (mm)
January	199	0	0	0
February	193	0	2	0
March	213	0	3	8
April	192	141	67	69
May	176	305	98	71
June	135	134	150	46
July	113	125	213	53
August	97	288	132	120
September	101	296	286	221
October	127	114	99	68
November	148	69	18	13
December	178	0	0	5
Annual Average	**1,872**	**1,472**	**1,069**	**674**

6.0 HISTORY

The Mineral Resource estimate (Section 13.1) forming the basis of this PFS was documented in MPR (2018) which is a comprehensive report is provided on Sedar (https://www.sedar.com). The information in this Section is derived from Golder (2018) and MPR (2018).

6.1 Historic production

Prior to the agreements described in Section 4.2 and below there were no previous legal mining tenements.

There has been no previous commercial production from the site. Production to date has been limited to small-scale artisanal mining which has not been quantified. The quantity of gold extracted by the artisanal miners is expected to be negligible with minimal effect on the Ore Reserve.

Golder observed artisanal mining operations during the May and December 2017 site visits. The artisanal mining is very small-scale and targets visible gold in quartz veins near surface (Figure 11).

Figure 11: Artisanal miners selectively extracting mineralized material from quartz veins

6.2 Exploration and discovery

The earliest reported gold discoveries in the general Namdini region date from the 1930's. Historic regional exploration from outside the Namdini Gold Project area is of no relevance to Namdini resource estimates and is not detailed in this report.

All exploration at Namdini was completed by Cardinal. Prior to Cardinal conducting several drill programs, systematic exploration had not been undertaken on the property. Small-scale artisanal mining began on the property about 2013 following Cardinal's initial exploration activities.

The following summary of historical exploration in this region of north eastern Ghana is taken from Golder (2018). Golder has reviewed RPA's exploration history with Cardinal, who consider that this is still a good representation of the exploration history in the region.

Northern Ghana did not have extensive artisanal gold mining when compared to elsewhere in West Africa, such as southern Ghana, Côte d'Ivoire or southern Burkina Faso.

The discovery of gold in this region occurred in late 1930s when a British businessman was shown some gold-bearing quartz veins at Nangodi by a local farmer. A small underground operation was underway by 1934, which attracted the attention of Gold Coast Selection Trust ("GCST") who optioned the property in 1936 and acquired a large prospecting licence area which covered most of the belt. GCST boosted the underground production, which peaked at about 5,000 oz/year of gold in 1936-1937, but dropped thereafter as a result of lower grades (originally about 1 oz/ton and dropping to about 0.6 to 0.8 oz/ton in 1937-1939). GCST subsequently dropped the option in 1938, but mining continued on a very modest scale for a few years.

During the early 1960s, the Ghanaian Government was trying to stimulate interest and development in northern Ghana. The Ghana Geological Survey Department carried out limited shallow drilling around prospects which had been identified by earlier work in the 1930s. In the 1970s some soil geochemistry and trenching were carried out over a 7 km stretch in the Nangodi area where most of the known prospects occur.

Driven by activity elsewhere in Ghana and Burkina Faso during the mid-1990s, numerous Canadian and Australian junior explorers started to explore the north of Ghana, where the discovery of Youga deposit in Burkina Faso by International Gold Resources (IGR) is significant.

During this same period small-scale miners inundated the area as the traditional small-scale mining sites in southern Ghana were closed at Tarkwa, Obuasi, Konongo, etc. Environmental problems were created when the artisanal miners encroached on forest reserve areas southwest of Bolgatanga. Eventually, the Small-Scale Mining Division of the Minerals Commission set aside a 72 km^2 area south of Nangodi (Shiega-Datoko) for small-scale mining. A number of licences were taken out and up to several thousand people were living and working in the general area between 1996 and 1998.

BHP was the first to conduct a major reconnaissance exploration program in the mid-1990s, covering most of the Nangodi area. BHP's work was directed towards developing both gold and base metal prospects. After an initial regional program which identified promising geochemical and geophysical anomalies, the project was largely abandoned as a result of BHP's decision to cease exploration activity in Ghana. Other groups acquired prospecting concessions in the mid-1990s including IGR, who picked up two areas on the margins of the belt; the western area covered a large area around Navrongo and the eastern area extended to the Bawku area. Cluff Resources held two concessions on the eastern side of the belt, adjacent to BHP's Nangodi licence area, and Teberebie Goldfields acquired a concession from just north of Nangodi to the Burkina Faso border.

Subsequently, Ashanti Goldfields carried out extensive work on the IGR concessions after their takeover of the company, and an Australian junior, Africwest Gold, successfully applied for a reconnaissance concession in the Nangodi area in late 1996, after the BHP licence had lapsed.

The market downturn in 1997 seriously affected Africwest's ability to raise additional equity funding and their licence in the Nangodi area lapsed.

> *Renewed interest in the area began around 2004, with an increase in the gold price, and as a result of the development of mines on the Burkina Faso side of the border. In 2006 Etruscan Resources Inc., a Canadian mining company, carried out soil sampling, rock chip sampling, limited trenching, and reverse circulation (RC) and rotary air blast (RAB) drilling (139 holes) in the Zupliga, Fulani and Dumorlugu prospects. The best drill intercept was 18 m at 3.35 g/t Au.*
>
> *Randgold also explored the Nangodi-Bole belts from 2004 to 2009 with soil geochemistry, stream sediment sampling and rock chip sampling. The company identified eight areas, but left when these failed to meet their economic criteria. Red Back Mining commenced exploration work over the Nangodi Belt and adjacent areas in 2005. This included a desktop study of satellite imagery, data compilation, mapping and rock chip sampling.*
>
> *Significantly, none of the historical exploration had used a detailed airborne geophysical survey to identify structural-lithological targets to support the ground work. In 1999, the Finnish Government flew a low-resolution geophysical survey over selected areas of the country for the Geological Survey of Ghana as part of a World Bank-supported project.*

Modern exploration of the Namdini area prior to discovery of Namdini mineralization in 2013 is limited to regional geophysical surveys which are of little relevance to resource estimates and are not detailed in this report.

6.3 Current status

Savannah Mining Ghana Limited (Savannah) discovered the Namdini gold mineralization in 2013 under a Heads of Agreement with Cardinal. A small-scale mining license, of approximately 6.25 Ha, was applied for by Savannah and approved in 2014.

In 2013 artisanal miners began applying for additional small-scale mining licenses and exploited surface gold mineralization, resulting in an irregular shallow open pit in the center of the deposit, north of the Savannah small-scale mining license. Savannah subsequently entered into sale-and-purchase agreements and license relinquishment agreements with holders of small-scale mining licenses that cover the Namdini gold deposit and surrounding area.

During the December 2017 quarter, a Large-Scale Mining Licence covering the Namdini Mining Lease was assigned to Cardinal Namdini Mining Limited (Cardinal Namdini), a wholly owned subsidiary of Cardinal by the Minister of Lands and Natural Resources. The Large-Scale Mining Licence, which covers 19.54 km^2 is for an initial period of 15 years and is renewable.

6.4 Previous Mineral Resource estimates

On 7 November 2016 a Mineral Resource estimate was reported by Cardinal (2016) with an effective date of 31 October 2016 (Gossage, 2017) based on drilling information available up to August. This was reported in accordance with the JORC Code (2012) to the Australian Stock Exchange (ASX) as summarized in Table 7.

Table 7: Mineral Resource estimate at 31 October 2016 (Gossage, 2017; Cardinal 2016)

Resource Category	Tonnage Mt	Grade (g/t Au)	Contained Metal (Moz Au)
Indicated	6.22	1.2	0.24
Inferred	89.9	1.3	3.60

Notes: Mineral Resources were reported according to the JORC Code (2012).

Resources were estimated by Multiple Indicator Kriging (MIK) of 3 m down hole composites cut to 15 g/t.

Mineral Resources were reported at a cut-off grade of 05 g/t Au above a pit shell using a long-term gold price of US$1,550 per ounce.

Drill holes completed at August 12, 2016 were incorporated (NMDD034).

Numbers may not add due to rounding to appropriate significant figures.

In 2017, Cardinal commissioned Roscoe Postle Associates Inc (RPA) to prepare a Mineral Resource Technical Report for the Namdini Gold Project. This Mineral Resource estimate incorporated drilling information to December 2, 2016 consisting of 32,275 m of diamond and reverse circulation (RC) drilling, up to and including NMDD061, and was released by Cardinal as an NI 43-101 report on April 6, 2017.

RPA used geological wire-frames and grade shells at 0.1 g/t Au and 1.0 g/t Au to constrain grades estimated using Ordinary Kriging. High-grade assays were capped separately for the four indicator domains at 1 g/t Au, 10 g/t Au, 15 g/t Au, and 25 g/t, then composited to 3 m intervals. Densities were assigned to blocks based on lithological units and weathering horizons. Blocks were classified as Indicated and Inferred based on confidence in block estimates implied by the variogram, grade continuity, and drill hole spacing. A cut-off grade of 0.5 g/t Au was used for resource reporting and Mineral Resources were constrained by a Whittle pit optimization.

The Mineral Resource estimate was documented in a technical report (RPA, 2017) dated April 5. A summary of the results is provided in Table 8.

Table 8: RPA Mineral Resource estimate at 2 December 2016 (RPA, 2017)

Resource Category	Tonnage Mt	Grade (g/t Au)	Contained Metal (Moz Au)
Indicated	23.9	1.21	0.93
Inferred	100.1	1.13	3.63

Notes: Mineral Resources were reported according to the JORC Code (2012).
Mineral Resources were estimated at a cut-off grade of 0.5 g/t Au, constrained by a preliminary open pit shell.
Mineral Resources were estimated using a long-term gold price of US$1,500 per ounce.
Drill holes completed at December 2, 2016 were incorporated (up to and including NMDD061).
Numbers may not add due to rounding to appropriate significant figures.

The MPR (2017) Mineral Resource estimate was the basis of the PEA study lodged with effective date 5 February 2018, referred to in this PFS Report as Golder (2018).

Table 9: Mineral Resource estimate by MPR (2017) at 0.5 g/t cut-off, used for the PEA (Golder, 2018)

September 2017 Mineral Resource Estimates			
Category	Mt	Au g/t	Au Moz
Indicated	120	1.10	4.27
Inferred	84	1.2	3.1

Notes: Mineral Resources were estimated by Multiple Indicator Kriging of 2 m down-hole composited gold grades from RC and diamond drilling. They include a variance adjustment to give estimates of recoverable resources for selective mining dimensions of 5 by 10 by 2.5 m (east, north, elevation).
All figures were rounded to reflect the relative accuracy of the estimates.
Mineral Resources constrained by a pit shell generated at US$1,500/oz.
Densities were assigned by rock unit and oxidation zone.

The above publicly reported Mineral Resource estimate is now superseded, based on MPR (2018). The 2018 estimate is fully discussed in Section 13.1. There was a change in tonnes, grade and contained metal based on additional drilling, an alternative estimation method and revised parameters to constrain the estimate within a pit shell compared to the MPR (2017) estimate (see Section 14.14.4).

6.5 Preliminary Economic Assessment

In February 2018 Cardinal announced results of a Preliminary Economic Assessment (PEA) for Namdini (Cardinal, 2018, Golder, 2018). The PEA was based on the September 2017 Mineral Resource model reported at 0.5 g/t cut-off within a pit design (Table 10).

Table 10: February 2018 PEA summary

After Golder 2018			
Gold price	US$1,300 per ounce		
Resource at 0.5 Au g/t cut-off grade	Indicated: 91 Mt at 1.1 g/t Au Inferred: 22 Mt at 1.1 g/t Au		
Strip ratio	1.2:1 (waste:ore)		
Metallurgical recovery	90% Oxide, 86% Fresh		
	9.5 Mtpa	**7.0 Mtpa**	**4.5 Mtpa**
Life of gold mine production	3.52 Moz	3.51 Moz	3.52 Moz
Mine life	14 Years	19 Years	27 Years
Development capital cost	US$426 M	US$349 M	US$275 M
Life of mine sustaining capital cost	US$154 M	US$160 M	US$172 M
All in Sustaining Costs (AISC)	US$701/oz	US$736/oz	US$794/oz

The mine was conceived as a single open pit with an initial starter pit, to be mined by conventional excavator and truck haulage. The mining plan scheduled 112 Mt of ore and 135 Mt of waste to recover 3.5 Moz Au. The processing was considered as SABC grinding, flotation with regrind and CIL across a range of throughput rates: 9.5, 7.0 or 4.5 Mtpa.

The PEA pit design was conservative due to the lack of geotechnical slope stability data and analysis.

The PEA study was preliminary in nature. It included Inferred Mineral Resources that were considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Ore Reserves.

The positive result of the PEA confirmed that progression of the Namdini Gold Project to the Pre-feasibility Study (PFS) stage was warranted.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

The Mineral Resource estimate forming the basis for this PFS was carried out by MPR (2018) which is a comprehensive report provided on Sedar (https://www.sedar.com).The information in this Section is derived from Golder (2018) and MPR (2018).

7.1 Geological background

This description is based on MPR (2017, 2018):

The Namdini Gold Project lies within the Paleo-Proterozoic Nangodi Greenstone Belt, one of a series of southwest–northeast trending granite-greenstone belts which host significant gold mineralization in Ghana and Burkina Faso. These belts are interpreted to be fault bounded, both during their development and post-deposition. Much of northern Ghana is covered by post-Birimian Voltaian Basin sediments, and at Namdini this forms the southern limit of exposure of Birimian rocks (Figure 12).

Key units of the metamorphosed greenstone belts include greywackes and phyllites of the Tarkwaian Formation, which are overlain by volcanic and sediment sequences of Birimian age (2.2 to 2.1 Ga), characterized by interbedded mafic to intermediate volcanic flows, felsic to intermediate tuffs and fine-grained sediments. The greenstone belts are intruded by belt-type and basin-type granitic rocks and late stage diorites. Belt-type granites are metaluminous and commonly tonalitic. Basin-types are peraluminous with higher potassium and rubidium than the belt-type granites and are generally granodiorites.

The granite–greenstone terrain that hosts the Namdini Gold Project is in the North Eastern District of Ghana, close to the border with Burkina Faso. The region contains several producing mines both on the Ghana side of the border (the Shaanxi underground gold mine) and in Burkina Faso (the Youga open pit gold mine).

Locally the Nangodi Greenstone Belt trends north-northeast to south-southwest over a distance of 30 km and turns to an east-northeast to south-southwest trend in the south of the area around Namdini. Much of the area to the south of the tenements is covered by later Voltaian Basin sediments. The belts continue underneath this cover.

Structurally the north eastern region of Ghana is characterized by steep isoclinal folding with near vertical axial planes. The greenstone belts contain locally developed open symmetric folds with axial planar cleavages parallel to bedding in the steeply inclined sediments.

7.2 Geological setting

This description is based on RPA (2017) and MPR (2018):

The Namdini Gold Project covers the southern extension of the Nangodi Greenstone Belt (Figure 12 and Figure 13).

Namdini rock units comprise a steeply dipping sequence of Birimian inter-bedded meta-sedimentary and meta-volcanic units, which were intruded by tonalite and diorite. The meta-sedimentary and volcaniclastic rocks were intensely altered with a pyrite-carbonate-muscovite-chlorite-quartz assemblage. The tonalite is extensively altered and was overprinted by silica-sericite-carbonate assemblages.

Petrography and mineralogy in the following sections is based on thin and polished section work by Townend Mineralogy Laboratory (“TML”) of Perth.

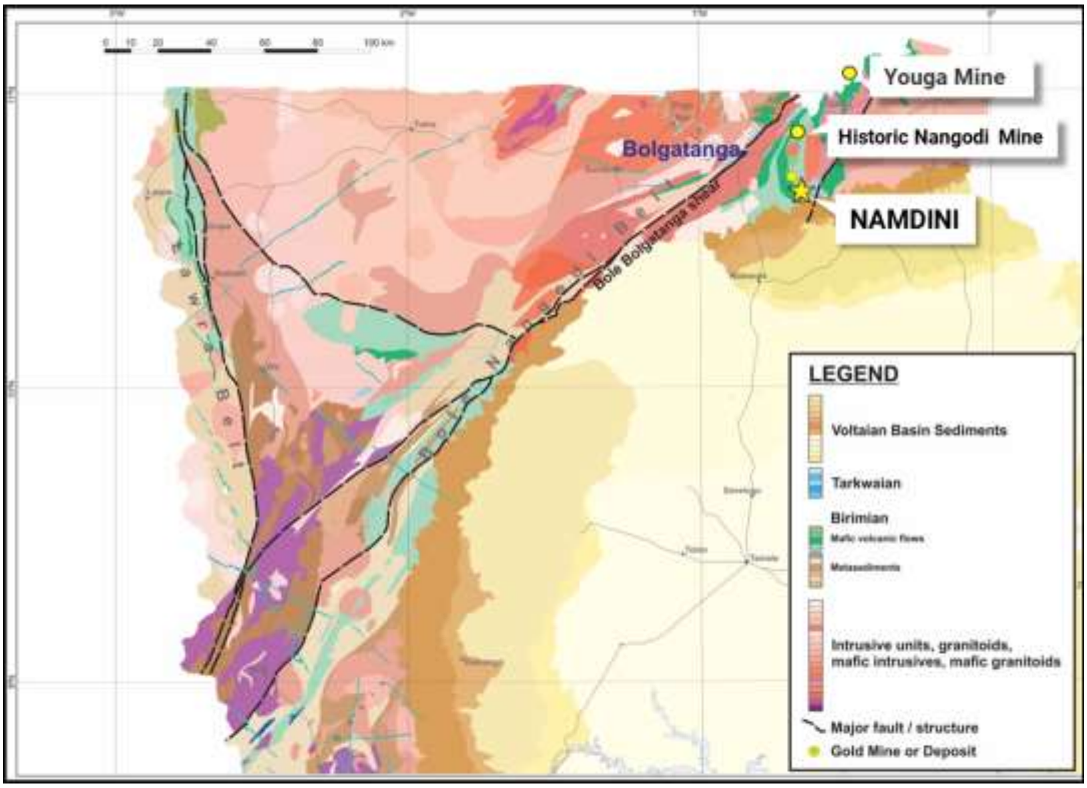


Figure 12: Regional geology of northern Ghana (source: Cardinal)

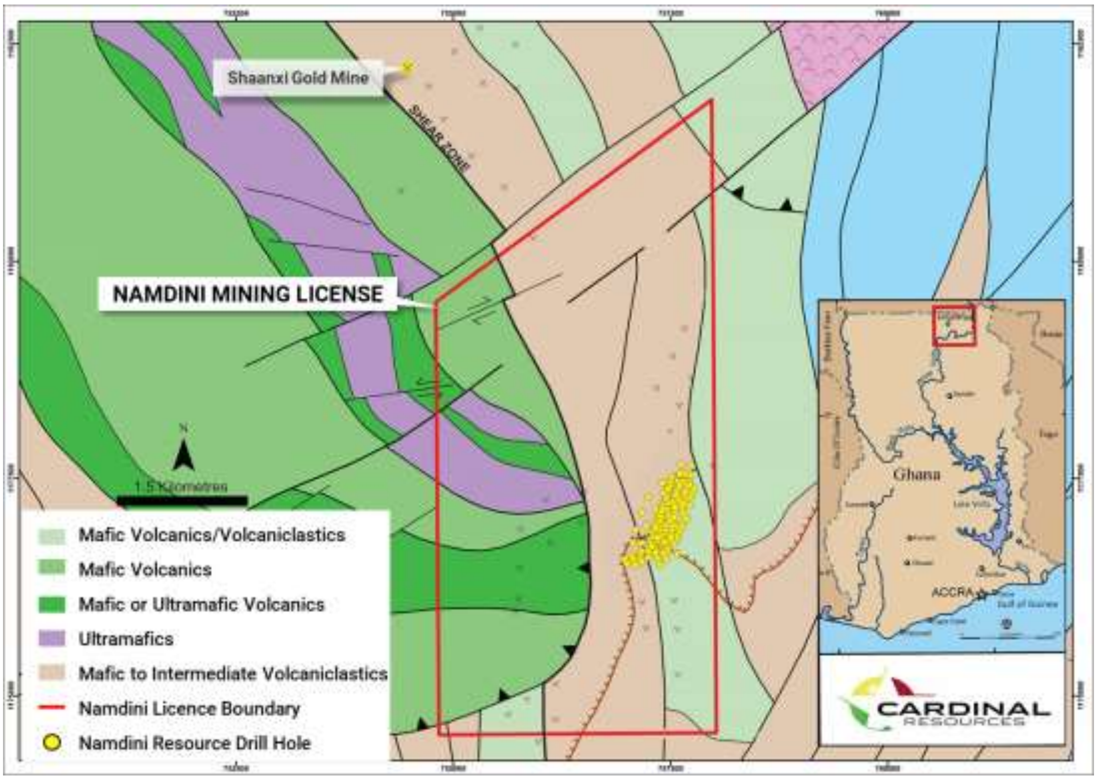


Figure 13: Local geology of the Namdini Gold Project (source: Cardinal)

7.2.1 Structural geology

In 2016 and 2017, geological consultants from Orefind Pty Ltd ("Orefind") conducted a structural study and then developed a structural framework with controls on, and geometry of, the gold mineralization comprising the Namdini deposit.

Orefind's review of diamond drill core and outcrops suggested two discrete stages of gold deposition and emplacement of vein minerals and associated alteration. Currently, the structural overprinting, geometries, and kinematics are thought to be consistent with deposition of mineralization in both D1 and D2 foliation-forming events. It is likely that this represents two stages of gold deposition punctuated by a deformation hiatus between D1 and D2. The emplacement of successive mineralization-associated vein stages does not represent a single event. The formation of intense D2 structures, in particular the penetrative S1-S2 composite foliation, have strongly modified or destroyed D1 overprinting and geometric relationships.

The Namdini mineralized system is located in a zone of oblique, sinistral, east side-up shearing that developed during D2. S1 is also an intense foliation and much of the foliation development is represented by a penetrative, composite S1-S2. This suggests a formation of the host shear zone during D1, with intense reactivation of the system in D2. Overall, the intensity of D2 has resulted in a strong preferred orientation of mineralized zones controlled by S2 and mesoscale F2 folds, with any D1 controls being preserved in local D2 low strain zones. Alternatively, the deposit may be located on the limb of a fold of a larger scale than the deposit.

S1 is a pervasive foliation and commonly occurs in the hinges of D2 differentiated crenulations. S2 manifests variably as an intense foliation, a spaced cleavage, a differentiated crenulation cleavage, and as a contributor to a composite S1-S2 produced by transposition of S1. Intense deformation during D2 has commonly resulted in rotation of the S1 into the S2 orientation, resulting in transposition of S1 with S2 and the formation of a pervasive composite S1-S2 foliation.

Orefind's traverses across the mineralized sequence determined that the S1-S2 asymmetry is consistent across the deposit. S1 strikes acutely clockwise of strike of S2. This asymmetry is consistent with D2 kinematic indicators, which indicate a constant sinistral sense of shear in plan. In section, the kinematics during D2 appear to be E-side-up. Overall, Orefind noted that the mineralized package has accommodated sinistral, E-side-up, oblique shear.

Foliations are less well developed in the tonalite, with most structures represented by quartz ± carbonate vein arrays and silica-sericite fractures or veinlets. Faults are uncommon in general. In the tonalite, faults tend to manifest as zones of enhanced quartz veining or local fracturing.

Parasitic folds are common in the core but were not observed in the field. This is interpreted as a function of exposure, rather than lack of development.

The majority of textural relationships are indicative of mineralization in D2, and this likely represents the period of greatest deposition. Structural orientation controls on the geometry of mineralized zones will be overwhelmingly along the north-northeast to south-southwest D2 trends of S2 and L12 intersection lines. Lesser orientation controls are likely to have a north-south trend indicative of vein orientations in D2 extensional jogs, especially for quartz veins in the tonalite.

7.2.2 Host rocks

Namdini gold mineralization is located in the Nangodi Greenstone Belt within a host sequence of meta-volcaniclastics, granitoids (tonalite), and diorites. The deposit is bounded on the hangingwall and footwall sides by metasediments.

Geological description of the Birimian units intersected during exploration drilling include:

- Meta-sedimentary rocks are fine-grained chlorite-muscovite schists.
- Meta-volcaniclastic rocks are very fine-grained, chlorite-muscovite phyllites.
- Granitoid samples are classified as altered, sheared, sulfide-bearing tonalite. A tectonic foliation is developed in the intrusive rocks, but is not pervasive in the granitoid.
- Diorite rocks are assumed to be late stage intermediate diorite stock and dykes. They occur as altered (shearing, silicification, chlorite, and sericites) or unaltered diorite, as well as Quartz Diorite speckled with quartz and feldspar.

Figure 14 shows a plan view of the key rock-units at Namdini relative to the surface expression of the mineralized domain used for the current estimates.

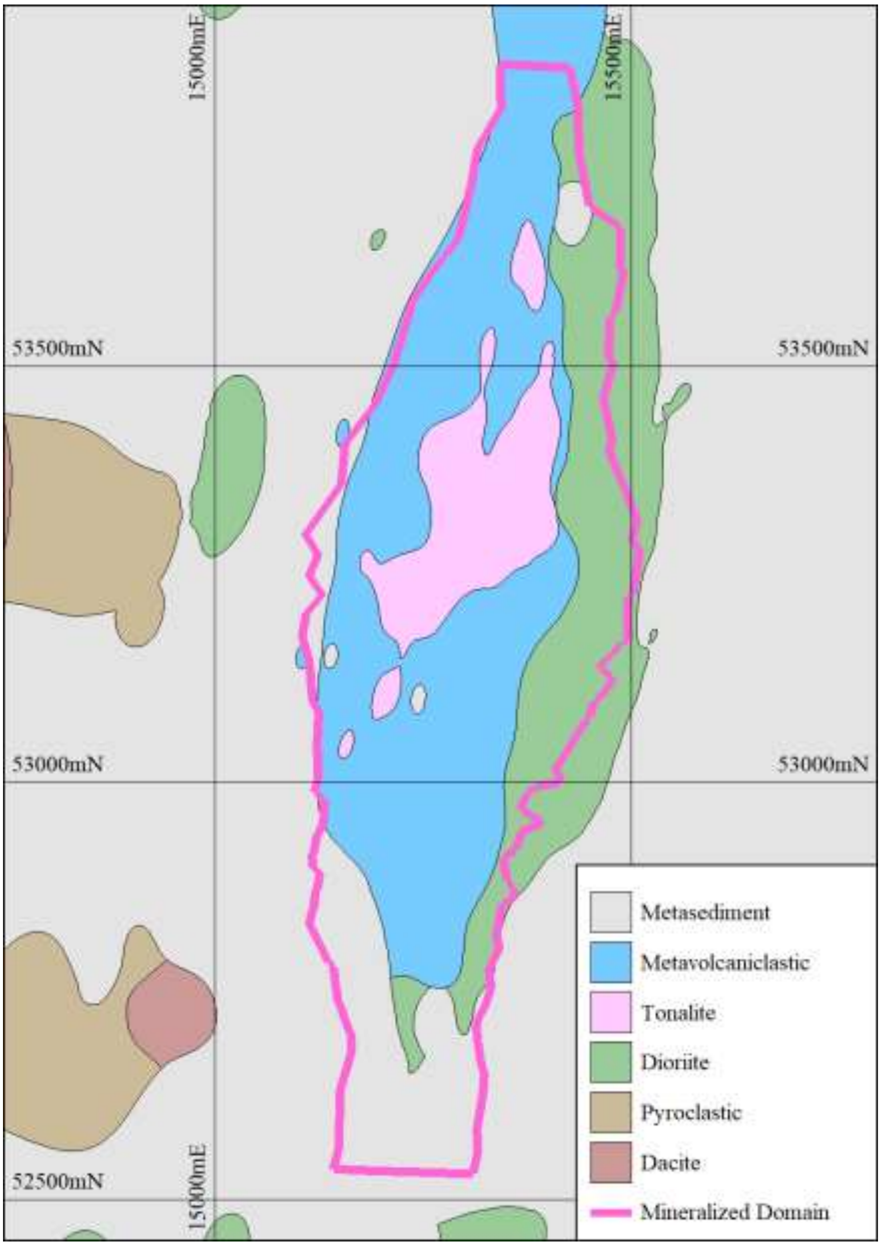


Figure 14: Namdini geological setting (source: MPR, 2018)

7.2.3 Weathering profile and hydrogeology

In the mineralized area, the tropical weathering profile extends to a maximum depth of around 30 m, comprising the following material types:

- **Strongly Oxidized (SOX):** total oxidation off all primary minerals with little or no primary rock texture. This zone ranges from 1.0 to 7.5 m in thickness.
- **Moderately Oxidized (MOX):** some primary rock texture, total oxidation of feldspar to clay, and total oxidation of sulfides. This zone ranges from 0.5 to 13.0 m in thickness.
- **Transition (TRANS):** strong primary rock textures with partial oxidation of feldspars and sulfides. The Transition zone ranges from 2.0 to 14.5 m in thickness.

Golder Africa (2018b) noted that most of the geological formations in the study area are overlain by a regolith comprising *in-situ* chemically weathered material and, to a lesser extent, transported surface material. Weathering processes involved are briefly summarized below. In the Bongo and Bolgatanga districts of the Upper East Region, it is reported that the thickness of the regolith developed over Birimian and Tarkwaian rocks ranges from 11.6 to 33.2 m with an average of 23.3 m, while it ranges from 2.7 to 37.0 m with an average of 19.5 m over granitoid intrusions. In sedimentary rock terrains, the regolith would be slightly thinner. Outcrop areas are relatively rare except where underlying rocks such as quartz-rich granitoids preferentially resist weathering and develop into inselbergs, in particular in the Upper East Region.

Regolith developing over bedrock as a result of chemical weathering usually exhibits a progressive degradation from fresh bedrock to residual soil. The typical zones forming the weathered profile are (from top to bottom):

- residual soil (usually sandy-clayey material possibly underlain by indurated layer)
- saprolite (completely to slightly decomposed rock with decreasing clay content with depth)
- saprock (remnants of unweathered bedrock in an altered matrix)
- fresh (variably fractured) bedrock.

While the thickness of these zones can vary, their sequence remains the same. The first two zones, i.e. residual soil and saprolite, form the regolith while the saprock is usually considered part of bedrock. The weathering front, that is the interface between fresh bedrock and saprock, is generally sharp in coarse-grained massive crystalline rocks and diffuse and gradational in fine-grained rocks. Degree and depth of weathering depends largely but not exclusively on bedrock lithological and structural characteristics (e.g. mineral grain size, mineral solubility, relative proportions of Fe-Mg minerals, intensity of fracturing etc.), climatic conditions, geomorphological characteristics, age of land surface and aquifer characteristics (e.g. flow, recharge, geochemistry etc.). While chemical weathering usually occurs without significant changes in volume, removal of dissolved material through groundwater flow reduces overall mass. As a result of this unloading of the bedrock, fractures can open to greater depth, thus allowing groundwater to flow deeper.

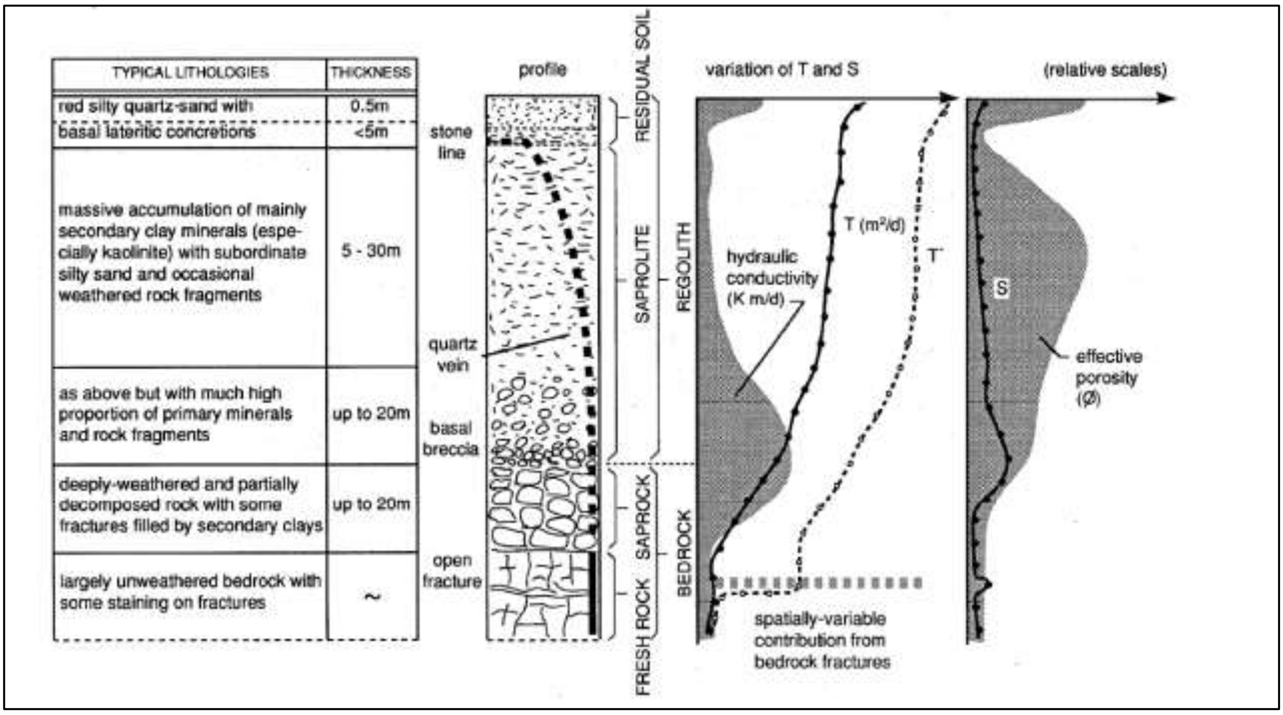


Figure 15: Conceptual model of weathered layer in Africa (from CIDA, 2011)

Groundwater flow is one of the principal geomorphological agents in the chemical weathering processes of bedrock through hydrolysis and dissolution. In basement rocks, which usually exhibit low primary porosity and hydraulic conductivity, weathering can cause substantial increases in both properties. Figure 15 shows a conceptual model of the weathered layer for basement rock areas. On the left of the figure, typical lithologies are described and illustrated while the two graphs on the right illustrate relative variations of material characteristics (i.e. transmissivity (T), storage (S), hydraulic conductivity (K) and effective porosity (Φ)) with depth. As these two graphs show (starting from fresh rock), porosity gradually increases while hydraulic conductivity rapidly increases at first as a result of fracturing and then gradually decreases with the formation of secondary clay minerals. At a later stage (i.e. near surface), more aggressive weathering can result in secondary clay minerals dissociation, thus re-increasing hydraulic conductivity. The distribution and extent of these zones of enhanced hydraulic conductivity within the regolith determines the presence of aquifers.

7.2.4 Alteration

The meta-sedimentary and volcaniclastic rocks were intensely altered with a pyrite-carbonate-muscovite-chlorite-quartz. Alteration is prevalent in the volcaniclastic rocks.

Similarly, the tonalite is extensively altered and was overprinted by silica-sericite-carbonate assemblages. The identity of carbonate alteration is difficult and is best described as iron-carbonate in the absence of petrological or geochemical characterization. Fe-dolomite and ankerite units were noted by TML, although these can be difficult to unequivocally identify.

7.2.5 Mineralization and gold particle size

Drilling has outlined mineralization with three-dimensional continuity, with a size of approximately 1,500 m long, 550 m wide, and 450 m in depth.

In all rock types, the mineralization is accompanied by visible disseminated sulfides of pyrite and arsenopyrite in both the veins and wall rocks. In diamond drill core, the mineralized zones are visually distinctive due to the presence of millimetre to centimetre wide quartz-carbonate veins that are commonly folded and possess yellow-brown sericite-carbonate selvedges (Figure 16).

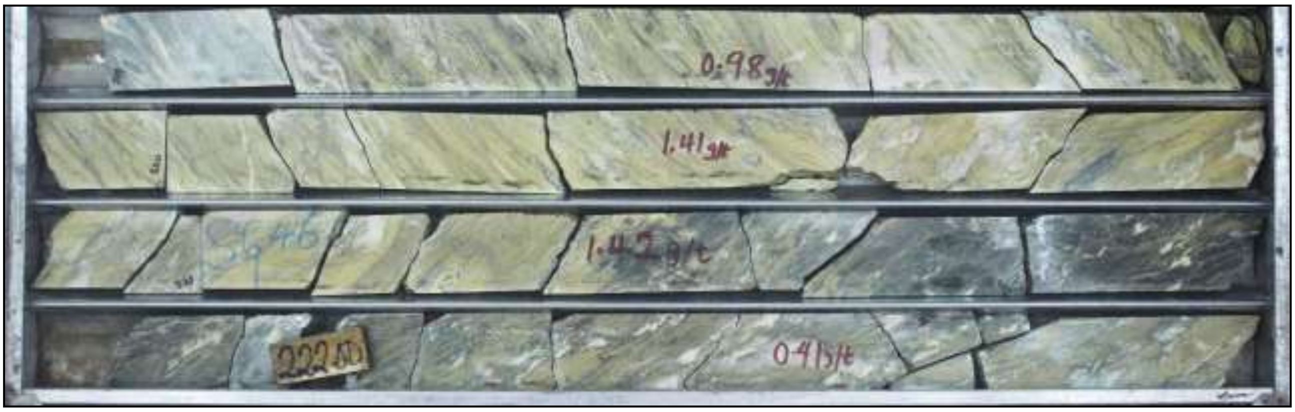


Figure 16: Alteration in drill hole NMDD007 at 227.33 m to 231.95 m (source: RPA, 2017)

Visible gold was identified by Cardinal, Orefind, and RPA. Its instances occurred in strongly altered granite and were associated with silica-sericite shears that had sub-millimetre widths, as well as in the diorite.

Petrological work by TML showed that gold is primarily associated with sulfides, in particular pyrite, where it commonly occurs as inclusions and on the crystal margins. Gold was also noted in phyllite matrix and, to a much lesser extent, in association with ilmenite.

Mineragraphic analysis by TML has shown that very fine-grained gold less than 5 µm is dominantly associated with, and as inclusions within, disseminated sulfides and less commonly silicate minerals (Figure 17).

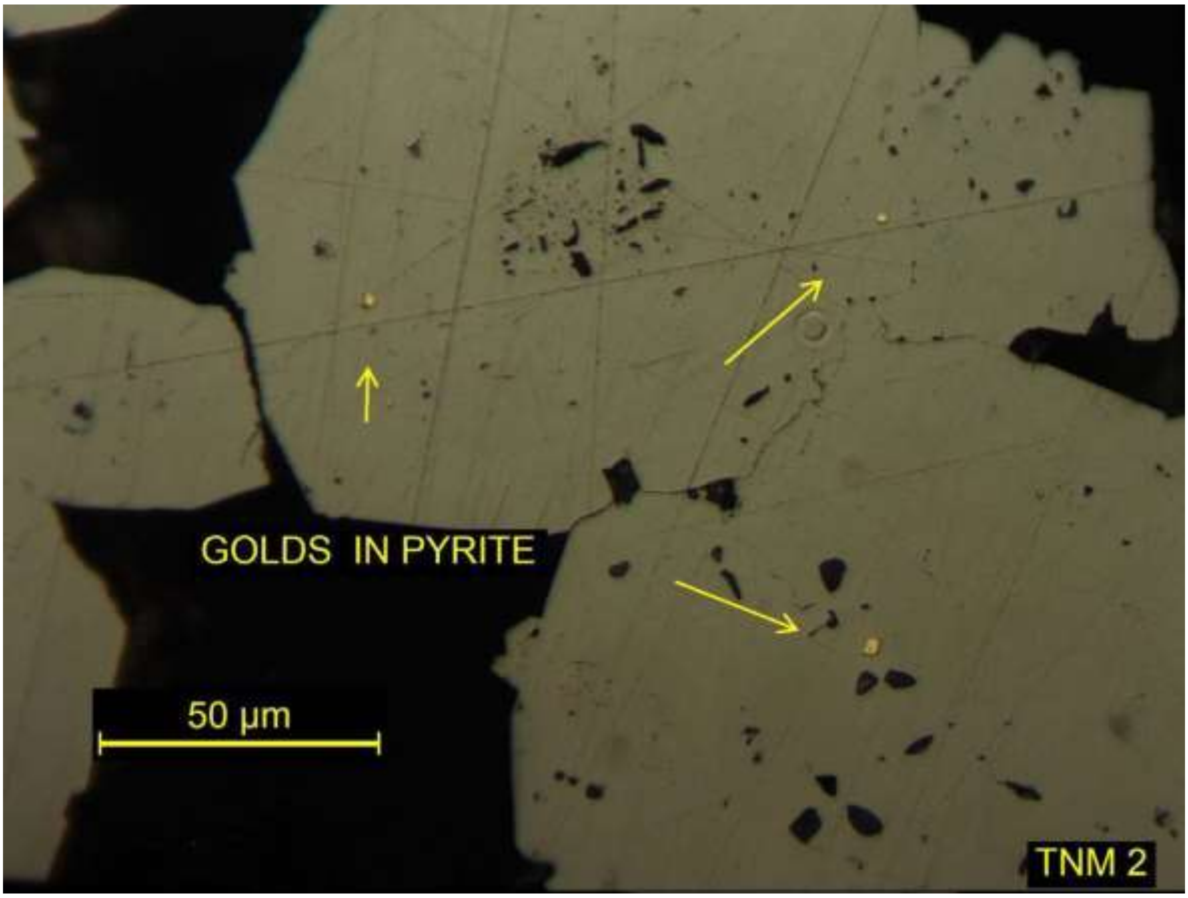


Figure 17: Gold associated with pyrite in sample TNM2 (source: TML in RPA, 2017)

The primary rock is a muscovite rich phyllite that shows extensive carbonate replacement. The phyllite hosts a major sulfide content and significant gold. The fine-grained muscovite shows a moderate preferred orientation and is heavily impregnated with ankerite carbonate.

The dominant ore minerals are pyrite and arsenopyrite that occur relatively commonly through much of the phyllite matrix. Tennantite, chalcopyrite, pyrrhotite, galena, and sphalerite all occur as fines within some pyrites. Magnetite was also detected once.

About 25 occurrences of gold were detected optically. The vast majority occur as inclusions in pyrite. The gold appeared low in silver. These included golds in pyrites that were predominantly under 5 µm, with the exceptions of a linear nature, reaching 25 µm. Most were single particles plus a rare trio.

The host pyrites had a wide range of grainsizes from 50 to 600 µm. Two occurrences of gold were single, fine grains of 2 µm within silicate.

7.2.6 Cardinal geological logging

Logging, interpretation and geological modelling were undertaken by Cardinal Resources' technical staff using Maxwell Geoservices (Perth) Logchief software and specialist structural consultants Orefind Pty Ltd resulting in a three-dimensional model of key lithologies, structures and weathering zones.

Table 11 presents a summary of the revised lithological codes and descriptions used by Cardinal geologists for geological logging purposes.

Table 11: Summary of Cardinal geological logging codes and descriptions

Code	Description
LAT	Laterite, ferruginous duricrust developed *in situ*
SPR	Saprock (<20% weatherable minerals altered)
SAP	Saprolite (>20% weatherable minerals altered)
GRA	Granodiorite or tonalite, altered Felsic rocks (sericites, muscovite, carbonate and K-feldspar)
DIO	Intermediate rocks, altered (shearing, silicification, chlorite and sericites) and unaltered diorite, Quartz Diorite speckled with quartz and feldspar
MVO	Mafic rocks, volcaniclastics, altered (sericites, chlorites + silicification)
MSE	Metasedimentary rocks, dominated by quartz-carbonate veining + haematite and chlorite
LTF	Pyroclastic rocks and tuffs
BX	Breccia
QTZ	Quartz

8.0 DEPOSIT TYPES

The Mineral Resource estimate forming the basis for this PFS was carried out by MPR (2018) which is a comprehensive report provided on Sedar (https://www.sedar.com). The information in this Section is derived from Golder (2018) and MPR (2018).

8.1 Gold mineralization

Gold mineralization in Ghana normally occurs as:

- Steeply dipping quartz veins with native gold in shear zones at Birimian belt/basin boundaries.
- Disseminated sulfide bodies, spatially (though not necessarily genetically) related with the shear zones and quartz veins, with arsenopyrite as the major host of gold.
- Disseminated and stockwork zones in late-kinematic ‘basin type’ granitoids.

The Birimian gold deposit types can be summarized as follows:

> *The majority of the gold occurs in two styles of mineralization: (1) mesothermal quartz vein-hosted and associated gold in metavolcanics and metasediments, and (2) modified palaeoplacer gold in conglomerates. These styles of mineralization occur in the Paleoproterozoic Birimian Supergroup and Tarkwaian Group that make up Ghana’s mainly southwest to northeast trending Birimian belts:*
>
> *Significant gold resources also occur as hydrothermal mineralization in basement-type granitoids which show some geological association with the Birimian Supergroup-hosted mesothermal mineralization. The majority of the gold mineralization is believed to have formed between approximately 2.15 and 2.6 Ga during the Eburnean orogeny.*
>
> *The mesothermal quartz vein gold mineralization is usually confined to tectonic corridors within the Birimian belts and is strongly associated with shear zones and fault systems. The quartz veins show multiple stages of formation and are steeply dipping, with the gold mineralization occurring either as free gold within fractures in the veins or as invisible gold within disseminated sulfides in the host rocks surrounding the veins. The vein- and sulfide-hosted gold is strongly associated with deformational fabrics formed by the Eburnean extensional and compressional events respectively, suggesting that disseminated sulfide mineralization predates quartz vein-hosted mineralization.*
>
> *The fluid from which the gold precipitated is believed to be of metamorphic origin and carbon dioxide (CO_2) dominated, with lesser water (H_2O) and nitrogen (N_2) and minor methane (CH_4). Gold precipitation was probably caused by decrease in pressure, temperature and CO_2-H_2O immiscibility, at depths of between 7 and 11 km.*
>
> *Hydrothermal gold mineralization occurs in the Paleoproterozoic belts and basin granitoids that intrude the Birimian belts, as well as in the sedimentary basins occurring between the belts. Gold mineralization within the granitoids occurs as micro-inclusions in sulfides in small, steeply dipping stockworks and as sulfide disseminations concordant with regional faults and shears. A gold-bearing fluid similar to that for the Birimian Supergroup-hosted quartz vein gold mineralization, but with a larger H_2O component, is proposed to have formed the granitoid-hosted gold mineralization.*

Birimian rocks are composed of granitic-gneiss terranes separated by linear greenstone belts of meta-sedimentary and meta-volcanic rocks.

Current exploration drilling has outlined mineralization with three-dimensional continuity, a size of approximately 1,500 m long, 550 m wide and 450 m in depth, and hosted within defined Birimian gold deposit lithologies.

The Namdini Gold Project appears to be a typical Birimian gold deposit and is hosted in a mixture of altered meta-volcanic sediments, diorite and tonalite. It is associated with quartz-carbonate veins and disseminated pyrite and arsenopyrite in both veins and wall rocks. The mineralization is strongly structurally controlled and the deposit appears to be located in an oblique, sinistral structure in a regionally extensive deformation zone.

8.2 Other deposit types

Exploration activities and mineralization outside the Namdini Gold Project are of no relevance to this PFS.

9.0 EXPLORATION

The Mineral Resource estimate forming the basis for this PFS was carried out by MPR (2018) which is a comprehensive report provided on Sedar (https://www.sedar.com). The information in this Section is derived from Golder (2018) and MPR (2018).

9.1 Summary of relevant work

Cardinal's regional exploration activities outside the Namdini deposit area are of little relevance to the current Project and are not detailed in this PFS Report.

Namdini was discovered in September 2013 by traditional prospecting methods. Follow-up exploration in the region has included aeromagnetic surveying, geological mapping, rock chip sampling and auger drilling.

Savannah subsequently excavated a shallow open pit to expose a westerly dipping gold mineralized zone. Follow up exploration by Cardinal included aeromagnetic surveying, regional and prospect-scale geological mapping and rock chip sampling.

Cardinal commenced RC and diamond drilling at Namdini in March 2014 and October 2015 respectively.

All exploration work on the Namdini Gold Project was completed by Cardinal. A field office with core logging and storage facilities is located near to the Namdini Gold Project site in Bolgatanga.

The primary objectives of Cardinal's exploration strategy are to:

- Improve understanding of the extent and style of mineralization in order to successfully increase the size and confidence level of the Mineral Resources for Namdini.
- Develop deposit models and use grassroots exploration methods to search for gold (and pathfinder elements) to potentially locate other deposits throughout Northern Ghana.

The Namdini Gold Project was first discovered in September 2013 by prospecting. A small-scale mining licence was approved in 2014 and Reverse Circulation ("RC") drilling began shortly thereafter. Cardinal drilled additional RC holes in the same licence area after reviewing initial RC results. At the conclusion of approximately 88 RC holes Cardinal had sufficient confidence in the potential size of the Namdini Gold Project to step out 600 m north along strike and drill a surface diamond drill hole (NMDD002) that intersected 87 m at 1.08 g/t Au and numerous other significant intersections.

Cardinal have built a team of site-based field geologists led by an experienced Exploration Manager. A Global Positioning System (GPS) on site is the most common navigation and survey tool in the field to locate and update geographic information.

9.2 Local grid

Consistent with supplied sampling information, the current study was undertaken in a local grid developed by Sahara. Unless specified, all coordinate references, and orientations in this report reflect the local grid.

The local grid transformation comprises an eight-degree rotation from WGS coordinates (Table 12), with no elevation change. The transformation rotates the obliquely (WGS) trending drill traverses to east-west (local) grid.

Table 12: Local grid translation details

Translation Point	UTM: WGS 84 Zone 30 N		Local	
	East (mE)	North (mN)	East (mE)	North (mN)
1	757,032.992	1,175,611.678	15,000.000	51,800.000
2	757,380.925	1,178,087.348	15,000.000	54,300.000
3	758,569.247	1,177,920.341	16,200.000	54,300.000
4	758,221.314	1,175,444.671	16,200.000	51,800.000

9.3 Geophysical surveying

A high-resolution, 100 m line-spaced airborne magnetic-radiometric survey was carried out by Terrascan Airborne from August to December 2013 over all of Cardinal’s properties in north eastern Ghana.

During 2016, Terratec Geophysical Services completed a ground magnetic survey and induced polarization (IP) survey over the Namdini deposit. The data was processed and interpreted by Southern Geoscience Consultants who generated a suite of digital images and contours, and a litho-structural interpretation at 1:50,000 scale over the area of the survey. This interpretation provided Cardinal with a detailed Project-wide geological and structural map for exploration target development and assessment activities (Figure 18 and Figure 19).

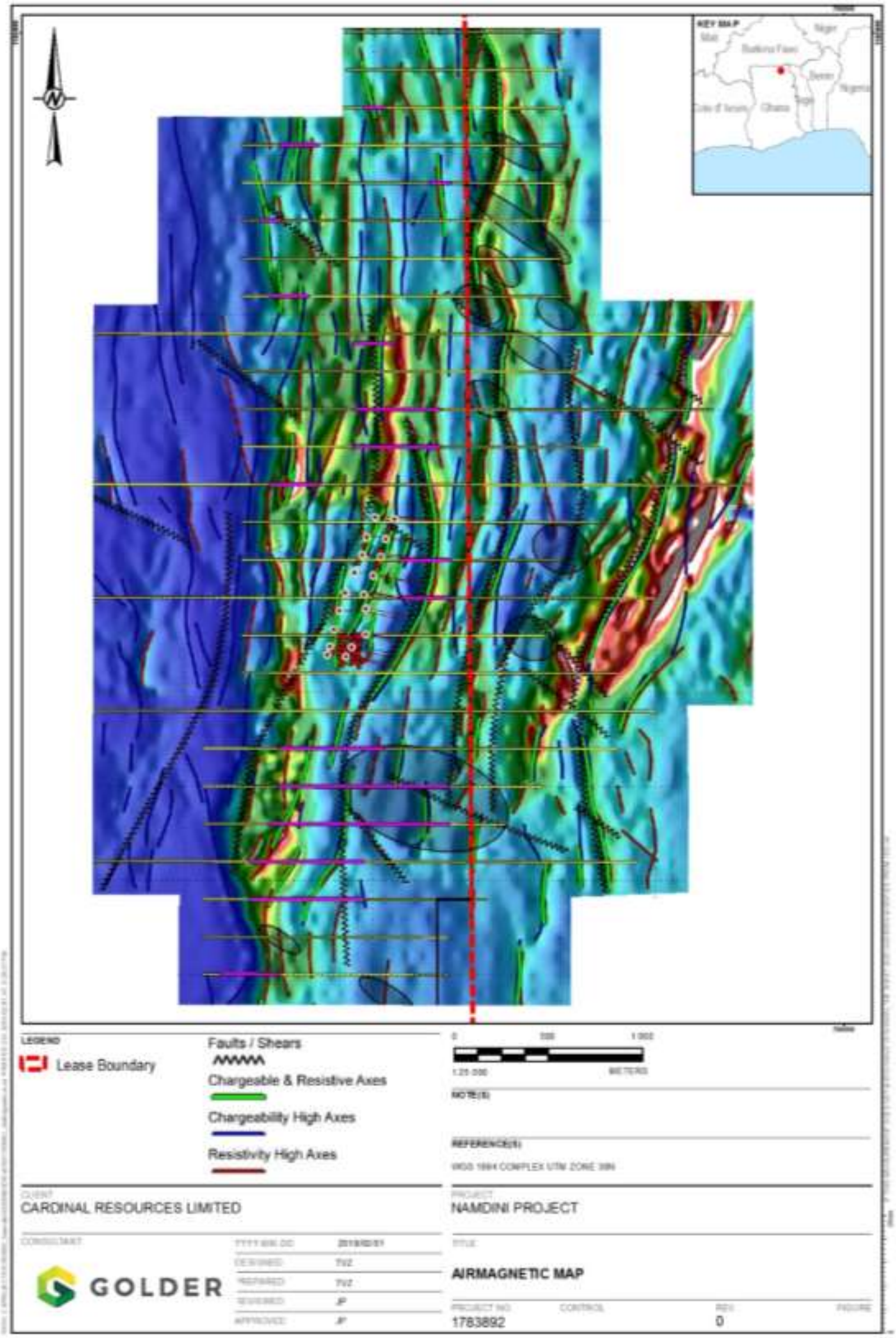


Figure 18: Aeromagnetic maps (source: Cardinal 2018)

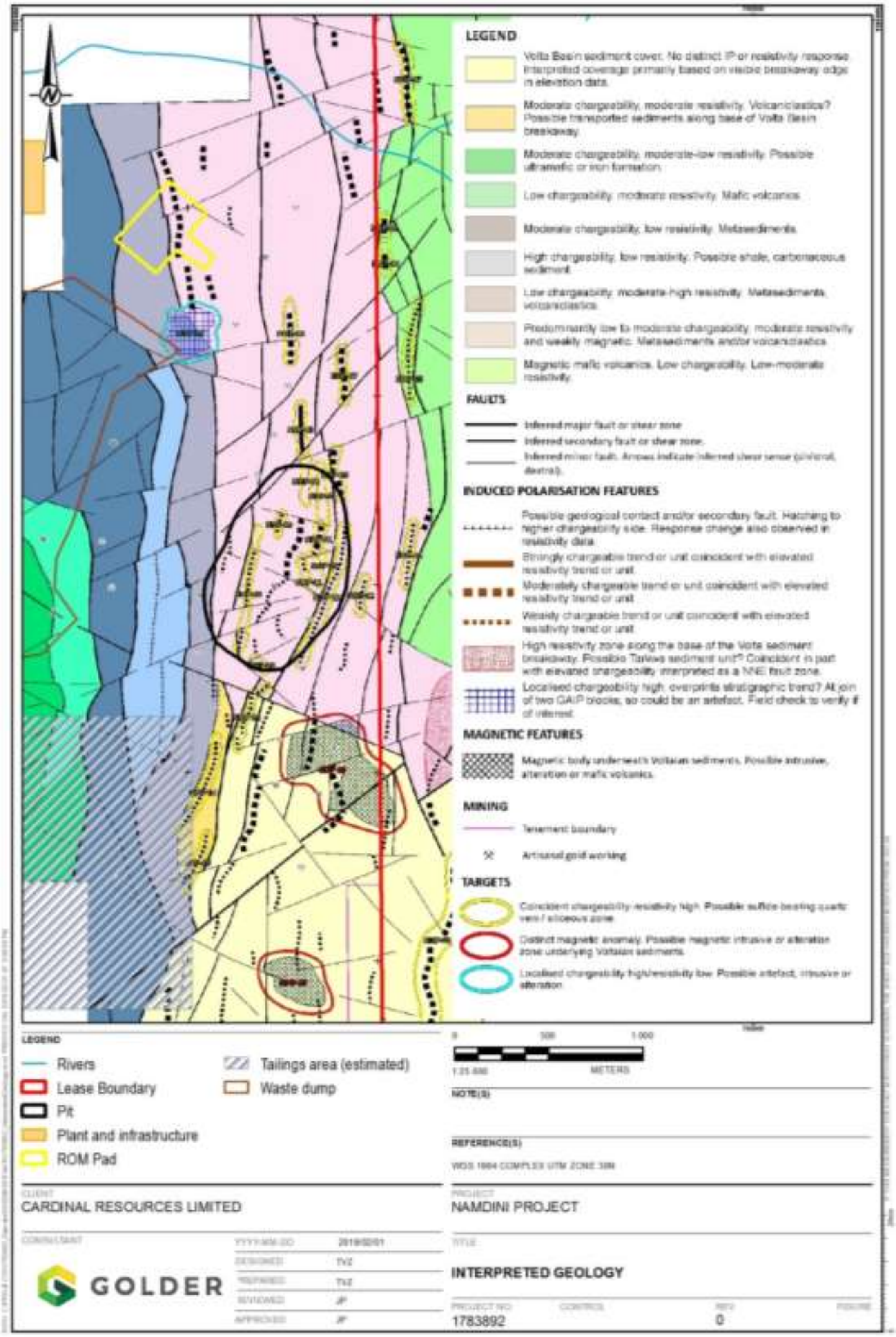


Figure 19: Interpreted geological structural features and lithology (source: Cardinal 2018)

10.0 DRILLING

The Mineral Resource estimate forming the basis for this PFS was carried out by MPR (2018) which is a comprehensive report provided on Sedar (https://www.sedar.com). The information in this Section is derived from MPR (2018).

10.1 Summary

The Mineral Resource estimates described in this report are based on RC and diamond information available for Namdini on 15 February 2018, totalling 311 holes for 82,870 m of drilling. RC and diamond drilling provide around one quarter and three quarters of the estimation dataset respectively.

Key aspects of the resource drilling and the information available to demonstrate sampling reliability include the following:

- Collar locations most resource holes were accurately surveyed by differential GPS techniques. For three resource holes the supplied database includes only hand-held GPS collar surveys. Collar locations were surveyed with sufficient accuracy for the current estimates.
- Most diamond holes and deeper RC holes were down-hole surveyed at generally 30 m intervals. These comprehensively surveyed holes provide around 79% of the mineralized domain estimation dataset. The remaining composites are from holes with no or incomplete down-hole surveying and have less reliably defined positions. Hole paths were located with sufficient accuracy for the current estimates.
- Core recovery measurements, which are available for around 96% of diamond drilling average 99.6% consistent with high quality, reliable diamond drilling.
- RC drilling is dominated by samples logged as dry with moist and wet samples representing insignificant proportions. Any uncertainty over the reliability of moist or wet samples does not affect general confidence in estimated resources.
- At around 85%, the average estimated RC sample recovery, is consistent with high quality RC sampling.
- Field duplicates show generally reasonable repeatability consistent with good quality RC sampling for comparable mineralization styles.
- Samples from Cardinal's RC rig show greater variability than those from other RC rigs employed at Namdini. This includes notably lower average estimated recoveries for the first sample of each drill rod and lower average gold grades for higher grade field duplicates.

The quality control measures adopted for the Namdini drilling have established that the RC and diamond sampling is representative and free of any biases or other factors that may materially impact the reliability of the sampling.

10.2 Available drilling

The Mineral Resource estimates described in this report are based on RC and diamond information available on the 15th of February 2018 as summarized in Table 13 and shown in Figure 20. RC drill metres in this table and figure include pre-collared portions of 39 diamond holes that average 97 m deep. In the right-hand plot in Figure 20, collars for additional drill holes available since the September 2017 estimates are shown as asterisks and older holes are shown as circles.

Mineral Resource estimates include only the resource RC and diamond drilling shown in Table 13 and Figure 20, and exclude non-resource drilling, comprising:

- 12 diamond holes drilled for geotechnical and metallurgical investigations for which routine down-hole assays are unavailable,
- 10 RC holes drilled for hydrological investigations without routine down-hole assays,
- 167 Sterilization and exploratory RC holes outside the resource area, and
- 317 closely spaced trial RC grade control holes.

The resource drilling comprises east-west trending traverses of easterly inclined holes. Hole spacing varies from around 12.5 by 25 metres in shallow portions of southern part of the deposit to around 50 by 50 metres and broader in the north and at depth.

Relative to drilling information available for the September 2017 resource estimates, the dataset available for the current estimates includes an additional 15,747 metres of drilling. This drilling, which represents an increase of around 23% is dominated by infill holes in central portions of the mineralized domain.

Several diamond and RC rigs were employed for the Namdini resource drilling. The database compiled for the current review indicates that the RC rigs included one rig owned by Cardinal, one rig from AMS drilling, one rig from Minerex and two rigs from Toomahit, along with several diamond rigs owned by AMS, Deeprock, Geodrill, Minerex, Toomahit and Cardinal.

Table 14 shows the proportion of the mineralized domain estimation dataset by drilling group. This table provides an indication of the relative contribution of each group to the resource estimates which is an important consideration for review of sampling quality information. Key features of this summary include the following:

- Diamond drilling provides around three quarters of the dataset with RC sampling contributing around one quarter.
- Half and quarter core samples represent around two thirds of the total sample data.
- Most mineralized domain RC composites are from holes drilled by the Cardinal (54%) and AMS rigs (29%) with the Minerex and Toomahit rigs contributing comparatively minor amounts.

Table 13: Namdini RC and diamond drilling

Group	Year	Count of Holes			Drilled Metres		
		RC	Diamond	Total	RC	Diamond	Total
Resource	2014	44	1	45	4,749.00	66.00	4,815.00
	2015	42	9	51	4,939.30	2,128.53	7,067.83
	2016	19	71	90	3,991.40	16,400.04	20,391.44
	2017[1]	36	53	89	8,208.70	26,639.25	34,847.95
	2017[2]	3	33	36	203.00	15,544.42	15,747.42
	Subtotal	***144***	***167***	***311***	***22,091.40***	***60,778.24***	***82,869.64***
Grade control		317	-	317	13,271.00	-	13,271.00
Metallurgical/Geotech/hydro		10	12	22	538.2	3644.99	4183.19
Sterilization/exploration		167	-	167	18,540.00	-	18,540.00
Subtotal		***494***	***12***	***506***	***32,349.20***	***3644.99***	***35,994.19***
Total		**638**	**179**	**817**	**54,440.60**	**64,423.23**	**118,863.83**

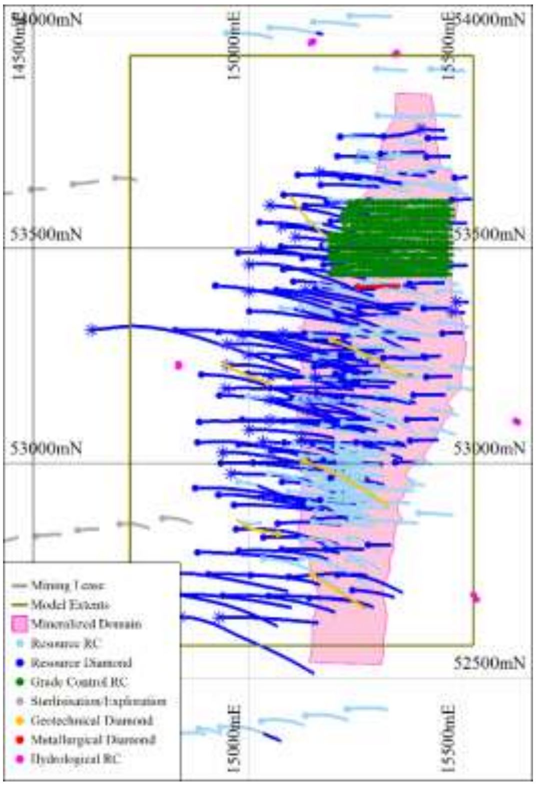
Notes: [1] Available for September 2017 estimate

[2] Additional drilling available for current estimate

Table 14: Mineralized domain composites by sampling type

Drilling Type	Group		Count of Composites	Proportion of Drill Type	Proportion of Total
RC	Drill Rig	Cardinal	4,285	54%	15%
		AMS	2,279	29%	8%
		Minerex	965	12%	3%
		Toomahit	426	5%	2%
		Subtotal	***7,955***	***100%***	***28%***
Diamond	Sample Type	Half core	12,986	64%	46%
		Quarter core	7,393	36%	26%
		Subtotal	***20,379***	***100%***	***72%***
Total			**28,334**		**100%**

Mining Lease



Resource Area

Figure 20: Namdini drilling, mineralized domain, model extents and tenement (source: MPR, 2018)

10.3 Drilling and sampling procedures

Key aspects of drilling and field sampling procedures for Namdini RC and diamond drilling include the following:

- Reverse circulation drilling:
- A Cardinal project geologist was at the rig site at all times while drilling.
- All holes were collared with six metres of PVC casing.
- Samples were collected over one metre down-hole intervals using a cyclone, with sub-sampling by a three tier-riffle splitter. Rare un-mineralized intervals were composited over four metre intervals for analysis.

- The riffle splitter was routinely cleaned with a rubber mallet and compressed air.
- Hole clearance and stabilization on every rod change and “blow-backs” for each metre.
- Recovered sample material was routinely weighed for each interval.
- Cessation of the hole if wet samples were encountered with completion by diamond drilling.
- Sieved samples were collected for geological logging in plastic chip trays, with initial logging at the drill site and follow up logging at the Bolgatanga office. The chip trays are securely stored for future reference.
- Diamond drilling:
- Cardinal core technicians were at the rig site at all times while drilling.
- Core orientation of every core run.
- Geotechnical logging at the rig prior to core being put in core boxes, including recording recovery for every core run.
- Core photography (wet and dry).
- Geological logging using tablet-based software.
- All diamond drilling was at HQ diameter, with soft near surface materials drilled with a triple tube core barrel to reduce core losses.
- For drilling to approximately April 2016 diamond core was halved for sub-sampling with a diamond saw. For later drilling, the core was quartered for assaying, until June 2017 when diamond core was halved for sub-sampling. Sample intervals range from 0.2 to 1.8 metres in length, with majority of samples assayed over one metre intervals.

The photographs in Figure 21 show a typical diamond hole drill site layout and typical RC drilling site at completion of sampling for the hole.

AMS Diamond drill rig

Arranged RC samples after splitting

Figure 21: Photographs of RC and diamond drilling sites in 2017 (source Cardinal)

10.4 Collar and down-hole surveying

At completion of drilling, hole collars were encased in concrete and hole details inscribed into the concrete and written on the collar pipe.

After completion of drilling all collar locations were surveyed with a hand-held GPS unit. Collar locations were generally subsequently surveyed by qualified independent surveyors from Sahara Mining Services using high accuracy differential GPS (DGPS) techniques.

For three diamond holes which provide 2% of the mineralized domain composite dataset for the current modelling includes only hand-held GPS collar surveys. Although hand-held GPS surveys generally provide reasonably accurate plan view coordinates, they commonly suffer from elevation inaccuracies.

The collars of some holes should be surveyed by an appropriately accurate method if possible Any resource hole collars that cannot be located should have their GPS elevations adjusted to the LIDAR triangulation.

Table 15 and Figure 22 show the number and proportion of main mineralized domain composites in the estimation dataset by down-hole survey availability.

Most diamond holes and deeper RC holes were surveyed by electronic Reflex Ez-Shot single-shot or gyroscopic tools with an initial survey at six, or rarely around 30 metres depth, and subsequent surveys at generally 30 metre intervals to hole end. Drilling intervals with such comprehensive down-hole surveying contribute 79% of mineralized domain composites. The remaining mineralized composites are from intervals with variable down-hole survey coverage including the following:

- Down-hole surveys are not available for holes providing around half of the mineralized domain RC estimation dataset.
- Rare RC holes, which provide 6% of the RC composites, were surveyed within the drill rods giving only inclinations and not azimuths. Locations of these hole paths have not been accurately defined.
- Portions of drill holes, including RC pre-collars of diamond holes surveyed at intervals of greater than 60 metres provide 6% of the combined mineralized composites. For diamond holes without poorly surveyed pre-collars, the locations of diamond tails of are not accurately defined.

The proportion of mineralized domain composites with incomplete or no down-hole surveying reduces with depth. To depths of around 60 metres, less than half of these data have comprehensive downhole surveys. Below 200 metres depth, around 90% have full down-hole survey information, and below approximately 550 metres, all composites are from comprehensively down-hole surveyed holes.

Due to the relatively wide drill hole spacing and broad mineralized zones, the lack of comprehensive accurate downhole surveys is of little concern for the current estimates. The hole paths were located with sufficient accuracy for the current estimates.

Table 15: Mineralized domain composites by downhole survey type

Downhole Survey Availability	Count of Composites			Proportion		
	RC	DDH	Total	RC	DDH	Total
Un-surveyed	3,740	-	3,740	47%	-	13%
Dip only	466	-	466	6%	-	2%
Sparsely surveyed > = 60 m spacing	640	1140	1780	8%	6%	6%
Comprehensive generally < = 30 m	3,109	19,239	22,348	39%	94%	79%
Total	**7,955**	**20,379**	**28,334**	**100%**	**100%**	**100%**

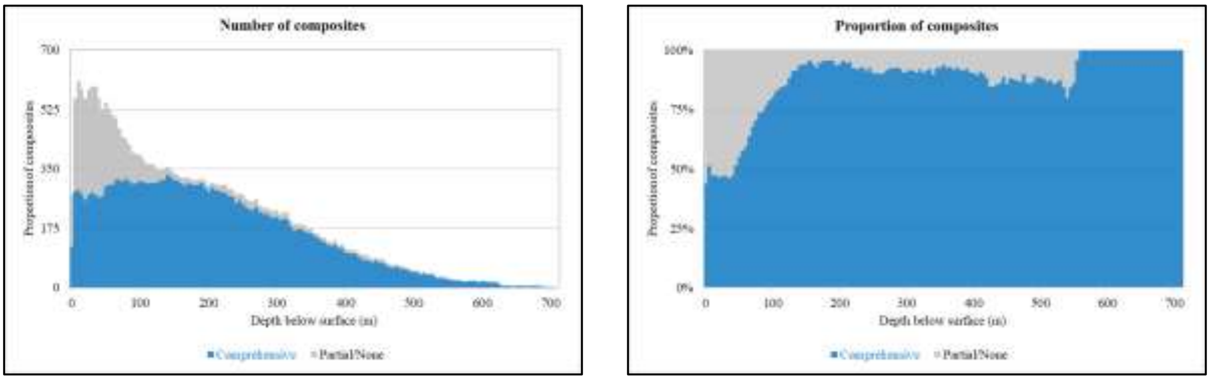


Figure 22: Down-hole surveying for mineralized domain composites (source: MPR, 2018)

10.5 Diamond core recovery

Core recovery measurements are available for around 96% of the Namdini resource diamond drilling. These measurements were supplied as recovered lengths for core runs which range from 0.1 to 8.5 m and are dominated by 3 m intervals. These data were composited to 3 m intervals to provide a consistent basis for analysis. Table 16 summarises core recoveries for the three metre composites by modelling domain.

The combined dataset of fresh rock core recoveries averages 99.9% with only approximately 1% of composites showing recoveries of less than 99%. These recoveries are consistent with high quality diamond drilling. Although lower than for Fresh rock, average core recoveries for the Oxide and Transition zones are within the range of high-quality diamond drilling.

Table 16: Diamond core recovery by domain

Oxidation Domain	Background		Mineralized		Total	
	Count	Ave. Recov.	Count	Ave. Recov.	Count	Ave. Recov.
Oxide	291	93.5%	237	95.0%	528	94.2%
Transition	453	97.1%	228	96.6%	681	97.0%
Fresh	5,969	99.8%	13,156	99.9%	19,125	99.9%
Total	**6,713**	**99.4%**	**13,621**	**99.8%**	**20,334**	**99.6%**

10.6 RC sample reliability

10.6.1 RC sample condition logging

Sample condition is an important factor in the reliability of RC sampling, and wet samples can be associated with unrepresentative, potentially biased samples.

Table 17 summarises sample condition logging for assayed RC samples demonstrating that samples logged as moist or wet represent an insignificant proportion of RC drilling. Any uncertainty over the reliability of moist or wet samples does not affect general confidence in estimated resources.

Table 17: RC Sample condition logging

Drill Rig	Count of Composites				Proportion			
	Dry	Moist	Wet	Total	Dry	Moist	Wet	Total
Cardinal	10,147	-	1	10,148	99.99%	-	0.01%	100.00%
AMS	6,334	29	4	6,367	99.48%	0.46%	0.06%	100.00%
Minerex	3,374	7	2	3,383	99.73%	0.21%	0.06%	100.00%
Toomahit	1,397	-	-	1,397	100.00%	-	-	100.00%
Total	**21,252**	**36**	**7**	**21,295**	**99.80%**	**0.17%**	**0.03%**	**100.00%**

10.6.2 RC sample recovery

Recovered RC sample weights are available for around half of the Namdini resource RC drilling. The following summaries of RC sample recovery are derived from MPR (2017), reflecting the limited additional RC information available since that time.

In conjunction with bit diameters, density measurements and moisture content estimates, recovered sample weights provide an indication of sample recovery for RC drilling which is an important factor for assessment of the reliability of the sampling. Sample recovery for high quality RC drilling typically averages around 80%. Experience also suggests that sample recoveries of consistently less than approximately 70% can be associated with unrepresentative samples and significantly biased assay grades.

Due to progressive wear, average bit diameters are likely to be less than the nominal values of 140 mm (5¼ inch) or 125 mm shown in supplied data files. For assessment of sample recovery, specified diameters were reduced by 1/8” (~3.2 mm).

For each weighed RC sample, recoveries were estimated using the bulk densities assigned to resource estimates with no allowance for moisture content. The range of moisture contents for Namdini RC samples is uncertain.

Table 18 summarises average estimated RC sample recovery by drilling rig and oxidation domain. At around 86%, average estimated recovery for Transition and Fresh material which dominates the estimated resources, is consistent with high quality RC sampling. At 74%, average recovery estimated for oxide material is somewhat lower, though still within the range of reasonable quality RC sampling.

Figure 23 shows example plots of average estimated RC sample recovery by down-hole depth. The plots for Cardinal’s rig show a distinctly cyclic trend with lower values at six metre increments representing the first sample of 6 m drill rod.

Cyclic recovery trends are common for RC drilling and generally reflect material lost as the driller blows the hole clean at the start of each rod. In cases where the down-hole recovery variability is greater than around 15% it can reflect depth measurement inaccuracies.

At around 30%, the difference between average estimated recovery for the first and subsequent samples of each drilling rod shown by Cardinal’s rig is outside the expected range for high-quality RC samples. Reasons for this trend are uncertain. As shown by the lower-right plot in Figure 23, mineralized domain samples from the Cardinal rig show no consistent variability in average gold grade with rod position, suggesting the variability in sample recovery does not significantly affect the RC samples. Additional investigations of this trend may be warranted as evaluation of the project continues.

Table 18: Average RC sample recovery by domain

	Oxide		Transition		Fresh		Total	
	Count	Ave.	Count	Ave.	Count	Ave.	Count	Ave.
AMS	197	78%	268	90%	4,557	85%	5,022	85%
Cardinal	478	74%	459	86%	2,598	86%	3,535	84%
Minerex	346	73%	208	90%	1,976	88%	2,530	86%
Toomahit	82	74%	78	84%	784	81%	944	81%
Combined	**1,103**	**74%**	**1,013**	**87%**	**9,915**	**86%**	**12,031**	**85%**

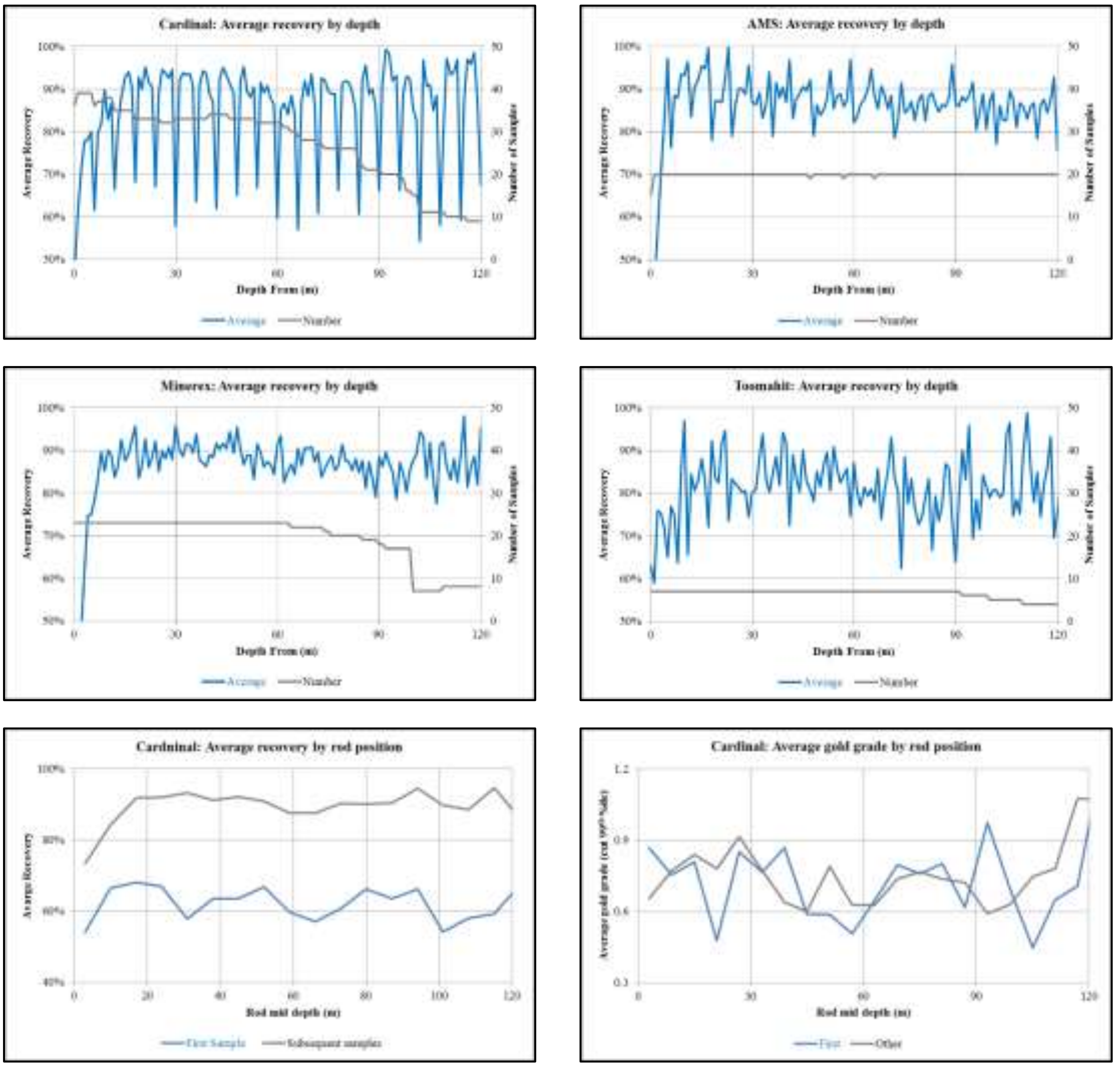


Figure 23: RC sample recovery by drilling depth (source: MPR, 2018)

10.6.3 RC field duplicates

Namdini RC sampling included routine collection of field duplicate samples which were collected consistently with and assayed in the same batch as original samples providing an indication of the repeatability of field sampling.

Field duplicate assays described in this report include all data from holes drilled by the rigs used for the resource drilling, including data from outside the resource area. Duplicates from drilling rigs used for only sterilization and grade control drilling are not included. This approach reflects the consistency of sampling and assaying and maximizes the size of the review dataset. The compiled field duplicates represent a frequency of around one duplicate per 15 primary samples.

Table 19 and Figure 24 compare gold assays for original and field duplicate samples subdivided by drilling rig. The duplicate assays show generally reasonable repeatability which is consistent with good quality RC sampling for comparable mineralization styles and confirms the repeatability of the RC field sampling.

For gold grades greater than 2.0 g/t, duplicates from Cardinal's rig show lower average grades than original samples. Reasons for this trend, which includes comparatively few samples, are uncertain.

Table 19: RC field duplicates

Au g/t	Cardinal		AMS		Minerex		Toomahit	
	Full Range		Full Range		Full Range		Full Range	
	Orig.	Dup.	Orig.	Dup.	Orig.	Dup.	Orig.	Dup.
Number	507		1,239		153		250	
Mean	**0.72**	**0.67**	**0.71**	**0.66**	**0.53**	**0.52**	**0.18**	**0.20**
Mean dif.		**-7%**		**-7%**		**-1%**		**13%**
CV	2.40	2.17	2.42	2.20	5.04	4.04	4.76	5.07
Minimum	0.00	0.01	0.00	0.01	0.01	0.01	0.01	0.01
1st Quartile	0.03	0.03	0.03	0.03	0.01	0.01	0.01	0.01
Median	0.18	0.19	0.17	0.18	0.01	0.02	0.01	0.01
3rd Quartile	0.61	0.63	0.58	0.61	0.17	0.19	0.04	0.04
Maximum	15.8	19.2	15.8	19.2	75.1	42.5	10.3	12.9
Pearson Correl.	0.85		0.91		0.94		0.99	

Au g/t	Cardinal		AMS		Minerex		Toomahit	
	0.10 to 10.0 g/t		0.10 to 6.0 g/t		0.10 to 8.0 g/t		0.1 to 6.0 g/t	
	Orig.	Dup.	Orig.	Dup.	Orig.	Dup.	Orig.	Dup.
Number	287		345		75		2	
Mean	**0.98**	**0.97**	**1.60**	**1.61**	**0.76**	**0.78**	**0.72**	**0.75**
Mean dif.		**-1%**		**0%**		**3%**		**4%**
CV	1.29	1.20	1.69	1.69	1.39	1.37	1.17	1.05
Minimum	0.10	0.10	0.10	0.10	0.10	0.11	0.11	0.10
1st Quartile	0.28	0.27	0.30	0.29	0.22	0.25	0.17	0.19
Median	0.50	0.51	0.60	0.64	0.40	0.41	0.26	0.32
3rd Quartile	1.12	1.25	1.59	1.50	0.74	0.77	0.96	1.17
Maximum	8.62	7.94	20.0	20.2	6.00	6.65	3.20	2.72
Pearson Correl.	0.93		0.94		0.91		0.92	

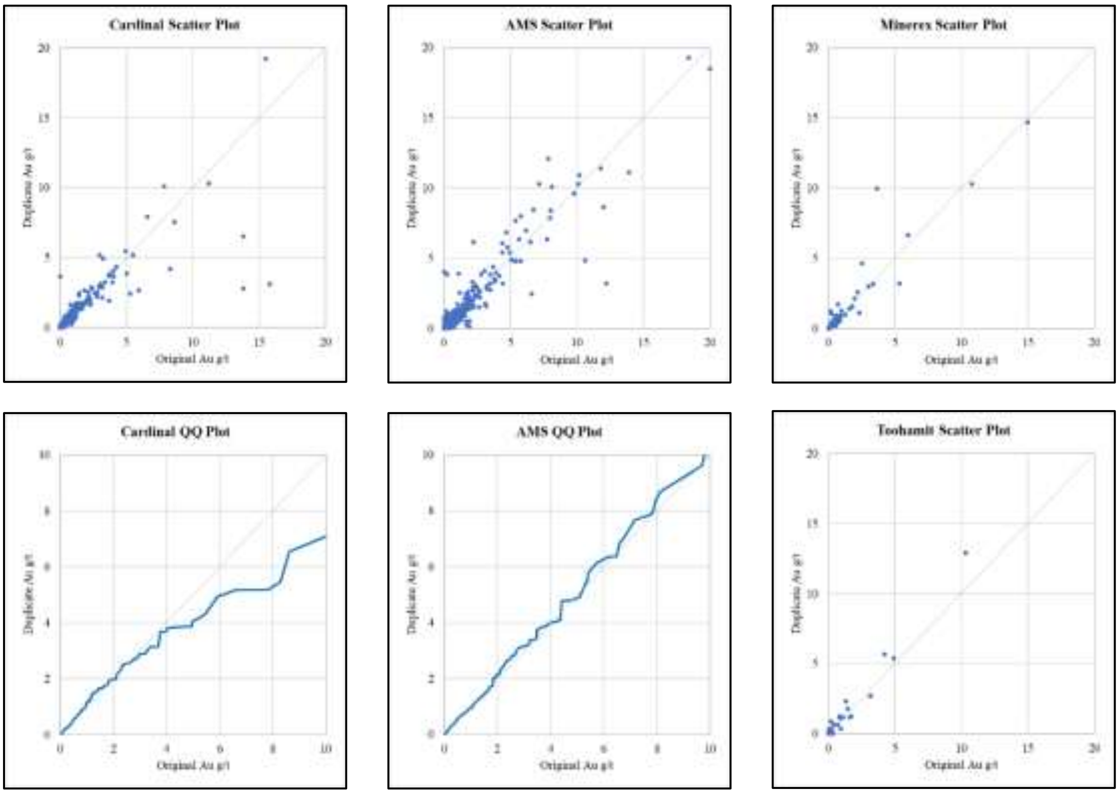


Figure 24: RC field duplicates (source: MPR, 2018)

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

The Mineral Resource estimate forming the basis for this PFS was carried out by MPR (2018) which is a comprehensive report provided on Sedar (https://www.sedar.com). The information in this Section is derived from Golder (2018) and MPR (2018).

11.1 Sample security

All sample preparation and gold analyses of resource samples was undertaken by independent commercial laboratories.

Diamond core and RC samples were transported from the drill site by Cardinal vehicle to secure storage at the Bolgatanga field exploration office. Core yard technicians, field technicians and geologists ensured samples were logged, prepared and securely stored until collected for transportation to the assay laboratories by personnel employed by the assay laboratory.

All samples collected for assaying were retained in a locked secure shed until collected by laboratory personnel. Retained drill core and RC chips are securely stored in the core storage compound, and pulps are securely stored in the core shed (Figure 25).

At the time of sample collection, a sign-off process between Cardinal and the laboratory truck driver ensured samples and paper work correspond. The samples were then transported to the laboratory where they were receipted against the dispatch documents. The assay laboratories were responsible for the samples from the time of collection from the exploration office.

No employee, officer, director, or associate of Cardinal carried out any sample preparation on samples from the Namdini Gold Project exploration programme.

Core trays laid out for logging

Core tray secure storage

Figure 25: Core tray logging and storage in 2017 (source: Cardinal)

11.2 Sample preparation and analysis for gold

Drill sampling procedures are described in Section 10.3. All sample preparation and gold assaying of primary samples from the Namdini resource drilling was undertaken by independent commercial laboratories. Analyses by Cardinal were limited to around 83% of the density samples.

Samples from sterilization drilling were analysed by ALS in Kumasi, Ghana. These samples are not included in estimated Mineral Resources and are not discussed in this report.

Primary gold analyses of samples from resource drilling were undertaken at SGS laboratories in Ouagadougou in Burkina Faso or SGS Tarkwa in Ghana. Selected samples from the resource drilling were assayed for additional attributes including sulfur and arsenic. These data were not included in the current estimates.

Table 20 summarises the contribution of each laboratory to assayed composites from the estimation dataset. This table demonstrates that samples analysed by SGS Tarkwa represent around two thirds of the estimation dataset with SGS Ouagadougou contributing around one third. The subdivision of SGS Ouagadougou assays by phase reflects variability in the information available to demonstrate assay reliability.

SGS Tarkwa and Ouagadougou are accredited by the South African National Accreditation System ("SANAS") for meeting the requirements of the ISO/IEC 17025 standard for specific registered tests for the minerals industry.

SGS Ouagadougou and SGS Tarkwa employed consistent sample preparation and analytical procedures as follows:

- Samples were sorted and weighed before being oven dried before and crushed to 75% passing 2 mm.
- A 1.5 kg riffle split sub sample was pulverized to nominally 85% at 75 µm. The remaining coarse reject was retained.
- The pulverized samples were thoroughly mixed on a rolling mat ('carpet rolled') and then 200 g of sub-sample was collected.
- Samples were fire assayed for gold using a 30 or 50 g charge with an atomic absorption finish, with a detection limit of 0.01 g/t. Assays of greater than 100 g/t were re-analysed with a gravimetric finish.
- Remaining reject and pulverized samples were returned to Cardinal's Bolgatanga Exploration Office for secure storage.

Table 20: Estimation dataset by assay laboratory

Assay Laboratory		Count of Composites			Proportion		
		Background	Mineralized	Total	Background	Mineralized	Total
SGS Ouaga.	April 2014 to May 2016	1,342	6,935	8,277	11%	24%	21%
	July 2017 to January 2018	1395	2982	4,377	12%	11%	11%
	Subtotal	***2,737***	***9,917***	***12,654***	***23%***	***35%***	***32%***
SGS Tarkwa		9,018	18,414	27,432	77%	65%	68%
Total		**11,755**	**28,331**	**40,086**	**100%**	**100%**	**100%**

11.3 Monitoring of assay reliability

11.3.1 Blanks

Cardinal routinely included samples of un-mineralized granite collected from a quarry outside the Namdini area in assay batches. Blank assays described in this report include all results from the assay laboratories used for resource drilling, including data from outside the resource area. Submission frequency of these samples averaged one blank per 30 primary samples.

For sampling prior to May 2015 and after September 2017, the inserted blanks comprised coarse rock chips. For sampling between these dates, samples of pulverized blank material prepared by SGS Tarkwa were used.

The coarse blank samples, which require crushing and pulverization prior to analysis test for contamination during sample preparation and provide a check of sample misallocation by field staff, the laboratory and during database compilation. Fine blanks do not require preparation by the analytical laboratory and do not test for contamination during sample preparation. These samples primarily test for sample misallocation.

Table 21 summarises the composite estimation dataset subdivided by assay laboratory and the type of blanks inserted in each assay batch. This indicates that assay batches with coarse blanks represent around 27% of the estimation dataset, and for the remaining 73% of resource composites, no information is available to check for contamination during sample preparation.

Testing for sample contamination is an important aspect of sample quality monitoring for resource datasets, and the lack of coarse blanks for the majority of assaying of Namdini resource samples is unusual.

Routine submission of appropriate coarse blank material consistent with the protocols adopted for recent drilling should be continued for future drilling.

Modifications to the supplied dataset of blank assay results were limited assigning all below detection assays a value representing half the dominant detection limit (0.005 g/t Au).

Table 22 summarises blank assay results by type and assay laboratory and Figure 26 shows blank assay results by type and assay date. This table and Figure demonstrate that the blank assays show generally low gold grades with no indication of common contamination or sample misallocation. Additional features shown by Table 22 and Figure 26 include the following:

- Relative to SGS Ouagadougou, SGS Tarkwa reports notably higher proportion of coarse and fine blank assays as above detection limit. Reasons for this trend are uncertain, however it is suggestive of comparatively greater low-level contamination by SGS Tarkwa.
- Most of the samples with comparatively elevated gold grades are from later batches assayed between August and December 2017. Although uncertain, this trend appears consistent with increased sample mis-allocation or contamination for a comparatively short period.

Table 21: Estimation dataset by blank types

Assay Laboratory	Count of Composites				Proportion			
	Coarse	Fine	None	Total	Coarse	Fine	None	Total
SGS Ouagadougou	3,275	9,064	315	12,654	26%	72%	2%	100%
SGS Tarkwa	7,491	19,582	359	27,432	27%	71%	1%	100%
Total	**10,766**	**28,646**	**674**	**40,086**	**27%**	**71%**	**2%**	**100%**

Table 22: Blank assay results

Blank Type	Assay Laboratory (SGS)	Assays (g/t Au)			Above Detection	
		Assays	Average	Maximum	Count	%
Coarse	Ouagadougou	165	0.005	0.01	-	-
	Tarkwa	384	0.01	0.17	73	19%
	Subtotal	*549*	*0.01*	*0.17*	*73*	*13%*
Fine	Ouagadougou	612	0.01	0.16	17	3%
	Tarkwa	1,031	0.01	0.04	158	15%
	Subtotal	*1,643*	*0.01*	*0.16*	*175*	*11%*
Combined	Ouagadougou	777	0.00	0.16	17	2%
	Tarkwa	1,415	0.01	0.17	231	16%
	Total	**2,192**	**0.01**	**0.17**	**248**	**11%**

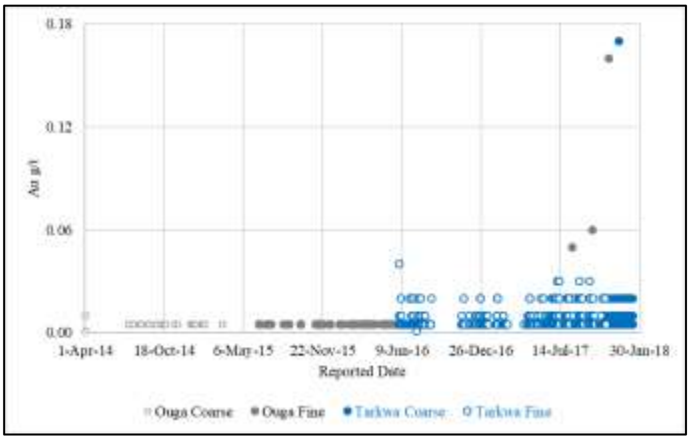


Figure 26: Blank assay results by date (source: MPR, 2018)

11.3.2 Certified reference standards

Cardinal's monitoring of assay reliability included insertion of samples of certified reference standards prepared by Geostats Pty Ltd, Perth, Western Australia in assay batches. The standards, which were inserted at an average rate of around 1 standard per 41 primary samples, have expected gold grades of 0.27 to 6.70 g/t Au.

Table 23 and Figure 27 summarise reference standard assays by laboratory. This table and Figure demonstrate that although there is some variability for individual samples, for both laboratories, average assay results generally reasonably reflect expected values, with no evidence of material biases. However, the results show some variability including the following:

- For standards with expected gold grades of greater than 1.5 g/t, SGS Tarkwa reports average results around 3% higher than expected values, with no notable variability with time.
- For 2015 and 2016 assaying, of standards with expected gold grades of less than 1.5 g/t SGS Ouagadougou reports average grades around 3% lower than expected values. For assaying after June 2017, the laboratory reports average grades around 2% higher than expected values for these standards.

Magnitudes of these differences are not significant at the current level of project assessment.

Table 23: Summary of reference standards assays

Period	Expected Value Range	Count of Assays	Average Grade Au g/t		Ave. *vs* Expected
			Expected	Assay	
SGS Ouagadougou					
2015 and 2016	<1.5 g/t	249	0.50	0.48	-3%
	>1.5 g/t	160	3.35	3.38	1%
	Subtotal	***409***	***1.61***	***1.62***	***0%***
2017	<1.5 g/t	195	0.51	0.52	2%
	>1.5 g/t	176	4.02	4.04	1%
	Subtotal	***371***	***2.17***	***2.19***	***1%***
Combined	<1.5 g/t	444	0.50	0.50	-1%
	>1.5 g/t	336	3.70	3.73	1%
	Subtotal	***780***	***1.88***	***1.89***	***1%***
SGS Tarkwa					
2016 and 2017	<1.5 g/t	503	0.69	0.69	0%
	>1.5 g/t	920	3.35	3.44	2%
	Subtotal	***1,423***	***2.41***	***2.47***	***2%***

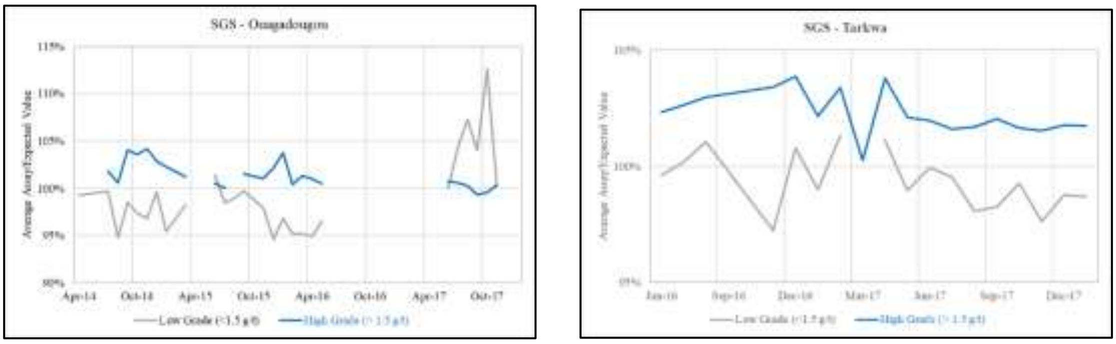


Figure 27: Reference standards results by date (source: MPR, 2018)

11.3.3 Inter-laboratory repeats

Cardinal's monitoring of assay reliability included submitting 2,709 pulp samples initially assayed by SGS Ouagadougou and SGS Tarkwa to Intertek for check assaying, comprising the following:

- 746 assays from June 2016 which test samples initially assayed by SGS Ouagadougou between 2014 and mid-2016
- 961 assays performed since October 2017 which test samples initially assayed by SGS Ouagadougou between July 2017 and November 2017
- 1,002 assays performed since October 2017 which test samples initially assayed by SGS Tarkwa between June 2016 and August 2017.

Intertek assayed the pulp samples by fire assay using a comparable method to SGS Ouagadougou and SGS Tarkwa.

The combined inter-laboratory pulp repeats represent around one repeat per 30 primary resource assays and provides reasonably representative coverage of all sampling phases, except for assaying since December 2017 which provides around 15% of the estimation dataset.

For preparation of the tables and figures in this section, the repeats of SGS Ouagadougou assays were subdivided into early (April 2014 to May 2017) and late (July to November 2017) phases based on primary assay date. This subdivision is consistent with the variability in standards assay results described above.

No reference standards assays were supplied for the 2016 Intertek repeat assay batches. Reliability of these assays have not been independently established. The later batches included reference standards. Assays for these samples are summarized in Table 24 with subdivision into Intertek batches which primarily repeated samples initially assayed by SGS Tarkwa and SGS Ouagadougou respectively. This table demonstrates the following:

- For both datasets, Intertek reports higher average gold grades than the expected values.
- The magnitude of this difference increases with increasing gold grade.
- For assay batches repeating original SGS Ouagadougou assays, the differences in average grade are generally greater than for assay batches repeating original SGS Tarkwa assays.

Although uncertain, standards results are suggestive of a potential positive bias in the later groups of Intertek repeat assaying. Suitability of these assays for detailed monitoring of assay accuracy of the primary assays are uncertain. It is uncertain whether differences in average grade shown by the repeats relative to the original assays reflect biases in the original or repeat assays.

The supplied 2016 inter-laboratory repeats include 12 particularly poorly correlating pairs which are shown in the scatter plots in Figure 28 as blue dots. The QQ plots in Figure 28 and summary statistics in Table 25 exclude these anomalous pairs. For many of these pairs the magnitude of the difference is suggestive of sample misallocation.

Cardinal should investigate database entries for all anomalous repeats.

Notable features of the inter-laboratory pulp repeats of SGS Tarkwa assays include the following:

- With an average gold grade of greater than 5 g/t, the repeated samples are notably higher grade than typical Namdini mineralization, and representativity of these repeats are uncertain.
- Intertek reports similar average gold grades to SGS Tarkwa. This supports the general accuracy of SGS Tarkwa assaying. However, due to the variability in standards assays for both SGS Tarkwa and Intertek, there is some uncertainty over the detailed accuracy of SGS Tarkwa assaying.

Notable features of the inter-laboratory pulp repeats of SGS Ouagadougou assays include the following:

- Excluding the anomalous results, for pre-May 2017 primary assaying Intertek reports very similar average grades to SGS supporting the general accuracy of earlier SGS Ouagadougou assaying.
- For post July 2017 primary assaying, Intertek reports approximately 9% higher average gold grades than SGS Ouagadougou. Although of greater magnitude, this difference in broadly comparable to trends shown by Intertek standards assays.

Each set of inter-laboratory repeats broadly support the general reliability of the original SGS assaying. However, detailed accuracy of the Intertek assays is uncertain, and their suitability for use as inter-laboratory repeats are uncertain.

Future work should include additional investigations of primary assay accuracy, such as further inter-laboratory repeats with comprehensive QAQC monitoring.

Table 24: Reference standards results for Intertek repeat batches

Standard	Expected (g/t Au)	Assay Results (g/t Au)				Ave. *vs* Expected
		Count	Minimum	Average	Maximum	
Batches repeating SGS Tarkwa assays						
STD501	0.43	9	0.40	0.42	0.45	-2%
STD413	0.79	14	0.72	0.82	0.91	3%
STD588	1.60	13	1.56	1.69	1.81	6%
STD611	4.03	15	4.14	4.29	4.57	7%
STD640	6.70	13	6.33	6.87	7.29	3%
Combined	**2.86**	**64**	**0.40**	**2.98**	**7.29**	**4%**
Batches repeating SGS Ouagadougou (July to November 2017) assays						
STD710	0.30	7	0.30	0.31	0.34	3%
STD501	0.43	7	0.40	0.44	0.47	1%
STD684	0.75	8	0.75	0.81	0.84	7%
STD413	0.79	6	0.76	0.83	0.87	5%
STD588	1.60	17	1.52	1.70	1.85	6%
STD611	4.03	20	3.90	4.28	4.59	6%
STD640	6.70	8	6.61	7.04	7.34	5%
Combined	**2.43**	**73**	**0.30**	**2.57**	**7.34**	**6%**

Table 25: Intertek inter-laboratory repeats

Intertek *vs* SGS Tarkwa				
	Full set		0.1 to 50 g/t Au	
	SGS	Intertek	SGS	Intertek
Number	1,002		967	
Mean	**2.89**	**2.96**	**2.54**	**2.58**
Mean difference		**2%**		**2%**
CV	2.79	2.62	1.64	1.61
Minimum	0.01	0.01	0.10	0.10
1st Quartile	0.51	0.51	0.54	0.56
Median	1.12	1.17	1.18	1.21
3rd Quartile	2.80	2.95	2.84	3.00
Maximum	166.0	134.3	49.6	46.8
Pearson Correl.	0.96		0.97	

Intertek *vs* SGS Ouagadougou								
	April 2014 to May 2016				July 2017 to November 2017			
	Full set		0.1 to 50 g/t		Full set		0.1 to 50 g/t	
	SGS.	Inter.	SGS.	Inter.	SGS.	Inter.	SGS.	Inter.
Number	734		595		961		899	
Mean	**3.81**	**3.80**	**3.52**	**3.57**	**6.15**	**6.68**	**6.32**	**6.86**
Mean difference		**0%**		**1%**		**9%**		**9%**
CV	3.17	2.99	1.37	1.39	1.22	1.24	1.02	1.02
Minimum	0.01	0.01	0.10	0.10	0.01	0.01	0.10	0.10
1st Quartile	0.23	0.22	0.56	0.57	0.53	0.58	0.74	0.83
Median	1.12	1.13	1.53	1.59	5.40	5.94	5.71	6.28
3rd Quartile	3.42	3.57	4.06	4.24	8.45	9.28	8.70	9.50
Maximum	220.0	197.3	43.0	48.0	89.7	115.6	43.8	44.6
Pearson Correl.	0.99		0.98		0.97		0.97	

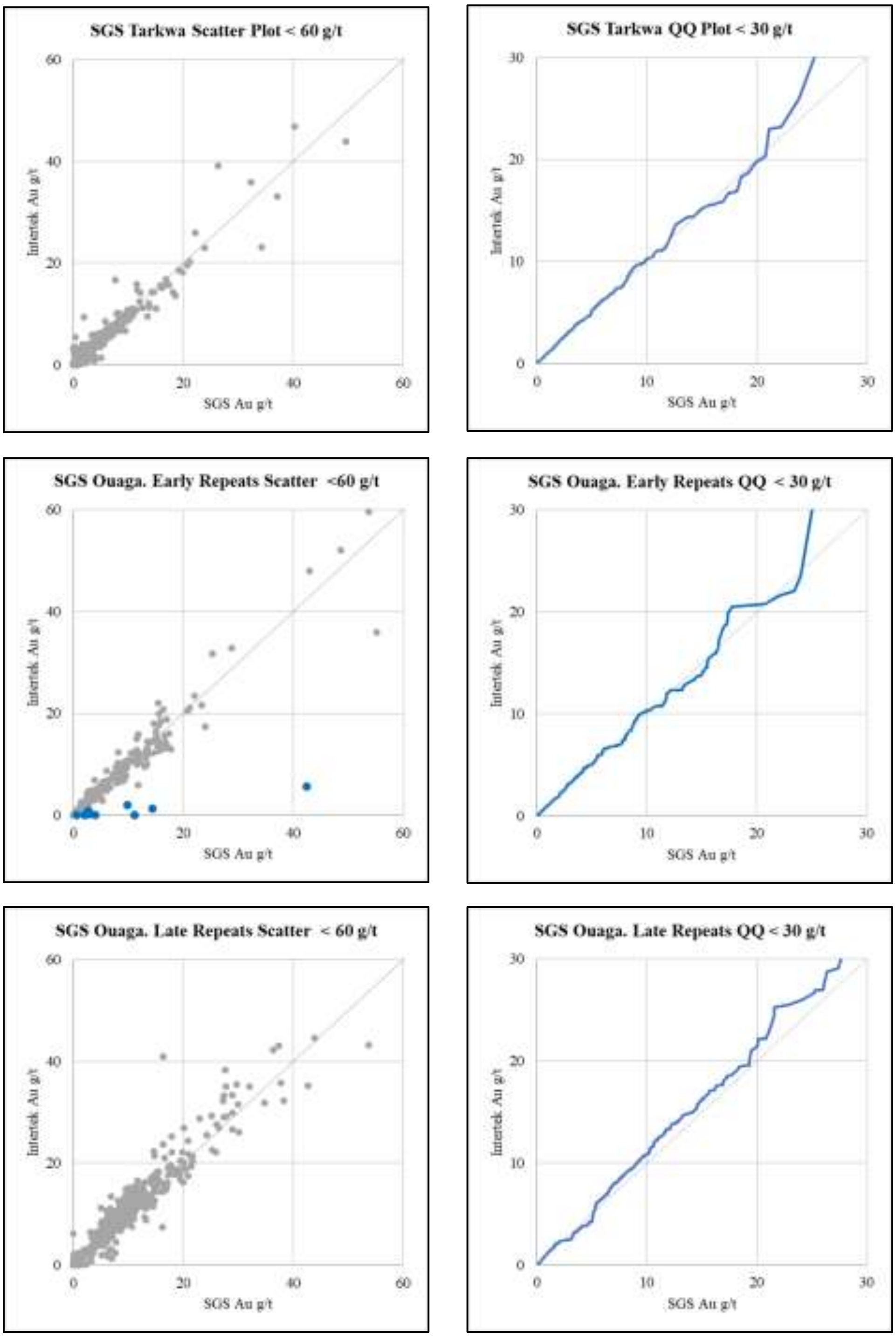


Figure 28: Intertek inter-laboratory repeats (source: MPR, 2018)

11.3.4 Inter-laboratory core duplicates

For 443 intervals of diamond core from hole NMD118, quarter core samples were assayed by SGS Tarkwa and half core samples were assayed by SGS Intertek. These results supplement the inter-laboratory pulp repeats described above.

As demonstrated by Table 26 and Figure 29, the Intertek core duplicate assays show similar average grades to those reported by SGS Tarkwa, providing additional support for SGS Tarkwa analyses.

Table 26: Intertek core duplicates

Au g/t	Full Set		0.1 to 9.0 g/t Au	
	SGS Tarkwa	Intertek	SGS Tarkwa	Intertek
Number	433		233	
Mean	**0.76**	**0.77**	**1.26**	**1.26**
Mean difference		**2%**		**0%**
CV	1.88	2.18	1.15	1.27
Minimum	0.01	0.01	0.11	0.10
1st Quartile	0.04	0.01	0.36	0.29
Median	0.21	0.17	0.64	0.60
3rd Quartile	0.72	0.72	1.72	1.42
Maximum	14.8	15.3	7.72	8.30
Pearson Correl.	0.68		0.78	

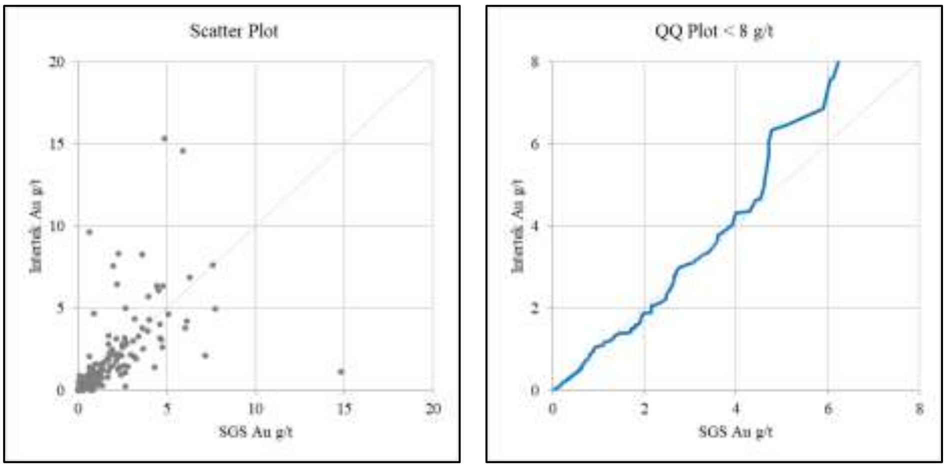


Figure 29: Intertek core duplicates (source: MPR, 2018)

11.3.5 Independent core duplicate sampling

Two sets of duplicate assays from independent quarter core check sampling by consultant geologists employed by Cardinal, are available comprising the following:

- 49 duplicates collected in July 2016, which were assayed by SGS Tarkwa and include 34 samples with original assays by SGS Ouagadougou and 15 samples with original SGS Tarkwa analyses.
- 165 duplicates collected in January 2017 and assayed by ALS Ireland, all of which have original assays by SGS Tarkwa.

11.3.5.1 July 2016 duplicates

Sample identifiers assigned to duplicate samples in the supplied data represent original sample identifiers with a suffix of "/1". The duplicates do not appear to strictly represent blind checks, reducing the integrity of the duplicate results.

For intervals with original SGS Tarkwa assays, the duplicate results closely match the original assays (Table 27, Figure 30).

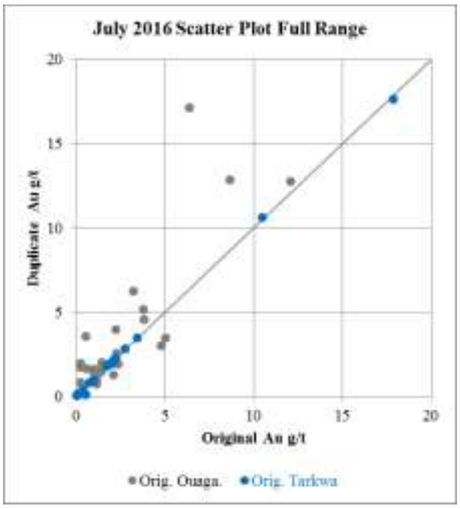
For intervals with original SGS Ouagadougou assays, duplicate results average around 35% higher than the original assays. Reasons for this trend are unclear. It may simply represent an artefact of the small dataset which is too small for the results to be material at the current level of project evaluation.

11.3.5.2 January 2017 duplicates

The duplicate ALS assay results generally reasonably match original SGS Tarkwa assays, with no notable difference in mean grades (Table 27, Figure 30). These data provide additional confidence in the general reliability of SGS Tarkwa assaying.

Table 27: Independent core duplicates

Au g/t	July 2016 Duplicates				January 2017 Duplicates			
	Original SGS Ouagadougou		Original SGS Tarkwa		Full Dataset		< 15.0 g/t	
	Ouaga.	Tark.	Tark.	Tark.	Tark.	ALS.	Tark.	ALS.
Number	34		15		165		164	
Mean	**2.26**	**3.06**	**3.15**	**3.11**	**1.41**	**1.34**	**1.26**	**1.28**
Mean difference		**35%**		**-1%**		**-5%**		**1%**
CV	1.13	1.24	1.47	1.48	1.76	1.42	1.33	1.34
Minimum	0.10	0.09	0.01	0.02	0.01	0.01	0.01	0.01
1st Quartile	0.55	1.25	0.37	0.12	0.24	0.19	0.24	0.19
Median	1.30	1.62	1.72	1.74	0.62	0.56	0.62	0.56
3rd Quartile	2.35	2.96	2.24	2.25	1.53	1.66	1.53	1.66
Maximum	12.1	17.1	17.9	17.6	24.6	12.5	11.4	12.5
Pearson Correl.	0.85		1.00		0.83		0.83	
Spearman. Correl.	0.77		0.99		0.89		0.88	



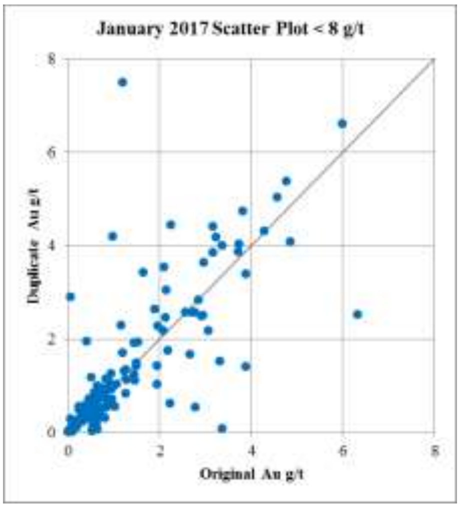


Figure 30: Independent core duplicates (source: MPR, 2018)

11.4 Bulk density measurements

Density measurements available for Namdini comprise 8,181 immersion density measurements performed by Cardinal (6,787) SGS Tarkwa (1,129) and SGS Ouagadougou (265). Oxidized and porous samples were wax-coated prior to density measurement. Lengths specified for these samples range from 0.01 to 1.4 m and average 0.3 m.

The Tables and Figures in this Section exclude anomalous density measurements with supplied values of less than 0.3 t/bcm or greater than 8.5 t/bcm giving a review dataset of 8,175 measurements.

Table 28 summarises the density measurements by rock type and weathering type, based on the wire-frames used for the resource model. For groups with reasonable numbers of samples the combined SGS measurements give very similar average values to Cardinal's measurements. This comparison provides some confidence in the reliability of Cardinal's measurements. For the Oxide and Transition subset, Cardinal's measurements average slightly higher than SGS's. Each grouping contains too few measurements for the results to be substantive and the magnitude of the differences are not significant at the current level of project assessment.

Density data on drill samples should be further investigated, including repeat measurements of selected representative intervals.

Table 28: Density measurements by rock type

Rock Unit	Weath. Zone	SGS		Cardinal		Combined		Cardinal *vs* SGS
		No.	Ave. t/bcm	No.	Ave. t/bcm	No.	Ave. t/bcm	
Metavolcanic	Oxide	7	1.98	33	2.18	40	2.14	10%
	Trans.	26	2.57	29	2.70	55	2.64	5%
	Fresh	620	2.82	1,823	2.81	2,443	2.81	0%
Tonalite	Oxide	8	2.44	12	2.51	20	2.48	3%
	Trans.	23	2.52	14	2.64	37	2.57	5%
	Fresh	227	2.73	532	2.73	759	2.73	0%
Diorite	Oxide	-	-	42	2.03	42	2.03	-
	Trans.	-	-	49	2.62	49	2.62	-
	Fresh	329	2.82	2,359	2.80	2,688	2.81	-1%
Metasediment	Oxide	4	2.22	65	2.28	69	2.28	3%
	Trans.	59	2.60	118	2.60	177	2.60	-
	Fresh	91	2.83	1,526	2.81	1,617	2.81	-
Dacite	Oxide	-	-	4	2.37	4	2.37	-
	Trans.	-	-	9	2.58	9	2.58	-
	Fresh	-	-	113	2.78	113	2.78	-
Pyroclastic	Oxide	-	-	7	2.18	7	2.18	-
	Trans.	-	-	20	2.60	20	2.60	-
	Fresh	-	-	26	2.73	26	2.73	-

The trend plots in Figure 31 compare density and gold grade for Fresh samples. This demonstrates that density measurements are not strongly correlated with gold grades. Tonalite samples show no notable association between density and grade. Metavolcanic and diorite samples demonstrate a slight general increase in average density with increasing grade. Measurements for these units show an increase in average density from around 2.80 t/bcm at low grades to 2.83 t/bcm at gold grades of around 4 g/t, an increase of around 1%. These trends are not significant at the current level of project assessment.

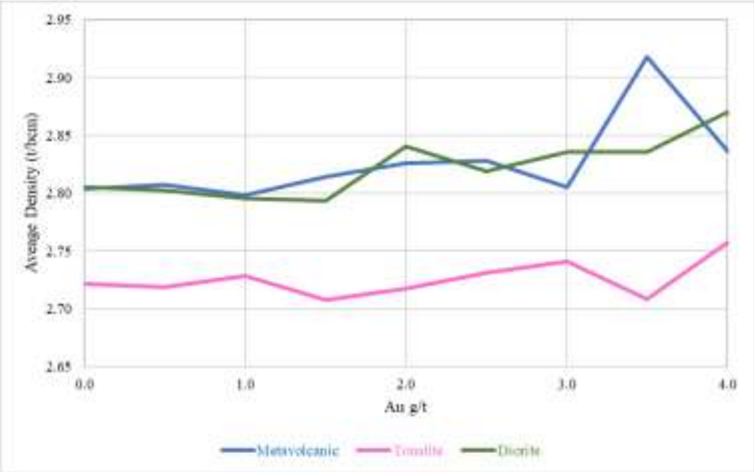


Figure 31: Density *vs* gold grade for Fresh rock types (source: MPR, 2018)

12.0 DATA VERIFICATION

The Mineral Resource estimate forming the basis for this PFS was carried out by MPR (2018) which is a comprehensive report provided on Sedar (https://www.sedar.com). The information in this Section is derived from Golder (2018) and MPR (2018).

12.1 Procedures

Cardinal maintains strict protocols with respect to the review and validation of assay results prior to importing into the drill hole database as detailed in Sections 10.3 and 11.2.

12.2 Verification of assay records

Checks undertaken by MPR (2018) to confirm the validity of the database compiled for the current study included:

- Checking for internal consistency between, and within database tables
- Comparison of assay values between nearby holes
- Comparison of assay entries with laboratory source files supplied by Cardinal.

The consistency checks showed no significant issues.

Laboratory source files supplied by Cardinal included gold assay results for 96% of primary assays in the compiled database (Table 29). Comparing assay entries in these files with database entries showed no significant inconsistencies.

The available information indicates that the drilling database was carefully compiled and validated, and that it was a reliable basis for Mineral Resource estimation.

Table 29: Laboratory source file checks

Comment	Count of Assays	Proportion
No significant differences noted	78,654	96%
Not checked	3,524	4%
Total	**82,178**	**100%**

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

This section is based on results from an extensive metallurgical testwork programme undertaken by ALS Metallurgy laboratory in Perth under the supervision of Cardinal and its consultants.

13.1 Introduction

The PFS phase of metallurgical testwork focused on development of the flowsheet as presented in Cardinal's PEA (Golder, 2018). The flowsheet is described as a conventional primary crush, SAG/Ball mill and re-crush ("SABC"), gravity, flotation, regrind and carbon-in-leach process. Metallurgical results proved to be consistent and strongly supported the flowsheet as described above.

All Fresh rock metallurgical testwork for the PFS was carried out by ALS Laboratory ("ALS") in Perth, Australia. This work followed on from the PEA metallurgical testwork completed by Suntech Geomet Laboratories ("SGL") in Johannesburg, South Africa.

Further Oxide (weathered rock) metallurgical testwork was not necessary. The Oxide PEA results were carried into the PFS.

The PFS Fresh rock was metallurgically tested under four main campaigns:

- Starter Pit and Flotation specific testwork (to test for initial ore metallurgical response):
  - Mineralogy and gold deportment
  - Gravity recoverable gold
  - Flotation
  - Cyanide leach testwork of bulk rougher flotation concentrate at various regrind sizes.
- Life of Mine ("LOM") testwork (to test for ore metallurgical response for the entire mine life):
  - Mineralogy and gold deportment
  - Gravity recoverable gold
  - Flotation
  - Cyanide leach testwork of bulk rougher flotation concentrate at various regrind sizes.
- Variability testwork (to test for combinations of gold and varying sulfur grades):
  - Mineralogy and gold deportment
  - Gravity recoverable gold
  - Flotation
  - Cyanide leach testwork of bulk rougher flotation concentrate at various regrind sizes.
- Comminution testwork:
  - Semi-autogenous mill comminution ("SMC") variability testwork using HQ core samples
  - Julius Kruttschnitt Drop Weight Test ("JKDWT") testwork using PQ core samples.

The ALS test scope focused on optimization of the flotation and cyanide leach response across a range of representative Master and Variability Composites formed from selected drill core samples.

Overall recovery is calculated as the sum of the up-front gravity recovery and the product of separate flotation and flotation concentrate leach recoveries as follows:

Overall Recovery (%) = (G + (F x L)*100)

Where:

G = Gravity (G) (%)

F = Flotation (F) (%)

L = Leach (L) (%)

Metallurgical flowsheet development testwork has been supported based on a wide range of testwork, including:

- Comprehensive Master and Variability Composite head assay analysis, XRD and QEMSCAN mineralogy, and diagnostic leach characterisation.
- Coupled gravity recovery and flotation optimization across a range of primary grind sizes and including assessment of alternative reagent regimes and conditions.
- Flotation Variability testing based on the optimal regime derived from the Master Composite tests.
- Comparative whole ore leach testing over a range of grind sizes, including: direct cyanidation (“DCN”), agitated (vat), carbon-in-leach (“CIL”), lead nitrate dosing, air and oxygen sparging.
- Flotation concentrate leaching based on bulk flotation concentrates formed from Master Composites, Life of Mine (“LOM”) and Variability composites. The test scope covered a wide range of regrind sizes and leach methodologies including lead nitrate assisted leaching.
- Comminution characterisation, including: JKDWT, Uniaxial Compressive Strength (“UCS”), Bond suite and SMC style testwork applied to HQ drill core variability lithological composites and PQ drill core lithological composites.

13.2 Sample provenance

In selecting representative samples, a notional 5 m downhole composite length was used to incorporate internal dilution that would occur during normal mining operations. Diamond drill (DD) core was used to produce the majority of the metallurgical composite samples of the different lithologies, with only a small proportion of reverse circulation (RC) drill chips being selected.

13.2.1 Samples for the Starter Pit composite (SP-MC)

Granite (“GRA”) and metavolcanic (“MVO”) predominate in the Starter Pit mining plan. Therefore, these two lithologies were selected and sampled for the Starter Pit master composite (“SP-MC”) as follows (Table 30):

- GRA: A total of 205 m of intersection were collected for the GRA sample, weighing 378 kg with a predicted mean grade of 1.45 g/t Au.
- MVO: 205 m of intersection were collected, weighing 407 kg with a predicted mean grade of 2.0 g/t Au.

Table 30: Starter pit composite selection

Element	Count of Intervals	Total Weight (kg)	Weighted Mean	Weighted Standard Deviation	Minimum	Maximum
Starter Pit Composite (SP-MC)						
Au (g/t)	74	700.9	1.21	0.89	0.12	6.24
As (ppm)			558	530	160	3290
S (ppm)			980	509	150	2990

13.2.2 Flotation samples (S-MC)

The holes, intersection lengths, samples and predicted grades to make the LOM Flotation Optimization master composite sample ("S-MC") as shown in Table 31, were:

- GRA: five holes, 60 m (30% of the combined sample), 12, 5 m composite samples, grading 1.05 g/t Au and 0.89% S.
- MVO: ten holes, 120 m (60%), 24 samples, grading 1.49 g/t Au and 1.01% S.
- DIO: two holes, 20 m (10%), 4 samples, grading 1.43 g/t Au and 1.30% S.
- Overall sample: twelve holes (some holes were used for multiple lithologies), 200 m, 40 samples, grading 1.35 g/t Au and 1.00% S.

Table 31: Flotation Optimization composite selection

Element	Count of Intervals	Total Weight (kg)	Weighted Mean	Weighted Standard Deviation	Minimum	Maximum
Flotation Optimization composite (S-MC)						
Au (g/t)	39	376.7	1.07	0.71	0.01	2.94
As (ppm)			586	299	10	1340
S (ppm)			991	338	300	1940

13.2.3 Life of Mine samples (LOM)

The holes sampled to produce the LOM metallurgical sample, subdivided by dominant lithology as shown in Table 32, were:

- GRA: 13 holes for 220 m, 44 composite samples of 5 m and 396 kg (target grade 1.12 g/t Au).
- MVO: 16 holes for 215 m, 43 composite samples of 5 m and 411 kg (target grade 1.28 g/t Au).
- DIO: 13 holes for 220 m, 46 composite samples of 5 m and 419 kg (target grade 1.1 g/t Au).

Table 32: LOM Lithology composite selection

Element	Number Intervals	Total Weight (kg)	Weighted Mean	Weighted Standard Deviation	Minimum	Maximum
LOM Metavolcanics Composite (MVO)						
Au (g/t)	196	238.6	1.26	1.75	0.01	11.10
As (ppm)			630	638	30	6,406
S (ppm)			12,665	7,910	1,900	31,700

Element	Number Intervals	Total Weight (kg)	Weighted Mean	Weighted Standard Deviation	Minimum	Maximum
LOM Granite Composite (GRA)						
Au (g/t)	214	280.6	1.03	1.56	0.07	24.60
As (ppm)			406	544	42	3,490
S (ppm)			8,369	1,992	3,500	13,500
LOM Diorite Composite (DIO)						
Au (g/t)	143	183.4	1.09	2.03	0.01	15.20
As (ppm)			297	950	21	13,428
S (ppm)			14,206	9,992	500	41,000

13.2.4 Variability samples

The variability samples were selected and composited from a combination of the LOM, Starter Pit and Flotation samples to target various Au, S and As grades as presented in Table 33.

Table 33: Variability sample – target grade ranges

Metavolcanic, Granite and Diorite	
Gold (g/t Au)	**Sulfur (% S)**
0.5-0.8	>0.6 (Hi)
0.5-0.8	<0.6 (Lo)
0.8-1.0	>0.6 (Hi)
0.5-0.8	>0.6 (Hi)

Table 34 shows the statistics of the samples tested for metallurgical analysis.

Table 34: Sample statistics for the variability composites

Range Au (g/t)	Range S (%)	Ave Au (g/t)	Ave S (%)	Ave As (ppm)	Mass (g)
Variability sample composite Metavolcanics (MVO)					
0.5-0.8	>0.6 (Hi)	0.61	1.52	581	63,214
0.5-0.8	<0.6 (Lo)	0.70	0.34	2610	9,066
0.8-1.0	>0.6 (Hi)	0.95	1.18	452	20,186
0.8-1.0	<0.6 (Lo)	0.88	0.40	609	10,338
Variability Sample Composite Granite (GRA)					
0.5-0.8	>0.6 (Hi)	0.64	0.80	453	190,120
0.5-0.8	<0.6 (Lo)	-	-	-	-
0.8-1.0	>0.6 (Hi)	0.90	0.90	290	89,969
0.8-1.0	<0.6 (Lo)	0.91	0.52	123	16,533
Variability Sample Composite Diorite (DIO)					
0.5-0.8	>0.6 (Hi)	0.61	1.89	179	26,548
0.5-0.8	<0.6 (Lo)	0.70	0.48	195	26,116
0.8-1.0	>0.6 (Hi)	0.97	1.44	96	9,208
0.8-1.0	<0.6 (Lo)	0.90	0.43	248	2,250

There was insufficient mass to test the Diorite sample 0.8-1.0 g/t Au <0.6% S (Lo) and Granite sample 0.5-0.8 g/t Au <0.6% S (Lo). It was determined that the available samples were sufficient to allow for statistically meaningful results from regression analysis.

13.2.5 Comminution samples

13.2.5.1 Comminution variability (SMC)

The SMC variability samples were selected from different areas of the pit at varying depths, by lithology. A total of twelve samples from eleven HQ drill-holes were selected for this testwork. The samples are defined in Table 35 and Figure 32 to Figure 34 shows typical core selected.

Table 35: Sample provenance for the Comminution Variability (SMC) testwork

Lithology	Drill Hole	Interval
MVO	NMDD002	61 to 76 m
MVO	NMDD008	57 to 72 m
MVO	NMDD009	115 to 130 m
MVO	NMDD028	92 to 107 m
GRA	NMDD029	89 to 104 m
GRA	NMDD091	38 to 53 m
GRA	NMDD005	129 to 144 m
GRA	NMDD005	204 to 219 m
DIO	NMDD040	55 to 70 m
DIO	NMDD054	44 to 59 m
DIO	NMDD046	47 to 62 m
DIO	NMDD044	40 to 55 m

Figure 32: Variability Comminution (SMC) sample – Metavolcanic HQ NMDD008

Figure 33: Variability Comminution (SMC) sample – Granite HQ NMDD005

Figure 34: Comminution (SMC) sample – Diorite HQ NMDD040

13.2.5.2 JKDWT testwork

Whole PQ core (150 mm nominal diameter) for JKDWT testing was produced from a single drill hole, NMDD153, and was composited at varying depth intervals to provide three sample specimens for each of the three lithology types as summarised in Table 36 and shown for diorite in Figure 35.

Table 36: Samples selected for the JKDWT Comminution testwork

Lithology	Drill Hole	Interval (m)	Mass (kg)
MVO	NMDD153	84 to 106	262
GRA	NMDD153	51 to 70	322
DIO	NMDD153	159 to 180	328

Figure 35: JKDWT comminution sample – Diorite PQ NMDD153

13.2.5.3 Sample spatial distribution indication

The spatial distribution of all the metallurgical sampling is shown in Figure 36 to Figure 40 in relation to the proposed Namdini Pit.

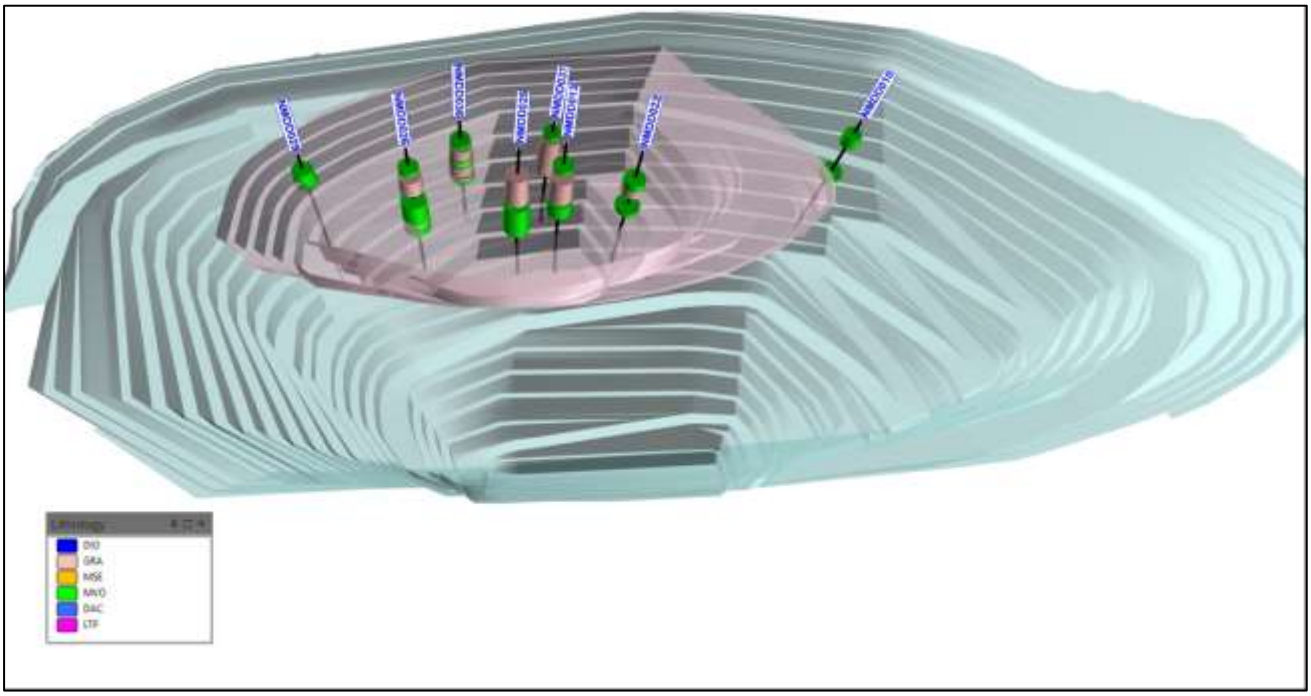


Figure 36: Provenance of Starter Pit Composites in the Namdini Starter Pit

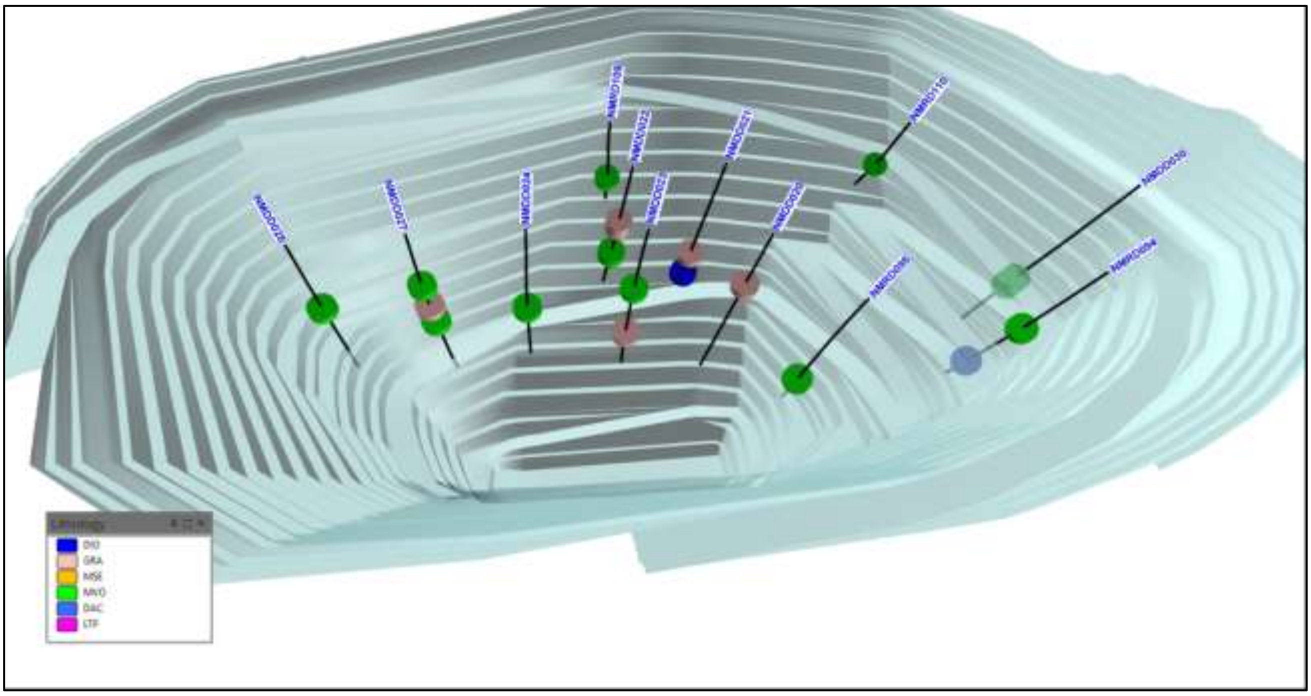


Figure 37: Provenance of the Flotation Composites in the Namdini Pit

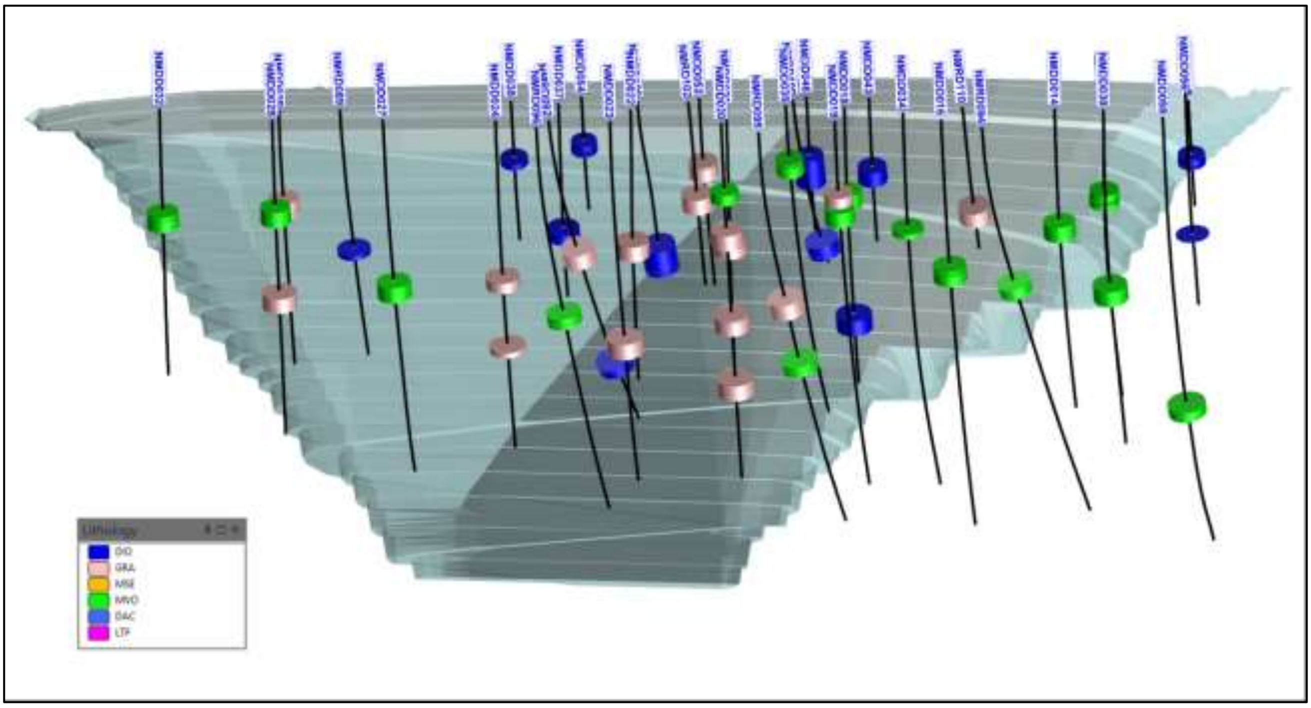


Figure 38: Provenance of the Life of Mine Composites in the Namdini Pit

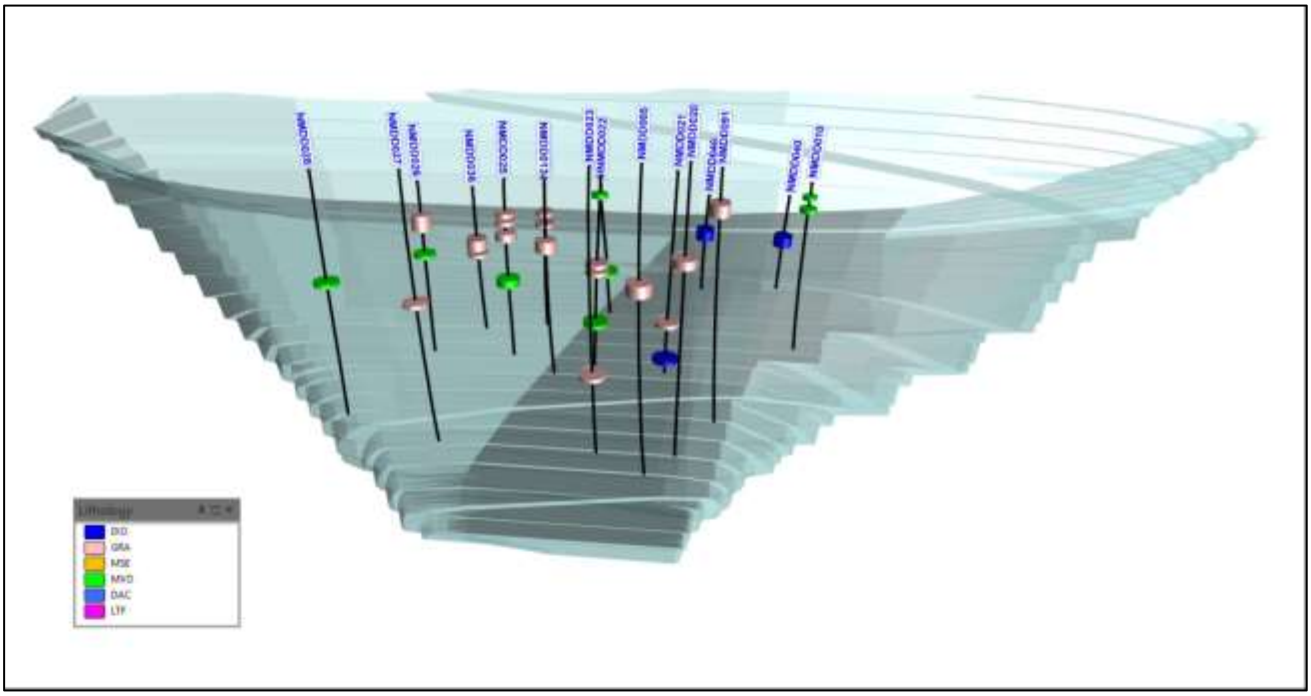


Figure 39: Provenance of the Variability Composites in the Namdini Pit

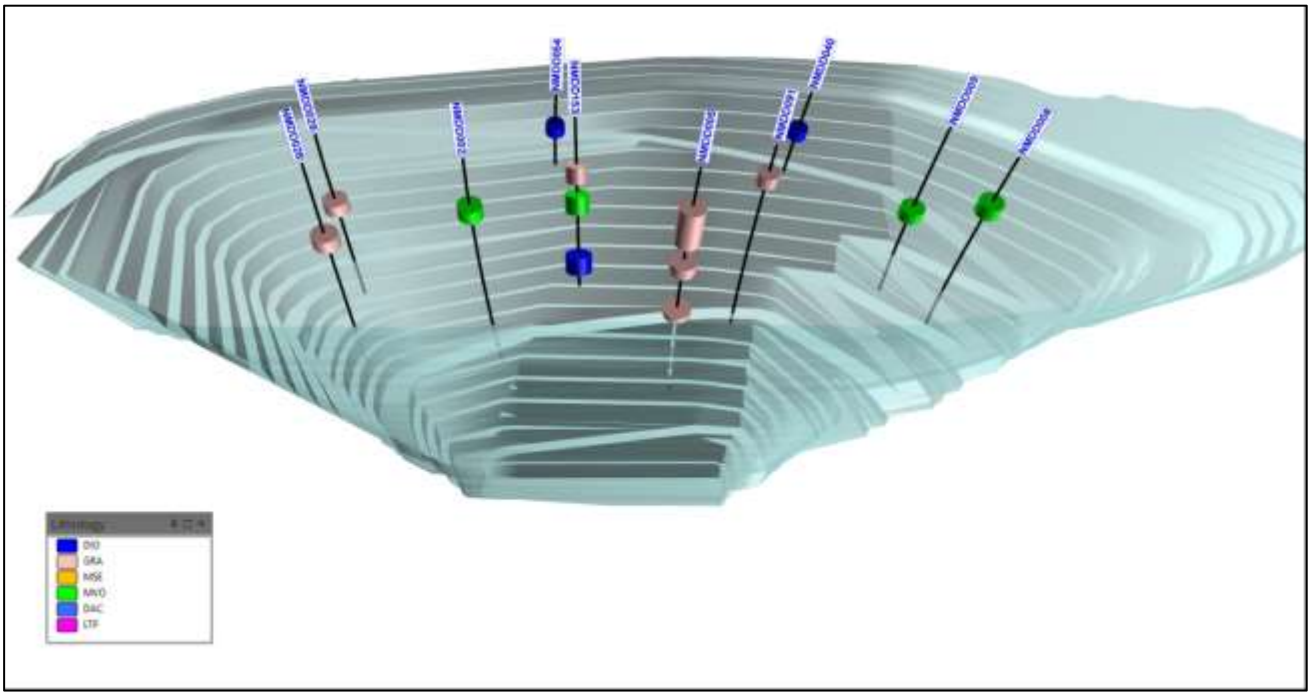


Figure 40: Provenance of the Comminution Composites in the Namdini Pit

13.3 Head analysis

13.3.1 Master composites

Composite head assays produced are indicated in Table 37. The analytical method can be summarised as:

- Gold by fire assay and screen fire assay
- Other metals by multi-element ICP assay
- Total sulfur, organics and carbon by LECO (CS2000) combustion assay.

Table 37: Composite head analysis – assay methods and results

Analyte	Method	Unit	Composite		LOM Lithology Composite		
			Starter Pit (SP-MC)	Flotation (S-MC)	MVO	GRA	DIO
Au	Fire Assay	g/t	1.32	1.02	1.32	1.50	1.01
Au	Screen Fire Assay	g/t	2.02	1.08	1.23	0.85	1.01
Average Au Head (Assay)		g/t	1.67	1.05	1.27	1.17	1.01
Au (Composite Intervals)		g/t	1.71	1.35	1.26	1.03	1.09
Ag	D3 digest/ICP	ppm	<2	<2	<2	<2	<2
As	D3 digest/ICP	ppm	660	430	750	260	320
Sb	D1 digest/ICP	ppm	3.0	3.1	2.7	4.9	1.7
Te	D1 digest/ICP	ppm	0.4	0.2	0.4	0.4	0.2
Cu	D3 digest/ICP	ppm	27	68	48	18	68
Pb	D3 digest/ICP	ppm	5	<5	10	10	<5
Ni	D3 digest/ICP	ppm	130	70	150	30	130
Zn	D3 digest/ICP	ppm	64	60	68	36	76
Fe	D4Z digest/ICP	%	3.76	4.58	5.32	1.72	6.98
S Total	CS2000	%	0.82	0.86	0.92	0.74	1.30
Sulfide	CS2000-Sherritt	%	0.72	0.78	0.80	0.66	1.18

Analyte	Method	Unit	Composite		LOM Lithology Composite		
			Starter Pit (SP-MC)	Flotation (S-MC)	MVO	GRA	DIO
C Total	CS2000	%	1.56	1.71	2.22	0.87	2.01
Organic C	CS2000	%	0.03	0.06	0.09	0.03	0.06
Carbonate	Calculated	%	1.53	1.65	2.13	0.84	1.95

The key results presented in Table 37 are summarised as follows:

- The difference between fire assays and screen fire assays for gold indicates the presence of free gold.
- Silver was reported to a 2 ppm detection limit. On-going silver assays at a lower detection limit are recommended to confirm requirements for elution and electrowinning circuits.
- Total sulfur levels ranged from 0.74 to 1.30%. Associated sulfide assays indicate the composites are amenable to flotation recovery.
- Arsenic ranged from 260 to 750 ppm. Levels reported for the Starter Pit (SP-MC) and MVO LOM Composites were 660 ppm and 750 ppm respectively. Arsenic levels are not excessive.
- Total Carbon ranged from 0.87 to 2.22%. Organic Carbon ranged from 0.03% to a maximum of 0.09%. Organic Carbon levels indicate little propensity for preg-robbing.
- Cyanide consuming elements: Cu, Zn, Pb, Ni and Sb all reported low, also indicating no adverse impact on cyanide consumption, loaded carbon or electrowinning.

13.3.2 Variability composites

A range of lithology variability composites were selected representing lower gold and a range of lower and higher sulfur and arsenic grades. The grades for all assays are summarised in Table 38.

The gold and sulfur grade averages are shown in Figure 41. Other multi-element grades were similar to the grade ranges of the LOM lithology composites.

Table 38: Variability Composite head grades

Assay	Units	Metavolcanics (MVO)				Granite (GRA)			Diorite (DIO)		
		V1	V2	V3	V4	V5	V6	V7	V8	V9	V10
Au	**g/t**	**0.58**	**0.86**	**0.64**	**1.58**	**0.63**	**1.04**	**0.93**	**0.57**	**0.60**	**0.90**
Ag	ppm	<2	<2	<2	<2	<2	<2	<2	<2	<2	<2
As	ppm	320	1040	230	580	320	180	200	110	170	140
Sb	ppm	2.30	2.10	1.50	2.10	5.20	4.80	3.80	1.00	1.20	1.50
Te	ppm	0.20	0.20	<0.2	0.20	0.40	0.40	0.40	0.20	0.40	0.40
Cu	ppm	44	40	32	38	18	16	16	60	40	46
Pb	ppm	<5	5.00	<5	<5	10.0	5.00	5.00	<5	10.0	5.00
Ni	ppm	165	175	120	165	25.0	20.0	25.0	155	155	150
Zn	ppm	78	72	64	74	40	38	34	84	70	64
Fe	%	5.22	5.08	5.16	5.68	1.84	1.60	1.74	7.34	5.74	4.90
S Total	%	0.72	0.42	0.52	0.68	0.88	0.88	0.88	1.48	0.46	1.38
Sulfide	%	0.74	0.44	0.50	0.70	0.88	0.84	0.86	1.48	0.50	1.40
C Total	%	2.37	2.19	1.80	3.27	0.96	0.84	0.96	1.89	2.61	2.73
Organic C	%	<0.03	<0.03	<0.03	<0.03	<0.03	<0.03	<0.03	<0.03	<0.03	<0.03
Carbonate	%	2.36	2.18	1.79	3.26	0.95	0.83	0.95	1.88	2.60	2.72

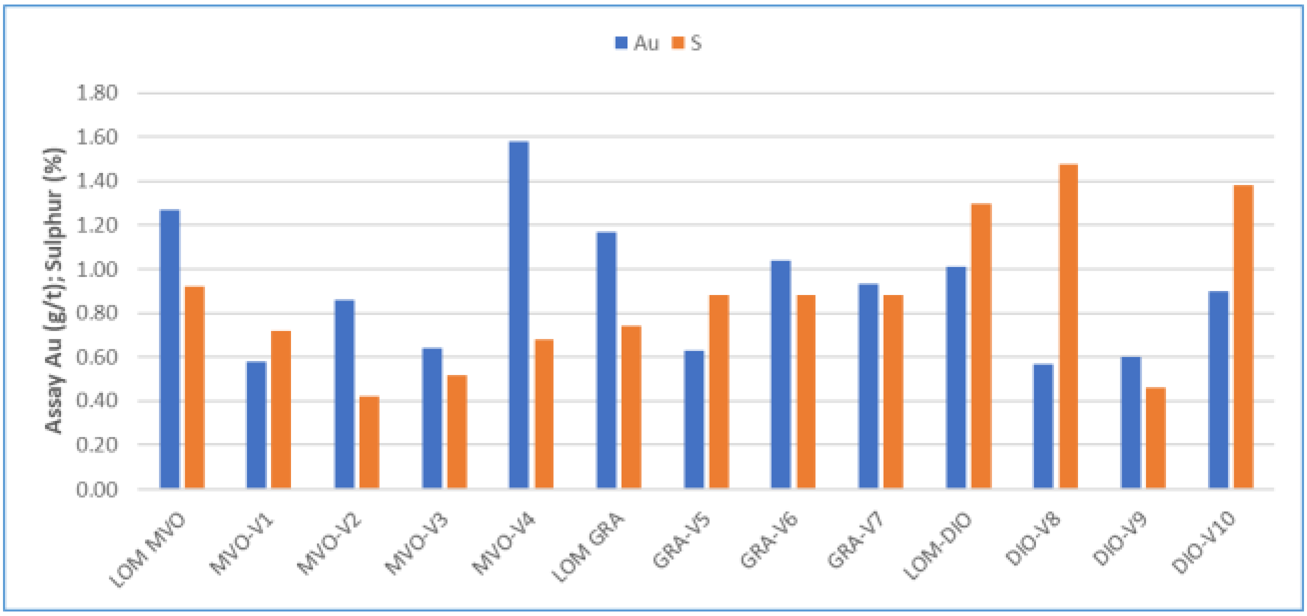


Figure 41: LOM and variability composite – gold and sulfur distribution

13.4 Mineralogy

13.4.1 Composite head mineralogy

Composite mineralogy was characterised by QEMSCAN and XRD. In each case samples were ground to a P_{80} of 75 µm and subjected to Knelson gravity upgrade to generate concentrate and tailings for the following:

- QEMSCAN analysis of concentrate, including optical analysis of free gold grain size
- XRD analysis of the Knelson gravity tailings.

Gravity concentrate QEMSCAN mineralogy is reported in Table 39 with the following key comments:

- Pyrite was approximately 1% in Knelson head samples
- Pyrite is the dominant sulfide and ranged from 16.2% in the Flotation Composite to 21.3% in the LOM Diorite Composite
- Arsenopyrite was significantly less abundant.
- Trace chalcopyrite and other sulfides, including sphalerite, galena and complex sulfosalts were observed
- Silicate modal abundance reported from ~60% in Diorite to a maximum of 68% in Granite. Carbonates, mainly intermediate ankerite/dolomite ranged from 10 to 15%.

Table 39: Variability composite head analysis

Mineral Group	Composite Modal Mass (%)				
	Flotation (S-MC)	Starter Pit (SP-MC)	LOM MVO	LOM GRA	LOM DIO
native gold	0.01	0.01	0.01	0.01	0.00
pyrite	16.2	18.9	16.5	20.3	21.3
arsenopyrite	0.93	0.99	1.33	0.90	0.22
chalcopyrite	0.01	0.01	0.04	0.01	0.08
other sulfides	0.01	0.01	0.00	0.01	0.01
quartz	21.5	23.4	21.2	26.3	17.9

Mineral Group	Composite Modal Mass (%)				
	Flotation (S-MC)	Starter Pit (SP-MC)	LOM MVO	LOM GRA	LOM DIO
albite	15.8	17.9	10.2	29.1	10.1
chlorite	9.36	5.72	13.6	0.67	19.4
micas	15.4	12.8	15.6	11.9	12.1
epidote	1.68	1.63	0.50	0.01	0.34
other silicates	0.26	0.27	0.13	0.04	0.11
ankerite/dolomite	10.5	10.9	15.4	6.96	8.15
amphibole	0.83	1.31	-	-	-
kaolinite/tourmaline	0.28	0.17	-	-	-
calcite	2.07	1.72	2.39	0.79	4.49
titanium minerals	1.82	1.51	1.77	1.12	3.09
Fe-oxides/oxyhydroxides	0.42	0.53	0.24	0.04	1.89
apatite	0.35	0.47	0.37	0.58	0.25
steel	2.30	1.33	0.62	1.24	0.44
other minerals	0.25	0.36	0.07	0.04	0.03
Total	**100**	**100**	**100**	**100**	**100**

Gravity concentrate liberation data is presented in Table 40 (see also Figure 42 and Figure 43) and summarised as follows:

- Composite mineralogy indicates the potential for up-front recovery of gravity gold. Fine locking of gold associated with pyrite and particularly arsenopyrite indicates a requirement for fine grinding of sulfide flotation concentrate to promote liberation ahead of leaching
- Pyrite was well liberated based on a P_{80} of 75 µm. The proportion exceeding 90% liberation ranged from 74.2% in LOM Metavolcanics (MVO) to 86.5% in Flotation Optimization Composite (S-MC). MVO and DIO composites presented a lower mass of greater than 90% liberated fraction, 74.2% and 77.5% respectively
- Arsenopyrite was less liberated than pyrite. The proportion exceeding 90% ranged from a minimum of 46.5% in DIO to a maximum of 69.4% in GRA. Unliberated arsenopyrite was also noted typically associated with silicates and pyrite
- Indicative gold locking within the Flotation Optimization Composite (S-MC) was predominantly gold-pyrite (98.5%) with minor composite gold-pyrite-silicate locking (1.5%). Comparatively, the Starter Pit Composite presented 15.9% liberated gold and a range of gold locking, including gold-pyrite (50.9%), arsenopyrite (3.2%), non-sulfide gangue (29.4%) and pyrite-silicate (0.5%)
- Optical examination identified coarse gold grains (150 to 200 µm) in Flotation Optimization (S-MC) and Starter Pit (SP-MC) composites. Similar analysis also identified coarse gold grains ranging from 150 to 300 µm in GRA and DIO composites
- The size and shape of coarse gravity gold grains indicates this fraction may be physically difficult to recover effectively by Flotation, also supporting recovery by the proposed Knelson gravity circuit.

Table 40: Gravity concentrate mineral liberation

Liberation Class	Composite Mass (%)				
	Flotation (S-MC)	Starter Pit (SP-MC)	LOM MVO	LOM GRA	LOM DIO
Pyrite (Dominant)					
Well liberated (> 90%)	86.5	84.6	74.2	83.4	77.5
High-grade middlings (60-90%)	9.44	11.8	19.9	13.7	18.0
Medium-grade middlings (30-60%)	1.85	1.93	3.05	1.85	2.71
Low-grade middlings (10-30%)	1.14	0.99	1.45	0.69	1.05
Locked (< 10%)	1.05	0.68	1.40	0.40	0.77
Total	**100**	**100**	**100**	**100**	**100**
Arsenopyrite (Low Abundance)					
Well liberated (> 90%)	60.6	66.3	63.8	69.4	46.5
High-grade middlings (60-90%)	28.3	23.1	21.2	22.1	20.2
Medium-grade middlings (30-60%)	4.43	3.12	5.26	4.47	17.8
Low-grade middlings (10-30%)	3.08	3.42	5.33	2.67	6.23
Locked (< 10%)	3.63	4.12	4.43	1.38	9.21
Total	**100**	**100**	**100**	**100**	**100**

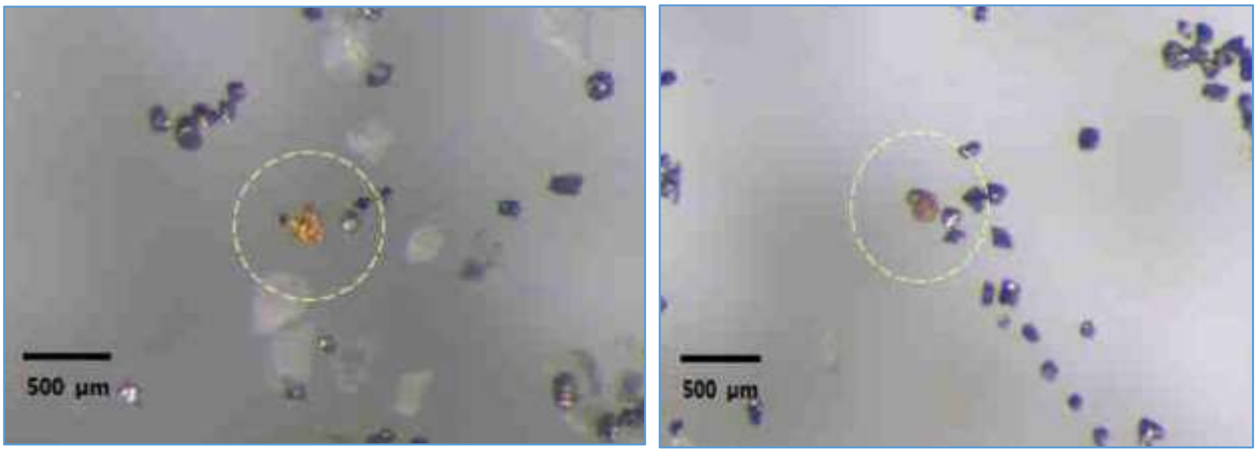
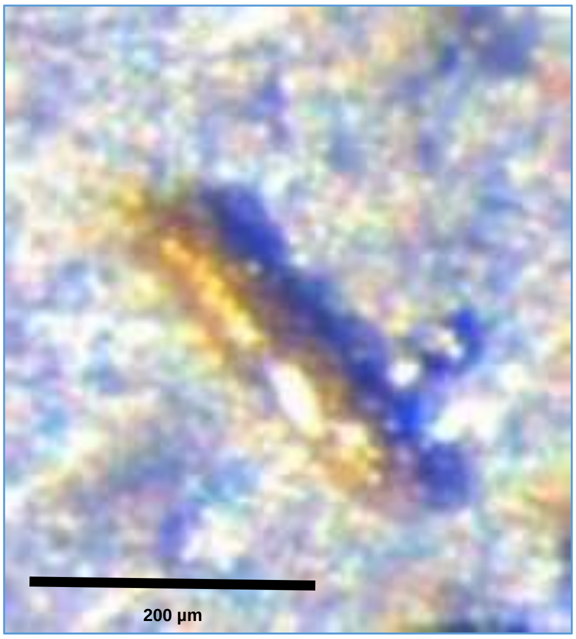


Figure 42: Coarse native gold – Flotation Composite S-MC (left) and Starter Pit SP-MC (right)



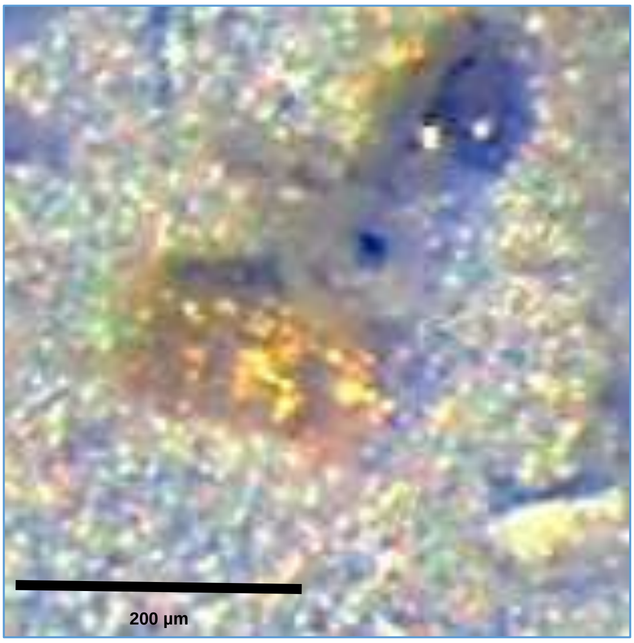


Figure 43: Coarse native gold – Composite GRA (left) and DIO (right)

13.4.2 Diagnostic leach characterisation

Composites were subjected to diagnostic leach characterisation to determine gold deportment based on the following sequenced treatment:

- Free gold – Knelson gravity recovery and mercury (Hg) amalgamation
- Cyanide recoverable gold – cyanidation leach
- Carbonate locked gold – sequenced dilute hydrochloric acid (HCl) leach and cyanidation
- Arsenopyrite locked gold – dilute nitric acid digestion and cyanidation
- Pyrite locked gold – *aqua regia* digestion.
- Silicate encapsulated gold – fire assay performed on final leach residue.

Diagnostic leach results shown in Table 41 indicate the following:

- All composites reported gravity recoverable gold (“GRG”). Based on a P_{80} of 75 µm, GRG varied widely, ranging from 19.6% in Granite (GRA) to 49% in Metavolcanics (MVO). The Starter Pit Composite reported 26.5%. The reduced GRG reported at P_{98} 15 µm was a result of the finer grind.
- Total gravity and cyanide recoverable gold tended higher in Flotation and Starter Pit master composites relative to LOM lithology composites based on a 75 µm P_{80} grind, also indicative of higher recovery expected for the Starter Pit. LOM lithology composites reported significant increase based on a grind of P_{98} 15 µm.
- Locked and occluded gold was mainly associated with arsenopyrite. Notably pyrite-locked gold reported very low across all composites at a coarse grind P_{80} of 75 µm and nil at a P_{98} of 15 µm.
- In overall terms the results indicate leach residue grades (and recovery) are determined by the gold associated with a small percentage of arsenopyrite.

Table 41: Composite diagnostic leach gold deportment

Gold Deportment	Master Composite P_{80} 75 µm		Lithology Composite P_{80} 75 µm			Lithology Composite P_{98} 15 µm		
	Flotation (S-MC)	Starter Pit (SP-MC)	MVO	GRA	DIO	MVO	GRA	DIO
Gravity gold	13.8	26.5	49.0	19.6	20.3	9.4	6.2	3.7
Cyanide recoverable	57.6	51.4	17.8	45.6	34.1	78.5	71.3	76.0
Gravity and cyanide recoverable	**71.4**	**77.9**	**66.8**	**65.2**	**54.4**	**87.9**	**77.5**	**79.7**
Locked carbonate associated	0.0	0.0	0.0	1.1	0.0	1.9	0.0	12.2
Locked arsenical	28.4	21.0	31.0	30.5	43.0	8.7	20.7	5.8
Locked pyritic	0.0	0.6	1.7	2.1	2.1	0.0	0.0	0.0
Silicate occluded	0.3	0.6	0.5	1.1	0.6	1.5	1.9	2.2
Locked and occluded	**28.6**	**22.1**	**33.2**	**34.8**	**45.6**	**12.1**	**22.5**	**20.3**
Total	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**	**100.0**
Calculated head grade (g/t Au)	**1.12**	**1.30**	**1.17**	**0.81**	**0.88**	**1.17**	**0.89**	**0.93**
Assay head (g/t Au)	**1.02**	**1.32**	**1.23**	**0.85**	**1.01**	**1.23**	**0.85**	**1.01**

13.4.3 Preg-robbing index (PRI) test results

The PRI test results for the starter pit and flotation master composites are presented in Table 42 below.

Table 42: Summary of PRI test results

Composite	Au Grade (mg/L)			PRI
	Au Spike	AuCN	AuPR	
S-MC	3.38	0.27	3.59	0.06
SP-MC	3.37	0.40	3.74	0.03

PRI values of 0.06 and 0.03 indicate both composites had a low preg-robbing potential. No further PRI tests were deemed necessary as multiple assay results have shown no significant organic carbon.

13.5 Comminution

13.5.1 Testwork scope – SMC variability testwork

The SMC variability testwork included twelve (12) sample composites comprising the three separate lithologies selected over a range of eleven (11) HQ drill-holes for the following scope of work:

- Bond Rod and Ball Mill Work Index and Bond Abrasion Index (Ai) determination.
- SAG Mill Comminution (SMC) Index.

13.5.2 Testwork scope – JKDWT testwork

The JKDWT PQ drill-hole (NMDD153) selected for comminution characterisation was located through the centre of the pit and included the following scope:

- Unconfined Compressive Strength (UCS).
- Bond Impact Crushing Work Index.
- Bond Rod and Ball Mill Work Index and Bond Abrasion Index characterisation.
- Drop Weight Index (DWi).

13.5.3 SMC variability testwork – bond work index

Comminution variability based on Bond Work Index testing across eleven (11) HQ drill-holes is summarised in Table 43. Key observations include the following:

- Work index values were relatively consistent with no clear depth trend.
- MVO and DIO lithologies reported higher Rod Mill Work Indices (RWi) relative to Ball Mill Work Index (BWi) indicating the suitability of recycle pebble crushing adopted in the comminution flowsheet.
- GRA lithology is indicated as most abrasive with an average Abrasion Index (Ai) of 0.274 relative to respective average values for MVO and DIO at 0.057 and 0.063.

Table 43: Bond Milling and Abrasion Index variability

Lithology	Drill Hole	Interval			Bond Milling and Abrasion Indices		
		From (m)	To (m)	Meters	BRWi (kWh/t)	BBWi (kWh/t)	Ai
MVO	NMDD002	61	76	15	21.0	15.7	0.051
	NMDD008	57	72	15	19.2	16.8	0.045
	NMDD009	115	130	15	20.0	15.7	0.064
	NMDD028	92	107	15	21.3	15.8	0.033
	NMDD153	84.33	105.77	21.44	26.2	16.8	0.090
85th Percentile					**23.2**	**16.8**	**0.074**
Average					**21.5**	**16.2**	**0.057**
GRA	NMDD005	129	144	15	16.9	17.0	0.277
	NMDD005	204	219	15	18.6	17.6	0.269
	NMDD029	89	104	15	18.5	18.4	0.291
	NMDD091	38	53	15	17.9	18.7	0.238
	NMDD153	51.24	70.40	19.16	18.3	18.1	0.294
85th Percentile					**18.5**	**18.5**	**0.292**
Average					**18.0**	**18.0**	**0.274**
DIO	NMDD040	55	70	15	20.3	16.3	0.040
	NMDD044	40	55	15	23.0	19.7	0.108
	NMDD046	47	62	15	18.4	14.7	0.026
	NMDD054	44	59	15	19.4	16.6	0.067
	NMDD153	158.73	180.44	21.71	19.9	15.7	0.075
85th Percentile					**21.4**	**17.8**	**0.088**
Average					**20.2**	**16.6**	**0.063**

13.5.4 SMC variability testwork – SMC Index

SMC tests performed over the eleven HQ drill-holes are reported in Table 44. Key results are summarised as follows:

- In overall terms the results support the project proposed SABC style comminution circuit.
- Relative lithology comminution energy (kWh/t) requirements are noted similar based on average and 85th percentile values for the SAG Circuit Specific Energy (SCSE) index.
- Consistent with results of Drop Weight (DWi) characterisation, MVO and DIO lithologies present similarly. MVO exhibits a slightly wider range of variability than DIO.
- GRA tends to produce more fines on impact breakage than MVO and DIO lithologies.

Table 44: SAG Mill Comminution (SMC) breakage parameters and Specific Energy (SCSE)

Lithology	Drill Hole	Interval		Breakage Parameter				SCSE (kWh/t)
		From (m)	To (m)	A	b	A*b	ta	
MVO	NMDD002	61	76	60.8	0.64	38.9	0.37	10.00
	NMDD008	57	72	61.5	0.65	40.0	0.37	10.04
	NMDD009	115	130	59.5	0.67	39.9	0.36	10.17
	NMDD028	92	107	57.1	0.60	34.3	0.32	10.84
	NMDD153	84.33	105.77	71.8	0.45	32.3	0.30	11.25
85th Percentile								**11.00**
Average								**10.46**
GRA	NMDD005	129	144	84.5	0.43	36.3	0.35	10.34
	NMDD005	204	219	100.0	0.30	30.0	0.29	11.38
	NMDD029	89	104	100.0	0.33	33.0	0.32	10.79
	NMDD091	38	53	83.3	0.44	36.7	0.35	10.32
	NMDD153	51.24	70.4	76.7	0.54	41.4	0.40	9.73
85th Percentile								**11.03**
Average								**10.51**
DIO	NMDD040	55	70	60.1	0.54	32.5	0.30	11.25
	NMDD044	40	55	67.4	0.44	29.7	0.28	11.62
	NMDD046	47	62	55.4	0.84	46.5	0.44	9.34
	NMDD054	44	59	67.7	0.52	35.2	0.32	10.86
	NMDD153	158.73	180.44	65.8	0.53	34.9	0.32	10.88
85th Percentile								**11.40**
Average								**10.79**

13.5.5 JKDWT testwork

Respective results based on UCS, Bond Crushing and Drop Weight tests applied to centrally located drill hole NMDD153 are shown in Table 45 to Table 47. Key observations are as follows:

- Compressive strength ranged from a minimum 20 MPa in Diorite (DIO) to a maximum of ~80 MPa in Granite (GRA), also indicating the composites to be medium-strong to strong. The range in UCS indicates Namdini ore feed as amenable to conventional jaw or gyratory crushing.
- Bond Crushing Work Index (CWi) average values ranged from a minimum of 10.9 kWh/t in Granite to a maximum of 19.8 kWh/t in Diorite. Granite is identified as presenting generally lower impact strength relative to MVO and DIO, which were similar in terms of average and maximum crushing work index.
- Relativity in resistance to milling impact breakage is indicated based on A*b values derived from Drop Weight testing. Lower values indicate more competent ore. MVO is noted as most competent with an A*b of 29.9, Diorite reported at 34.0 and Granite, 42.1. The range indicates Namdini ore as amenable to conventional SABC milling, including SAG and Ball milling with recycle crushing of SAG mill scats.

Table 45: UCS by Lithology drill hole NMDD153

Lithology	Interval (m)			Failure Mode	UCS (MPa)	Specific Gravity (t/m^3)
	From	To	Metres			
MVO	84.55	84.97	0.42	Shear	72.2	2.82
	97.03	97.38	0.35	Shear	67.5	2.86
	105.38	105.77	0.39	Shear	52.5	2.86
GRA	56.1	56.54	0.44	Shear	80.3	2.71
	61.3	61.67	0.37	Shear	30.5	2.71
	62.95	63.24	0.29	Shear	31.9	2.70
DIO	163.04	163.35	0.31	Shear	20.3	2.79
	174.16	174.54	0.38	Shear	59.5	2.87
	165.63	165.93	0.30	Shear	65.1	2.82

Table 46: Bond Crushing Work Index by Lithology drill hole NMDD153

Lithology	Interval		Average Depth (m)	Bond Crushing Work Index (kWh/t)			
	From	To		Maximum	85th Percentile	Average	Minimum
GRA	53.72	67.74	60.73	14.9	13.1	10.9	7.2
MVO	86.10	105.19	95.65	26.3	25.2	18.2	12.1
DIO	159.00	178.81	168.91	29.7	25.9	19.8	10.9

Table 47: Drop Weight Index by Lithology drill hole NMDD153

Lithology	Interval		Drop Weight Test Summary					
	From (m)	To (m)	A	b	A*b	ta	Specific Gravity (t/m^3)	SCSE (kWh/t)
MVO	84.33	105.77	61.0	0.49	29.9	0.49	2.82	11.69
GRA	51.24	70.4	71.3	0.59	42.1	0.43	2.71	9.66
DIO	158.73	180.44	58.7	0.58	34.0	0.55	2.81	10.92

13.5.6 Conclusions

The comminution testwork results are summarised below:

- Unconfined Compressive Strength (UCS) ranged from 20 MPa in Diorite (DIO) to a maximum of 80 MPa in Granite (GRA), also indicating Namdini ore feed as amenable to conventional jaw or gyratory crushing.
- SAG Mill Comminution (SMC) test A*b values ranged from 29.9 to 42.1, indicating the ore feed to be competent. Combined results derived via SMC, Drop Weight (DWi) Index and Bond suite testwork support configuration of the comminution circuit based on the proposed SABC flowsheet, incorporating SAG milling and recycle crushing of SAG mill scats, coupled with Ball milling to a finished primary grind (P_{80}) size of 106 µm.
- Granite (GRA) had a low Bond Abrasion Index (Ai) of 0.274; lower average values were obtained for MVO at 0.057 and DIO at 0.063.
- Bond crushing index (CWi) presented trending indicating increasing crushing index with increasing depth. Comparatively Bond Rod and Ball Mill index and SMC index values did not present clear trending with depth.

- Overall variability in SAG Mill comminution specific energy is noted relatively low based on SMC and Drop Weight Index (DWi) test data across a total of 11 HQ drill-holes.

13.6 Gravity Recovery and Flotation

13.6.1 Scope of Testwork

The process flowsheet incorporates gravity recovery, rougher flotation, concentrate regrinding and cyanide leach. Knelson gravity and flotation optimization testing was performed on Flotation and Starter Pit composites to assess response relative to the following variables:

- Primary grind P_{80}
- Prior gravity gold recovery
- Impact of site water
- Reagent regime and dosage.

On establishing an optimal response, testing was extended to include the following lithology composites:

- LOM Metavolcanics (MVO)
- LOM Granite (GRA)
- LOM Diorite (DIO).

Lithology composites representative of the Life of Mine (LOM) and variability over a range of gold and sulfur head grades were assessed based on the optimal primary grind size and flotation regime established for Flotation Optimization and Starter Pit composites.

Final bulk flotation testing was conducted on the following composites to provide concentrate for leach testing:

- Starter Pit (SP-MC)
- Flotation (S-MC)
- Metavolcanics (MVO) – LOM and variability composites
- Granite (GRA) – LOM and variability composites
- Diorite (DIO) – LOM and variability composites.

Results, including bulk flotation for each composite are presented in Table 48.

Table 48: Gravity and Flotation key physicals

Criteria	Statistic	Unit	Composite/Lithology				
			S-MC Flotation	SP-MC Starter Pit	MVO Metavolcanic	GRA Granite	DIO Diorite
Gold head grade	Number		14	18	6	6	5
	LOM	g/t	-	1.31	1.18	1.13	1.07
	Min		1.02	1.43	0.62	0.62	0.62
	Average		**1.14**	**2.00**	**1.10**	**1.01**	**1.01**
	Max		1.31	2.36	1.54	1.21	1.48
	SD (*)		0.09	0.24	0.33	0.19	0.32

Criteria	Statistic	Unit	Composite/Lithology				
			S-MC Flotation	SP-MC Starter Pit	MVO Metavolcanic	GRA Granite	DIO Diorite
Gravity recovery	Number		5	9	6	6	5
	Min	%	7.8	22.1	3.6	2.4	6.0
	Average	%	**8.8**	**30.0**	**9.2**	**14.0**	**17.3**
	Max	%	9.9	34.7	18.6	27.8	58.1
	SD	%	0.7	3.5	5.4	11.1	20.4
Flotation tailing gold grade	Number		14	18	6	6	5
	Min	g/t	0.05	0.07	0.06	0.03	0.05
	Average	g/t	**0.07**	**0.14**	**0.11**	**0.05**	**0.09**
	Max	g/t	0.10	0.31	0.15	0.07	0.13
	SD	g/t	0.02	0.07	0.03	0.01	0.03
Flotation Recovery	Number		14	18	6	6	5
	Min	%	89.8	82.9	82.7	92.0	81.6
	Average	%	**94.4**	**92.5**	**89.0**	**94.6**	**89.5**
	Max	%	96.0	96.0	92.9	97.2	92.5
	SD	%	1.6	3.0	3.5	1.8	4.0
Total Flotation and Gravity Recovery	Number		14	18	6	6	5
	Min	%	90.8	82.9	85.1	93.3	83.1
	Average	%	**94.5**	**93.5**	**90.0**	**95.5**	**91.0**
	Max	%	96.0	97.0	94.2	97.2	96.9
	SD	%	1.4	3.3	3.2	1.4	4.5

13.6.2 Starter Pit Composite – results summary

Results derived for the Starter Pit composite with prior gravity recovery and under the optimal flotation regime are presented in Table 49. Key results support the following:

- Primary grind P_{80} of 106 µm
- Reduction in flotation tailings grade and improved recovery across all grind sizes with gravity treatment
- Flotation based on 50 g/t copper sulfate activator; 50-100 g/t potassium amyl xanthate (PAX) collector and either MIBC or Polyfroth H27 frothing agent
- Flotation based on the natural pH (8) with potential to improve recovery at reduced pH (6.5).

Table 49: Composite (SP-MC) gravity and variable grind size flotation – grade recovery summary

P_{80} Size (µm)	Concentrate Grade and Recovery							Rougher Tailing Grade		
	Mass Pull	Gold		Arsenic		Sulfur		Gold	Arsenic	Sulfur
	%	(g/t)	Rec (%)	(%)	Rec (%)	(%)	Rec (%)	(g/t)	(%)	(%)
212	6.7	20.6	95.4	0.81	92.0	13.1	95.9	0.11	0.01	0.04
150	5.2	21.8	93.5	0.92	91.1	14.6	96.4	0.13	0.01	0.03
106	4.9	27.5	96.7	1.06	91.6	17.5	97.8	0.07	0.01	0.02
75	6.3	21.5	96.2	0.90	92.4	14.5	98.0	0.09	0.01	0.02

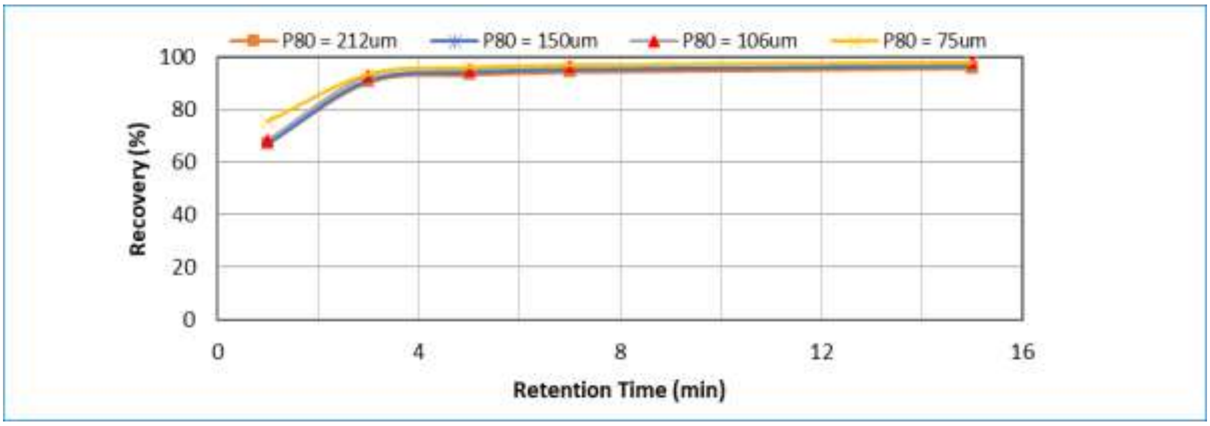


Figure 44: Composite (SP-MC) cumulative gravity and rougher flotation – sulfur kinetic recovery

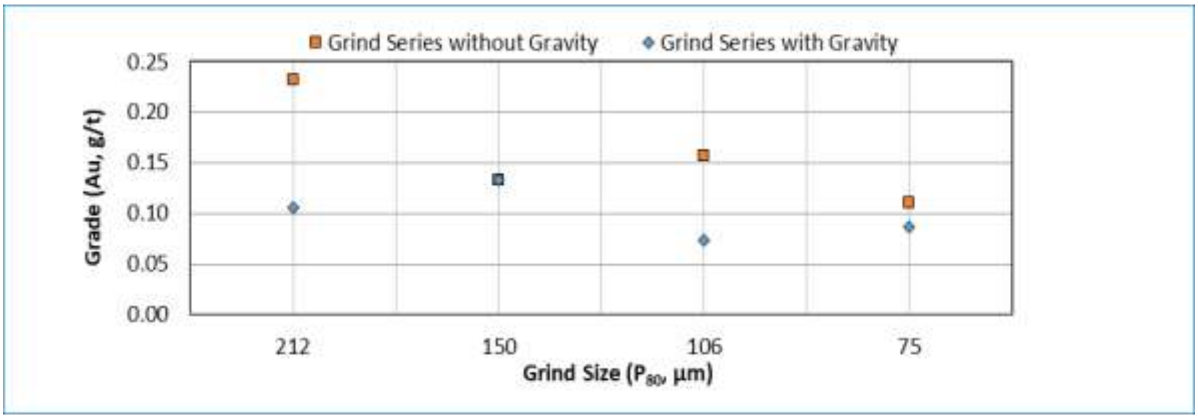


Figure 45: Composite (SP-MC) rougher tailings grade – with and without gravity recovery

13.6.2.1 Lithology Composite – LOM and Variability Batch Flotation

LOM lithology and variability composites were tested to assess Knelson gravity upgrade and flotation recovery. Batch flotation was conducted at a grind size of 106 µm under the flotation regime established during the optimization test phase.

13.6.2.2 Metavolcanics (MVO)

Knelson gravity and batch flotation results for LOM Metavolcanics (MVO) and four MVO variability composites are presented in Table 50 and Table 51. The following comments are relevant:

- LOM MVO reported a calculated head grade of 1.54 g/t and gravity recovered grade of 0.29 g/t, representing a recovery of 18.6%. Subsequent rougher flotation resulted in total recovery of 94.2% (including gravity) with flotation tailings grading 0.10 g/t.
- MVO variability composite head grade ranged from 0.62-1.43 g/t. For LOM MVO, gravity recovery was proportional to gold head grade except for composite MVO-V4, grading at 1.43 g/t Au with low recovered grade at 0.05 g/t.
- Relative to LOM MVO, lower grade variability composites reported a range of recoveries from 85.1-91.6% with flotation tailings grades from 0.06-0.15 g/t. Notably higher flotation tailings grades tended to be correlated with lower arsenic recovery, indicative of lower arsenopyrite recovery.
- The results were achieved based on a flotation pH of ~8.0 to 8.2. Improved recoveries may be possible based on a reduced pH, nominally ~6.5.

Table 50: Metavolcanics LOM and Variability gravity recovery

Composite	Calculated Head Grade			Gravity Recovery	
	Gold (g/t)	Arsenic (%)	Sulfur (%)	Gold (g/t)	Rec (%)
LOM MVO	**1.54**	**0.07**	**0.98**	**0.29**	**18.6**
MVO-V1	0.63	0.03	0.75	0.03	4.9
MVO-V2	0.84	0.10	0.42	0.04	4.8
MVO-V3	0.92	0.03	0.51	0.13	13.7
MVO-V4	1.44	0.05	0.66	0.05	3.5

Table 51: Metavolcanics LOM and Variability composite flotation grade recovery

Comp	Calculated Head Grade			Concentrate Grade and Recovery							Tailing
	Au	As	S_T	Mass	Gold		Arsenic		Total Sulfur		
	(g/t)	(%)	(%)	(%)	(g/t)	Rec (%)	(g/t)	Rec (%)	(g/t)	Rec (%)	Au (g/t)
LOM-MVO	**1.54**	**0.07**	**0.98**	**7.7**	**15.1**	**94.2**	**0.83**	**93.3**	**12.3**	**96.2**	**0.10**
MVO-V1	0.62	0.03	0.75	5.0	10.8	91.6	0.54	85.2	13.9	93.6	0.06
MVO-V2	0.84	0.10	0.42	5.6	12.4	86.5	1.54	81.9	7.2	95.5	0.12
MVO-V3	0.92	0.03	0.51	5.2	12.7	85.1	0.33	64.3	9.4	96.3	0.15
MVO-V4	1.43	0.05	0.66	6.5	19.3	91.2	0.71	90.9	9.6	94.3	0.14

13.6.2.3 Granite (GRA)

Key results for Granite (GRA) composites are presented in Table 52 and Table 53 and summarised as follows:

- LOM GRA returned a calculated head grade of 1.21 g/t and gravity recovered grade of 0.34/t, representing a recovery of 27.8%. Rougher flotation applied to gravity tailings resulted in a total recovery of 94.3% with a tailings grade of 0.10 g/t.
- Relative to LOM GRA, variability composites reported lower head grade ranging from 0.62 to 1.04 g/t and higher sulfur. Gravity recovered grades were consistently low at 0.03 g/t and total recovery ranged from 93.3 to 97.2%.
- Variability composites reported low flotation tailings grades, ranging from 0.03 to 0.05 g/t, overall sulfur recovery was elevated relative to the LOM composite.
- Unlike MVO, GRA flotation recoveries did not appear to be correlated with arsenic recovery, albeit higher gold recovery corresponded to higher sulfur recovery, also indicating that improved flotation recovery may be possible by adopting a lower pH flotation.

Table 52: Granite LOM and Variability gravity recovery

Composite	Calculated Head Grade			Gravity Recovery	
	Gold (g/t)	Arsenic (%)	Sulfur (%)	Au (g/t)	Rec (%)
LOM GRA	**1.21**	**0.03**	**0.88**	**0.34**	**27.8**
GRA-V5	0.62	0.04	0.89	0.03	4.1
GRA-V6	1.04	0.03	0.96	0.03	2.4
GRA-V7	1.01	0.03	0.97	0.03	2.5

Table 53: Granite LOM and Variability flotation grade recovery summary

Comp	Calculated Head Grade			Concentrate Grade and Recovery							Tailing
	Au	As	S_T	Mass	Gold		Arsenic		Total Sulfur		
	(g/t)	(%)	(%)	(%)	(g/t)	Rec (%)	(g/t)	Rec (%)	(g/t)	Rec (%)	Au (g/t)
LOM GRA	**1.21**	**0.03**	**0.88**	**5.2**	**15.5**	**94.3**	**0.53**	**85.4**	**12.3**	**96.2**	**0.10**
GRA-V5	0.62	0.04	0.89	7.7	7.2	93.3	0.44	88.1	11.5	99.5	0.05
GRA-V6	1.04	0.03	0.96	6.1	16.1	96.4	0.35	81.8	15.6	99.5	0.04
GRA-V7	1.01	0.03	0.97	7.2	13.3	97.2	0.31	82.5	13.4	98.1	0.03

13.6.2.4 Diorite (DIO)

Diorite (DIO) gravity and flotation results are provided in Table 54 and Table 55. Relevant comments include the following:

- LOM DIO reported a calculated head grade of 0.97 g/t and gravity recovered grade of 0.07/t, representing a recovery of 6.8%. Subsequent flotation resulted in a recovery of 92.6% based on flotation tailings grading 0.08 g/t.
- DIO variability composite head grade bracketed the LOM composite head grade, ranging from 0.62-1.48 g/t. Gravity recovery tended to correlate with increased head grade, notably DIO-V8 reported elevated gravity recovery of 58.1% based on a head grade of 1.48 g/t.
- Variability composite total recovery varied between 83.1 and 96.9%. Lower recovery occurred in the case of DIO-V9 at 83.1% based on a calculated head grade of 0.62 g/t and flotation tailings grade of 0.11 g/t.
- Overall recovery tended to be correlated with gold head grade and sulfur recovery. As for other composites, improved gold recovery may be possible with lower pH flotation.

Table 54: Diorite LOM and Variability gravity recovery

Composite	Calculated Head Grade			Gravity Recovery	
	Gold (g/t)	Arsenic (%)	Sulfur (%)	Au (g/t)	Rec (%)
LOM DIO	**0.97**	**0.03**	**1.09**	**0.07**	**6.8**
DIO-V8	1.48	0.02	1.49	0.86	58.1
DIO-V9	0.62	0.02	0.56	0.05	8.2
DIO-V10	0.76	0.02	1.44	0.05	6.0

Table 55: Diorite LOM and Variability flotation grade recovery summary

Comp	Calculated Head Grade			Concentrate Grade and Recovery							Tailing
	Au	As	S_T	Mass	Gold		Arsenic		Total Sulfur		
	(g/t)	(%)	(%)	(%)	(g/t)	Rec (%)	(g/t)	Rec (%)	(g/t)	Rec (%)	Au (g/t)
LOM DIO	**0.97**	**0.03**	**1.09**	**6.9**	**12.1**	**92.6**	**0.33**	**70.7**	**15.3**	**96.6**	**0.08**
DIO-V8	1.48	0.02	1.49	7.0	8.2	96.9	0.15	52.6	21.0	98.8	0.05
DIO-V9	0.62	0.02	0.56	5.5	8.5	83.1	0.24	57.8	9.7	94.9	0.11
DIO-V10	0.76	0.02	1.44	8.1	8.0	91.5	0.14	71.2	16.9	94.9	0.07

13.6.3 Gravity and flotation – regression analysis

Test results for all composites were subjected to a multivariable regression analysis to determine the main variables driving gravity and flotation recovery and grade. The key regression outputs were compared to actual results obtained by Knelson gravity, batch and bulk flotation tests.

13.6.3.1 Gravity recovered grade regression

Gravity recovered grade correlated strongly with head grade and gold to arsenic ratio.

Regression estimates across all composites and 41 × 3” Knelson concentrator tests are compared to actual in Figure 46. Overall correlation was reasonable with a coefficient of determination (R^2) of ~0.8. Increased variability was noted for lower grade lithology variability composites.

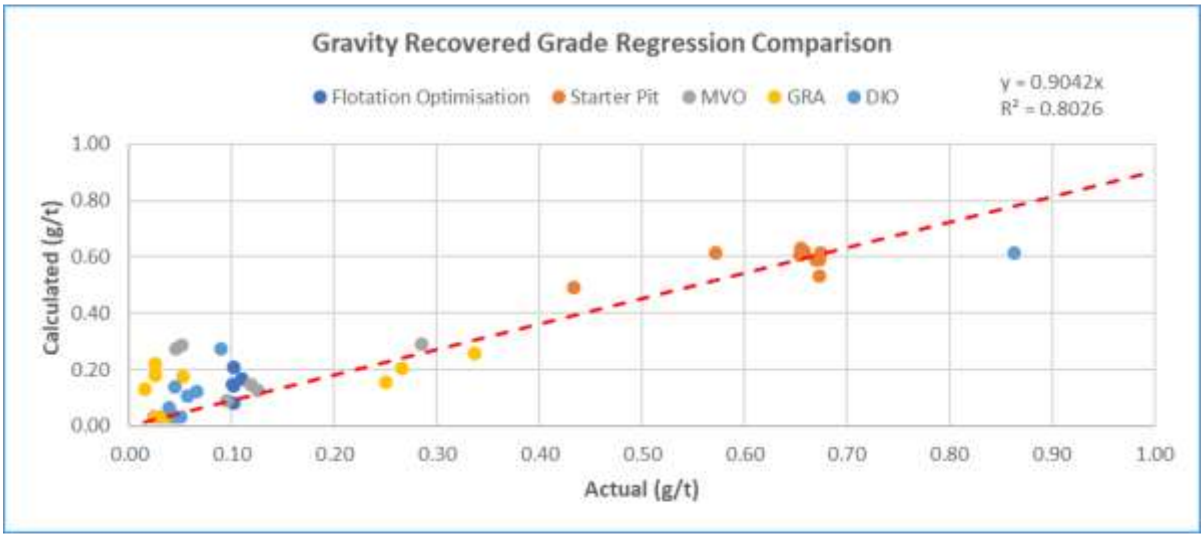


Figure 46: Gravity recovered grade – regression comparison

13.6.3.2 Flotation Concentrate grade regression

Rougher flotation concentrate grade indicated close correlation with flotation feed grade and concentrate mass, as evidenced in Figure 47 with a correlation coefficient of ~0.95 across all composites and a total of 59 flotation tests.

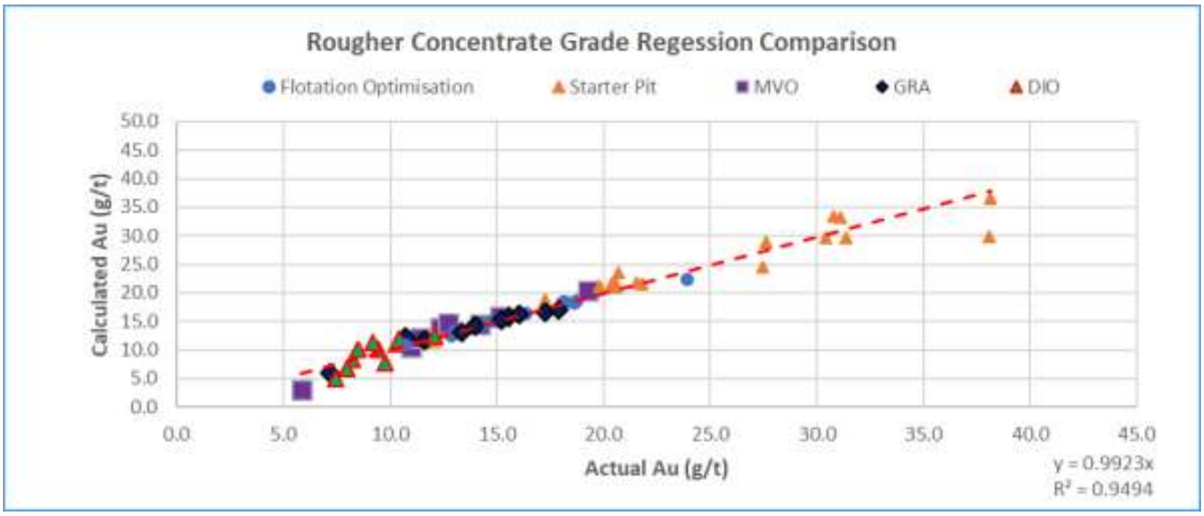


Figure 47: Rougher Flotation Concentrate grade regression comparison

13.6.3.3 Flotation Concentrate recovered grade regression

Rougher concentrate recovered grade (g/t feed) was found to be closely correlated with flotation feed grade (Figure 48) based on the difference between mill feed grade and gravity recovered grade.

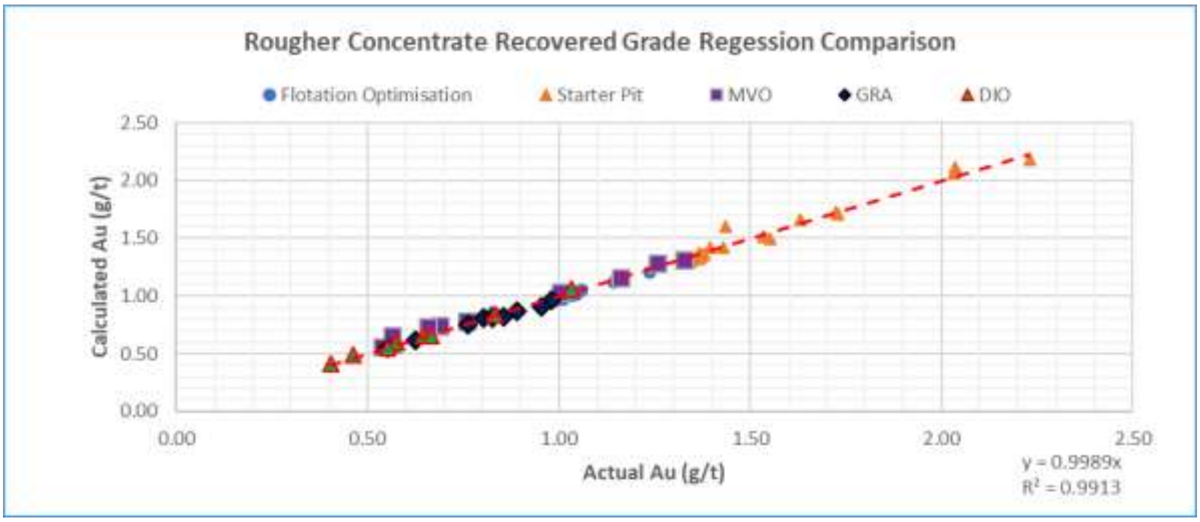


Figure 48: Rougher Flotation Concentrate recovered grade regression comparison

13.6.4 Conclusions

13.6.4.1 Gravity recovery

- Prior gravity treatment lifted base-line recovery, mitigating lower flotation recovery of free gold. The inclusion of a gravity circuit is justified based on recoveries achieved for the Starter Pit composite.
- Knelson gravity recovery was closely correlated with mill head grade and arsenic to gold ratio. Starter Pit composite reported an average recovery of 30% based on an average head grade of 2.0 g/t Au.
- LOM and variable grade lithology composites showed an increasing recovery trend with lower average recovery reported for Metavolcanics (MVO) at 9.2%, relative to Granite (14.0%) and Diorite (DIO) 17.3%. Variability between separate lithologies followed the same trend, increasing with increased head grade. Diorite (DIO) exhibited the highest variability ranging from 6.0 to 58.1%.
- Gravity recovery to date has been based on 3” Knelson batch tests. Staged grinding, cyclic recovery and modelling based on CONSEP method will be required for: equipment selection, assessment of impact on the mill water balance and estimation of expected gravity recovery at full-scale.

13.6.4.2 Flotation optimization

- Gold and sulfur flotation kinetics were rapid with high recovery (> 90%) achieved after five minutes with an industry standard reagent regime comprising: copper sulfate activator, potassium amyl xanthate (PAX) collector and either methyl isobutyl carbinol (MIBC) or Polyfroth H27 frother.
- Variable grind testing confirmed a primary grind P_{80} of 106 µm suitable for on-going development. However, scope for increasing the primary grind to P_{80} of 150 µm without compromising flotation recovery is a possibility.
- Site water analysis indicated low concentrations of sulfate and multi-valent cations, also indicating little impact on flotation. Subsequent testing in site water confirmed no measurable impact on flotation kinetics or recovery.

- Visual observations during reagent testing identified Huntsman Polyfroth H27 as providing a more stable froth than MIBC.
- Collector screening identified PAX as suitable based on dosage in the range 50 g/t to 100 g/t; notably reduced PAX (50 g/t) reported lower gold recovery when treating ore with a higher proportion of free gold. The inclusion of Knelson pre-concentration was shown to mitigate this impact at low dosage.
- While the introduction co-collector Cytec A7249 blended with 100 g/t PAX did offer improved selectivity, gold recoveries were in-line with the range of recoveries achievable with PAX.

13.6.4.3 Gravity and Flotation Recovery

- Starter Pit composite generated an average gravity and flotation recovery of 93.5% based on an average head grading 2.00 g/t Au. Overall recovery ranged from 82.9 to 97.0%.
- Granite (GRA) provided the highest gravity and flotation recovery, averaging 95.5% relative to Diorite (DIO) at 91.0% and Metavolcanics (MVO) at 90.0%. Lower flotation recoveries for MVO and DIO lithologies correlated with lower arsenic (arsenopyrite) recovery.
- Metallurgical response was very consistent for all lithologies. Regression analysis across all composites identified close correlation between flotation recovered gold grade and flotation feed grade with a coefficient of determination (R^2) of ~ 0.99. Concentrate grade was closely correlated with flotation feed gold grade and concentrate mass with R^2 of 0.95. Specific variations in flotation tailing grade (and recovery) were a result of spotty free gold where prior gravity recovery had not been conducted and variance in arsenic recovery.
- To date flotation optimization has been conducted at the natural pH 8.0-8.2. Improved base-line arsenic and sulfur (gold) recovery may be viable based on a reduced pH in the range 6.5-7.0. The addition of copper sulfate activator to the ball mill circuit is also endorsed.
- The process flowsheet is based on rougher only flotation, fine concentrate re-grinding and leaching. A trade-off exists in terms of concentrate mass, grade/recovery and downstream grinding. Inclusion of a cleaner flotation circuit with recirculation of cleaner tailings may act to mitigate the risk of constrained throughput when operating at high rougher recovery and concentrate mass.

13.7 Whole ore leaching

Extensive whole ore leaching was conducted across the following composites to benchmark extraction:

- Starter Pit Composite (SP-MC).
- Metavolcanics (MVO) LOM and variability composites
- Granite (GRA) LOM and variability composites
- Diorite (DIO) LOM and variability composites.

The scope included Direct Cyanidation (DCN) bottle-roll leach tests, with and without gravity recovery, CIL leaches and VAT (agitated) tests at a grind P_{80} of 75 µm. Variability lithology composites were subjected to additional direct cyanidation testing at a grind of P_{98} 15 µm.

The testwork covered a wide range of variables including: grind size, air/oxygen sparging, lead nitrate addition and the impact of gravity recovery. Direct cyanidation tests were based on the following standardised conditions:

- Leach solids loading, 45% solids (w/w) in Perth tap water.

- Initial pH target 10.5, maintained above pH 9.8, dosing lime. Selected variable pH tests were conducted with lead nitrate up to pH 12.5 (maintaining 0.05% free lime).
- Initial cyanide target, 1000 ppm, maintained above 500 ppm, dosing sodium cyanide (NaCN).
- Oxygen sparging to 20 ppm dissolved oxygen; air sparging to ~9 ppm.
- Kinetic leach solution sampling based on a total retention time of 72 hours.
- CIL conducted under identical conditions described above, including 20 g/L activated carbon.

13.7.1 Starter Pit Composite (SP-MC), P_{80} 75 µm

Leach test results for the Starter Pit composite are presented in based on the following leach methods:

- Direct Cyanidation (DCN) bottle-roll leaching.
- CIL based (20 g/L) bottle-roll leaching.

Gravity recovery and Direct Cyanidation (DCN) bottle-roll leaching.

Table 56: Starter Pit Composite (SP-MC) whole ore leach summary P_{80} 75 µm

Test and Conditions				Leach Grade and Extraction (Ext)				Reagent Consumption	
Basis	O_2/Air	pH	Lead Nitrate	Assay Head Au (g/t)	Calc Head Au (g/t)	Leach Residue Au (g/t)	Ext (%)	NaCN (kg/t)	Lime (kg/t)
DCN	O_2	10.5	-	2.02	1.92	0.35	81.7	0.44	0.35
CIL	O_2	10.5	-		2.21	0.36	83.7	1.02	0.35
G-DCN (*)	O_2	10.5	-		2.21	0.35	84.2	0.47	0.30

Notes: *G-DCN = Prior gravity recovery and direct cyanidation (bottle roll) leach.

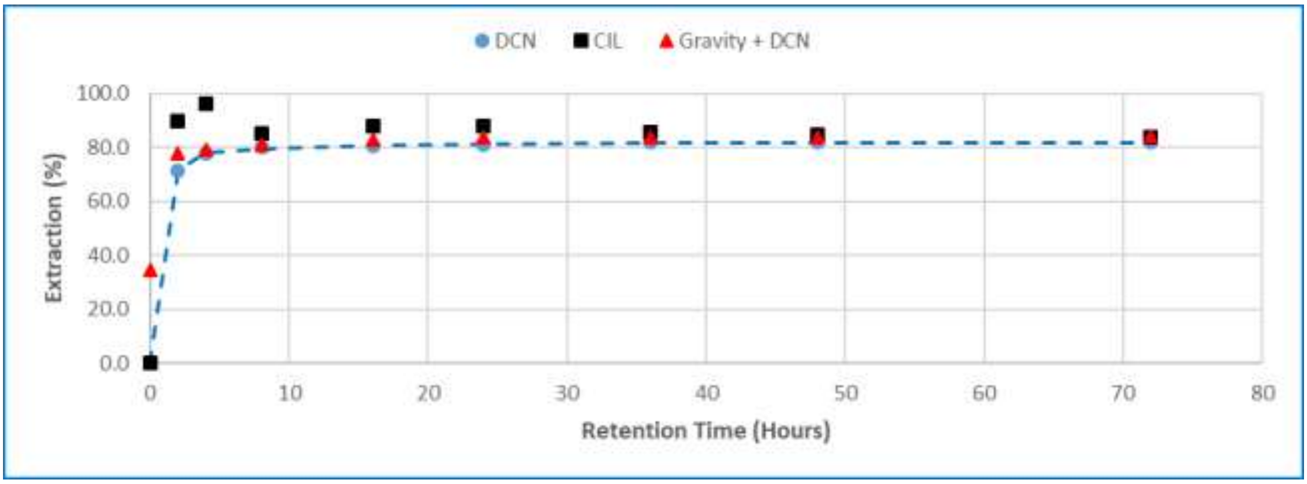


Figure 49: Composite (SP-MC) whole ore; P_{80} 75 µm leach kinetics

The following comments summarise the results:

- Consistent with sighter testing, leach residue grade is insensitive to leach methodology.
- Direct cyanidation leaching achieved a (72 hour) residue grade of 0.35 g/t from a calculated head of 1.92 g/t or 81.7% extraction. The residue grade is consistent with that reported for the Flotation Optimization composite, also indicating both composites leached to a relatively constant residue grade.
- CIL leaching resulted in slightly improved extraction (83.7%) relative to direct cyanidation (81.7%).

- Prior Knelson gravity treatment improved recovery returning a gravity recovered grade of 0.77 g/t representing 34.6% recovery. Final (72-hour) residue grade was consistent with other tests at 0.35 g/t.
- Reagent consumption was low to moderate based on Direct Cyanidation (DCN) leach results with respective cyanide and lime consumptions were 0.44 kg/t and 0.35 kg/t. Elevated cyanide consumption in the case of CIL leaching is a function of cyanide oxidation by activated carbon, typical under laboratory conditions, but not expected in practice.

13.7.2 LOM Lithology Composites, P_{80} 75 µm

The following series of whole ore leaches were conducted on LOM lithology composites: MVO, GRA and DIO.

- Direct Cyanidation (DCN) bottle-roll leaching.
- CIL based (20 g/L) bottle-roll leaching.
- Gravity recovery and Direct Cyanidation (DCN) bottle-roll leaching.

Table 57: LOM Lithology Composite whole ore leach summary P_{80} 75 µm

Test and Conditions				Leach Grade and Extraction (Ext)				Reagent Consumption	
Basis	O_2/Air	pH	Lead Nitrate	Assay Head Au (g/t)	Calc Head Au (g/t)	Leach Residue Au (g/t)	Ext (%)	NaCN (kg/t)	Lime (kg/t)
Lithology Composite: LOM Metavolcanics (MVO)									
DCN	O_2	10.5	-	1.27	1.11	0.42	62.2	0.45	0.38
CIL	O_2	10.5	-		1.69	0.47	72.3	0.81	0.37
G-DCN (*)	O_2	10.5	-		1.27	0.41	67.8	0.45	0.32
Lithology Composite: LOM Granite (GRA)									
DCN	O_2	10.5	-	1.17	0.88	0.28	68.2	0.14	0.45
CIL	O_2	10.5	-		0.86	0.29	66.2	0.64	0.36
G-DCN (*)	O_2	10.5	-		0.91	0.31	65.8	0.33	0.28
Lithology Composite: LOM Diorite (DIO)									
DCN	O_2	10.5	-	1.01	1.07	0.44	58.8	0.41	0.52
CIL	O_2	10.5	-		1.03	0.42	59.3	0.76	0.38
G-DCN (*)	O_2	10.5	-		1.32	0.44	66.6	0.36	0.32

Notes: *G-DCN = Prior gravity recovery and direct cyanidation (bottle roll) leach.

Key summary comments include the following:

- Direct cyanidation leach extraction was highest in the case of Granite relative to Metavolcanics, with lowest extraction achieved for Diorite.
- As for previous composites, leach residue and extraction were insensitive to leach method; initial leach kinetics were rapid, followed by a slow reduction after ~24 to 36 hours, consistent with leach kinetics reported for the Starter Pit composite in Figure 49.
- Extraction generally improved with prior gravity recovery and under CIL conditions; MVO reported a gravity recovered grade of 0.19 g/t, respective results for GRA and DIO composites were 0.145 g/t and 0.31 g/t.
- Leach residue grade varied dependent on arsenic content; LOM Diorite (DIO) and Metavolcanics (MVO) reported higher residue grades (0.44 and 0.42 g/t respectively) relative to Granite (GRA) at 0.28 g/t. MVO and DIO composites also reported respective arsenic head grades of 750 ppm and 320 ppm relative to GRA at 260 ppm.

- Cyanide and lime consumptions were low to moderate based on Direct Cyanidation (DCN) leach results; cyanide consumption ranged from 0.14 kg/t to 0.45 kg/t and lime from 0.38 kg/t to 0.52 kg/t.

13.7.3 Variability Lithology Composites – Variable Grind Size

13.7.3.1 Whole Ore Leach, P_{80} 75 µm

Gravity and whole ore leach results derived for lithology variability composites based on a grind (P_{80}) size of 75 µm are provided in Table 58 and compared to previous results reported for LOM lithology composites based on prior gravity recovery and direct cyanidation leaching. Key summary comments include the following:

- Gravity recoveries were similar, averaging 9.9%; total extraction ranged from 56.6% to 72.7% and reagent consumption was consistent with previous tests on Starter Pit and LOM composites.
- Trending between leach feed and 72-hour residue grade illustrated below in Figure 50 shows that residue grades are logarithmically related to leach feed grade down to a limiting feed grade of ~ 0.6 g/t; below this, the residue grade achieved for Diorite (DIO) was relatively constant at ~ 0.2 g/t.
- On the basis that Diorite reported lower extraction based on previous tests at an identical grind size, it is inferred that terminal residue grades for MVO and particularly Granite at low feed grade may have been possible below 0.2 g/t.
- The relationship between leach feed grade and residue grade indicates that the total extraction is predicated on primarily on grind size, gravity recovery and resulting leach feed grade. The specific mineralogy of separate lithologies and proportion of finely encapsulated gold determines the minimum terminal residue grade (and extraction) at low leach feed grades.

Table 58: Lithology Variability Composite whole ore leach P_{80} 75 µm

Composite	Head Grade		Gravity		Extraction at P_{80} 75 µm		Reagents (kg/t)	
	Assay	Calculated	(g/t)	(%)	Residue (g/t)	Extraction (%)	NaCN	Lime
Metavolcanics (MVO)								
LOM-MVO	**1.27**	**1.27**	**0.19**	**14.9**	**0.41**	**67.8**	**0.45**	**0.32**
MVO-V1	0.58	0.72	0.06	8.3	0.20	72.3	0.64	0.31
MVO-V2	0.86	0.94	0.08	8.3	0.33	65.0	0.35	0.29
MVO-V3	0.64	0.75	0.09	11.3	0.21	72.7	0.36	0.27
MVO-V4	1.58	1.77	0.10	5.7	0.53	70.0	0.41	0.30
Average	**0.99**	**1.09**	**0.10**	**9.7**	**0.34**	**69.6**	**0.44**	**0.30**
Granite (GRA)								
LOM-GRA	**1.17**	**0.91**	**0.15**	**16.0**	**0.31**	**65.8**	**0.33**	**0.28**
GRA-V5	0.63	0.74	0.05	6.7	0.23	69.0	0.16	0.29
GRA-V6	1.04	0.97	0.11	11.4	0.31	68.0	0.21	0.32
GRA-V7	0.93	0.94	0.04	3.7	0.41	56.6	0.28	0.24
Average	**0.94**	**0.89**	**0.09**	**9.5**	**0.32**	**64.9**	**0.25**	**0.28**
Diorite (DIO)								
LOM-DIO	**1.01**	**1.32**	**0.31**	**23.5**	**0.44**	**66.6**	**0.36**	**0.32**
DIO-V8	0.57	0.58	0.05	7.7	0.21	64.0	0.31	0.24
DIO-V9	0.60	0.49	0.04	7.2	0.21	57.0	0.33	0.22
DIO-V10	0.90	0.71	0.05	6.4	0.20	71.7	0.30	0.25
DIO-V11	0.68	0.78	0.07	8.3	0.32	59.6	0.30	0.24
Average	**0.75**	**0.78**	**0.10**	**10.6**	**0.28**	**63.8**	**0.32**	**0.25**

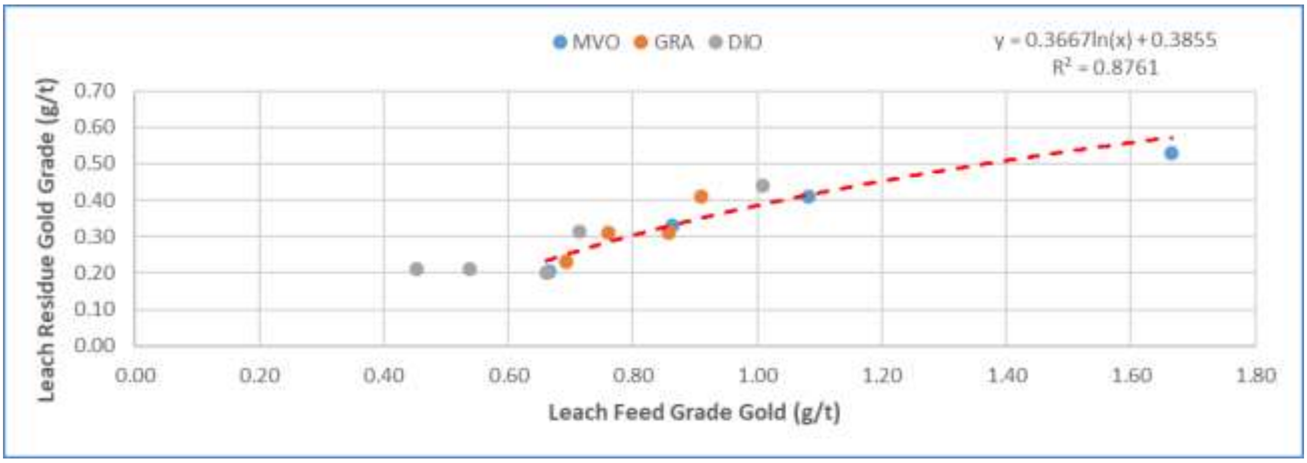


Figure 50: Lithology Variability Composite leach feed and 72-hour residue; P_{80} 75 µm

13.7.3.2 Whole ore leach P_{98} 15 µm

Whole ore cyanidation leach data based on a P_{98} grind of 15 µm is presented in Table 59. Finer grinding targeting a P_{98} size of 15 µm supports fine regrinding of flotation concentrate proposed in the process flowsheet; relative to previous tests at coarser size, total extraction increased with leach feed grade ranges of 0.61 to 1.66 g/t.

The scatter plot in Figure 51 indicates a similar trend shown previously for whole ore leaching at a 75 µm grind. Leach residue grade is shown to be proportional to leach feed grade, tending toward a lower limit of ~0.10 g/t at low leach feed grade. Notably this series of tests were performed without prior gravity recovery. Scatter in Figure 51 and relative extraction variance in Table 59 are likely to be a result of the impact of spotty gravity recoverable gold.

Table 59: Lithology Variability Composite whole ore leach P_{98} 15 µm

Composite	Head Grade		Leach		Reagents (kg/t)	
	Assay	Calculated	Residue (g/t)	Extraction (%)	NaCN	Lime
Metavolcanic						
MVO-V1	0.58	0.67	0.09	86.5	1.43	5.28
MVO-V2	0.86	0.96	0.22	77.0	1.26	2.63
MVO-V3	0.64	0.75	0.10	86.6	1.43	2.86
MVO-V4	1.58	1.66	0.29	82.6	1.54	3.43
Average	**0.92**	**1.01**	**0.18**	**83.2**	**1.42**	**3.55**
Granite						
GRA-V5	0.63	0.70	0.11	84.4	1.99	1.31
GRA-V6	1.04	0.82	0.14	82.9	0.88	1.96
GRA-V7	0.93	1.02	0.19	81.3	0.97	1.87
Average	**0.87**		**0.15**	**82.9**	**1.28**	**1.71**
Diorite						
DIO-V8	0.57	0.62	0.15	76.0	1.47	3.51
DIO-V9	0.60	0.61	0.11	81.8	1.57	1.58
DIO-V10	0.90	0.81	0.11	86.4	1.62	3.40
Average	**0.69**	**0.68**	**0.12**	**81.4**	**1.55**	**2.83**

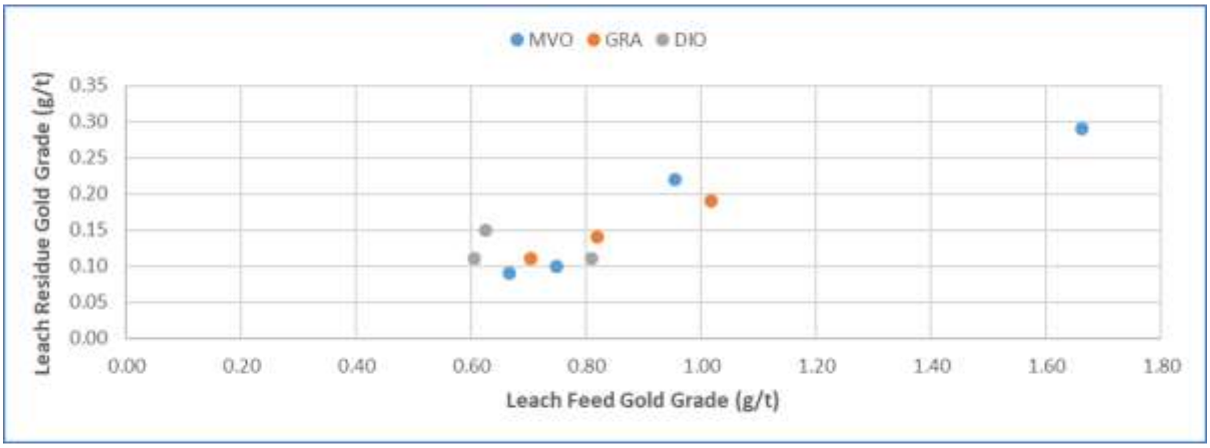


Figure 51: Lithology Variability Composite – Leach Feed and 72-hour Residue, P_{98} 15 µm

13.7.4 Whole Ore Leach – Recovered Grade Regression Analysis

A regression analysis was performed to generate a correlation describing leach recovered gold grade based on key variables: leach feed gold, arsenic and total sulfur grades. The comparison between actual and regression calculated recovered grade is presented in Figure 52. Key findings include:

- Recovered grade increases with increasing leach feed Au grade. Counter to this, increasing arsenic increases final residue grade, also reducing recovered grade.
- In overall terms the analysis indicates that at fixed grind size the various composites leach to a relatively constant residue grade that is dependent on the leach feed grade and proportion of arsenic.
- The high degree of correlation shown in the case of the recovered grade regression (R^2 = 0.99) is supported based on the results of diagnostic leaches performed on leach residues. Sulfides dominate in terms of leach residue modal gold deportment.

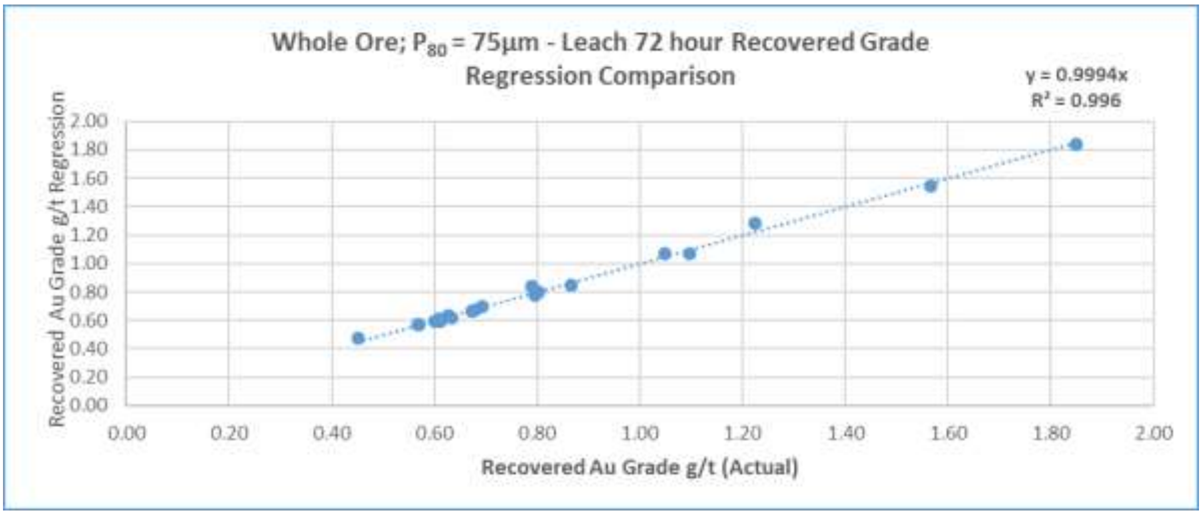


Figure 52: Whole ore P_{80} 75 µm leach 72 hour recovered grade regression comparison

13.7.5 Conclusions

Sighter testing on flotation optimised composites indicated that leach residue grade showed little sensitivity to leach method; residue grades based on direct cyanidation (bottle roll), agitated (VAT) leach and CIL methods averaged 0.35 g/t. Oxygen sparging resulted in a slightly lower residue grade of 0.37 g/t compared to air sparging (0.40 g/t) based on VAT leach test data, and addition of lead nitrate at 100 g/t had no impact on final leach residue grade over a wide pH range from 10.5 through 12.5 (maintaining 0.05% free lime).

The following conclusions are presented based on the results:

- Gravity recovery and oxygen sparging increase extraction; CIL treatment is supported.
- Leach residue grade is predicated primarily on grind size and leach feed grade; separate lithology residues tended toward a relatively constant (terminal) grade at low leach feed grade.
- Regression analysis indicated leach feed recovered gold grade increases with increasing leach feed gold grade. Counter to this, increasing arsenic acts to increase leach residue grade, also reducing recovered grade.
- Starter Pit leach extraction at a P_{80} of 75 µm reported at 84.2% with gravity recovery and a calculated leach feed grade of 2.21 g/t.
- Fine grinding of flotation concentrate proposed in the process flowsheet is supported based on lithology composite results at a coarse grind P_{80} of 75 µm and a regrind P_{98} of 15 µm. Leach extraction improved from 76.0% to 86.6% based on a P_{98} of 15 µm and a calculated leach feed grade range of 0.6 to 1.66 g/t.

13.8 Flotation Concentrate Leaching

Concentrate leach tests were performed on bulk flotation concentrates derived from the following composites:

- Flotation composite (S-MC).
- Starter Pit Composite (SP-MC).
- LOM Lithology composites representing: MVO, GRA and DIO lithologies.
- Lithology variability composites over a range of gold, arsenic and sulfur grade.

Bulk flotation was consistent with the optimal reagent regime reported in Section 13.6 with leaching conducted under on the following standard conditions:

- Variable P_{90} grind ranging from 63 µm to 5 µm.
- Leach solids loading, 30% solids (w/w) in Perth tap water.
- Leach pH target 11.0, maintained at pH 11, dosing lime.
- Initial cyanide target, 1500 ppm, maintained at 1500 ppm, dosing Sodium Cyanide (NaCN).
- Oxygen sparging to > 20 ppm dissolved oxygen.
- Kinetic leach solution sampling based on a total retention time of 72 hours.

Final comparative leach tests were also conducted to assess the following:

- Improvement in leach extraction based on Lead Nitrate dosing.

13.8.1 Flotation Concentrate Leaching – Flotation Optimization and Starter Pit Composites

Key results from flotation concentrate bottle-roll tests are summarised in Table 60 with kinetic leach curves depicted in Figure 53.

The following observations are provided based on the test data:

- Starter Pit total extraction reported at 87.9% based on a calculated head grade of 1.96 g/t and a regrind P_{90} of 6 µm. Lower grade Flotation Optimization Composite reported 80.9% from 1.24 g/t at a 9 µm grind
- Leach kinetics were rapid with extraction plateauing after 24 to 36 hours retention
- Leach kinetics increase with decreasing grind size. Notably Composite S-MC exhibited a significant increase below P_{90} of 10 µm
- Overall the results indicate leach extraction exceeding 80% with a concentrate regrind P_{90} at or below 10 µm. Nominal regrind sizes in the range 6 to 10 µm generated acceptable leach extraction.

Table 60: Master Composite – flotation concentrate direct cyanidation leach

Head Au (g/t)	Gravity (g/t)	Bulk Flotation		P_{90} (µm)	Leach			Total Residue and Extraction	
		Mass Pull (%)	Tail Au (g/t) Feed		Flotation Con (g/t)	Residue (g/t) Con	Residue (g/t) Feed	Total Residue (g/t) Feed	Total Ext (%)
Starter Pit Composite (SP-MC)									
1.96	0.43	7.18	0.10	30	18.46	4.48	0.32	0.42	78.6
				23	18.59	3.83	0.27	0.37	81.0
				13	19.24	2.93	0.21	0.31	84.3
				10	18.07	2.70	0.19	0.29	85.1
				6	16.34	1.94	0.14	0.24	87.9
Flotation Optimization Composite (S-MC)									
1.24	0.00	6.44	0.07	63	18.38	6.17	0.39	0.46	62.8
				40	17.79	5.29	0.35	0.41	66.6
				15	17.91	5.07	0.32	0.39	68.5
				9	13.94	2.66	0.17	0.24	80.9

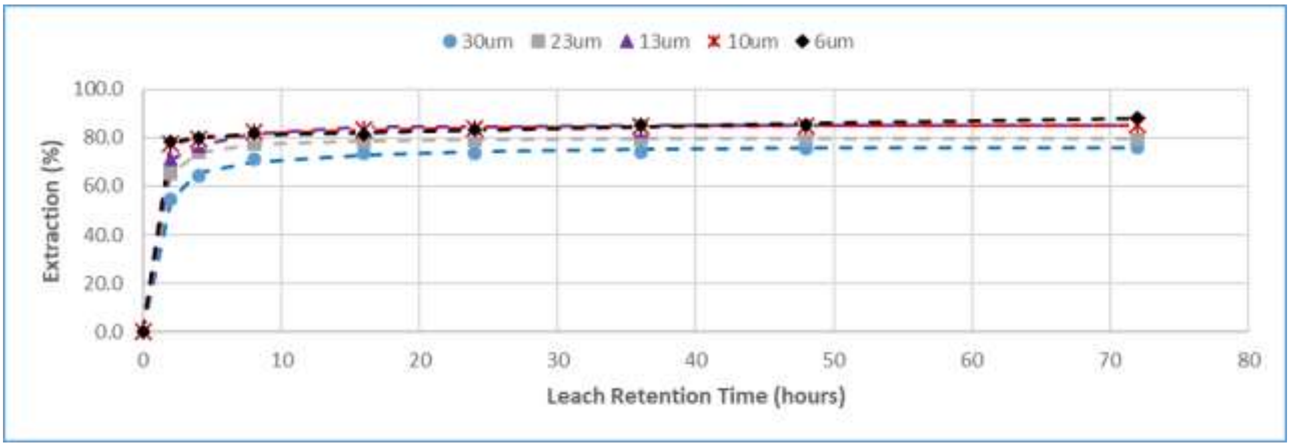


Figure 53: Starter Pit Composite (SP-MC) flotation concentrate leach kinetics

13.8.2 Flotation Concentrate Leaching – LOM Lithology Composites

LOM Lithology bulk flotation concentrates were subjected to the following range of leach tests:

- Oxygen sparged Direct Cyanidation leaching across a regrind size range of 75 µm to 9 µm
- Comparative CIL leach tests
- Comparative Direct Cyanidation leaching with and without Lead Nitrate addition.

Key results from each test phase are summarised below.

13.8.2.1 Direct Cyanidation Leaching – variable grind size

Results derived via direct cyanidation are presented below in Table 61, summarised as follows:

- Leach extraction increased with decreasing grind size; leach kinetics shown for Granite (GRA) below in Figure 54 were typical with a significant increase in extraction at and below a concentrate regrind P_{98} of 15 µm
- Final leach extraction was highest in the case of Granite (GRA) at 87.0% with a P_{98} of 9µm relative to Metavolcanics (MVO) at 82.5%; Diorite (DIO) achieved lowest extraction at 60.4%
- Final leach residue grade ranged from 2.07 g/t to 3.54 g/t at a P_{98} of 9 µm; equivalent to 0.11 g/t to 0.38 g/t whole ore feed.

Table 61: LOM Lithology Composite – Flotation Concentrate Direct Cyanidation Leach

Regrind P_{98} (µm)	Au Grade (g/t)		Extraction (%)	Residue Au Deportment (g/t)		Reagent Consumption (kg/t)	
	Leach Feed	Leach Residue		Sulfide	Silicate	Cyanide	Lime
LOM Metavolcanics (MVO)							
75	12.0	4.60	61.6	-	-	1.15	0.95
45	11.5	3.79	67.1	-	-	1.37	0.80
15	12.1	2.06	82.9	-	-	3.31	5.30
9	11.8	2.07	82.5	-	-	4.38	8.00
LOM Granite (GRA)							
75	17.2	6.38	62.9	-	-	0.99	0.85
45	17.1	5.24	69.3	-	-	1.30	0.95
15	18.8	2.25	88.0	-	-	4.55	7.15
9	18.5	2.39	87.0	-	-	6.57	7.55
LOM Diorite (DIO)							
75	9.4	3.90	58.5	-	-	0.76	1.85
45	9.1	3.49	61.5	-	-	1.19	0.95
15	8.7	3.71	57.6	-	-	1.39	2.60
9	8.9	3.54	60.4	-	-	1.54	4.15

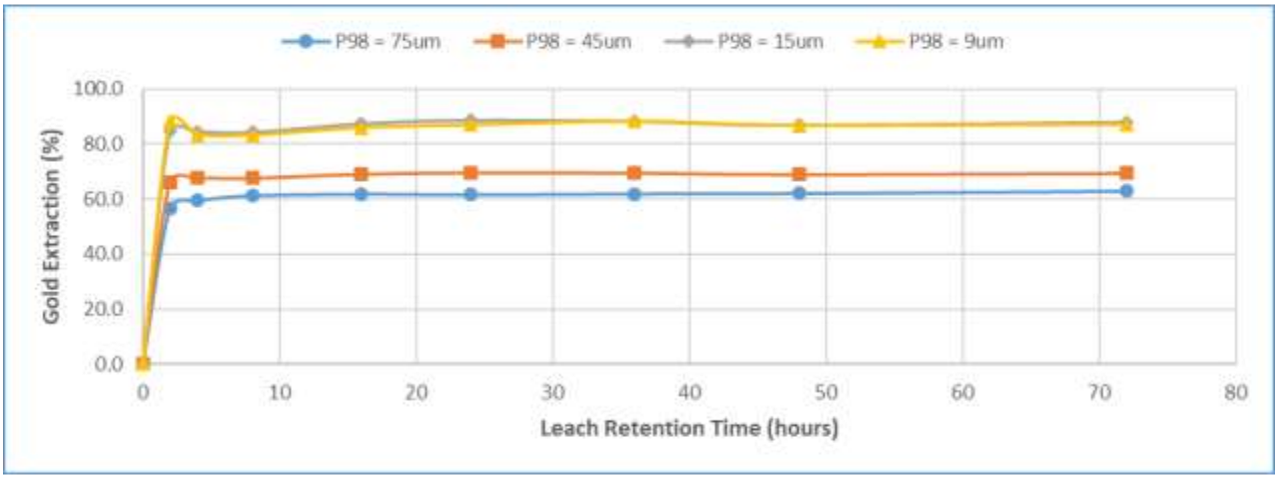


Figure 54: LOM GRA Direct Cyanidation Leach Kinetics – Variable P_{98} regrind size; pH 11

13.8.2.2 Direct Cyanidation – CIL leach tests, P_{98} 15 µm

Flotation concentrate direct cyanidation and CIL leach test results are presented in Table 62. Extraction generally improved based on CIL leaching. Diorite (DIO) CIL leach extraction was lower relative to direct cyanidation and likely impacted due to low cyanide tenor during this specific test.

Table 62: LOM Lithology Composite – CIL and direct cyanidation leach comparison

Test and Conditions				Leach Grade and Extraction (Ext)				Reagent Consumption	
Basis	O_2/Air	pH	Lead Nitrate	Assay Head Au (g/t)	Calc Head Au (g/t)	Leach Residue Au (g/t)	Ext (%)	NaCN (kg/t)	Lime (kg/t)
Lithology Composite: LOM Metavolcanics (MVO)									
DCN	O_2	11.0	-	11.0	12.6	2.95	76.7	4.07	4.15
CIL	O_2	11.0	-		11.2	2.51	77.5	9.65	4.35
Lithology Composite: LOM Granite (GRA)									
DCN	O_2	11.0	-	17.3	18.2	3.30	81.8	10.75	5.60
CIL	O_2	11.0	-		16.0	2.81	82.5	13.55	4.60
Lithology Composite: LOM Diorite (DIO)									
DCN	O_2	11.0	-	9.8	9.6	2.58	73.1	5.78	4.45
CIL	O_2	11.0	-		8.9	3.56	60.1	6.40	4.35

13.8.3 Flotation Concentrate Leaching – Lithology LOM and Variability Composites

Concentrate leach results from lithology variability composite bulk flotation tests are reported in Table 63 relative to results for LOM lithology composites. Table 63 also provides an overarching summary, including bulk flotation: head grade, gravity recovered grade, concentrate re-grind (P_{90}) size, flotation tailing grade and resultant total extraction, summarised as follows:

- Based on LOM blend: proportions: 60% MVO; 30% GRA and 10% DIO – total extraction of 72.0% with an average concentrate regrind P_{90} of 13 µm and head grade of 1.03 g/t
- Reported average total extraction at 72% is noted a function of the lower average head grade of the composites, 1.03 g/t relative to the established LOM blend head grade of 1.15 g/t
- In overall terms the results indicate higher extraction at the established LOM head grade based on a concentrate regrind P_{90} below 13 µm.

Table 63: Lithology LOM and Variability Composite flotation concentrate leach

Comp	Head Au (g/t)	Gravity (g/t)	P_{90} (µm)	Bulk Flotation			Leach Residue		Total Residue and Extraction	
				Mass Pull (%)	Conc Au (g/t)	Tail Au (g/t) Feed	Au (g/t) Conc	Au (g/t) Feed	Total Residue (g/t)	Total Ext (%)
Metavolcanics (MVO)										
LOM-MVO	1.23	0.12	14	9.1	12.01	0.11	3.09	0.28	0.39	68.4
MVO-V1	0.64	0.02	13	5.4	10.44	0.05	2.06	0.11	0.17	74.3
MVO-V1	0.64	0.02	8	5.4	10.22	0.05	1.78	0.10	0.15	76.6
MVO-V2	0.89	0.05	7	6.7	11.55	0.09	3.77	0.25	0.34	62.1
MVO-V3	0.80	0.10	14	9.7	5.68	0.14	1.49	0.14	0.28	64.9
MVO-V3	0.80	0.10	8	9.7	5.68	0.14	1.21	0.12	0.25	68.3
MVO-V4	1.46	0.05	18	9.4	14.64	0.08	3.70	0.35	0.43	70.5
MVO-V4	1.46	0.05	8	9.4	13.90	0.08	2.72	0.25	0.34	76.8
Average	**1.07**	**0.08**	**12**	**8.4**	**11.01**	**0.10**	**2.68**	**0.23**	**0.32**	**69.6**
Granite (GRA)										
LOM-GRA	1.13	0.27	17	4.8	17.94	0.04	3.91	0.19	0.23	79.8
GRA-V5	0.70	0.04	15	4.5	12.85	0.04	2.59	0.12	0.15	78.0
GRA-V5	0.70	0.04	10	4.5	12.40	0.04	2.27	0.10	0.14	80.0
GRA-V6	0.99	0.05	17	5.9	12.52	0.05	2.90	0.17	0.22	77.8
GRA-V6	0.99	0.05	10	5.9	12.46	0.05	2.41	0.14	0.19	80.8
GRA-V7	0.90	0.01	14	4.8	17.77	0.03	4.88	0.23	0.26	70.6
GRA-V7	0.90	0.01	10	4.8	17.75	0.03	4.27	0.20	0.24	73.9
Average	**0.99**	**0.14**	**15**	**4.9**	**15.95**	**0.04**	**3.53**	**0.17**	**0.21**	**78.2**
Diorite (DIO)										
LOM-DIO	1.24	0.09	21	10.6	9.30	0.11	3.44	0.36	0.48	61.5
DIO-V8	0.63	0.04	18	5.9	8.27	0.04	2.63	0.15	0.19	69.5
DIO-V8	0.63	0.04	8	5.9	8.35	0.04	2.18	0.13	0.17	73.7
DIO-V9	0.49	0.05	17	4.4	9.90	0.04	3.05	0.13	0.17	64.7
DIO-V9	0.49	0.05	9	4.4	9.82	0.04	2.60	0.11	0.15	68.7
DIO-V10	0.77	0.06	19	8.9	6.48	0.05	1.02	0.09	0.14	82.2
DIO-V10	0.77	0.06	10	8.9	7.32	0.05	0.86	0.08	0.12	84.1
Average	**0.90**	**0.07**	**17**	**8.3**	**8.79**	**0.07**	**2.69**	**0.23**	**0.30**	**68.2**
LOM Blend	**1.03**	**0.10**	**13**	**7.4**	**12.27**	**0.08**	**2.94**	**0.21**	**0.29**	**72.0**

13.8.3.1 Flotation Concentrate leach regression analysis

Concentrate leach results for separate Starter Pit and lithology composites indicated leach residue grade was sensitive to leach feed (flotation concentrate) grade and re-grind size; variability was also indicated based on leach feed arsenic grade. On this basis, separate analysis was conducted based on the Starter Pit Composite test data reported in Section 13.8.2 and Lithology Composite data in Section 13.8.4.

13.8.3.2 Starter Pit Composite

Starter Pit Composite leach residue grade was found to be strongly correlated with concentrate regrind (P_{90}) size and leach feed grade. Figure 55 illustrates the trend relative to regrind size and Figure 56 shows the comparison between actual leach residue grade reported in Section 13.8.2 relative to the regression calculated residue grade based on variable grind size and leach feed grade.

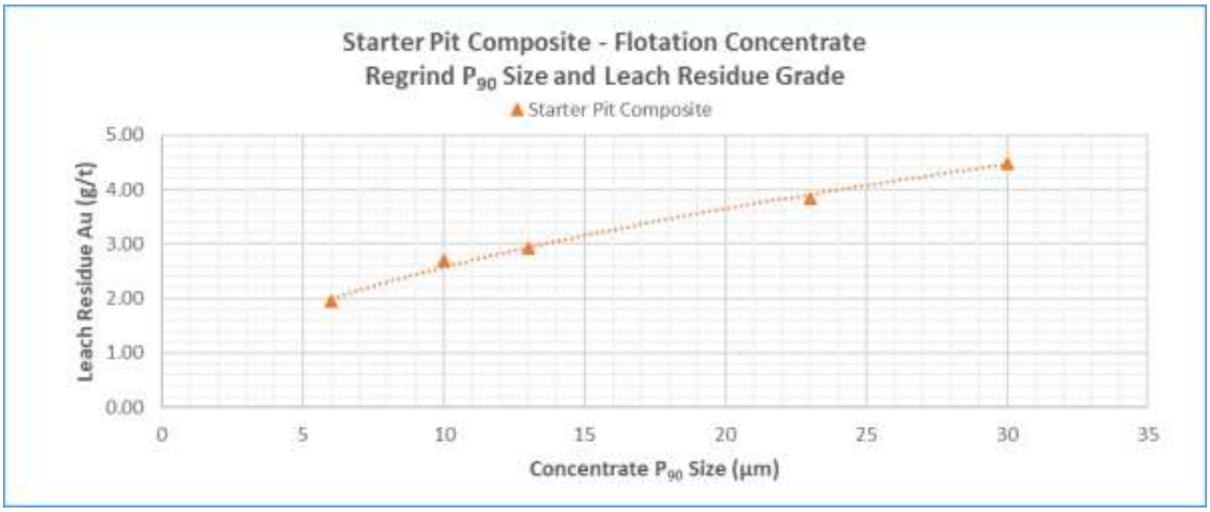


Figure 55: Starter Pit Composite – leach residue gold grade and regrind (P_{90})

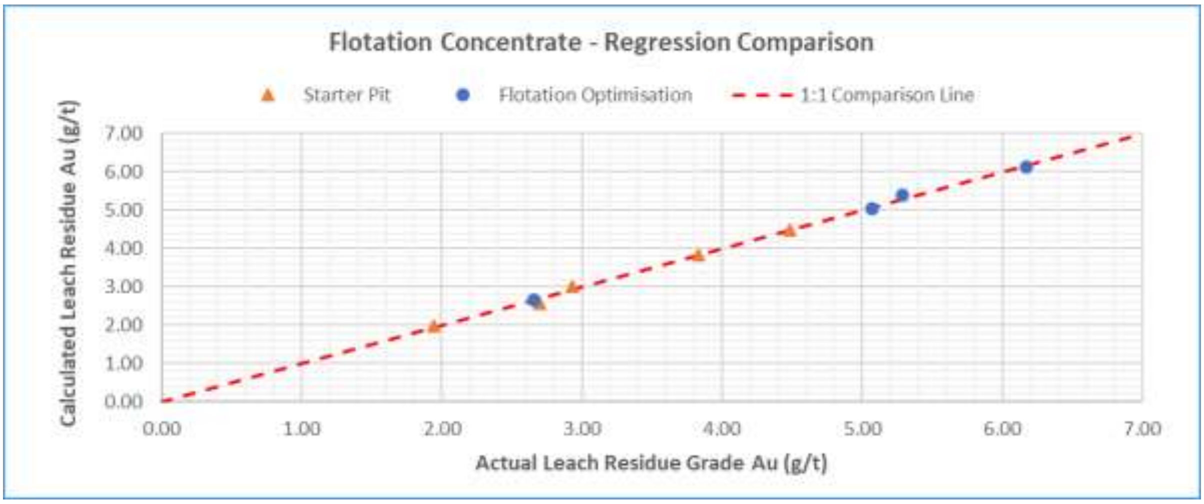


Figure 56: Starter Pit and Flotation Composite leach residue regression comparison to actual

13.8.3.3 Lithology LOM and Variability Composites

Analysis of variability in lithology composite test data reported previously in Table 63 identified the following:

- Metavolcanics (MVO) was closely correlated with regrind size and leach feed gold and arsenic grade
- Granite (GRA) correlated based on regrind size and leach feed gold grade
- Diorite (DIO) correlated with regrind size and leach feed gold and arsenic grade.

Actual leach residue grades are compared to regression calculated results in Figure 57. Reasonable correlation is demonstrated across all composites based on a total of 34 bulk flotation concentrate leach tests.

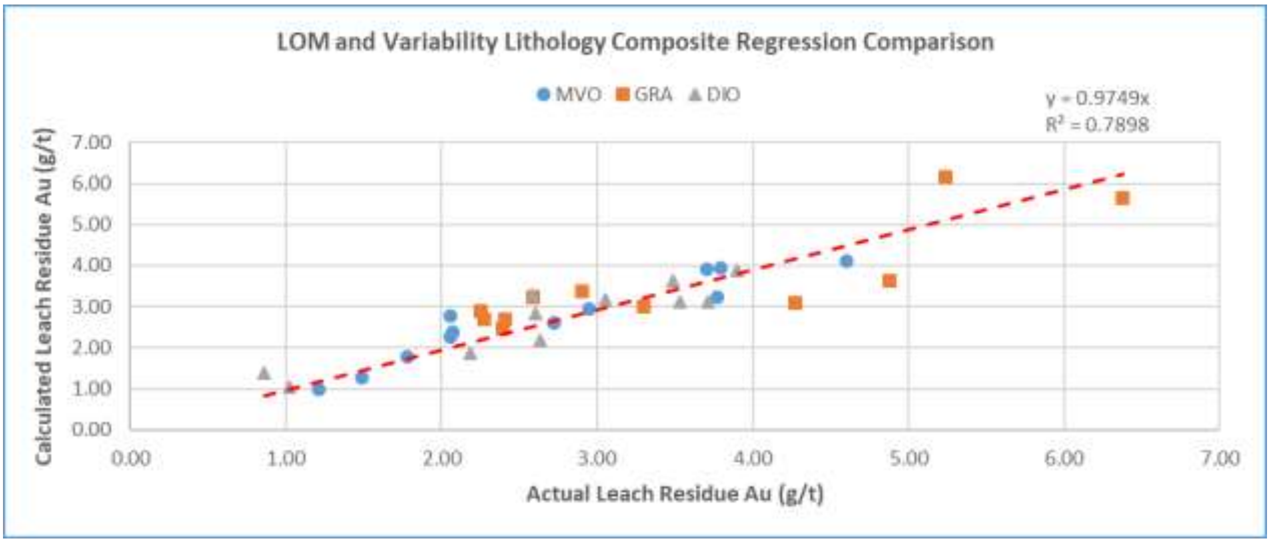


Figure 57: LOM and Variability Lithology Composite – leach residue regression comparison to actual

13.8.4 Flotation and Concentrate Leach – grade recovery

Starter Pit and lithology, LOM and variability composite test data was utilised in conjunction with previously reported flotation and concentrate leach regression analysis to generate grade recovery curves for the Starter Pit and LOM blend based on a concentrate regrind P_{90} ranging from 6 µm to 9 µm (equivalent to a P_{98} 15 µm).

Key results relative to the grade recovery curves are illustrated in Figure 58 at P_{90} ranging from 6 µm to 9 µm:

- Starter Pit recovery at an average head grade of 1.31 g/t ranges from 83.4% to 85.1%.
- LOM Blend recovery with an average head grading 1.15 g/t ranges from 79.3% to 83.0%.

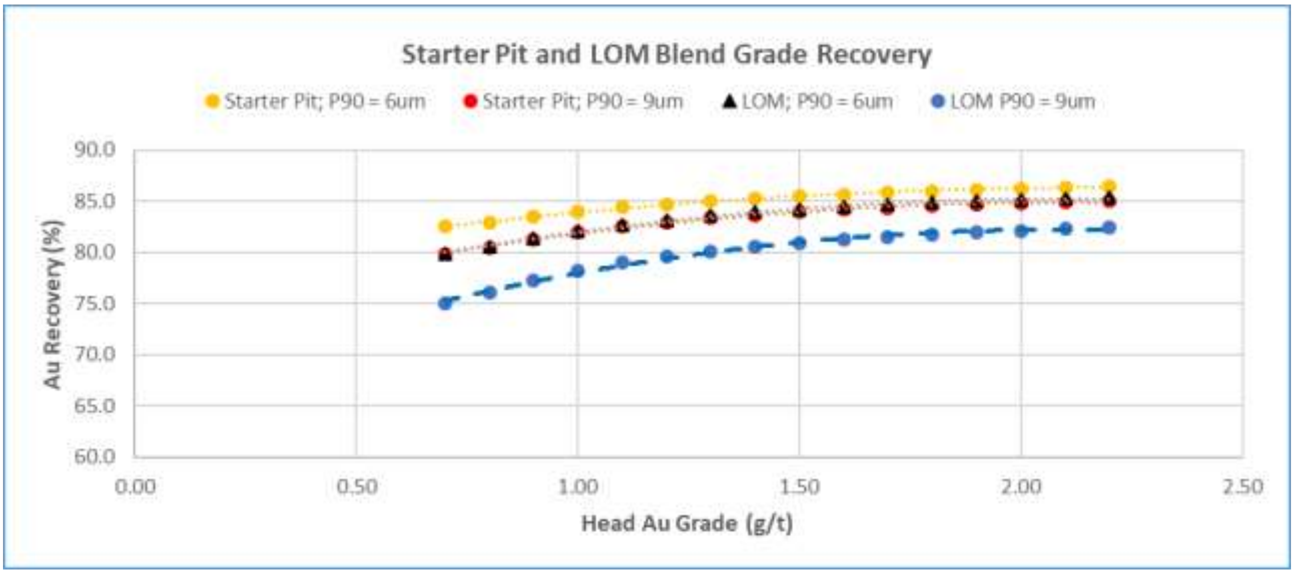


Figure 58: Starter Pit and LOM blend – grade recovery curves

13.8.5 LOM Lithology Composite – lead nitrate assisted leaching

Potential for improved flotation concentrate leach extraction at a grind (P_{98} 15 µm) size was assessed based on direct cyanidation at pH 11 applied to LOM lithology composites with and without 500 g/t lead nitrate. Similar testing applied on a whole ore basis in the case of Flotation and Starter Pit composites did not indicate any measurable difference. Table 64 indicates:

- Improvement in MVO gold extraction (+4.1%) and silver (+0.9%). Separate sub-sample results report a lower gold residue grade (2.53 g/t) with lead nitrate addition from a higher head grade (13.16 g/t) relative to direct cyanidation without lead nitrate.
- Granite reported slight improvement (+0.6%) gold and (+0.9%) silver, while DIO reported only a slight increase in silver extraction with no improvement in gold extraction.
- Overall the results indicate that MVO flotation concentrate kinetics and extraction may respond positively to lead nitrate and reduced passivation as evidenced in the kinetic profile (Figure 59).

Table 64: LOM Lithology Flotation Concentrate lead nitrate assisted leach summary

Lithology	$Pb(NO_3)_2$ (g/t)	Head Grade (g/t)		Residue Grade (g/t)		Extraction (%)		Extraction Delta (%)	
		Gold	Silver	Gold	Silver	Gold	Silver	Gold	Silver
MVO	500	13.16	3.04	2.53	0.60	80.8	80.3	+4.1	+0.9
	0	12.64	2.91	2.95	0.60	76.7	79.4		
GRA	500	18.33	6.25	3.22	1.20	82.4	80.8	+0.6	+0.9
	0	18.18	5.99	3.30	1.20	81.8	80.0		
DIO	500	9.73	2.86	2.70	0.60	72.3	79.0	-0.8	+0.4
	0	9.59	2.80	2.58	0.60	73.1	78.6		

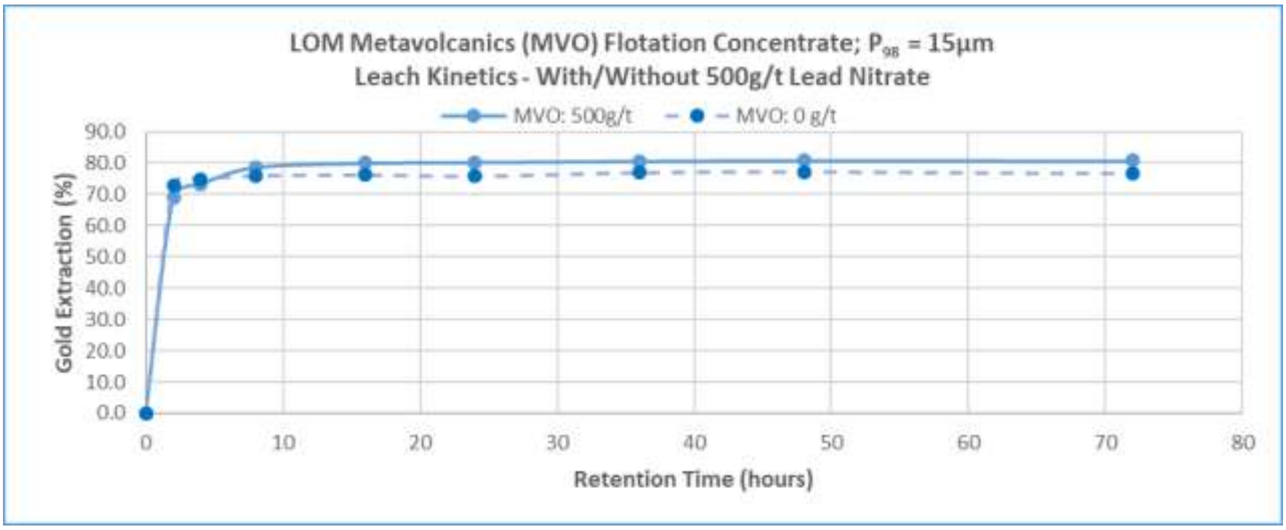


Figure 59: MVO flotation concentrate – leach kinetics with/without 500 g/t lead nitrate

13.8.6 Conclusions

The following conclusions are presented based on the results and analyses discussed:

- Leach kinetics were rapid with extraction plateauing after 24-36 hours retention. Consistent with the current flowsheet, CIL leaching and recovery are supported. However, an opportunity exists to reduce the process design leach residence time allowed which is currently 72 hours.

- Leach residue grade reduces with decreasing grind (P_{90}) size. Analysis of separate lithology composite data indicates that Metavolcanics (MVO) and Diorite (DIO) leach residue grades are controlled based on: leach feed grade, regrind size and arsenic head grade. Granite (GRA) leach residue was controlled based on leach feed grade and regrind size.
- Starter Pit composite reported 87.9% extraction at a calculated head of 1.96 g/t and a regrind P_{98} of 15 µm. Gold recovery based on the average Starter Pit head grade of 1.31 g/t is calculated at 85.1% based on a concentrate regrind P_{98} of 15 µm. A combination of oxide and fresh material yields an overall recovery of for the starter pit of 86%.
- Gold recovery based on LOM blend proportions: 60% Metavolcanics, 30% Granite and 10% Diorite and LOM head grading 1.15 g/t is calculated at 84% based on a fine concentrate regrind P_{98} of 15 µm.
- Lead nitrate dosed at 500 g/t resulted in significant improvement in flotation concentrate leach extractions for MVO and GRA lithologies. MVO reported +4.1% higher and GRA, +0.6%.

13.9 Summary of conclusions

Key results supporting selection of the process flowsheet from the metallurgical testing are as follows:

Comminution:

- From UCS testing, the Namdini ore is amenable to conventional jaw or gyratory crushing
- SAG Mill Comminution (SMC) testing supports configuration of the comminution circuit based on the proposed SABC flowsheet, incorporating SAG milling and recycle crushing of SAG mill scats, coupled with Ball milling to a finished primary grind (P_{80}) size of 106 µm.

Gravity:

- Gravity testwork proved the requirement of an upfront gravity recovery process. Prior gravity treatment lifted base-line recovery, mitigating lower flotation recovery of free gold. The inclusion of a gravity circuit is justified based on recoveries achieved for the Starter Pit composite.

Flotation:

- Metallurgical response was very consistent for all lithologies
- Gold and sulfur flotation kinetics were rapid with high recovery (> 90%) achieved after five minutes with an industry standard reagent regime
- Variable primary grind and flotation testing confirmed a primary grind and flotation P_{80} of 106 µm suitable for on-going development. However, scope for increasing the primary grind to P_{80} of 150 µm without compromising flotation recovery is a possibility
- Site water analysis confirmed no measurable impact on flotation kinetics or recovery.

Leach:

- Leach kinetics were rapid with extraction plateauing after 24-36 hours retention. Consistent with the current flowsheet, CIL leaching and recovery are supported. However, an opportunity exists to reduce the process design leach residence time allowed which is currently 72 hours
- Leach residue grade is predicated primarily on grind size and leach feed grade; separate lithology residues tended toward a relatively constant (terminal) grade at low leach feed grade
- Leach residue grade reduces with decreasing regrind size

- Starter Pit composite reported 87.9% extraction at a calculated head of 1.96 g/t and a regrind P_{98} of 15 µm. Gold recovery based on the average Starter Pit head grade of 1.31 g/t is calculated at 85.1% based on a concentrate regrind P_{98} of 15 µm. A combination of oxide and fresh material yields an overall recovery of 86% for the starter pit
- Gold recovery based on LOM blend proportions: 60% Metavolcanics, 30% Granite and 10% Diorite and LOM head grading 1.15 g/t is calculated at 84% based on a fine concentrate regrind P_{98} of 15 µm.

13.10 Recommendations

The following recommendations are offered based on outcomes derived from the testwork:

- Confirmatory Knelson gravity testing based on CONSEP method, including stage grinding, cyclic recovery and modelling is recommended to support equipment selection and to quantify the recovery expected at full-scale. Testing is recommended based on separate Starter Pit and lithology composites
- Comparative gravity and batch flotation tests are recommended over the pH range 6.5-8.0 to assess any relative improvement in base-line gold recovery
- Further study work should consider the incremental operating cost associated with Polyfroth H27 and MIBC
- Testwork incorporating a cleaner circuit is recommended to assess the economic trade-off between concentrate mass, concentrate fine regrind size and gold grade/recovery. Review of historical cleaner flotation testwork will assess if cleaner tailings largely comprise physically entrained (siliceous) gangue at high rougher concentrate mass
- Based on diagnostic leach results reported in Section 13.4.2 differential testing of pyrite-arsenopyrite flotation is recommended to assess the ability to concentrate the fine grinding effort on arsenopyrite to reduce overall regrind power consumption. This assessment should be compared to results based on prior cleaner stage flotation
- Lead nitrate reagent dosing optimization should be explored to determine optimum metallurgical response to improve flotation concentrate leach extraction
- Pre-leach oxidative testwork utilising shear reactor technology is recommended for further evaluation to enhance leach kinetics and gold recovery.

13.11 Design criteria development

The comminution and metallurgical testwork has provided detailed and fundamental information about the physical characteristics and metallurgical response of the three Namdini lithologies. The process design criteria have been developed based on the available testwork, Cardinal's advice and industry-based assumptions. The processing route selected for the Namdini ores is recognised to be commercially robust using known unit operations. The process flowsheet incorporates rougher flotation followed by concentrate regrind and CIL cyanidation of flotation concentrate.

Comminution process selection and sizing is detailed in Section 17.1.3. Orway Mineral Consultants ("OMC") used the testwork results for grinding plant definition and mill sizing. A primary crushing and SABC comminution circuit (open circuit SAG mill followed by closed circuit ball mill and recycle pebble crushing) was selected by OMC based on all available comminution parameters.

Grind sensitivity testwork indicated that ~95% flotation recovery could consistently be achieved at coarse grinds. For purposes of the PFS design, it was agreed with Cardinal that pre-flotation grind size P_{80} of 106 µm be selected for the study.

Gravity recovery was included in the flowsheet on the basis that ~15% gold for the Starter Pit and ~5% gold for the LOM ore was potentially recoverable. The gravity plant will enable any spikes of high-grade material that may be fed to the process plant to be smoothed out prior to flotation, thereby allowing gravity gold to be recovered and allowing more consistent operation of the CIL and elution unit operations.

The rougher rate flotation testwork indicated fast flotation kinetics, with >90% of the overall recovered gold occurring after five minutes of flotation with no significant improvement evident post 15 minutes. A testwork flotation time of 15 minutes with a process plant scale-up factor of 2 was allowed for the PFS study. Mass recovery to concentrate of 7.5% of rougher feed was used for design of the rougher flotation cells.

The concentrate regrind and cyanidation testwork indicated regrinding the rougher concentrate followed by oxygen aeration prior to cyanide leach achieved the highest gold extractions. The concentrate leach testwork was completed as a direct leach (without carbon) and hence it is expected that a CIL test may achieve higher gold extraction. A 72-hour leach residence time has been allowed for in the design, but leach kinetics indicate that this residence can be shortened.

Industry typical design parameters were assumed for the PFS study where testwork has not yet been completed. Metallurgical testwork is continuing for the Namdini Gold Project under the direction of Cardinal at ALS in Perth, Western Australia.

14.0 MINERAL RESOURCE ESTIMATES

This Section is based on the Mineral Resource estimate produced by MPR (2018) announced on 5 March 2018. The resulting block model was used for the mining planning studies leading to the Ore Reserve (Section 15.0).

14.1 Approach

The Mineral Resource was estimated by Multiple Indicator Kriging ("MIK") incorporating a variance adjustment to reflect open pit mining selectivity, a method that has been demonstrated to provide reliable estimates of gold resources achieved in open pit mining for a wide range of mineralization styles.

The estimates are based on RC and diamond information available on the 15 of February 2018. Relative to drilling information available for the September 2017 resource estimates, the dataset available for the current estimates includes an additional 15,747 m of drilling. This additional drilling is dominated by infill holes in central portions of the mineralized domain.

Micromine software was used for data compilation, domain wire-framing and coding of composite values and GS3M software from FSSI Consultants (Australia) Pty Ltd was used for resource estimation. The resulting estimates were imported into Micromine for resource reporting.

The Qualified Person responsible for the Mineral Resources is Nicolas Johnson who is a full-time employee of MPR Geological Consultants Pty Ltd and a member of the Australian Institute of Geoscientists. Mr Johnson visited the project between the 11 and 14 January 2017.

Mineral Resources were previously estimated for Namdini in November 2016, April 2017 and September 2017 (Section 6.4) in accordance with the JORC Code (2012) and NI 43-101. The basis, method of estimation and classification of the previous Mineral Resource estimates under the JORC Code do not materially vary from the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves. The present Mineral Resource estimates are classified and reported in accordance with NI 43-101 guidelines and classifications adopted by CIM (2014).

Mineral Resources that are not Ore Reserves do not have demonstrated economic viability. The extent to which mining, metallurgical, marketing, infrastructure, permitting, marketing and other financial factors may affect the Mineral Resource estimates are not defined. An approach was used (Section 14.10) to demonstrate that the Mineral Resource estimates have reasonable prospects for eventual economic extraction as required under relevant codes (JORC, 2012; CIM, 2014).

14.2 Estimation dataset

The current estimates are based on 2 m down-hole composited gold grades from RC and diamond drilling with not-sampled intervals generally assigned gold grades of 0.001 g/t Au. Peripheral, un-mineralized drill holes not relevant to the resource estimates were removed from the resource dataset, along with the following:

- 12 diamond holes drilled for geotechnical or metallurgical purposes for which routine down-hole assays are un-available,
- 10 not-assayed RC holes drilled for hydrological investigations
- 167 sterilization and exploratory RC holes from outside the resource area, and
- 317 closely spaced holes, drilled as part of a grade control trial program.

The compiled resource dataset comprises 40,089 composites with gold grades from 0.00 to 242.05 g/t Au, averaging 0.58 g/t Au. Additional information compiled since construction of the September 2017 resource estimates represents around 19% of the dataset.

The spatial configuration of the drill holes with gold composites is shown in Figure 60.

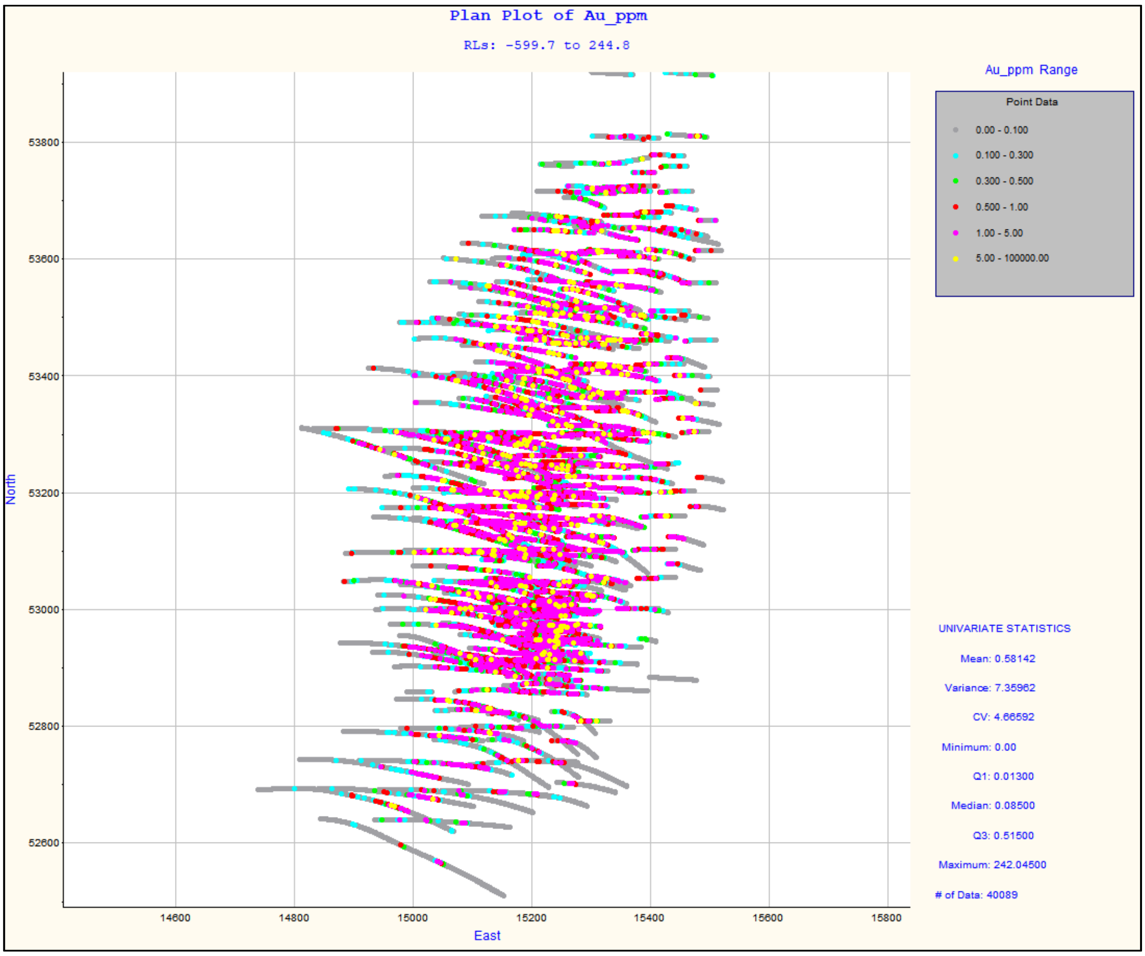


Figure 60: Gold composites (source: MPR, 2018)

14.3 Modelling domains

The transition from gold mineralization to barren host rock is generally characterized by diffuse grade boundaries.

The current estimates are based on a mineralized domain interpreted on the basis of composited gold grades. Domain boundaries were digitized on cross sections, snapped to drill hole traces where appropriate, then wire-framed into a three-dimensional solid designated Domain 2. Domain 1 represents a background domain capturing generally non-mineralized composites outside the mineralized domain wire-frame.

The mineralized domain captures zones of continuous mineralization with composite grades of greater than nominally 0.1 g/t Au. This domain trends north-northeast over a strike length of approximately 1,330 m with horizontal widths ranging from around 90 to 400 m and averaging approximately 240 m. The domain dips to the west at around 70° and is interpreted to a constant elevation of -500 m RL, which represents an average depth of around 710 m.

Figure 61 is a plan view of the surface expression of the mineralized domain relative to drill hole traces.

Cardinal supplied closed wire-frames representing the main rock units and wire-framed surfaces representing the base of oxidation and the top of fresh rock interpreted from drill hole logging. Logging, interpretation and modelling were undertaken by Cardinal technical staff and specialist structural consultants Orefind Pty Ltd updated the three-dimensional model of key rock types, structures and weathering zones. These wire-frames were used for flagging of the resource composites into Oxide, Transition and Fresh subdomains, and assigning rock types and oxidation zones to the block model for density assignment and to partition final resources by oxidation type. Depth to the interpreted base of complete oxidation ranges from 2 to 20 m and averages approximately 10 m. Interpreted depth to fresh rock ranges from 8 to 30 m depth and averages 18 m.

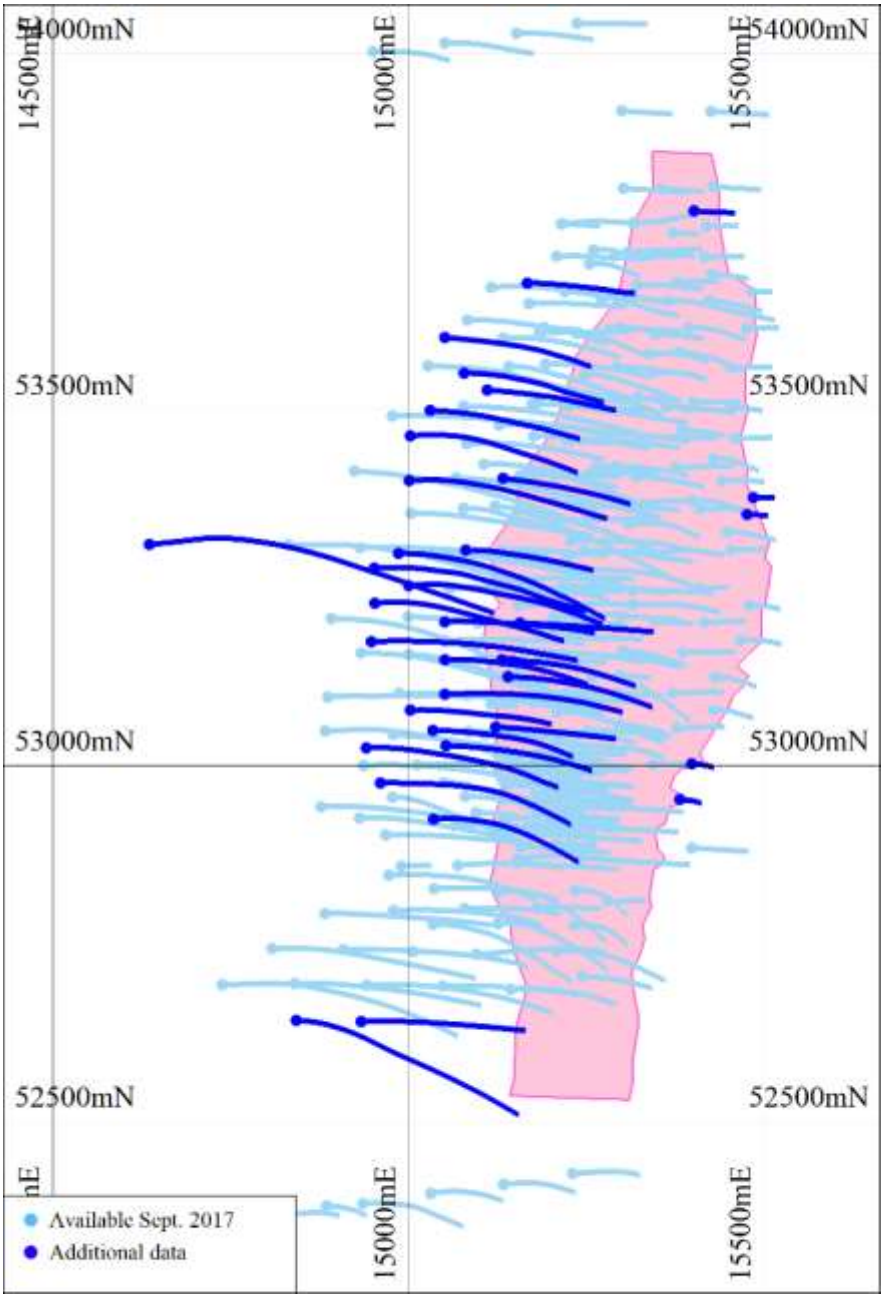


Figure 61: Mineralized domain and drill hole traces (source: MPR, 2018)

14.4 Exploratory data analysis

Table 65 shows univariate statistics of composite gold grades for the resource dataset subdivided by mineralization and oxidation domain. Notable features of these statistics include the following:

- At 0.05 g/t, the mean gold grade for Domain 1 composites is notably lower than for Domain 2 demonstrating that the domaining was effective in assigning most mineralized composites into the mineralized domain.
- Typical of many gold deposits, all populations of gold grades show strong positive skewness with coefficient of variation ("CV") generally greater than 2.0, indicating that MIK is an appropriate estimation technique and that selective mining above elevated cut-off grades will be difficult.

Table 65: Composite statistics by Domain and oxidation type

Au g/t	Oxide	Transition	Fresh	Combined
Domain 1				
Number	557	851	10,347	11,755
Mean	**0.07**	**0.03**	**0.04**	**0.04**
Variance	0.08	0.01	1.06	0.94
CV	**3.97**	**3.25**	**25.5**	**23.5**
Minimum	0.01	0.01	0.00	0.00
1st Quartile	0.01	0.01	0.01	0.01
Median	0.01	0.01	0.01	0.01
3rd Quartile	0.04	0.02	0.02	0.02
Maximum	4.87	2.11	103.09	103.09
Domain 2				
Number	1,275	1,042	26,017	28,334
Mean	**0.84**	**0.75**	**0.81**	**0.81**
Variance	2.27	2.37	10.5	9.85
CV	**1.80**	**2.06**	**4.02**	**3.90**
Minimum	0.00	0.01	0.00	0.00
1st Quartile	0.13	0.08	0.05	0.05
Median	0.38	0.30	0.26	0.27
3rd Quartile	0.93	0.77	0.79	0.79
Maximum	21.20	22.49	242.05	242.05

14.5 Indicator thresholds and bin average grades

For datasets formed from each mineralization and oxidation domain combination, indicator thresholds were defined using a consistent set of percentiles representing probability thresholds of 0.1, 0.2, 0.3, 0.4, 0.5, 0.6, 0.7, 0.75, 0.8, 0.85, 0.9, 0.95, 0.97 and 0.99 for data in each data subset.

All class average grades were determined from bin mean grades with the exception of the upper bins, which were reviewed on a case by case basis and bin grades selected on the basis of bin mean, or median with or without exclusion of high-grade composites. This approach was adopted to reduce the impact of a small number of outlier composites. This approach is appropriate for MIK modelling of highly variable mineralization such as Namdini. No composites were excluded from the estimation dataset and the entire composite dataset was used for the MIK modelling.

Table 66 presents the grade thresholds and class mean gold grades, with the value assigned to the upper bin shown in brackets along with methodology used to determine upper bin grades. The exclusion threshold value of 50 g/t Au selected for determining class grades for the fresh Domain 2 dataset represents the 99.95 percentile of this subset.

Table 66: Indicator thresholds (TH) and class grades (median or cut mean for highest bin)

Percentile	Oxide		Transition		Fresh	
	TH (Au g/t)	Mean (Au g/t)	TH (Au g/t)	Mean (Au g/t)	TH (Au g/t)	Mean (Au g/t)
Domain 1 (Background)						
10%	0.005	0.005	0.005	0.005	0.005	0.005
20%	0.005	0.005	0.005	0.005	0.005	0.005
30%	0.005	0.005	0.005	0.005	0.005	0.005
40%	0.008	0.007	0.005	0.005	0.005	0.005
50%	0.013	0.012	0.008	0.007	0.006	0.005
60%	0.020	0.017	0.013	0.012	0.008	0.007
70%	0.030	0.025	0.018	0.015	0.013	0.012
75%	0.043	0.036	0.023	0.020	0.018	0.015
80%	0.055	0.048	0.025	0.024	0.023	0.019
85%	0.075	0.064	0.035	0.031	0.030	0.026
90%	0.125	0.094	0.055	0.046	0.045	0.036
95%	0.220	0.166	0.110	0.075	0.085	0.061
97%	0.285	0.250	0.200	0.162	0.145	0.108
99%	1.115	0.689	0.320	0.268	0.390	0.229
100%	4.870	2.212 (1.865)	2.110	0.815 (0.658)	103.085	2.115 (0.710)
		(Median)		(Median)		(Median)
Domain 2 (Mineralized)						
10%	0.040	0.019	0.015	0.007	0.008	0.006
20%	0.090	0.064	0.050	0.032	0.030	0.019
30%	0.165	0.127	0.110	0.076	0.070	0.046
40%	0.255	0.211	0.200	0.152	0.148	0.105
50%	0.375	0.315	0.295	0.249	0.260	0.201
60%	0.505	0.436	0.425	0.355	0.410	0.334
70%	0.745	0.609	0.625	0.521	0.630	0.511
75%	0.930	0.835	0.765	0.688	0.781	0.702
80%	1.185	1.034	0.995	0.872	1.000	0.886
85%	1.415	1.295	1.265	1.128	1.315	1.145
90%	2.005	1.721	1.705	1.457	1.840	1.552
95%	3.070	2.468	2.742	2.143	2.945	2.302
97%	4.190	3.564	3.985	3.379	3.950	3.402
99%	7.370	5.372	7.705	5.365	7.225	5.201
100%	21.200	11.138 (9.925)	22.490	11.401 (8.955)	46.185	12.833 (9.775)
		(Median)		(Median)		(Mean excluding > 50 g/t Au)

14.6 Variogram models

Domain 2 indicator variograms were modelled for thresholds defined using percentiles representing probability thresholds of 0.1, 0.2, 0.3, 0.4, 0.5, 0.6, 0.7, 0.75, 0.8, 0.85, 0.9, 0.95, 0.97 and 0.99 for the dataset of combined oxidation subdomains (Table 67). For determination of variance adjustment factors a variogram model of composite gold grades was also developed for the dataset. The variograms modelled for Domain 2 were used for modelling Domain 1.

Spatial continuity observed in the variograms is consistent with geological interpretation and trends shown by resource composite gold grades. As an example of the variogram models, Figure 62 presents a three-dimensional variogram surface map of the median indicator variogram model at variogram ranges at 90% of sill.

Table 67: Variogram models

Rotation: Z+34,Y+41,X+77							
Percentile	Nugget	First Structure (Exponential)		Second Structure Spherical		Third Structure Spherical	
		Sill	Range (x,y,z)	Sill	Range (x,y,z)	Sill	Range (x,y,z)
10%	0.24	0.45	4.5,6.5,9.0	0.27	47,157,256	0.04	1860,2209,291
20%	0.24	0.45	6.0,10.5,16	0.27	57,268,261	0.04	2114,2885,363
30%	0.24	0.45	6.0,8.0,20	0.27	57,372,302	0.04	1467,2631,330
40%	0.24	0.45	5.5,9.5,12.5	0.27	55,320,344	0.04	1475,2324,377
50%	0.26	0.45	5.0,10,15	0.27	58,389,285	0.02	688,666,604
60%	0.26	0.50	6.0,6.5,41.5	0.18	37,297,180	0.06	575,2354,490
70%	0.27	0.50	4.5,36.5,11.5	0.18	32,257,150	0.05	142,1137,1136
75%	0.27	0.50	4.5,7.5,36.5	0.18	25,201,42	0.05	87,692,662
80%	0.27	0.50	4.0,9.5,13.5	0.18	18,97,56	0.05	136,1026,192
85%	0.27	0.50	4.0,4.0,4.0	0.18	16,59,129	0.05	205,1308,179
90%	0.29	0.53	4.0,4.5,5.5	0.15	19,135,57	0.03	136,836,215
95%	0.29	0.57	4.0,4.0,4.0	0.11	12,35,80	0.03	1174,2045,257
97%	0.29	0.62	4.0,4.0,4.0	0.06	38,16,103	0.03	48,137,381
99%	0.34	0.61	4.0,4.0,21.5	0.05	201,180,294	-	-
Au g/t	0.21	0.58	4.5,5.5,7.5	0.10	13,105,54	0.11	36,284,55

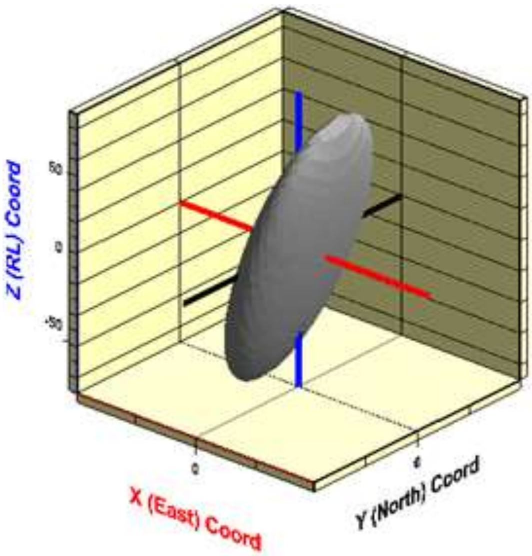


Figure 62: Three-dimensional variogram plot (source: MPR, 2018)

14.7 Estimation parameters

The block model framework used for MIK estimation covers the full extent of the composite dataset (Figure 20). It includes panels with dimensions of 12.5 m east by 25 m north by 5 m vertical. The plan view panel dimensions are consistent with drill hole spacing in the more closely drilled portion of the deposit.

The three progressively more relaxed search criteria used for MIK estimation are presented in Table 68. The search ellipsoids were aligned with the general mineralization orientation.

Table 68: Search criteria

Search	Radii (m) (x,y,z)	Minimum Data	Minimum Octants	Maximum Data
1	65,65,15	16	4	48
2	97.5,97.5,22.5	16	4	48
3	97.5,97.5,22.5	8	2	48

Estimated resources include a variance adjustment to give estimates of recoverable resources for SMU dimensions of 5 by 10 by 2.5 m with high quality grade control sampling on an 8 by 12 by 1.25 m pattern. Variance adjustments were applied using the Direct Lognormal Correction method. The recoverable resource estimates can be reasonably expected to provide appropriately reliable estimates of potential mining outcomes at the assumed selectivity without application of additional mining dilution, or mining recovery factors.

The Direct Lognormal Correction factors are provided in Table 69.

Table 69: Variance adjustment factors for the Direct Lognormal Correction method

Domain	Block/Panel	Information Effect	Total Adjustment
1	0.137	0.434	0.059
2	0.137	0.434	0.059

14.8 Bulk density assignment

Collection and validation of the bulk density data was discussed in Section 11.4.

Bulk densities were assigned to the block model by rock type and oxidation domain. The assigned values (Table 70) were derived from the average of the available measurements (Table 28).

Table 70: Bulk density assignment

Rock Type	Bulk Density (t/bcm)		
	Oxide	Transition	Fresh
Metavolcanic	2.00	2.54	2.81
Tonalite	2.51	2.54	2.73
Diorite	2.27	2.58	2.82
Metasediment	2.25	2.58	2.82
Dacite	2.40	2.40	2.78
Pyroclastic	2.39	2.60	2.73

14.9 Mineral Resource classification

Resource model blocks were classified as Indicated or Inferred (Figure 63), on the basis of search pass and a wire-frame outlining the more closely drilled portions of the mineralization as follows:

- All panels informed by search passes 1 and 2 were initially classified as Indicated, with all search pass 3 panels classified as Inferred.
- Panels within the classification wire-frame informed by search pass 3 were re-assigned to the Indicated category. These panels represent around 0.1% of the Indicated Resource.

- A small number of search pass 3 panels outside the classification wire-frame surrounded by Indicated panels were re-classified as Indicated. These panels represent around 0.01% of the Indicated Resource.

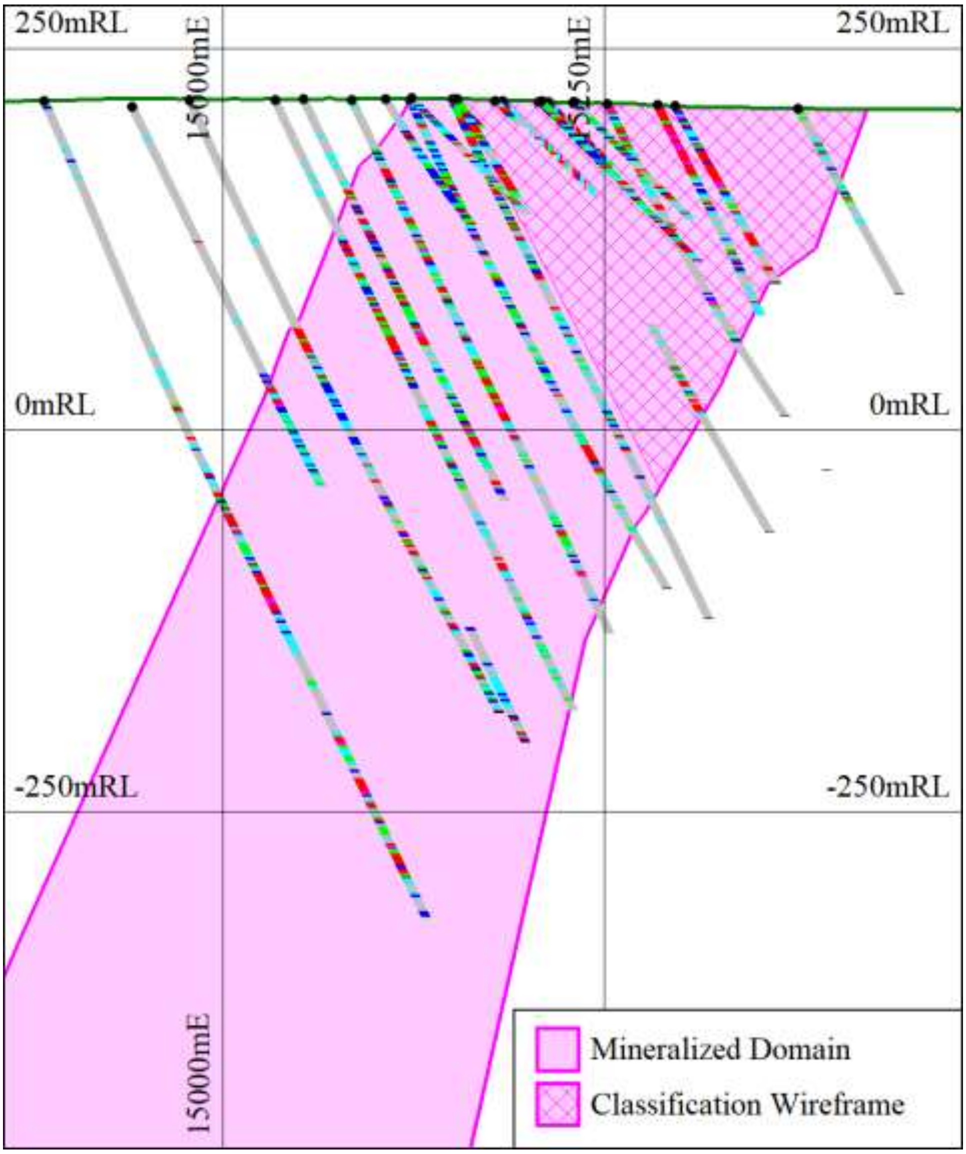


Figure 63: Classification and mineralized domain wire-frames, section 53,037.50 mN (source: MPR, 2018)

14.10 Pit shell constraint

To provide estimates with reasonable prospects for eventual economic extraction ("RPEEE") as required for Public Reporting, the Mineral Resources are reported within an optimized pit shell. The optimization parameters reflect a large scale conventional open pit operation with the cost and revenue parameters detailed in Table 71.

Table 71: Resource pit shell optimization parameters

Description		Value
Gold price		US$1,500/oz
NSR royalty		5.00%
Pit slope angle		45°
Ore and waste mining cost		US$2.88/t
Mill Recovery	Oxide	90.0%
	Transition and Fresh	86.0%

Description		Value
Mill Processing Cost	Oxide	US$11.61/t
	Transition and Fresh	US$12.08/t

14.11 Model validation

14.11.1 Comparison with informing data

Block model reviews included comparison of estimated block grades with informing composites. These checks comprised inspection of sectional plots of the model and drill data and review of swath plots. These reviews showed no significant issues.

Figure 64 is a representative cross section showing resource model panels scaled by the estimated proportion above 0.5 g/t Au colored by resource category with the average estimated gold grade at 0.5 g/t cut-off relative to resource domains and drill holes traces colored by two-metre composited gold grades.

The swath plots in Figure 65 compare average estimated panel grades for Indicated Resources and average composite grades by easting, northing and elevation. The average composite grades include an upper cut of 50 g/t Au which is consistent with the upper limit used for generating the statistics for Domain 2 and reduces the impact of a small number of outlier composite grades.

The swath plots show that although, as expected, average block grades are smoothed compared to the average composite grades they generally closely follow the trends shown by the composite mean grades, except for areas of variably spaced or limited sampling. There are minor local deviations between the model and composite trends seen on the plots.

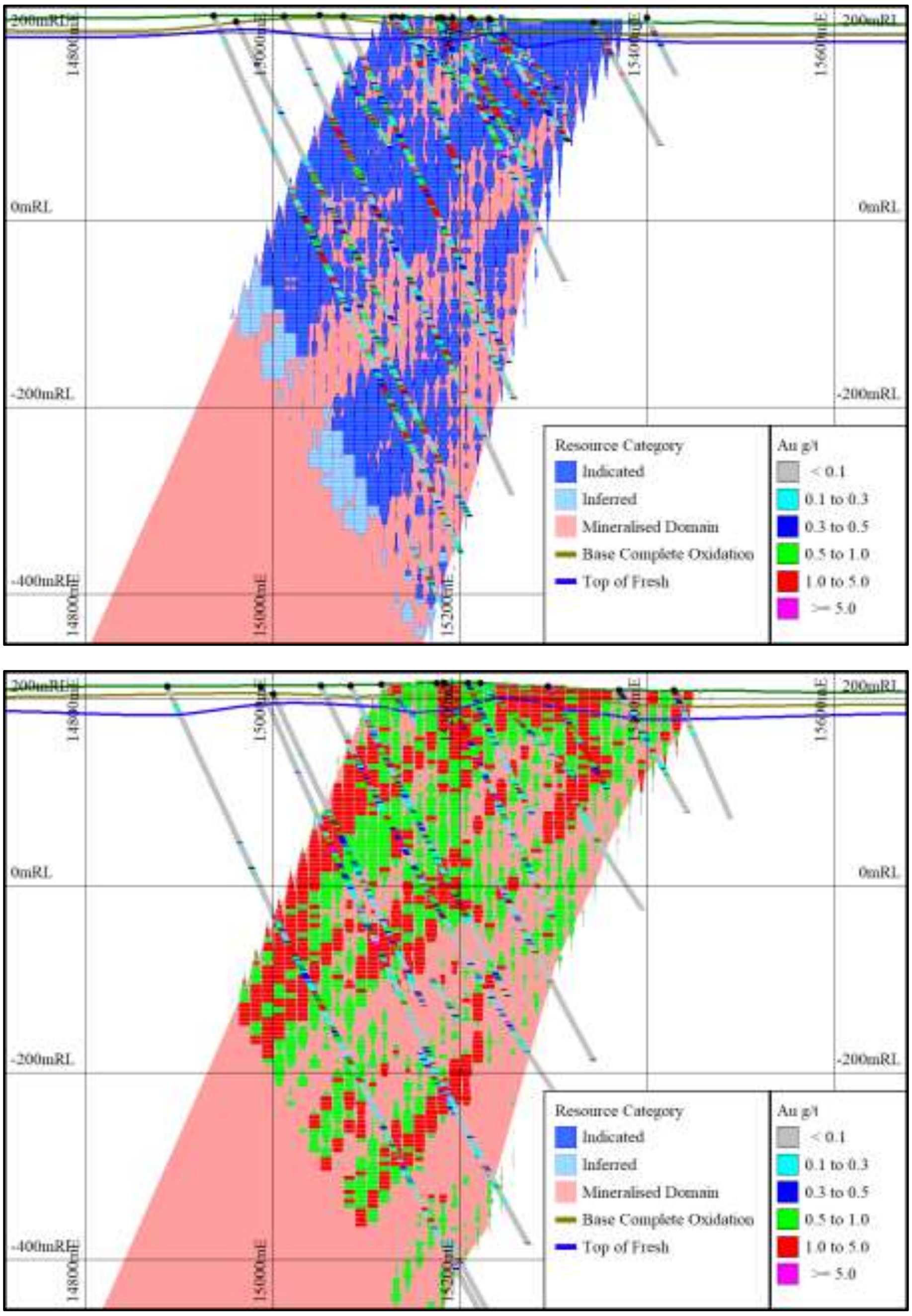


Figure 64: Block model colored by resource category (Top) and gold grade (Bottom) (source: MPR, 2018)

Gold grades at 0.5 g/t cut-off, 53,300 mN blocks scaled to 0.5 g/t Au cut-off

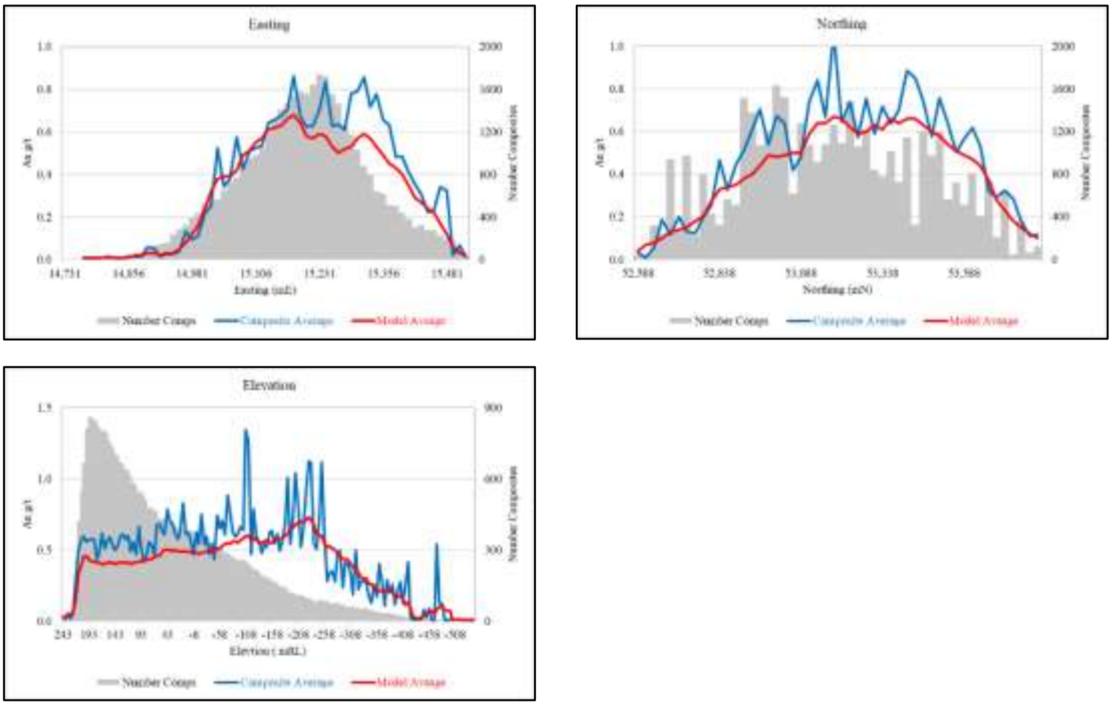


Figure 65: Swath plots showing average estimated panel grades *vs* composite grades

14.11.2 Comparison with grade control estimates

There were 317 RC grade control holes on a 10 by 15 m grid drilled to around 40 m vertical depth in the central-northern part of the deposit. These trial grade control holes were not included in the resource estimation dataset. Comparison with estimates from a Grade Control Model constructed from these data using conditional simulation provides an independent check of the resource model estimates.

Table 72 compares estimates for the Grade Control Model with the MIK Resource Model. For preparation of this table, the two models were reported within a consistent volume with densities assigned to the Grade Control Model by rock type and oxidation zone consistent with the Resource Model. All Resource Model estimates within this volume were classified as Indicated and represent around 2% of the Indicated Resource.

Table 72 demonstrates that estimates from the Grade Control Model closely match Resource Model estimates at the target cut-off grade of 0.5 g/t Au, providing an independent check on the Mineral Resource.

Table 72: Comparison of Grade Control Model and MIK Resource Model estimates

Cut-off (g/t Au)	MIK Resource Model			Grade Control Model			Difference		
	Tonnes (Mt)	Grade (g/t Au)	Gold (koz)	Tonnes (Mt)	Grade (g/t Au)	Gold (koz)	Tonnes (%)	Grade (%)	Gold (%)
0.3	4.04	1.43	186	4.12	1.41	187	2%	-1%	1%
0.5	3.47	1.59	177	3.50	1.59	179	1%	0%	1%
0.7	3.00	1.75	169	2.94	1.78	168	-2%	2%	-1%

The grade control trial validated the Mineral Resource estimate by testing the full thickness of Oxide/Transition material and 35 m (3 benches) of Fresh material.

14.12 Mineral Resource estimate

Table 73 shows the Mineral Resource estimate for Namdini at a range of cut-off grades. The figures are rounded to reflect the precision of the estimates and include rounding errors.

The Mineral Resources are reported within the resource pit shell and extend from natural surface to 570 metres depth with around 90% of the Indicated Resource and 60% of the Inferred Resource occurring at a depth of less than 400 m. The estimates make no allowance for depletion by currently active artisanal mining, which does not significantly impact the reported estimates.

Oxide and Transition material types each host around 2.5% of the Indicated Resource, with the remainder (95%) being Fresh material.

Oxide and Transition material types host around 0.6% and 0.2% respectively of the Indicated Resource, with the remainder (99%) being Fresh material.

Table 73: March 2018 Namdini Mineral Resource estimate

Cut-off (g/t Au)	Tonnes (Mt)	Grade (g/t Au)	Contained gold (Moz)
Indicated Mineral Resources			
0.2	269	0.9	7.5
0.3	240	0.9	7.2
0.4	210	1.0	6.9
0.5	**180**	**1.1**	6.5
0.6	152	1.2	6.0
0.7	128	1.3	5.5
0.8	107	1.4	5.0
0.9	90	1.6	4.5
1.0	76	1.7	4.1
Inferred Mineral Resources			
0.2	20	0.9	0.6
0.3	18	1.0	0.6
0.4	15	1.1	0.6
0.5	**13**	**1.2**	**0.5**
0.6	11	1.4	0.5
0.7	9	1.5	0.5
0.8	8	1.6	0.4
0.9	7	1.8	0.4
1.0	6	1.9	0.4

14.13 Deleterious elements – sulfur and arsenic

The number of sulfur and arsenic assays is approximately one quarter that of the gold assays. Whereas gold assays are in an extensive drilling pattern of 311 holes spaced generally 50 by 50 m, at this stage sulfur and arsenic is only available for 77 holes spaced on sections at approximately 100 to 200 m on northings. Along cross sections the sulfur and arsenic drilling is spaced at 100 m or greater.

Figure 66 shows plan projections of sulfur and arsenic assays which can be directly compared to that for gold in Figure 60.

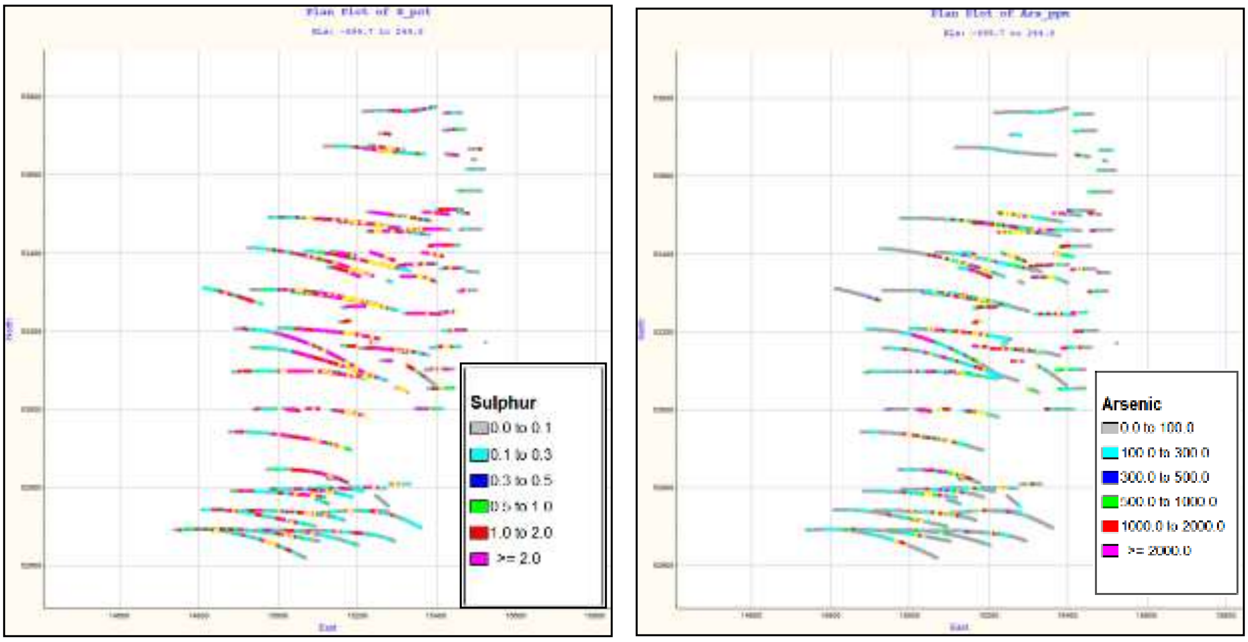


Figure 66: Plan projections of drill hole assays for sulfur and arsenic

The Quality Assurance procedures and Quality Control data supporting the sulfur and arsenic sampling were not reviewed by MPR. The preliminary S and As modelling results were not used for resource estimation.

Preliminary estimation of sulfur and arsenic grades was done by MPR using Ordinary Kriging ("OK") into blocks consistent with those used in the gold estimation. Data was composited to 2 m and variograms and kriging were done with no upper grade capping. Unadjusted block estimates of sulfur and arsenic are shown in Table 74.

Table 74: Estimates of sulfur and arsenic grades within the unclassified Mineral Resource

Cut-off (g/t Au)	Mineral Resource			
	Mt	Gold (g/t Au)	Sulfur (% S)	Arsenic (ppm As)
0.3	258	0.94	0.74	414
0.4	225	1.03	0.75	428
0.5	**193**	**1.13**	**0.76**	**440**
0.6	163	1.23	0.77	452
0.7	137	1.34	0.78	464
0.8	115	1.45	0.78	475

Examples of the OK model estimates are shown in cross section and plan in Figure 67 for sulfur and in Figure 68 for arsenic, with block grades and drill hole traces colored by composite grade. The resource pit shell (blue) used to constrain the Mineral Resource and the mineralization domain (magenta) are also shown on each Figure.

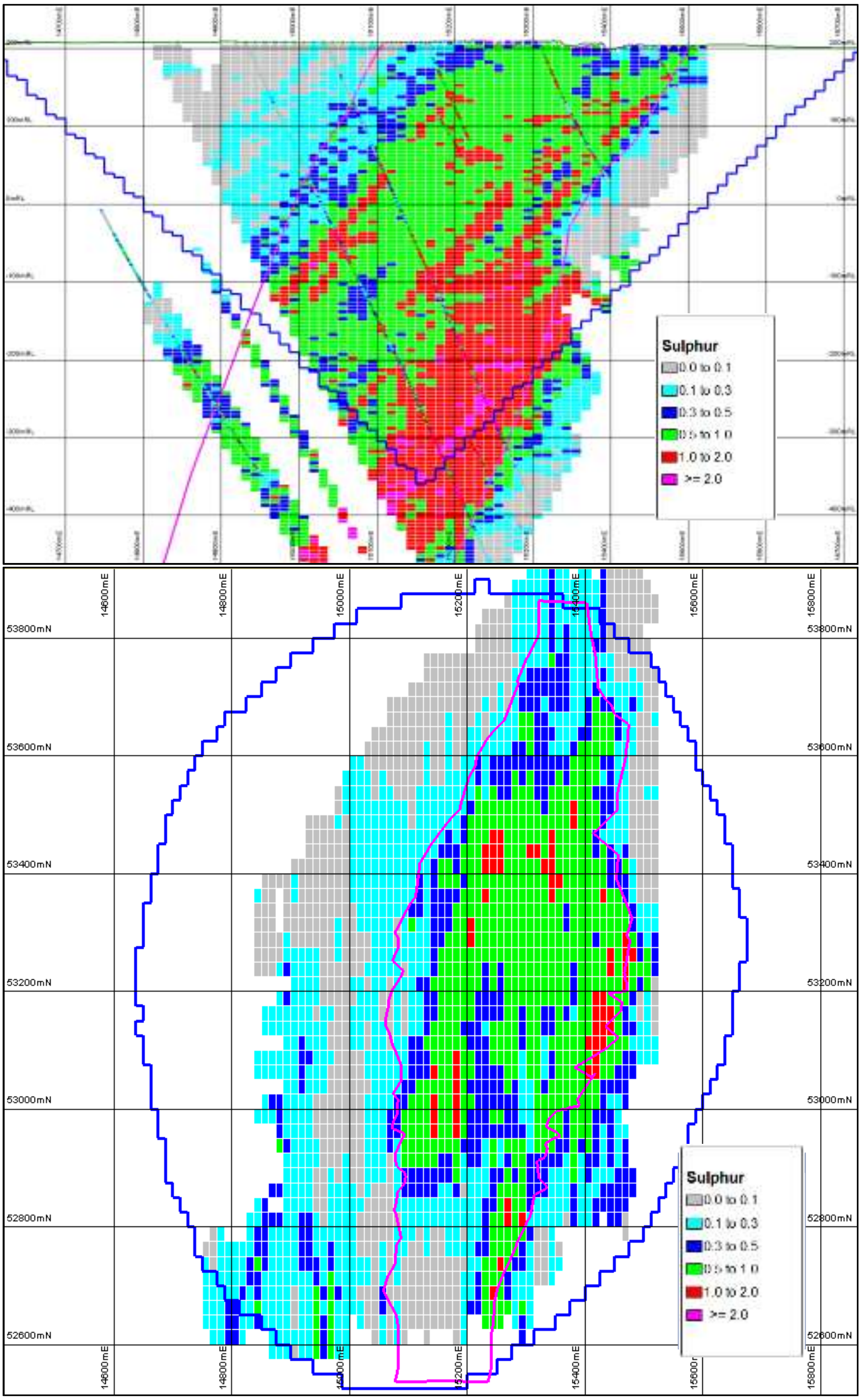


Figure 67: Sulfur estimates (Top: cross section 53,300 mN, Bottom: plan at 150 m RL)

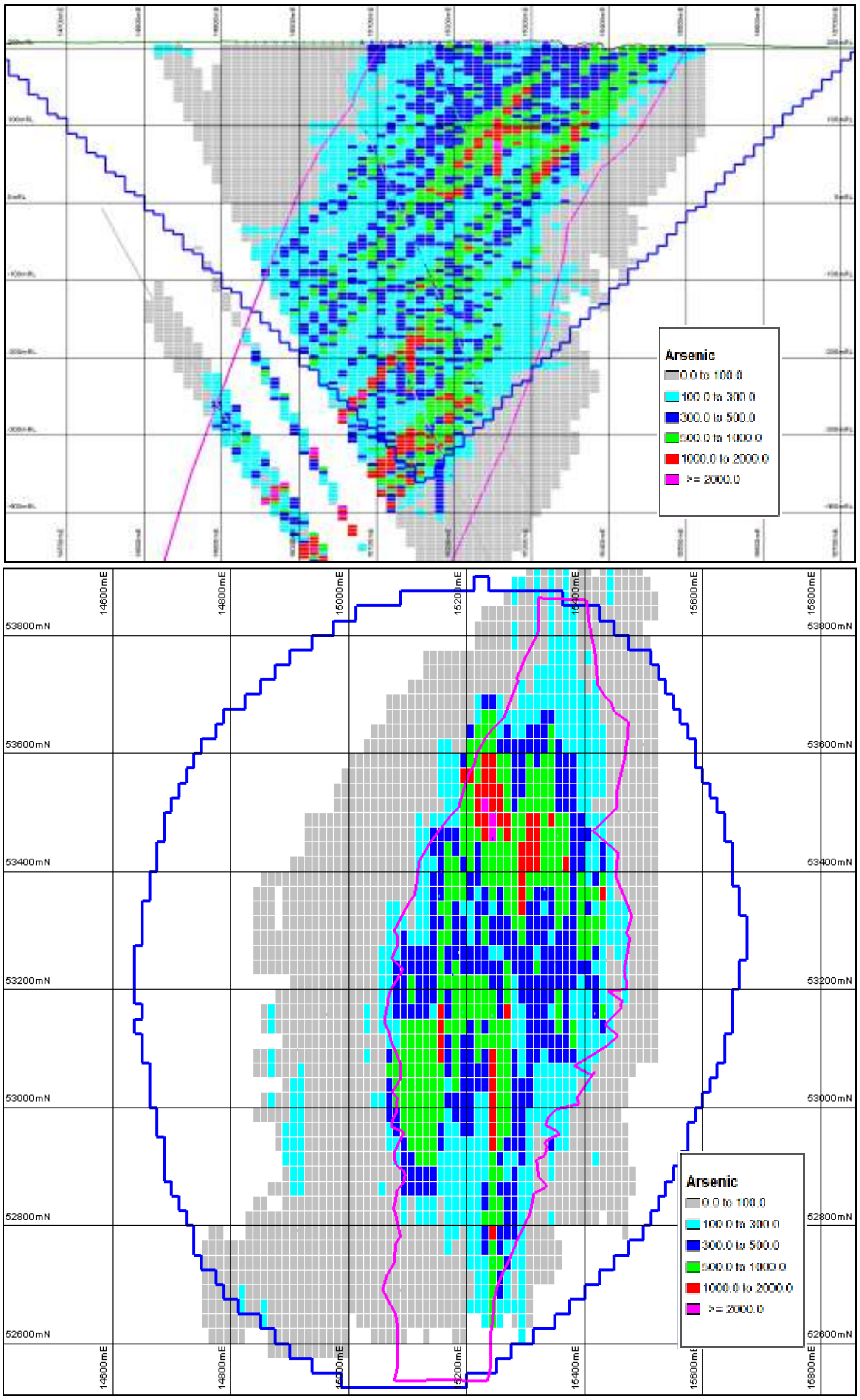


Figure 68: Arsenic estimates (Top: cross section 53,300 mN, Bottom: plan at 150 m RL)

Sensitivity of the mineable part of the Mineral Resource to local variations in the S and As should be considered at the Feasibility Study stage, by more formal modelling of this data and reporting of the results. A more accurate model with S and As grades will also be useful for understanding how these elements may present in the waste dumps and how to best manage this to prevent acid formation.

14.14 Review of the Mineral Resource by Golder

Prior to carrying out the mine planning studies leading to the Ore Reserve for this PFS, Golder reviewed the Mineral Resource model developed by MPR (2018). This Section details Golder's review of the changes to the Mineral Resource based on the additional drilling and re-modelling.

14.14.1 Data supplied

All files and data for this review were provided by Cardinal. Initially the review was based on an export of the updated model (*Full_Model_MIK_Feb_2018_Cleaned_no_Negatives.csv*), mineralized zone solid (*Min_Zone_Feb2018_topo.dxf*) and drill hole database (DH*.xlsx) files supplied in February 2018. The Mineral Resource Estimation technical report and the memo on estimation of sulfur and arsenic were subsequently supplied in July 2018. Golder's review was updated based on this additional information.

There were an additional 36 holes (an additional 15,747 m of drilling) used to update the Mineral Resource estimate. The majority of new holes were collared to the west (Figure 69) of the deposit and targeted the deeper mineralization that was in the Inferred Resource category in the previous model.

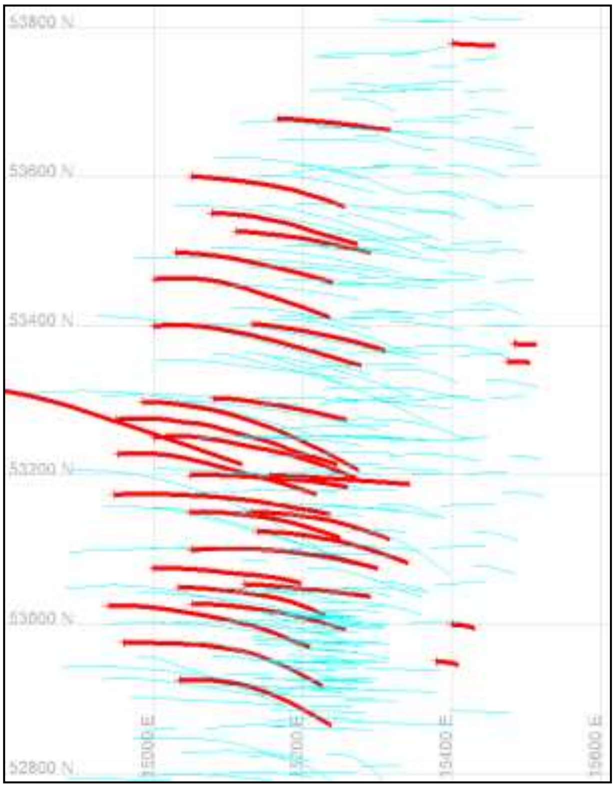


Figure 69: Drill hole location plan showing 2018 holes in red

14.14.2 Drill hole data

A total of 311 holes for 82,870 m of drilling was used in the Mineral Resource update reported by MPR (2018). Golder agrees with MPR's assessment that the drilling, sampling and assaying is of high quality and supports the Mineral Resource estimate.

Key factors underpinning the confidence in the data are:

- Accurate collar location survey backed up by comprehensive downhole survey of 79% of the mineralized domain dataset
- Three quarters of the drilling was conducted by diamond coring
- Detailed assessment of core and RC recovery. Generally, recovery and sample quality were good. An issue was identified for one drilling company where lower RC sample recovery was identified on the metre following a rod change. It was determined that this did not affect gold grade
- Good repeatability of RC field duplicate samples
- Certified Reference Material (standards) confirmed there was no bias in analysis at both SGS Ouagadougou and SGS Tarkwa laboratories
- Interlaboratory duplicate assays suggested a positive bias (higher values obtained in the umpire laboratory results). The accuracy of the results from the Intertek umpire laboratory was questioned and determined to be inconclusive as many batches did not contain reference standards and appeared to have some sample misallocations
- 214 quarter core duplicate samples were submitted to SGS Tarkwa and ALS Ireland. Significant scatter was observed suggesting short-scale grade variability. Only one small batch (35 samples) showed a bias between primary and duplicate results.
- Density was determined by immersion for 8,181 samples by Cardinal and SGS Tarkwa.

14.14.3 Grade estimation

Golder completed a thorough evaluation of the grade estimation applied in the previous (2017) model. Global statistical and swath plot comparisons between the drill hole composites and grade estimates for the previous model indicated the conformance between the drilling and the grade estimates was reasonable. Some changes in the domaining to improve local estimations were suggested in the previous review.

The majority of samples (99.5%) were less than 2 m in length and support the use of a 2 m composite for statistical analysis and grade estimation. Golder accepted that the 2 m composite is appropriate for the sample length and the estimation process applied.

No changes were made to the domaining applied to the 2018 model. The probability plot (Figure 70) of mineralized composites shows there is a significant difference in the grade distribution between composites of different lithology types. In the review of the 2017 model, Golder recommended that the impact of the use of these lithology types to control the estimation be evaluated. Contact analysis, as described below, highlights that the contact between the lithologies shows a gradational change and the impact of using separate lithology domains may be reduced due to the soft nature of the grade change over these boundaries.

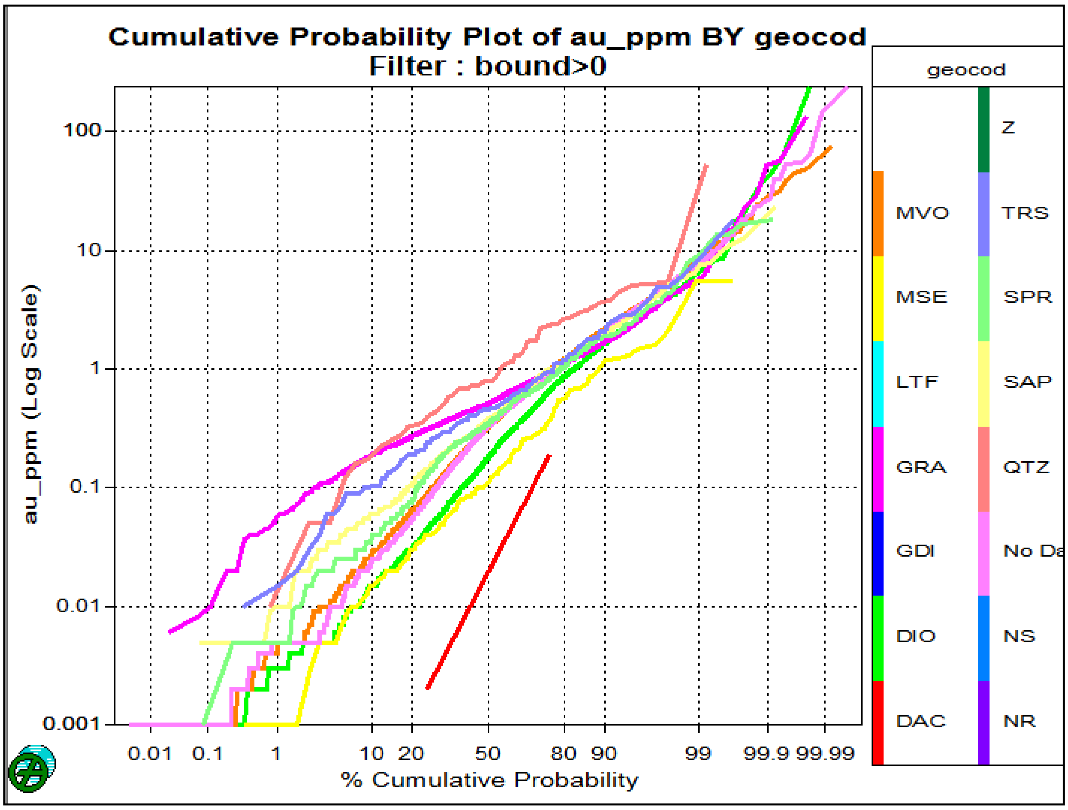


Figure 70: Probability plot of composite gold grade by logged lithology

Golder conducted contact analysis on the composites to test the grade contrast between composites on either side of the mineralized domain boundary. The contact plot for the mineralized domain, as shown in Figure 71, shows a marked contrast between the composites defined by the mineralized domain confirming this should be treated as a hard boundary in the estimation, as was done by MPR.

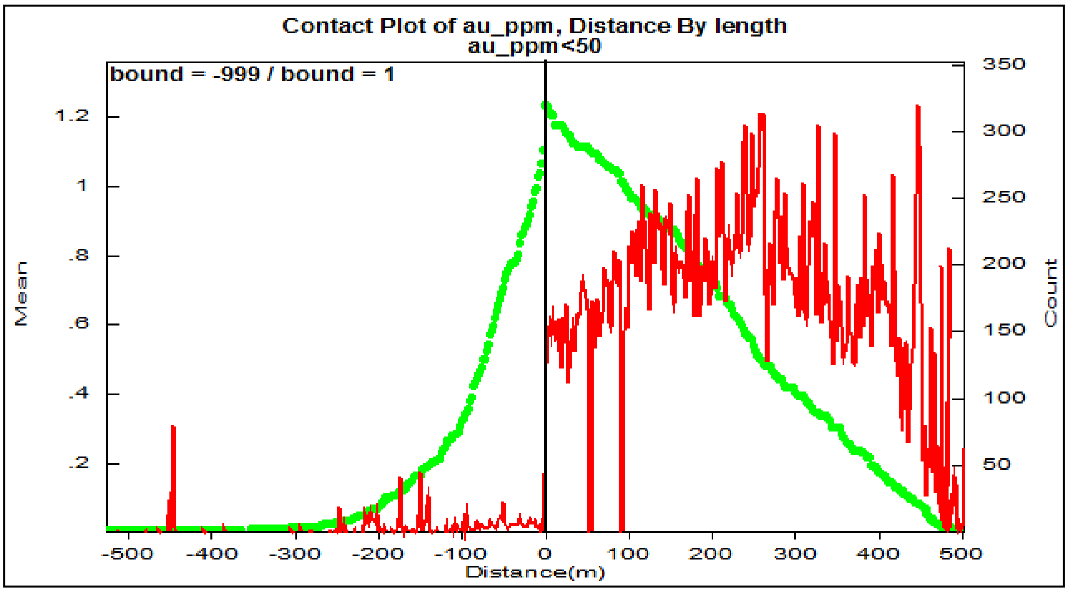


Figure 71: Contact analysis plot for the mineralized domain contact

Contact analysis between the various lithology types within the mineralized domain generally shows an erratic and gradational change over the lithology contacts. An example is the GRA to DIO contact shown in Figure 72. This suggest these contacts should be treated as soft boundaries in the grade estimation. However, these contacts were not used to control the grade estimation in the 2018 model.

Golder recommended that the impact of using separate lithology domains to control the grade estimation should be considered for future estimations.

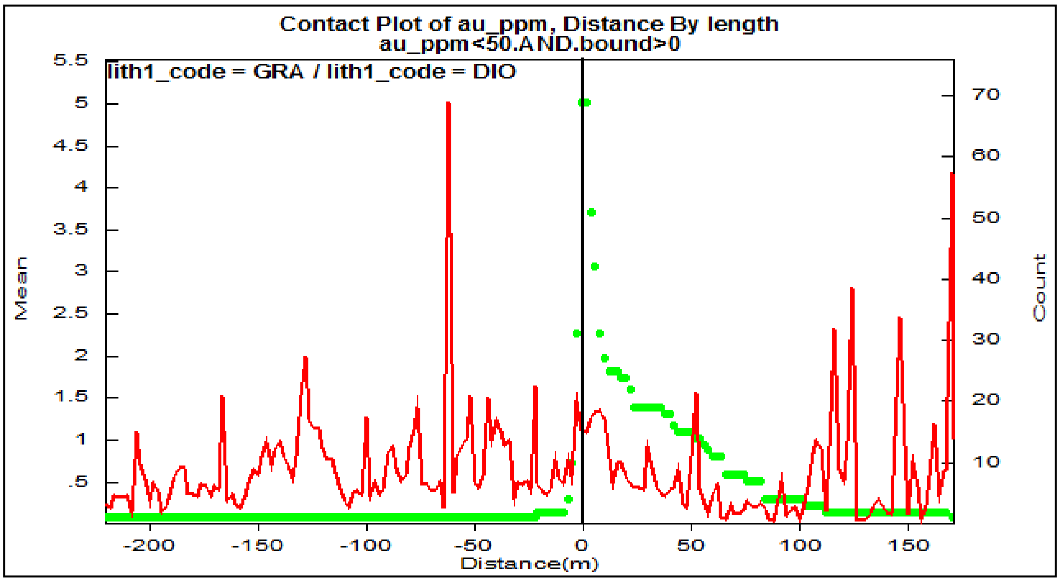


Figure 72: Contact analysis plot for GRA to DIO lithology contact

The 2018 model has used the same MIK estimation process as applied in the previous estimate. There were 14 grade thresholds used to model indicator variograms for each oxidation zone for the Mineralized Domain. Bin mean grades were used for all but the top bin. To reduce the influence of the small number of very high-grade composites in the top bin, the median value was used for all but the Fresh domain where the mean was applied after excluding composites >50 g/t Au. Golder accepts this is a reasonable approach to managing these high-grade outliers.

MIK was applied to a panel size of 12.5 by 25 by 5 m, which is a reasonable size for the closer drilled parts of the deposit. The estimation was conducted using three passes of varying search criteria. The initial pass used a search ellipse radius of 65 by 65 by 15 m with a limit on composites selected to a minimum of 16, maximum of 48 and minimum of 4 octants. The second pass used an ellipse of 97.5 by 97.5 by 22.5 m and the same composite limits. The third pass used the same ellipse as the second pass but reduced the minimum number of composites to 8 and octants to 2. Golder did not assess the kriging neighbourhood. The search criteria are reasonable for the drill spacing, composite size and style of mineralization.

14.14.4 Change from previous model

Globally the additional drilling and modified Mineralized Domain has resulted in a small increase in the global tonnage (reported at zero cut-off grade) at slightly lower grade (Table 75). When a cut-off of 0.5 g/t Au is applied to the etype (panel average) gold grade both the tonnage and grade are slightly lower in the 2018 model.

Table 75: Global change in Mineralized Domain (no cut-off grade)

	Tonnage (T)	Volume (m^3)	Grade (g/t Au)	Density (t/m^3)
2017	326,501,094	116,885,938	0.796	2.793
2018	329,242,359	118,653,125	0.763	2.775
Change	0.84%	1.51%	-4.15%	-0.64%

Notes: Restricted to the RPEEE pit shell

Table 76: Change in Mineralized Domain above 0.5 g/t Au for Panel (etype) estimate

	Tonnage (T)	Volume (m^3)	Grade (g/t Au)	Density (t/m^3)
2017	211,975,156	75,929,688	1.07	2.792
2018	206,392,375	74,565,625	1.047	2.768
Change	-2.63%	-1.80%	-2.15%	-0.86%

Golder compared the 2018 and 2017 models on a block by block basis to assess the local variation in the models. As expected, the greatest changes (Figure 73) are observed in the vicinity of the new drilling. Away from the new drilling there is still some difference between the models suggesting there has been a minor change in estimation parameters that influences the local estimated grades.

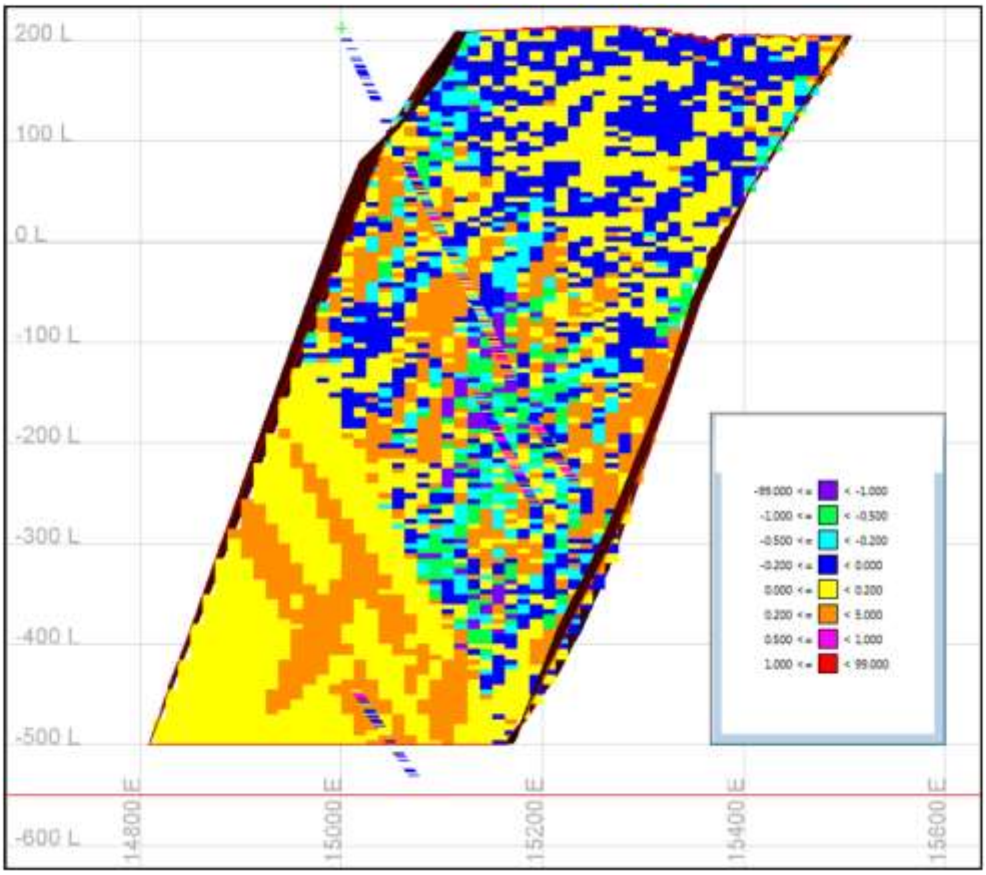


Figure 73: Section 53250N showing change in etype Au (g/t) estimate (2018 model minus 2017 model)

14.14.5 Model validation and recoverable resources

MPR validated the model using visual examination comparing cross sections of the block estimates and drill hole data, and by preparing swath plots over a range of eastings, northings and elevations. There appears to be a reasonable correspondence between the block grades and the input drilling data.

MPR (2018) states that the recoverable resources at an SMU of 5 by 10 by 2.5 m were determined by using post-processing of the MIK Model using the Direct Lognormal Correction method. The total adjustment applied was 0.059. This adjustment factor is a very low value given the small SMU considered and produces only a small difference from the panel results which is evidenced by the slight change between the panel etype grade and the recoverable resource grade determined at a 0.5 g/t Au cut-off grade. Table 77 shows the 'recoverable resource' based on the selected SMU adjustment is 7% lower in tonnage but at a 7.5% higher grade than the global Panel value at the 0.5 g/t Au cut-off grade. Overall both estimates have the same contained gold metal.

Table 77: Comparison of Panel estimate to SMU estimate

	Tonnage (Mt)	Grade (g/t Au)	Contained Gold (t)
SMU	191.856	1.126	216.0296
Panel	206.392	1.047	216.0928
Difference	**-7.043%**	**7.545%**	**-0.029%**

Notes: Some rounding, but apparent precision is retained for this comparison

There is an area of close spaced grade control drilling to a vertical depth of 40 m at a spacing of 10 by 15 m. MPR compared the resource estimate in this area to a conditional simulation Grade Control Model assumed to represent similar mining selectivity to the SMU adjustment applied to the MIK Model.

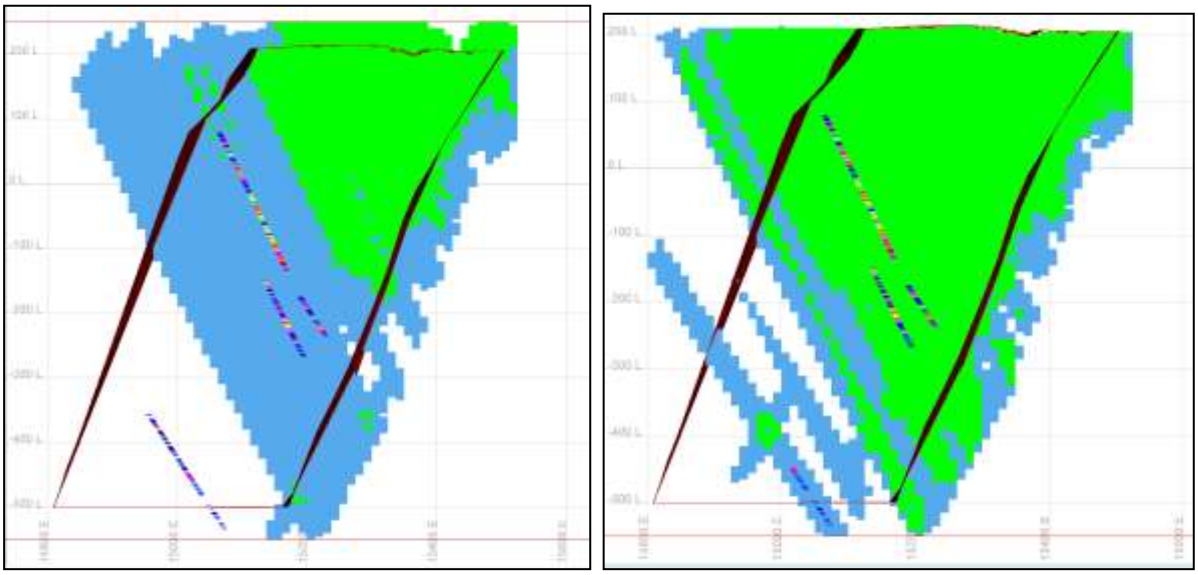
At the 0.5 g/t Au cut-off grade there is little difference (<1%) between the Resource Model and the Grade Control Model. This demonstrates that the resource model is giving a reasonable prediction of the recoverable resource for this area. However, this area represents only about 2% of the total resource and may not be representative of the mineralization at depth.

While Golder considers that there was relatively little value in the MIK variance adjustment approach as implemented to define the recoverable resource agrees that the Grade Control Model does validate the MIK based Mineral Resource Model.

14.14.6 Resource classification

The most significant impact of the additional drilling has been on the classification of the Mineral Resource at depth. The additional drilling at depth has converted a large proportion of the Inferred Resource to Indicated Resource. An example of this is shown in Section 53250N (Figure 74), where much of the Resource in the vicinity of new drilling changed from Inferred to Indicated.

In the current Mineral Resource there are isolated thin zones of Indicated Resource supported by individual drill holes. Golder recommends that a refinement of the resource classification would remove such isolated areas that do not demonstrate continuity and reassign these to Inferred.



(Left: 2017, Right: 2018; Green = Indicated; Blue = Inferred)

Figure 74: Section 53250N with Mineral Resource classification and additional drilling

14.14.7 Mineral Resource

Additional drilling has increased the confidence in the local geology and the grade estimation of the deeper part of the deposit. This has resulted in conversion of previous Inferred Resource material to Indicated Resource as shown in Table 78. Comparing the 2017 and 2018 MIK estimates at the 0.5 g/t Au cut-off grade there is a reduction in tonnage of 12 Mt at a similar grade, resulting in a reduction in the contained metal of 0.41 Moz.

Table 78: Mineral Resource changes at the 0.5 g/t Au cut-off for MIK estimates

Model	Classification	Tonnes (Mt)	Grade (Au g/t)	Metal (Au Moz)
2017	Indicated	120	1.10	4.27
2017	Inferred	84	1.15	3.12

Model	Classification	Tonnes (Mt)	Grade (Au g/t)	Metal (Au Moz)
2018	Indicated	180	1.12	6.46
2018	Inferred	13	1.25	0.52
Change	Indicated	59	0.01	2.18
Change	Inferred	-71	0.10	-2.60

Notes: Some rounding, but apparent precision is retained for this comparison

Restricted to RPEEE pit shell.

14.14.8 Sulfur and arsenic estimation

Global sulfur and arsenic grade estimations were discussed in Section 14.13. There is a significant difference in the distribution of the sulfur and arsenic grades between the mineralized (Domain 2) and non-mineralized (Domain -999) in the probability plots shown in Figure 75.

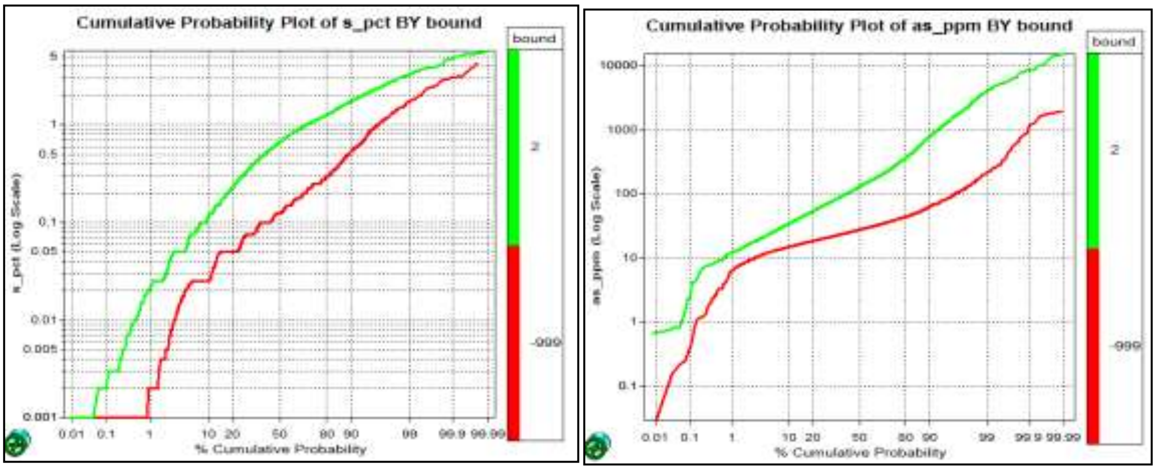


Figure 75: Probability plots of sulfur and arsenic by mineralized domain

MPR estimated grades for both arsenic and sulfur using OK from 2 m composites. Golder agrees with MPR that the sulfur and arsenic estimations should be considered global in nature. Due to the wide sample spacing and lack of geological domaining the individual block grades are likely to provide a poor estimate of the local sulfur and arsenic grades.

14.14.9 Validation of the Mineral Resource estimate

Golder considers that the updated Mineral Resource model has improved the confidence in the local grade estimate for the deeper section of the deposit, which is reflected in the revised resource classification at higher confidence.

The additional drilling and revised estimation parameters have reduced the tonnage and metal contained in the resource but have increased the confidence in the resource estimation. The 2018 resource estimate is supportable as an appropriate input to mining studies and is a reasonable development from the 2017 resource estimate.

14.15 Continuity, MIK estimation method and dilution

14.15.1 Introduction

MPR estimated the Mineral Resource at Namdini using a non-linear geostatistical method called Multiple Indicator Kriging (MIK). This method is well suited to the Namdini orebody where the mineralisation is disseminated, fine-grained, confined within definable rock types by a grade boundary or envelope, and where large-scale mining is anticipated. This estimation technique is considered appropriate for this style of mineralisation.

14.15.2 Continuity of mineralisation

14.15.2.1 Geological continuity

The mineralisation at Namdini shows good continuity at low cut-off grades, extending 1,150 m along strike, with up to 300 m width and 700 m depth. Using a cut-off grade of 0.5 g/t Au has established expected mining widths of up to 100 m. Demonstration that mining of these will be economically viable by this PFS has resulted in the establishment of the Probable Ore Reserve.

The mineralisation continuity was interpreted manually using geology and drill hole assays from 50 by 50 m resource drilling, guided by wireframed trends for lithology and grade established using Leapfrog software.

The expected continuity is illustrated at the resource modelling stage by Figure 76 and at the grade control modelling stage by Figure 77. The resource block model used 50 × 50 m spaced drill holes and 12.5 × 25 × 5 m blocks. The grade control model interpolated used 15 × 10 m spaced drill holes and 5 × 8 × 2.5 m blocks.

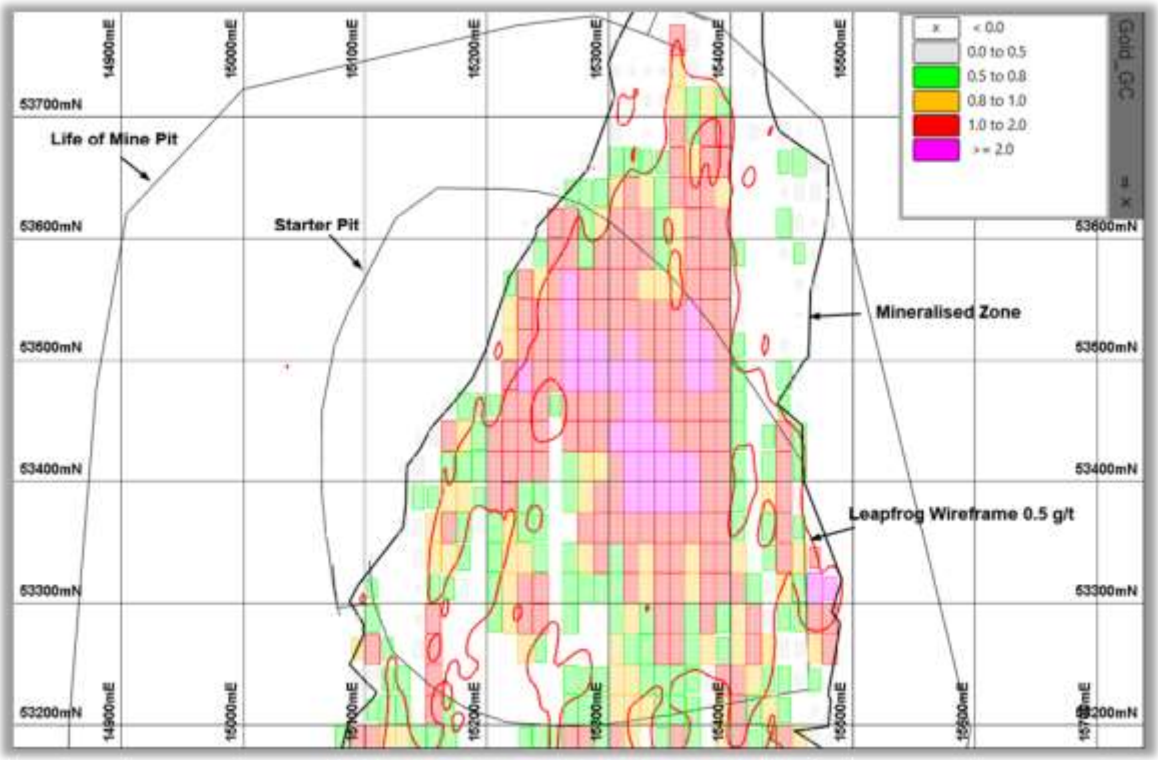


Figure 76: Resource Model at 175 m RL

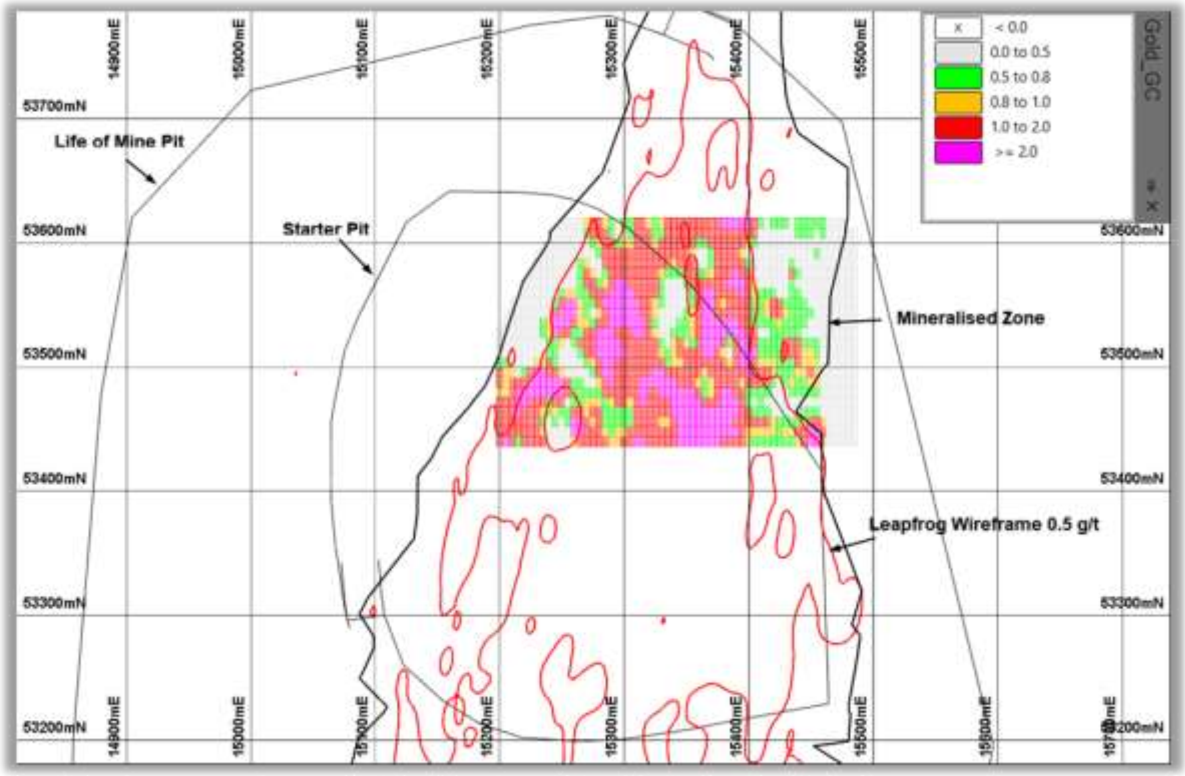


Figure 77: Grade Control Model at 175 m RL

14.15.2.2 Geostatistical assessment of grade continuity

Indicator variograms were generated and the directions of continuity were examined using variogram maps such as Figure 78. These clearly demonstrate the mineralisation continuity of the Namdini deposit.

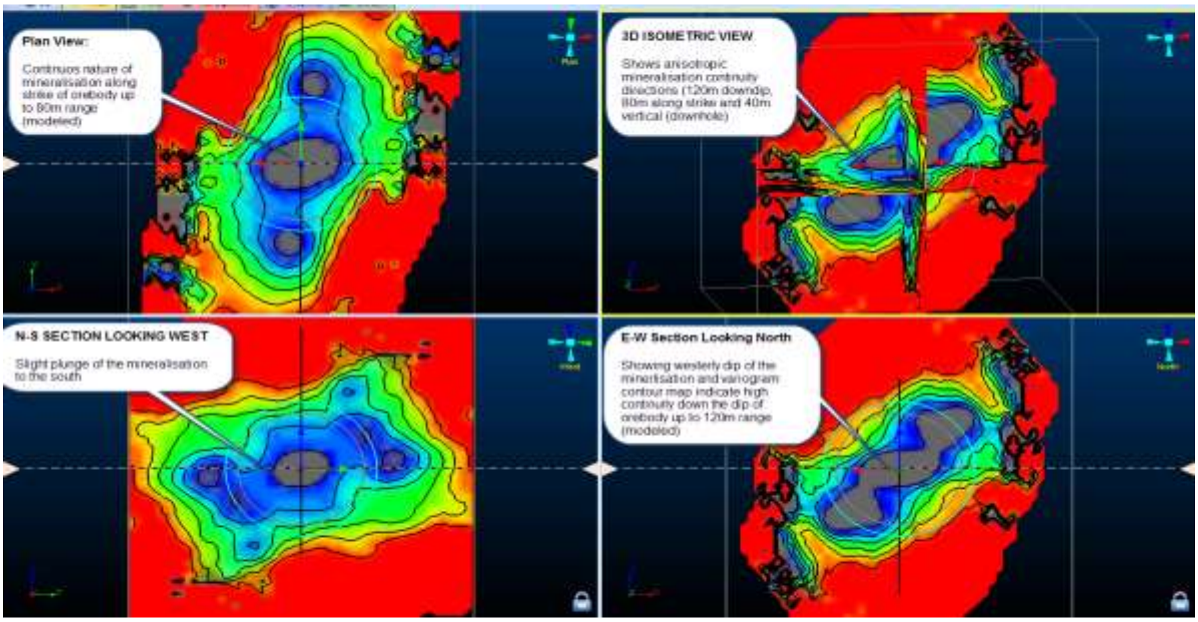


Figure 78: Variogram contour maps of gold grades

The mineralisation continuity can be quantified using the ranges of the modelled variograms. These show the distance at which grade relationships between samples become random. The variogram down the dip of the orebody shows maximum continuity with a short-range structure at 25 m and a long-range structure of 120 m. The downhole variogram has a short-range of 20 m and a long-range structure 40 m (Figure 79 and Table 79). Perpendicular to these, the deposit shows good continuity along strike with a short-range structure of 25 m and long-range structure 80 m.

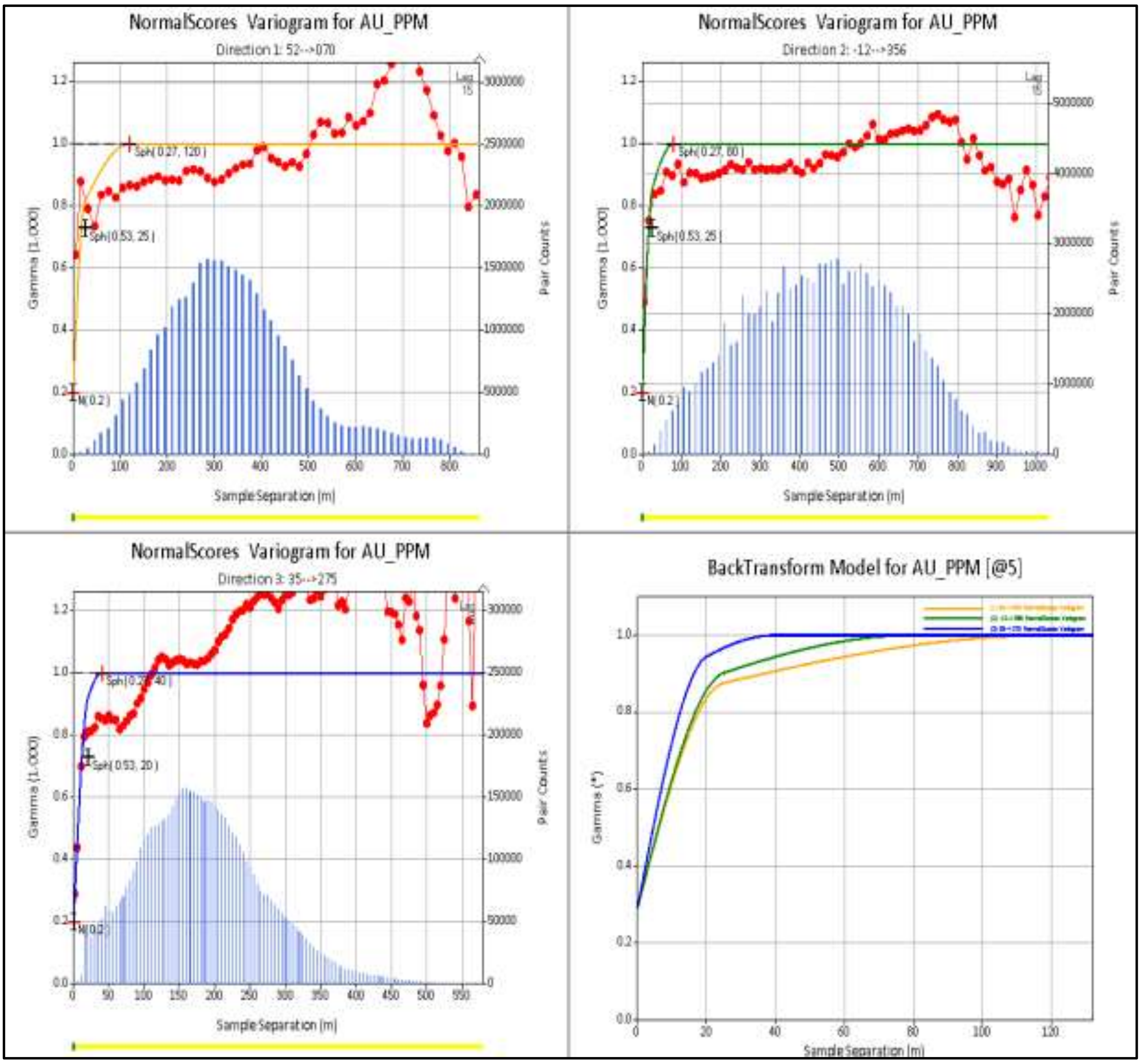


Figure 79: Variogram contour maps of gold grades

Table 79: Variogram model and continuity parameters

Structure	Variance	Along Strike	Down Dip	Orthogonal
Nugget	29%			
Spherical 1	53%	25 m	25 m	20 m
Spherical 2	18%	120 m	80 m	40 m

The grades are correlated for up to 120 m downdip, 80 m along strike and 40 m downhole within the mineralisation corridor. These are characteristics of an orebody with good continuity.

14.15.3 The MIK estimation method

14.15.3.1 Indicator Kriging

For each defined Domain and weathering combination, indicator thresholds were defined using a consistent set of percentiles representing probability thresholds of 0.1, 0.2, 0.3, 0.4, 0.5,0.6, 0.7, 0.75, 0.8, 0.85, 0.9, 0.95, 0.97 and 0.99 for data in each sample data subset. All class average grades were determined from bin mean grades with the exception of the upper grade bins, which were reviewed on a case by case basis and bin grades selected on the basis of bin mean, or median, with or without exclusion of high-grade composites.

This estimates into each block the expected proportion of that block that reports to a series of cut-off grades. For example, any block can be queried for the expected proportion (equivalent to tonnage) of ore above 0.5 g/t Au, the average grade of that proportion, and the resulting proportion and grade of waste in the block.

14.15.3.2 Variance adjustments applied to the Namdini Pit model

In estimating the Namdini recoverable gold resources, variance adjustments were applied using the Direct Lognormal Correction method as noted in Section 14.7 and shown in Table 69. The variance adjustments applied to the Namdini Recoverable Resource model are minimal and appropriate.

14.15.3.3 Correcting a distribution of samples to a distribution of blocks

The relationship between samples and mineable sized (SMU) blocks may be understood as follows:

- The average grade of sample-sized units and blocks within the panel is the same and is equal to the estimated average grade of the panel.
- The variance, or spread, of the block grades within the panel is less than the variance of grades of sample sized units within the panel and the change of variance from sample-sized units to blocks can be calculated from the variogram. This is based on Krige's Relationship (the volume-variance effect) which defines the relationship of the variance of samples to variance of blocks.
- The approximate shape of the histogram of block grades can be reasonably predicted by some appropriate assumptions that apply during the Change Of Support. This results in the choosing to use one of a number of variance adjustment methods, depending on the expected relationship between the sample distribution shape and the block distribution shape.
  - For deposits close to a normal distribution the overall shape of the distribution can be preserved even though the variance of the distribution changes. In going from sample to blocks the distribution becomes more peaked. A simplistic variance adjustment method such as the Affine Correction is appropriate. This is not the case for highly skewed distributions such as in gold deposits.
  - For deposits that have a lognormal distribution, again the overall shape of the distribution can be preserved by using the Direct Lognormal Correction method.
  - For deposits that have a variance adjustment ratio of less than 0.3, it is reasonable to assume there is a high degree of symmetrisation in the block grade histogram, i.e. that progressive reduction in the variance moves the shape of the distribution towards normality, in line with the Central Limit Theorem of statistics. This is the basis for MPR's implementation of the Direct Lognormal Correction in their MIK estimation method.

The Direct Lognormal Correction method works as follows:

- If the histogram of sample-size is positively skewed, the resultant histogram of block grades is assumed to be lognormal in shape and the appropriate variance is given by the variance ratio.

- If the histogram of sample-size unit is approximately symmetrical or negatively skewed, the resultant block grade histogram is assumed to be normal in shape with the appropriate variance given by the variance ratio.
- This is a local support correction, applied to the histogram of sample-sized units on a panel-by-panel basis.

14.15.3.4 The Information Effect

In many gold deposits the variance adjustment is not the only adjustment required because the short-scale variation in gold grades is extreme, as is the case at Namdini. This variance adjustment provides an estimate of the variance of true block grades of the SMUs under the assumption that grade control selection will operate with knowledge of the true block grades. While this assumption is never absolutely true, it can be a reasonable assumption in some deposits where the short scale variability is small and the grade control sampling density is high.

In many deposits, however, an additional variance adjustment must be undertaken to account for the "Information Effect". This quantifies the amount and the effect of the misclassification on the recoverable reserves to obtain a more realistic recoverable reserve estimate. The dispersion (variability) of the block grades is usually greater during mining than predicted by a kriged resource model. This is because the resource model has been estimated from more sparse drill sampling information. At the grade control (mining) stage more information is available from RC or blast holes on a closer drill spacing.

It is possible to estimate the dispersion of grades at the mining stage because the kriging variance of the block estimates only depends on the sample spacing. Knowing this future production sample spacing, MPR's implementation has incorporated the Information Effect in the estimate of the recoverable resources to more realistically differentiate between ore and waste at the SMU scale.

14.15.4 Dilution

MPR's MIK recoverable resource estimation implementation incorporates dilution automatically. The principal causes of dilution at the estimation stage are:

- SMU size
- Sample spacing
- Small scale continuity
- The position of the cut-off grade relative to the mean.

Dilution is a consequence of changing the support volume, e.g. from samples to blocks. As the volume increases, the selectivity decreases, there is a corresponding increase in the tonnage mined with decrease in the average grade and the variance decreases.

The Namdini Recoverable Mineral Resource estimates can be reasonably expected to provide appropriately reliable estimates of potential mining outcomes at the assumed selectivity, without application of additional mining dilution or mining recovery in the mine plan schedules.

Expected mining dilution is incorporated into the block model grade estimation. The resource model block size with its series of internal selectivity cut-offs, is considered an appropriate size and no further adjustment to the recoverable resource model is required. The resource block model is considered suitable for engineering purposes such as Whittle Optimization, pit designs and optimised mining schedules.

14.15.5 Grade control drilling

Cardinal has undertaken a 13,000 m close spaced trial grade control drilling program and compared the results with the MIK resource estimate. Section 14.11.2 demonstrates that the resource and grade control models compare extremely well.

The trial grade control drilling showed that with RC drilling on a 10 × 15 m pattern, with 1 m down hole sampling, large areas above 0.8 g/t Au can be delineated. Selective mining of such zones will be defined where possible and the ore above this cut-off grade may be selectively prioritised to the crusher with 0.5 g/t to 0.8 g/t material stockpiled.

It is envisaged that grade control drilling will test two to three benches in advance of the mining.

14.16 Summary of the Mineral Resource estimate

Gold mineralization occurs in altered meta-volcano sedimentary rocks and tonalite. The mineralized domain used for the current estimates trends NNE over approximately 1,330 m of strike and dips to the west at around 70°. It has an average width of approximately 240 m and extends to 710 m depth, around 25 m below the base of drilling.

The Mineral Resource estimates are based on RC and diamond information available for Namdini on 15 February 2018, totalling 311 holes for 82,870 metres of drilling. RC and diamond drilling provide around one quarter and three quarters of the estimation dataset respectively. The available drilling information includes an additional 15,747 metres relative to the dataset available for previous mineral Resource Estimates reported in September 2017 (MPR, 2017 and Golder, 2018).

Information available to demonstrate the reliability of field sampling for the resource drilling includes core recoveries, recovered RC sample weights, RC sample condition logs and RC field duplicates. These data have established that the RC and diamond sampling is representative and free of any biases or other factors that may materially impact the reliability of the sampling.

All sample preparation and primary gold analyses of samples from the resource drilling were undertaken by independent commercial laboratories. Primary samples were submitted to SGS Ouagadougou or SGS Tarkwa for analysis for gold by fire assay.

Information available to demonstrate reliability of sample preparation and analysis includes assays for blanks, reference standards, and inter-laboratory repeats. These data have established that the assaying is representative and free of any biases or other factors that may materially impact the reliability of the analytical results. The sample preparation, security and analytical procedures adopted for the Namdini drilling provide an adequate basis for the current Mineral Resource estimates.

Mineral Resources were estimated using the Multiple Indicator Kriging ("MIK") method based on 2 m down-hole composited gold grades from RC and diamond drilling. Estimated resources included a variance adjustment to give estimates of 'recoverable resources' for selective mining unit ("SMU") dimensions of 5 mE by 10 mN by 2.5 m in elevation, assuming there would be high-quality grade control sampling on an 8 by 12 by 1.25 m pattern. The Mineral Resource is reported within an optimal pit shell generated at a gold price of US$1,500/oz.

Resource model blocks were classified as Indicated or Inferred on the basis of search pass and a wire-frame defining the more closely drilled parts of the deposit. The Mineral Resource model was independently validated by Golder Associates Pty Ltd as being based on appropriate drill sampling, assaying and estimation methodology.

Table 80 shows Namdini Mineral Resource estimates for selected cut-off grades at 5 March 2018 (Cardinal, 2018a; MPR, 2018). The figures in this table are rounded to reflect the precision of the estimates and include rounding errors. The Mineral Resource estimates are classified and reported in accordance with NI 43-101 guidelines and classifications adopted by CIM (2014).

Table 80: March 2018 Namdini Mineral Resource Estimates for selected cut-off grades

Cut-off (Au g/t)	Tonnes (Mt)	Grade (Au g/t)	Metal (Au Moz)
Namdini Indicated Mineral Resource Estimates			
0.3	240	0.9	7.2
0.4	210	1.0	6.9
0.5	**180**	**1.1**	**6.5**
0.6	152	1.2	6.0
0.7	128	1.3	5.5
0.8	107	1.4	5.0
0.3	240	0.9	7.2
Namdini Inferred Mineral Resource Estimates			
0.3	18	1.0	0.6
0.4	15	1.1	0.6
0.5	**13**	**1.2**	**0.5**
0.6	11	1.4	0.5
0.7	9	1.5	0.5
0.8	8	1.6	0.4

Notes: Rounding may affect totals

Restricted to RPEEE pit shell.

For Public Reporting (Cardinal, 2018a; MPR, 2018) the estimated Mineral Resources at a cut-off grade of 0.5 g/t Au, using an appropriate level of precision, are:

Indicated Resources 180 Mt at 1.1 g/t for 6.5 Moz of contained gold.

Inferred Resources 13 Mt at 1.2 g/t for 0.5 Moz of contained gold.

The current data set of assays for sulfur and arsenic is insufficient for Mineral Resource estimation however it is sufficient to provide preliminary global estimates. At the 0.5 g/t Au cut-off grade the current Mineral Resource has grades of 0.76% S and 440 ppm As.

Table 81: Namdini Indicated Mineral Resource estimate at 0.5 g/t Au cut-off grade – March 2018

<table>
<tr><th colspan="2">Category and Weathering</th><th>Tonnage (Mt)</th><th>Grade (g/t Au)</th><th>Contained Gold (koz)</th></tr>
<tr><td rowspan="3">Indicated</td><td>Oxide</td><td>4.0</td><td>1.1</td><td>150</td></tr>
<tr><td>Transition</td><td>4.2</td><td>1.1</td><td>150</td></tr>
<tr><td>Fresh</td><td>171.4</td><td>1.13</td><td>6,160</td></tr>
<tr><td colspan="2">Total Indicated Mineral Resource</td><td>179.6</td><td>1.13</td><td>6,460</td></tr>
</table>

Notes: Mineral Resource estimates are reported inclusive of those Mineral Resources converted to Ore Reserves. The Mineral Resources and Ore Reserves conform with and use JORC Code (2012) recommendations and Canadian Institute of Mining, Metallurgy and Petroleum Standards (CIM, 2014)

Table 82: Namdini Inferred Mineral Resource estimate at 0.5 g/t Au cut-off grade – March 2018

Category and Weathering		Tonnage (Mt)	Grade (g/t Au)	Contained Gold (koz)
Inferred	*Oxide*	0.07	0.9	**2**
	Transition	0.02	0.7	**0.5**
	Fresh	12.9	1.3	**538**
Total Inferred Mineral Resource		**13.1**	**1.3**	**540**

Notes: Mineral Resource estimates are reported inclusive of those Mineral Resources converted to Ore Reserves. The Mineral Resources and Ore Reserves conform with and use JORC Code (2012) recommendations and Canadian Institute of Mining, Metallurgy and Petroleum Standards (CIM, 2014)

At this time, there are no known environmental, legal, socio-economic, marketing or other relevant conditions, that would put the Project at a higher level of risk than any other developing project within Ghana, West Africa. There are no known risk factors that would materially affect the estimated Mineral Resource of the Namdini Gold Project.

15.0 MINERAL RESERVE ESTIMATES

15.1 Introduction

No Ore Reserve was defined for the Namdini Gold Project at the previous PEA level of study (Golder, 2018). Subsequently Cardinal engaged Golder to estimate the Ore Reserve based on the latest available resource model.

An Ore Reserve was announced on 19 September 2018 (Cardinal, 2018c). Cardinal confirms that it is not aware of any new information or data that materially affects the information included in the corresponding market announcement, and that all material assumptions and technical parameters underpinning the Ore Reserve estimates in the market announcement continue to apply and have not materially changed. This is the maiden release of the Ore Reserve for the Namdini Gold Project.

Golder estimated the Ore Reserve in accordance with the JORC Code (2012). The term 'Ore Reserve' is synonymous with the term 'Mineral Reserve' as used by Canadian National Instrument 43-101 'Standards of Disclosure for Mineral Projects' (NI 43-101, 2014). The JORC Code (2012) is defined as an 'acceptable foreign code' under NI 43-101.

The Ore Reserve was estimated from the Mineral Resource after consideration of the level of confidence in the Mineral Resource and taking account of material and relevant modifying factors including mining, processing, infrastructure, environmental, legal, social and commercial factors. The Probable Ore Reserve estimate is based on Indicated Mineral Resources. No Inferred Mineral Resource was included in the Ore Reserve. The Ore Reserve represents the economically mineable part of the Indicated Mineral Resources. There is no Proved Ore Reserve since no Measured Mineral Resource has yet been defined.

At this time, there are no known environmental, legal, socio-economic, marketing or other relevant conditions, that would put the Project at a higher level of risk than any other developing project within Ghana, West Africa. There are no known risk factors that would materially affect the estimated Ore Reserve of the Namdini Gold Project.

The proposed mine plan is technically achievable. All technical proposals made for the operational phase involve the application of conventional technology that is widely utilized in the gold industry. Financial modelling completed as part of the PFS shows that the Project is economically viable under current assumptions. Material Modifying Factors (mining, processing, infrastructure, environmental, legal, social and commercial) were considered during the Ore Reserve estimation process.

15.2 Mining approach

The Mineral Resource model produced by MPR (2018) discussed in Section 13.1 was used to develop a mining model.

The mining model retained the Mineral Resource model framework using a parent block model size of 12.5 m (east) by 25.0 m (north) by 5 m (elevation). Mining is proposed on 5 m flitches in the ore within 10 m benches. The use of the multiple indicator kriging (MIK) method for the resource estimation provides a fair representation of the anticipated equivalent ore loss and dilution given the proposed low-selectivity, medium-scale mining method.

For the purposes of the mining study, it was assumed that a 'dry pit' will be operated and that sufficient dewatering will allow for the geotechnical design for dry slopes.

All tonnage is estimated on a dry basis with expected surface moisture associated with the hauled rock being less than 3% on average.

Using the marginal COG of 0.5 g/t Au, the proportion of ore, and the gold grade above the COG, were defined in the mining model and the ore and waste fraction within each block was calculated, based on the Mineral Resource MIK values of *pr05* and *au05*. The blocks were then exported for open pit optimization.

15.3 Processing approach

This PFS has considered three process rate throughput options, best described as a Low, Mid and High Case.

- The Low Case provides a process plant throughput limited to 4.5 Mtpa
- The Mid Case provides a process plant throughput limited to 7.0 Mtpa
- The High Case provides a process plant throughput limited to 9.5 Mtpa.

Capital and operating costs for these options are developed in Section 21.0.

15.4 Summary of the Ore Reserve estimate

The pit optimization used PFS level mining and processing costs, processing recoveries, recommended geotechnical slope angles and a gold price of US$1,300 per ounce to determine an optimum pit shell. The selected pit shell was used as a basis for the final open pit design. A mining schedule, processing schedule, operating cost model, and overall financial model were developed based on the mineral inventory contained within the open pit design. The mining model within the final pit is reported as the Probable Ore Reserve (Table 83).

Table 83: Ore Reserve estimate at September 2018

Class	Material	Tonnage (Mt)	Grade (g/t Au)	Contained Gold (koz)
Probable	Oxide	4.2	1.14	155.5
Probable	Transition	4.2	1.09	146.5
Probable	Fresh	121.2	1.14	4,458
Probable	**Total**	**129.6**	**1.14**	**4,760**

Notes: The Ore Reserve conforms with and uses the JORC Code (2012) and CIM (2014) definitions
The Ore Reserve was evaluated using a gold price of US$1,300 per ounce
The Ore Reserve was evaluated using a fixed cut-off grade of 0.5 g/t Au
Ore block grade and tonnage dilution was incorporated through the use of an MIK estimation model which was demonstrated to incorporate an expected level of equivalent ore loss and dilution for the scale of mining envisaged
All figures are rounded to reflect appropriate levels of confidence
Apparent differences may occur due to rounding.

Ore Reserves were estimated for the Namdini Gold Project as part of this PFS. The Probable Ore Reserve for the Namdini Gold Project is estimated at 129.6 Mt grading 1.14 g/t Au with a contained gold content of 4,760 koz.

The Namdini Recoverable Mineral Resource estimate is reasonably expected to provide appropriately reliable estimates of potential mining outcomes at the assumed selectivity, without application of additional mining dilution or mining recovery in the mine plan schedules. The Resource estimate is based on the MIK method which incorporates dilution at the proposed scale of mining.

From a geostatistical perspective dilution relates to:

- SMU size
- Sample spacing

- Small scale continuity
- The cut-off grades.

As the SMU size increases, selectivity decreases and there is a corresponding increase in the tonnage mined with a decrease in the average grade as dilution is progressively included. Such dilution is axiomatic in the volume-variance relation of geostatistics.

The MIK adjustment assumed a moderately selective SMU of 10 × 5 × 2.5 m, which has been applied to the low-grade large-tonnage disseminated Namdini deposit. The appropriately-sized equipment is of medium scale (150 t trucks and 400 t face-shovels) and so less amenable to selective mining.

At an RC grade control pattern of 10 × 15 m with 1 m sampling, the short-term model confirmed the MIK Recoverable Resource estimate based on the Resource drilling. The 10 × 15 m grade control pattern is a suitable grid for defining the ore zones at the mining scale anticipated. This will be tested further before mining commences. Section 14.11.2 discusses this further.

The presence of sulfur and arsenic associated with the ore is discussed in Section 14.13.

Mining selectivity is discussed in Section 16.8.3.

Based on the information presented in this PFS, the Ore Reserve estimation process has converted 73% of the Indicated Mineral Resources to Probable Ore Reserves.

16.0 MINING METHODS

16.1 Approach

The mining study, the mine design and costing assessments were based on the resource block model (MPR, 2018) discussed in Section 13.1. Inferred and Indicated Mineral Resources in this model were included in the mining assessment at a cut-off grade of 0.5 g/t Au. The geological block models were provided in the local grid system developed by Sahara Mining Services. The local grid transformation comprises an 8° rotation from WGS coordinates (Table 84), with no elevation change. The transformation rotates the obliquely (WGS) trending drill traverses to east-west (local) grid.

Table 84: Survey grid transformation from UTM (WGS84 Zone 30P) to Local Grid

Translation Point	UTM: WGS 84 Zone 30P		Local	
	East (mE)	North (mN)	East (mE)	North (mN)
1	757,032.992	1,175,611.678	15,000.000	51,800.000
2	757,380.925	1,178,087.348	15,000.000	54,300.000
3	758,569.247	1,177,920.341	16,200.000	54,300.000
4	758,221.314	1,175,444.671	16,200.000	51,800.000

The UTM (WGS 84 Zone 30P) coordinate system was used for mine planning and design to ensure pit layouts, waste dump locations and roads were in the survey system used for the plant layout and all other infrastructure.

16.2 Mining methodology

The mine design for the Namdini Gold Project consists of a series of nested conventional open pit layouts with orebody access provided through a series of ramps. For mining purposes, the Namdini orebody can be considered a layered sequence consisting of:

- strongly Oxidized upper weathered zone
- moderately Oxidized zone with total oxidization of sulfides extending to approximately 10 m average depth (this and the material above it comprise the Oxide zone)
- Transition zone with partial oxidization of sulfides extending to approximately 18 m average depth
- Fresh non-mineralized meta-sediment sequence
- Fresh mineralized host sequence of meta-volcaniclastics, granitoids (tonalite), and diorites.

In the Oxide and Transition zones, a mixture of free-digging, ripping and blasting (after drilling) will be used. In the Fresh zone, competent material at depth will need conventional pre-splitting, drilling and blasting to fragment the rock for excavation. The Oxide and Transition zones are limited in extent with the Oxide zone seldom greater than 20 m below surface. The Transition layer appears more pronounced at the southern end of the final pit mining area and is of limited thickness throughout the remainder of the planned mining area. Figure 80 shows the Oxide, Transition and Fresh zones within the planned final pit extent.

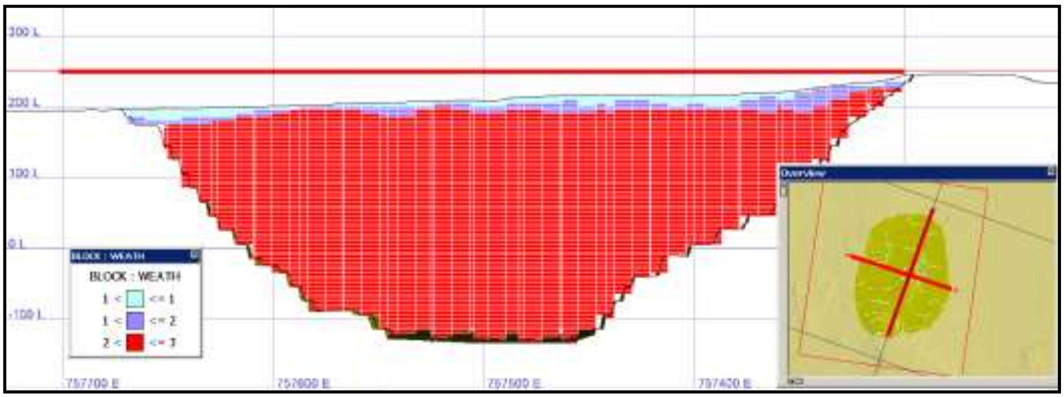


Figure 80: Oxide and Transition material within the final pit (July 2018)

Mining will consist of a conventional hydraulic shovel operation typically using 400 t class excavators in a face-shovel configuration and 150 t class (Cat 785 or similar) rigid body dump trucks hauling on designed access roads. An auxiliary mining fleet of dozers, graders, water carts and utility vehicles will support the mining operation.

No consideration has been made for underground extensions of the operation in this PFS.

Road width was based on 3.5 times the largest envisaged size truck width plus an allowance for the high wall drain (culvert) and the low wall safety berm. The road was designed at 32 m overall width including an allowance of 4 m for the safety berm that should be a minimum of half the height of the wheel for the haul truck (0.5 times 3.1 m, so about 1.5 m high). The criteria for the road width can be seen in Figure 81.

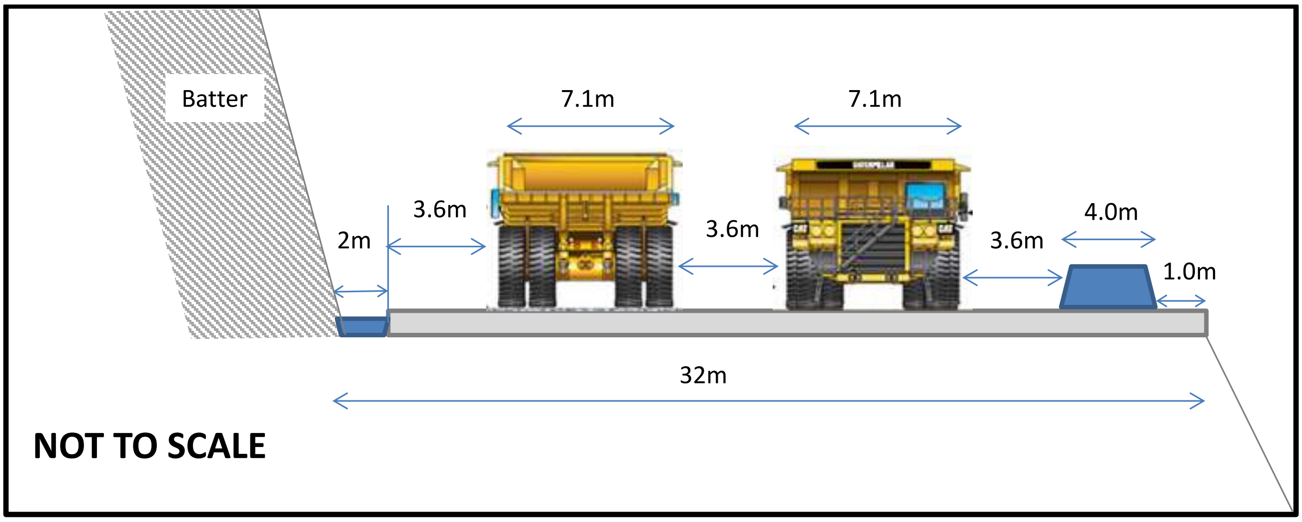


Figure 81: Road width design parameters for Cat 785 size haul trucks

16.3 Mining options

Mine design criteria were used for the generation of open pit layouts (mine design) and production scheduling (mine planning). The optimization strategy included testing a range of process plant throughputs to give an indication of maximum potential value against capital outlay. Capital and operating expenditure costs were applied to the optimization for three selected process plant throughput options: 4.5 Mtpa (Low), 7.0 Mtpa (Medium) and 9.5 Mtpa (High). The discounted Net Present Value ("NPV") from the optimization was used to assess the relative value of each option.

16.4 Mining model

The resource model was provided to Golder by Cardinal in February 2018 and is discussed in Section 13.1. The block model framework dimensions and block sizes are shown in Table 85.

Table 85: Mineral Resource block model framework (from MPR, 2018)

	Easting	Northing	Elevation
Start	14,387.5	52,450.0	-550.0
End	15,837.5	54,125.0	250.0
Block size (m)	12.5	25.0	5.0
Count of blocks	116.0	67.0	160.0
Dimensions (m)	1,450.0	1,675.0	800.0

The model was supplied in comma separated value ("CSV") format and was imported into Vulcan software. The supplied model included grade and resource variables defining the proportion of resource material at various cut-off grades, by resource category. The model as provided was first checked for anomalous values, e.g. proportions greater than 100, or density values outside the expected range.

The supplied block model stored the proportion and grade of material above various cut-off grades. The proportion and grade of material above the 'XX' cut-off grade was respectively stored in the fields 'prXX' and 'auXX'. The supplied data file included variables that are shown in Table 86.

Table 86: Mineral Resource model variable names and default values (*Feb V4 2018 Model*)

Variables	Default	Type	Description
lith	0	name	lith_code
density	0	float	density in t per bcm
weath	0	integer	weathering code 1=Oxide 2=Trans 3=Fresh
as	0	float	Grade of arsenic in ppm
s	0	float	Grade of sulfur in %
rockprop	0	float	Proportion of rock in block below topo
au_final	0	float	Au grade used for optimization Au00 for LUC and Au0
pr02	0	float	Proportion of block with grade above 0.2 g/t Au
au02	0	float	Average grade of block above 0.2 g/t Au
pr03	0	float	Proportion of block with grade above 0.3 g/t Au
au03	0	float	Average grade of block above 0.3 g/t Au
pr04	0	float	Proportion of block with grade above 0.4 g/t Au
au04	0	float	Average grade of block above 0.4 g/t Au
pr05	0	float	Proportion of block with grade above 0.5 g/t Au
au05	0	float	Average grade of block above 0.5 g/t Au
pr06	0	float	Proportion of block with grade above 0.6 g/t Au
au06	0	float	Average grade of block above 0.6 g/t Au

Variables	Default	Type	Description
pr07	0	float	Proportion of block with grade above 0.7 g/t Au
au07	0	float	Average grade of block above 0.7 g/t Au
pr08	0	float	Proportion of block with grade above 0.8 g/t Au
au08	0	float	Average grade of block above 0.8 g/t Au
pr09	0	float	Proportion of block with grade above 0.9 g/t Au
au09	0	float	Average grade of block above 0.9 g/t Au
pr10	0	float	Proportion of block with grade above 1.0 g/t Au
au10	0	float	Average grade of block above 1.0 g/t Au
rescat_final	0	integer	rescat_final (Cardinal estimate)
resource18	0	integer	within Cardinal resource shell

Additional variables were then created to facilitate the mining analysis, whilst retaining the original variables. For example, the resource classification fields were copied to an integer-type field. A list of the additional mining variables is shown in Table 87.

Table 87: Additional mining variables added to the Mineral Resource model

Variables	Default	Type	Description
bench	0	integer	Bench RL in metres
bench_num	0	integer	Bench number based upon 10 m benches
batter	60	integer	Face batter angle for pit design
benchheight	10	integer	Bench height for pit design
berm	8	integer	Berm width for pit design
ira	40	integer	Inter ramp angle for pit design
block_id	0	integer	Copy of hidden Block ID from Vulcan block model
au_cont	0	float	Contained Au in grams in the parcel
contoz	0	float	Contained Au in Troy ounces in the parcel
meastons	0	float	Ore tonnes in Measured
indtons	0	float	Ore tonnes in Indicated
inftons	0	float	Ore tonnes in Inferred
measgms	0	float	Gold grams in Measured
indgms	0	float	Gold grams in Indicated
infgms	0	float	Gold grams in Inferred
ore_t	0	float	Ore tonnes in block model
ore_vol	0	float	Ore volume in cubic metres
leach_t	0	float	Ore tonnes that can be fed to the Leach Process
mill_t	0	float	Ore tonnes that can be fed to the Mill Process
waste_t	0	float	Waste tonnes in block model
rock_t	0	float	Total tonnes of rock contained in parcel
waste_vol	0	float	Waste volume in m^3
total_t	0	float	Total tonnes in block model
mcaf	1	float	Mining Cost Adjustment Factor for cost calculations
o_mcaf	1	float	Mining Cost Adjustment Factor for cost calculations for Ore
w_mcaf	1	float	Mining Cost Adjustment factor for cost calculations for Waste
o_mcost	1	float	Mining Cost for Ore
w_mcost	1	float	Mining Cost for Waste
pit_num	0	integer	Pit Number or Pushback Number for scheduling purposes
rescatint	0	integer	Resource Category (1=meas,2=ind,3=inf,4-uncl)

Variables	Default	Type	Description
rockint	0	integer	Rock code converted to integer value
codeint	0	integer	Lithology code in integer format
rwit	0	integer	2-DIGIT Code for Whittle rescat-rtypes combination
rwita	0	integer	3 DIGIT Code for Whittle rescat-rtypes-lease combination
pcaf	0	float	Process Cost Adjustment Factor, used for Whittle and scheduling costs
phase	0	integer	Mining Phase or Pushback variable for flagging stages of mining
period	0	integer	Period when mined in Scheduler
uncltons	0	float	Ore Tonnes in Unclassified
wpit	0	integer	Whittle pit number for import from Whittle results
rpit	0	integer	Whittle Pit for using Measured plus Indicated only

Using the cut-off grade of 0.5 g/t Au, scripting was used to estimate the 'ore' and 'waste' tonnes on a block by block basis within the Measured, Indicated or Inferred classes. The Inferred Resource was only used to provide estimates of the potential ultimate pit shells. No Inferred Resource material was considered when determining the pit shell that is the basis for Ore Reserve reporting.

The mining model was exported to Whittle 4X software for pit optimization and assessment of the Ore Reserves. The block model was exported retaining the 'ore' and 'waste' fractions defined in the MIK resource model.

The Vulcan and Whittle models were compared to ensure there were no export or import errors. The check on the tonnage imported into Whittle showed that the Indicated plus Inferred material above cut-off was 218 Mt, with an identical ore tonnage being present in the Vulcan mining model, showing that there had been no material loss or gain during the export from Vulcan to Whittle.

Ore types to be used in the Whittle optimiser were created based on resource category and material type (weathering). Table 88 and Table 89 show the comparison check between the Vulcan mining model and the exported Whittle optimization model. This matches the original mining model's mineralized material. The check on the total mineralized material enables a comparison that is independent of the optimization envelope used by the resource geologist to estimate the Mineral Resource.

Table 88: Comparison of MIK resource model to Whittle mining model

>0.5 g/t Au	Indicated and Inferred for Namdini		
Model type	Vulcan	Whittle 4X	Difference (%)
Model name	*Namdini_LG_MIK_V4_FEB 2018_RPIT2.bmf*	*NAM2018_FEB_RWIT_ORE_ T_with_MCAF.mod*	
Tonnes (Mt)	218.9	218.9	0.0
Contained gold (koz)	7,844	7,844	0.0

Notes: This is not considered formal reporting of Indicated + Inferred, which is not permitted under NI 43-101

Table 89: Comparison of Ore Type by resource category and material type above 0.5 g/t Au

Ore Type (resource category + material type)	Code	Vulcan Model Contained gold (g)	Whittle Model Contained gold (g)	Difference (%)
Ind+Ox	21	4,855,091	4,855,091	0.0
Ind+Trans	22	4,616,868	4,616,868	0.0
Ind+Fresh	23	203,630,413	203,630,415	0.0
Inf+Ox	31	73,546	73,546	0.0

Ore Type (resource category + material type)	Code	Vulcan Model Contained gold (g)	Whittle Model Contained gold (g)	Difference (%)
Inf+Trans	32	16,306	16,306	0.0
Inf+Fresh	33	30,792,954	30,792,954	0.0

Source: "namdini_pfs_geology_resource_by_RWIT_2018June18.xls"

Table 90 summarises the model origin and dimensions of the Whittle model.

Table 90: Whittle model dimensions

Item	Easting	Northing	RL
Block size (m)	12.5	25	5
Model origin	14387.5	52450.0	-550.0
Extent (m)	1450	1675	800
Count of parent blocks	116	67	160

(NAM2018_FEB_RWIT_ORE_T_with_MCAF.mod)

The parameters used in the pit optimization analysis are summarized in Table 91. The full input parameters were documented and provided to Cardinal.

Table 91: Summary of the technical and economic parameters used in pit optimization studies

Parameter	Units	Value
Gold price	US$/oz	1,300
Ore lost during mining*	%	0
Dilution added during mining*	%	0
Mining rate	Mtpa	Varies with scenario
Recovery – Oxide	%	90
Recovery – Transition**	%	86
Recovery – Fresh**	%	86
Annual discount rate	%	10
Cut-off grade	g/t Au	0.5

Notes: * Ore loss and dilution that are the result of operational mining (whether due to mine, planning, grade control information or mining recovery) were allowed for in the MIK resource modelling process. No further allowances, adjustments or factoring were made during pit optimization, mine design and mine scheduling.

** Recovery used from the PEA that was applicable at the time of the pit optimization study.

16.5 Mining cut-off grade for pit optimization and mine planning

The cut-off grade ("COG") for mine design and mine planning was derived from estimates of the expected gold price, processing costs and processing recoveries in the Namdini PEA study (Golder, 2018).

The COG was established using an assumed long-term gold price of US$1,300/oz. Gold royalties were assumed at 5% of gold price, with payable gold estimated at 99.8% of doré exported. The net gold price in the required COG units (grams per tonne) was thus $39.63.

Process plant recovery was applied at an average of 86% (the value for Fresh material) as advised by Cardinal from the PEA testwork applicable at the point of time for the pit optimization study.

The input processing cost provided in May 2018 was $14.49 plus an additional $1.50/t allowed for stockpile reclaim giving a total of $15.99/t of mill feed (as dry tonnes).

Thus, the initial marginal COG was estimated as: process cost/(net gold price * process recovery)

i.e. COG = ($14.49 + $1.50) / ($39.63 * 86%)

giving 0.5 g/t (to one significant figure).

A summary of the criteria used to estimate the applicable marginal cut-off grade is provided in Table 92.

Table 92: Marginal cut-off grade determination

Item	Value	Unit
Gold price	1,300.00	US$/oz
Royalties	65.00	US$/oz
Gold price after royalties	1,235.00	US$/oz
Payable gold fraction	99.8%	
Net price per ounce	1,232.53	US$/oz
Net price per gram	39.63	US$/g Au
Process plant recovery (Fresh)	86%*	% (average)
Stockpile reclaim costs	1.50	per Tonne
Process costs	14.49	per Tonne
Total process costs	15.99	per Tonne
Marginal COG (Au g/t)	**0.48 (≡ 0.5)**	**g/t Au**

Notes: Note that the actual recovery varies in the model by lithology.

Using the calculated marginal COG of 0.5 g/t Au, the proportion of ore and the gold grade above the COG were defined in the mining model and the parcelled (ore + waste) blocks were exported for open pit optimization.

16.6 Pit optimization

The initial set of pit optimizations covered options with and without Inferred Resource material. The inclusion of the Inferred Resource material in the pit optimization was to assess likely areas that could provide possible future growth opportunities and to ensure that infrastructure and permanent waste dumps are outside the potential future pit footprint. The initial optimizations used a revenue factor (“RF”) ranging from 0.5 to 1.5, equivalent to an input gold price range of US$650 to US$1,950. The low-price pit shell (0.5 RF) indicated the target area for commencement of the initial Phase (or Stage or pushback), whilst the high price (1.5 RF) indicated the likely maximum potential pit limit.

Whittle produces a Best case and Worst-case schedule as a guide to determine a practical balance between high financial value and practical mining constraints, such as minimum mining width and bench advance (sink rate). The Whittle documentation describes the ‘Best case’ and ‘Worst case’ as follows:

The Best-case schedule consists of mining out pit 1, the smallest pit, and then mining out each subsequent pit shell from the top down, before starting the next pit shell. In other words, there are as many intermediate mining pushbacks as there are pit outlines within the one we are mining. This schedule is seldom feasible because the pushbacks are usually much too narrow. Its usefulness lies in setting an upper limit to the achievable Net Present Value. If, as is sometimes the case, Worst case and Best-case Net Present Values differ by only a percent or two, then you know that, for that pit, mining sequence is unimportant from an economic point of view.

The Worst-case schedule consists of mining each bench completely before starting on the next bench. This schedule, or something very close to it, is usually feasible. It also sets a lower limit to the Net Present Value. Unless you mine waste to the exclusion of ore, it is difficult to achieve a lower NPV.

Golder has found that using the average of the discounted Best case and discounted Worst case to provide a discounted average case (DAvg) has proved to be useful in providing an expected financial return that often aligns with more detailed schedules. The maximum DAvg is thus often used to identify the preferred revenue factor point at which to identify a final pit shell.

Pit optimizations were carried out using only the Measured plus Indicated resource categories with the Inferred Resource material being considered waste. This method is appropriate when determining likely pit shells for potential Ore Reserve reporting.

The Whittle optimization results show that for the given costs and pricing assumptions the discounted cashflow begins to erode from about the 0.9 RF indicating that any pit shell above 0.9 RF is not adding value. Figure 82 shows the undiscounted and discounted cashflows against the revenue factor for the M+I Resource categories.

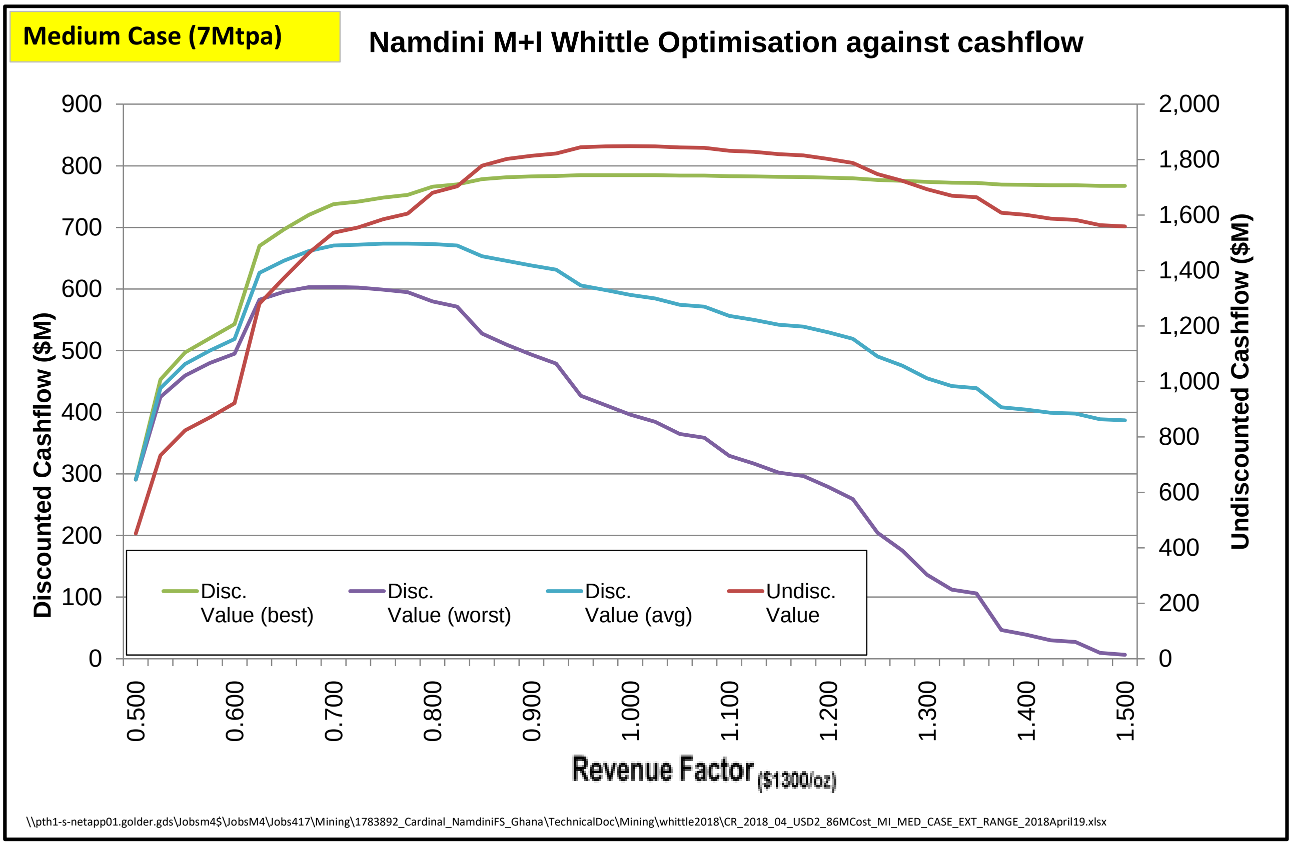


Figure 82: Pit optimization based on the Measured + Indicated Resources

In terms of comparison to 1.0 RF, it is often useful to assess and compare the relative changes in total rock movement, ore feed, contained metal and discounted average cashflow (Figure 83). The discounted average cashflow for Namdini rises rapidly up to around 0.75 RF and is then relatively constant before decreasing at around 0.95 RF. The fall-off in value at around 0.95 RF is due to only modest increases in contained gold in the ore feed for relatively large increases in total rock movement (i.e. the strip ratio increases markedly). The slope of the curve for the total rock increases much faster than that for Ore beyond 0.85 RF.

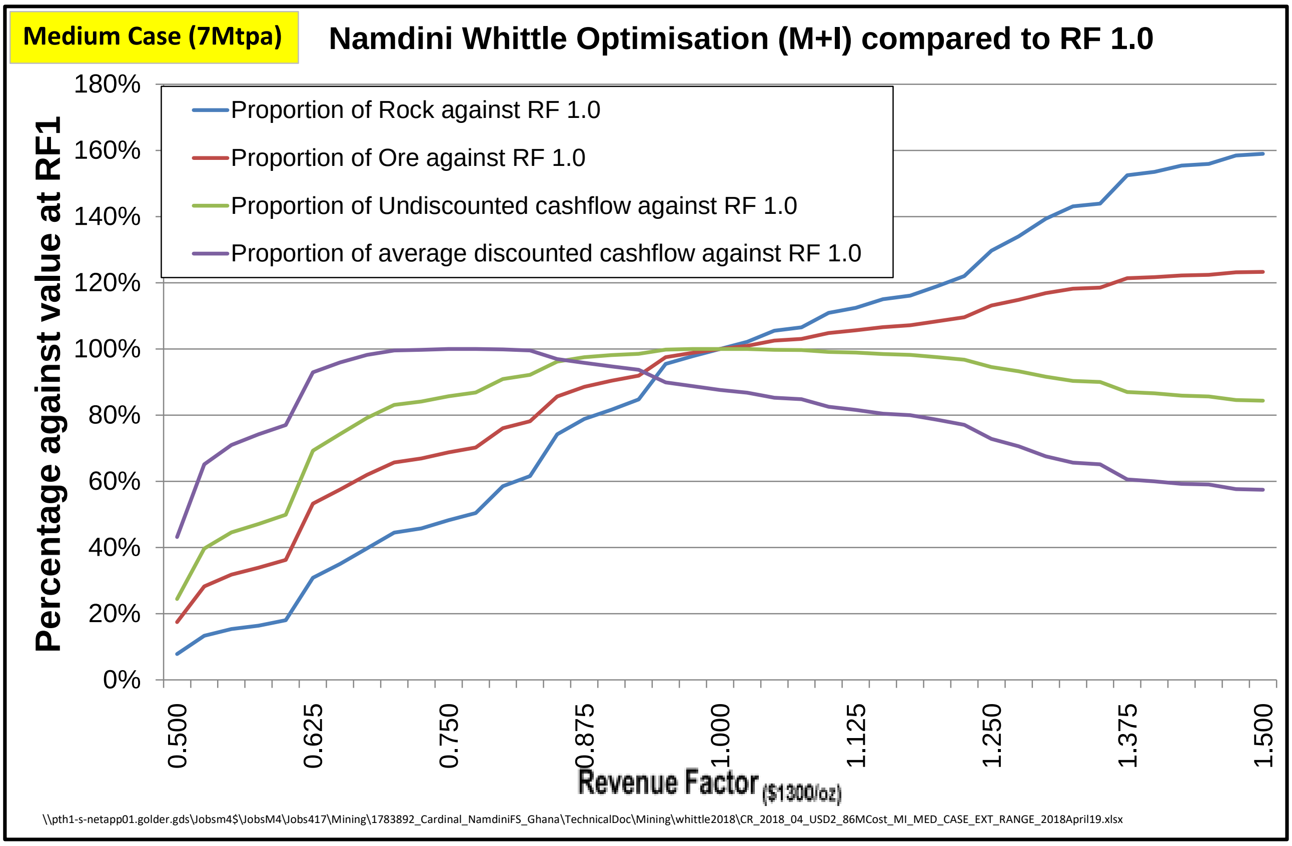


Figure 83: M+I pit optimization comparison against RF 1.0

A visual inspection of the pit shells in the Whittle viewer showed the pit expands from east to west as the gold price increases. This is expected as the gold deposit dips 70°W.

Figure 84 to Figure 86 show the pit shells in 3 orientations. The pit shells are colored by group (Table 93).

Table 93: Pit shells grouped by color for viewing

Pit Shell Group	Revenue Factor	Comment
Pits 1 to 9	0.50 to 0.70	Initial target pit
Pits 10 to 20	0.725 to 0.975	Final economic mineable reserve shell range
Pits 21 to 30	1.00 to 1.225	Infrastructure exclusion area for modest gold price increase
Pits 31 to 41	1.25 to 1.50	Infrastructure exclusion area for high gold price increase

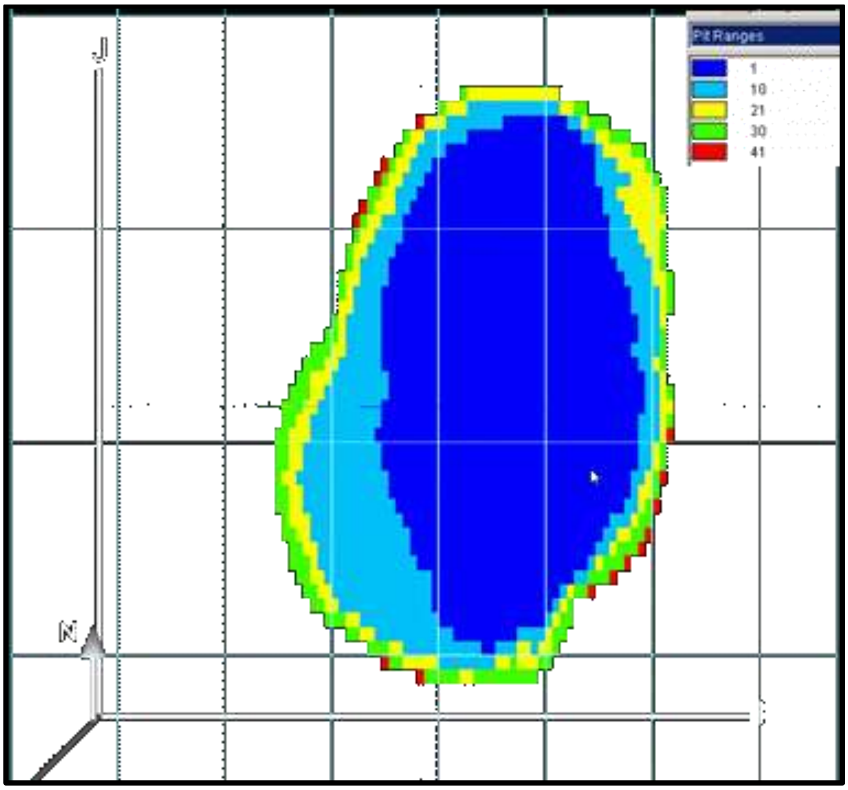


Figure 84:Grouped pit shells plan view (M+I, RF1.0 is US$1,300/oz)

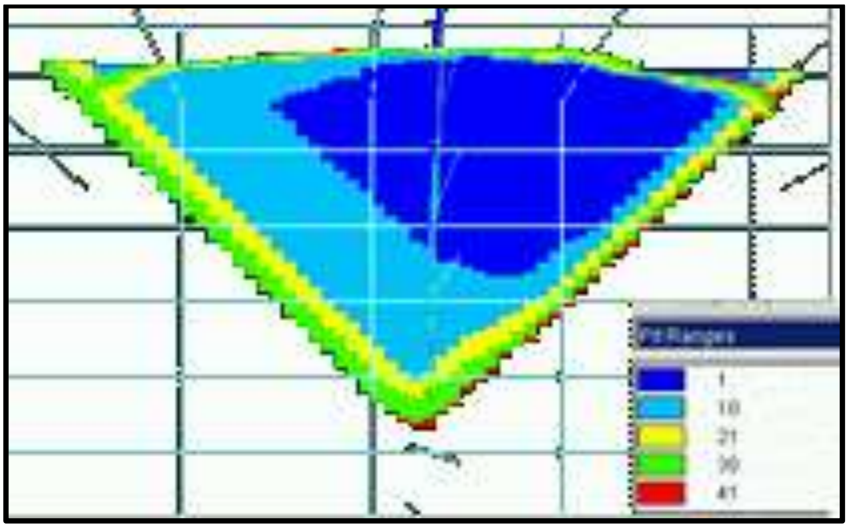


Figure 85: Grouped pit shells facing north (M+I, RF1.0 is US$1,300/oz)

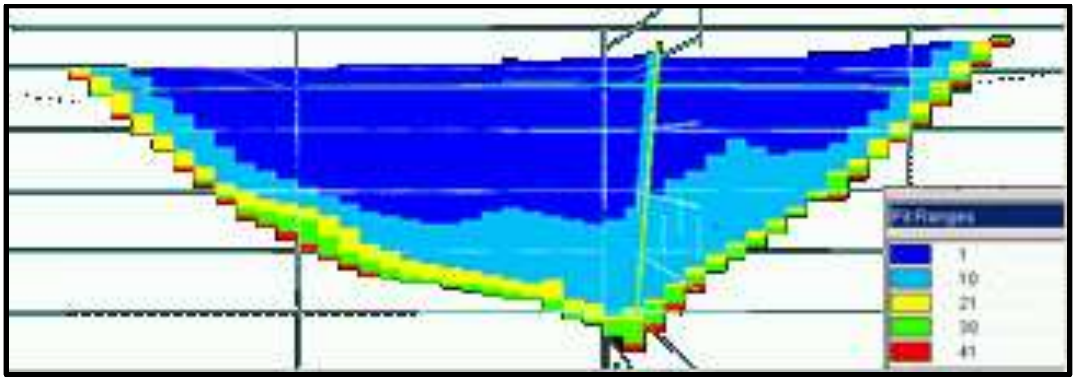


Figure 86: Grouped pit shells facing east (M+I, RF1.0 is US$1,300/oz)

A cross section of the block model (Figure 87) shows the general dip of the mineralization (the green ellipse) and the higher-grade material (>2 g/t Au) that drives the pit optimization.

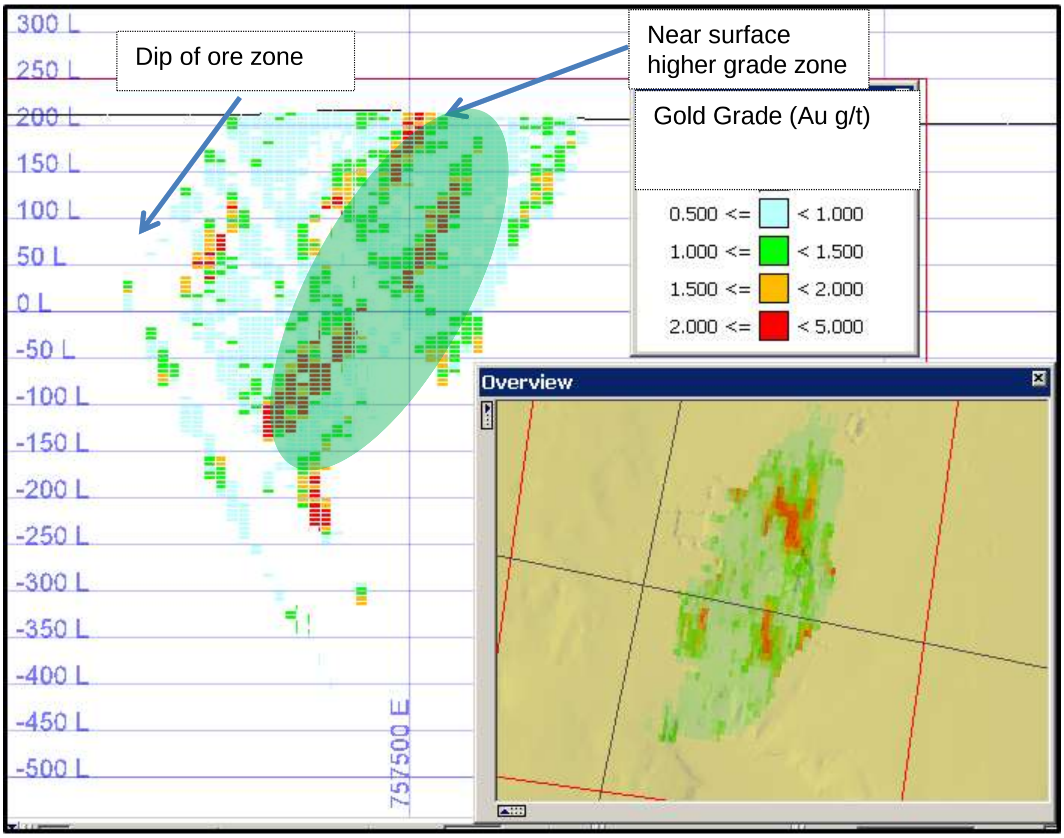


Figure 87: Cross section of the M+I Resource model with blocks colored by gold grade, facing north

A general plan view of the ore zone is shown in Figure 88 showing two of the larger higher-grade zones near surface.

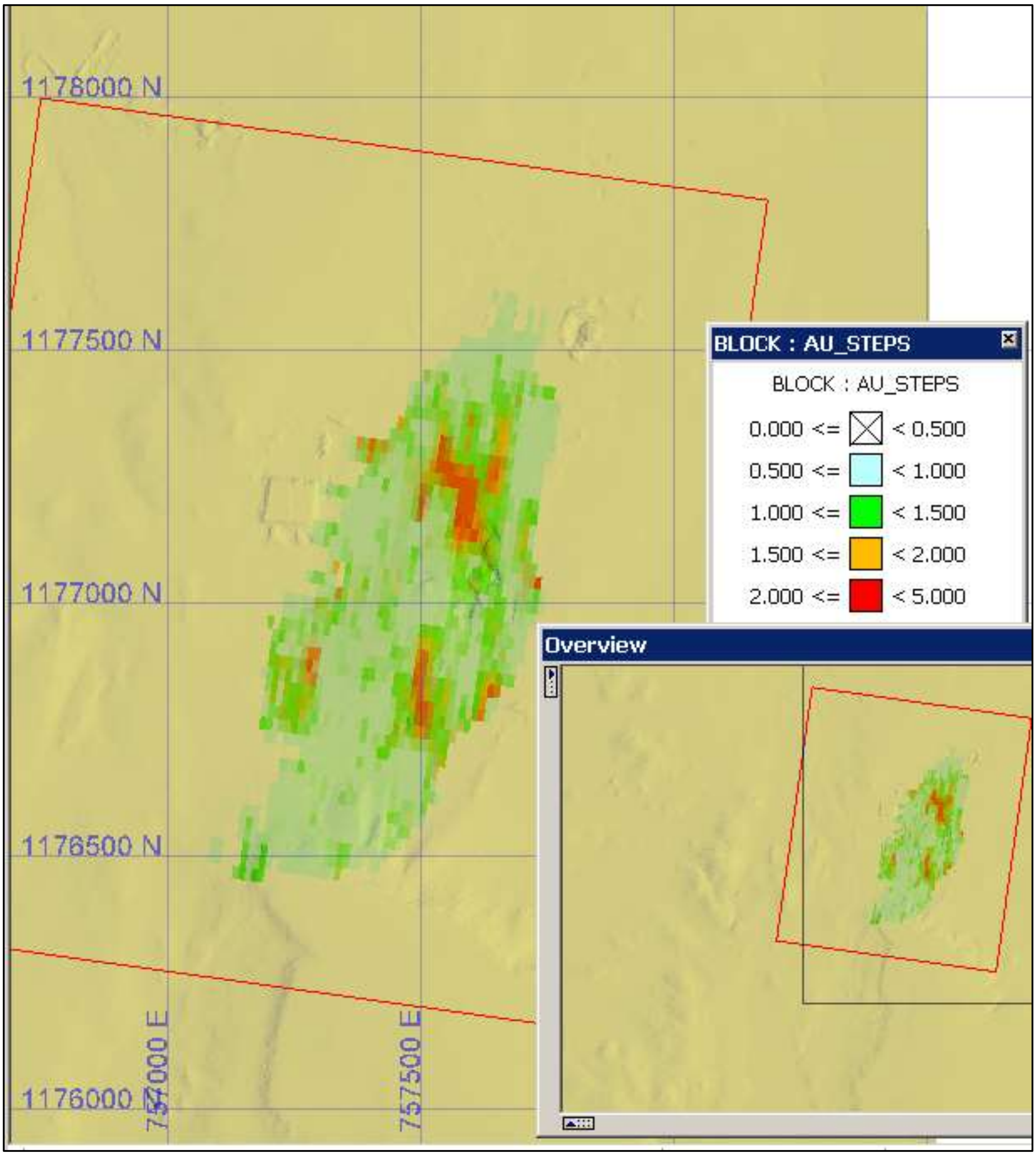


Figure 88: Plan view of the M+I Resource model with blocks colored by gold grade

The area of highest-grade potential was chosen to commence the 'Starter Pit'. This area consistently represents an economically viable mineable pit at very low gold prices.

The pit shells were used to design Phases (also known as stages, pushbacks or "PB"s) for the progressive development of the Namdini open pit.

16.7 Pit selection

To identify possible stages and the extent of a higher value 'starter pit' a further set of optimizations was run on the resource model using a closer-spaced RF range. The range was extended down to a RF 0.25 up to RF 1.0 in 0.0125 increments, equivalent to approximately US$16.25/oz shell increments. This extended RF range allows for greater resolution of the potential higher-grade starter pit and also allows greater resolution where incremental pushbacks or Phases can be sited to maximize the value per pushback.

The use of the fine RF increments allowed the identification of a possible starter pit area below the 0.5 RF. The Namdini Gold Project shows distinct step changes in ore and waste movements for the 0.525 and 0.725 RF shells. The step changes are useful to target pushbacks with potential economic value, in that the step changes generally indicate larger volumes of waste movement required to unlock the available ore. It can be seen from Figure 89 that up to around 0.475 RF there is potential for a high value 'starter pit' with very low waste removal requirements. Subsequent pushbacks would be based upon a combination of practical considerations such as minimum mining width, pit geometry and value changes in the pit design.

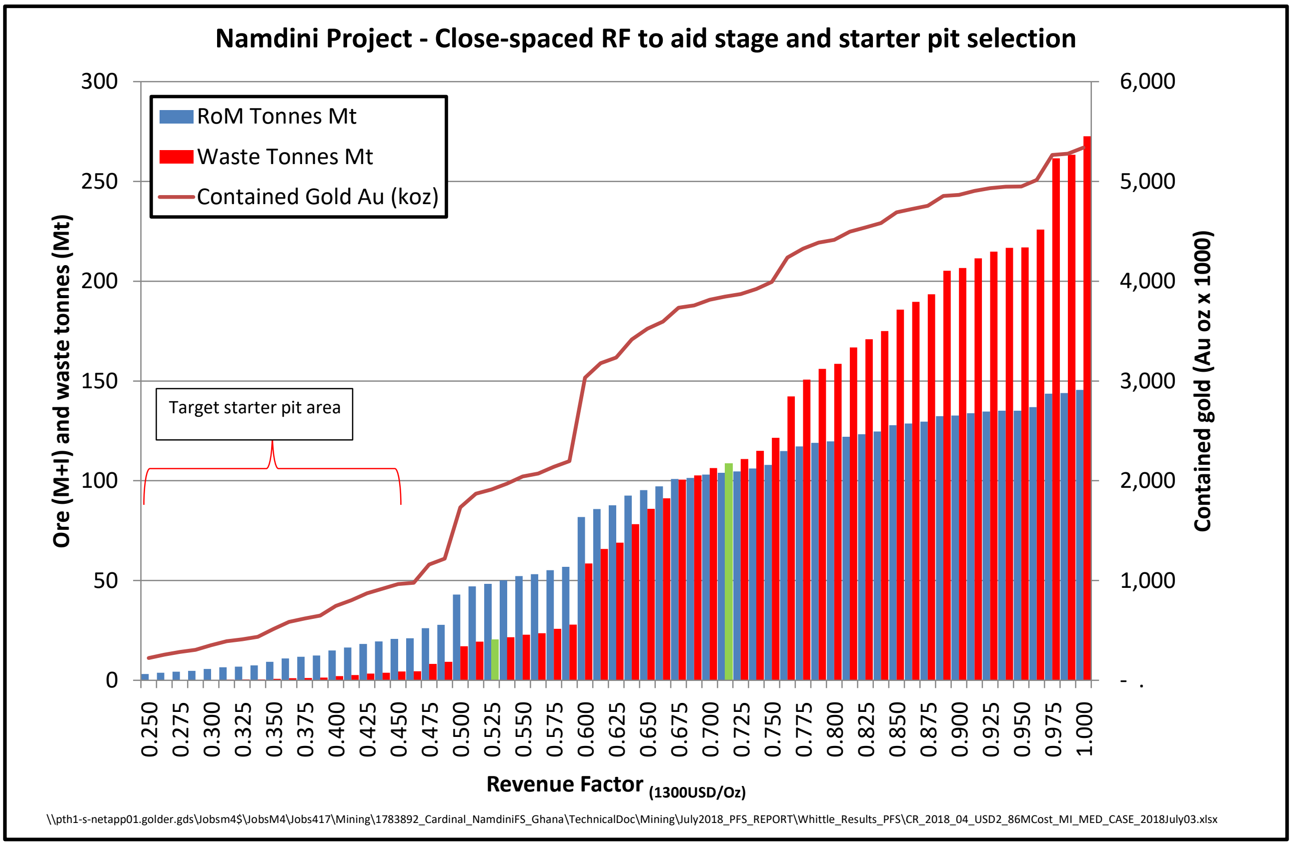


Figure 89: Close-spaced revenue factors showing ore, waste and contained gold

As identified during the previous PEA study (Golder, 2018) the Namdini Gold Project appears to offer a maximum return on capital employed ("ROCE") in terms of NPV at a 10% discount rate (NPV_{10}) at around 6 to 7 Mtpa of mill feed throughput. The ROCE was assessed over the range of Whittle optimizations for the Measured + Indicated ("M+I") only processing option indicating that the peak ROCE occurred at a revenue factor based on a gold price of US$1,300 of RF 0.813 and was similar up to a RF of 0.9. The ROCE graph for the M+I Whittle optimization is shown in Figure 90, which shows a similar outcome to that in the PEA (Golder, 2017).

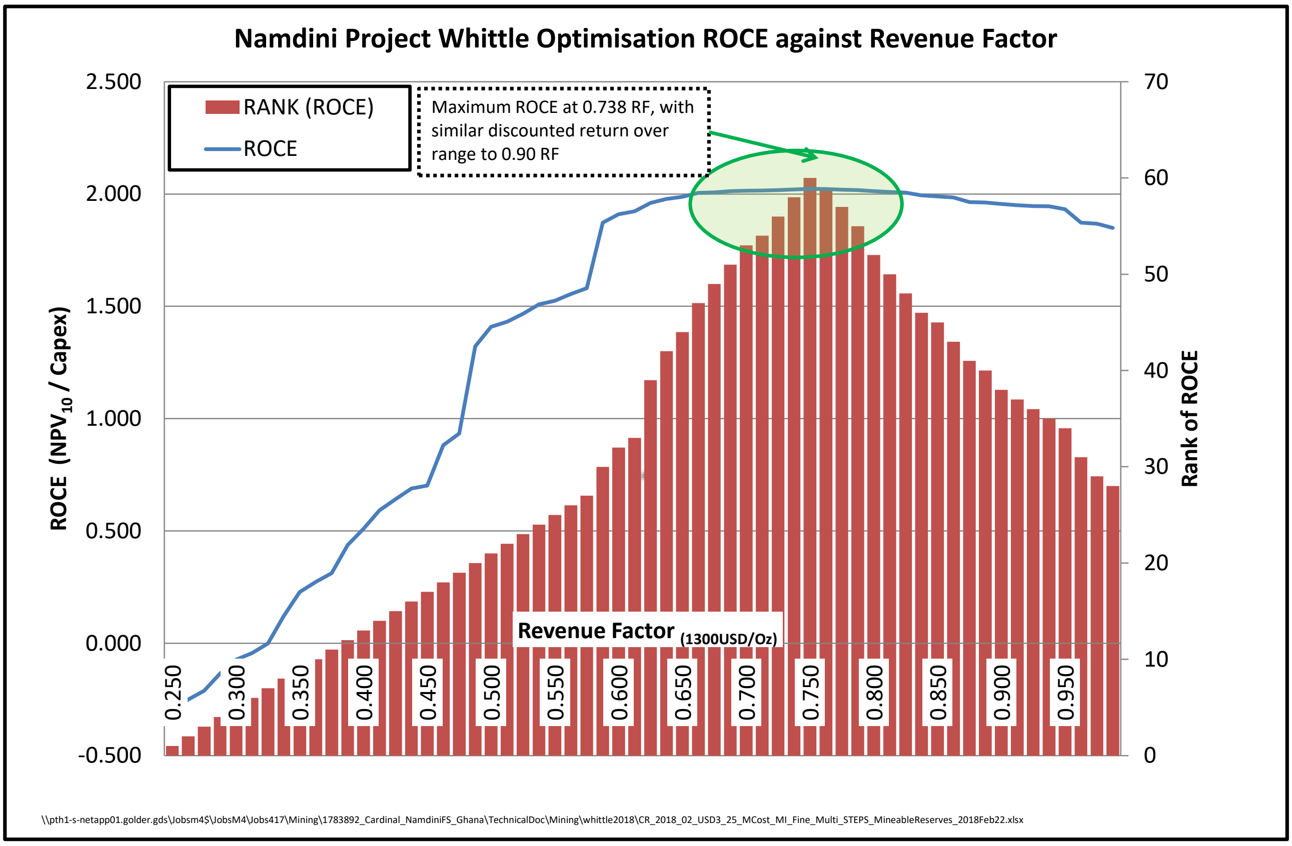


Figure 90: ROCE for M+I optimization

The pit shells from the Whittle optimiser were then exported and imported into Vulcan to enable Phase selection based on practical considerations. A combination of the value potential as indicated by the Whittle optimization results and the physical shapes viewed in the Vulcan mine modelling software enabled the selection of four practical Phases within the final pit. The final pit was selected at a RF of ~ 0.90, this being the point where the DAvg is close to the peak value prior to the point of erosion of value, enabling a balance between ore volume and financial return.

The project has considered three process plant throughput options based around step change limitations in the processing plant configuration (Section 17.1). The Low Case (4.5 Mtpa) option is considered the largest achievable throughput with the adoption of a jaw crusher as the primary crushing stage. The High Case (9.5 Mtpa) option is considered the maximum based on using dual pinion drives rather than the more expensive gearless drives for the grinding mills. An analysis of the throughput range using a cost of capital scaling factor of 0.6 indicates that the optimum process plant throughput appears to be around the Mid Case (7.0 Mtpa). The analysis taken from the PEA study stage is repeated here to illustrate the initial finding. The graph in Figure 91 shows the ROCE compared to the process plant throughput for the Namdini Gold Project, as derived during the PEA (Golder, 2018).

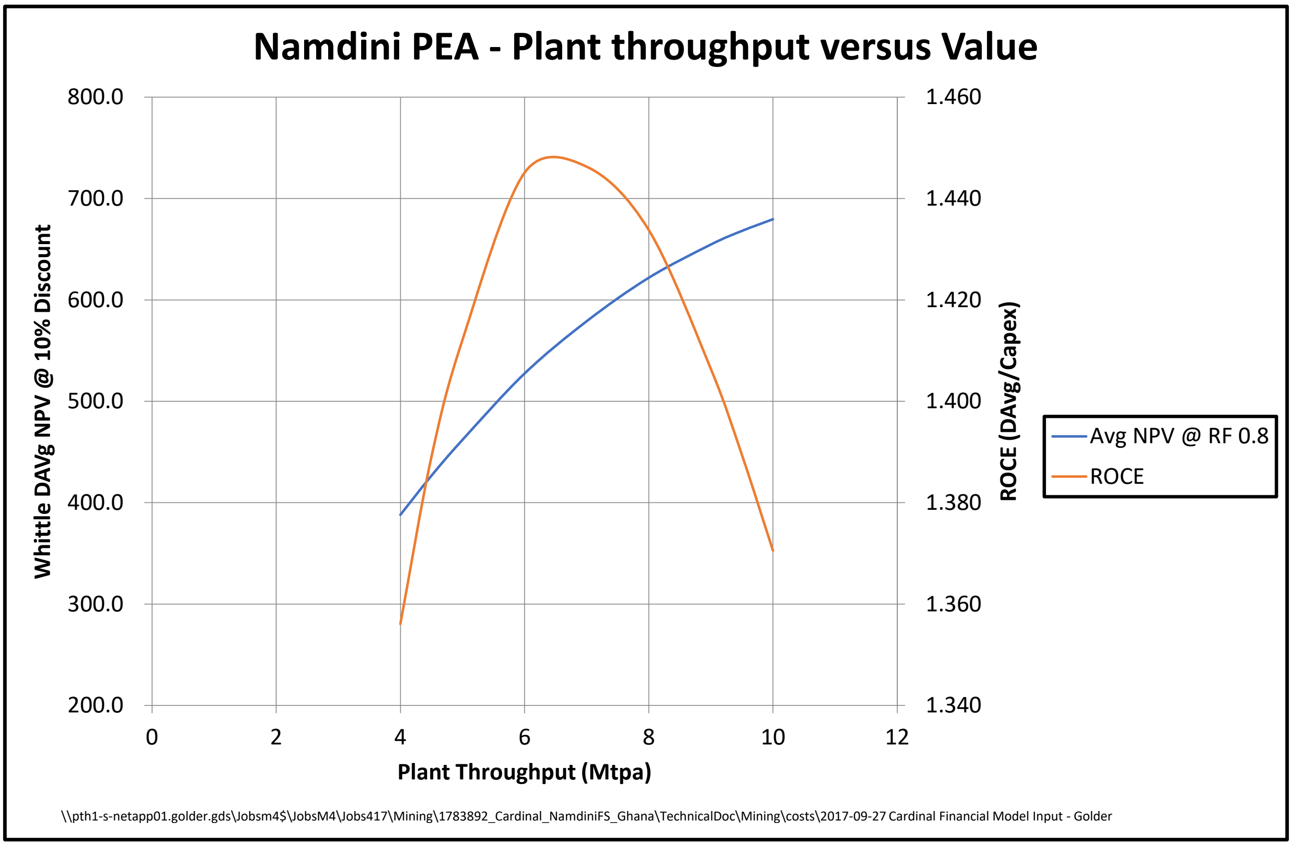


Figure 91: High-level project value *vs* mill throughput (Golder, 2018)

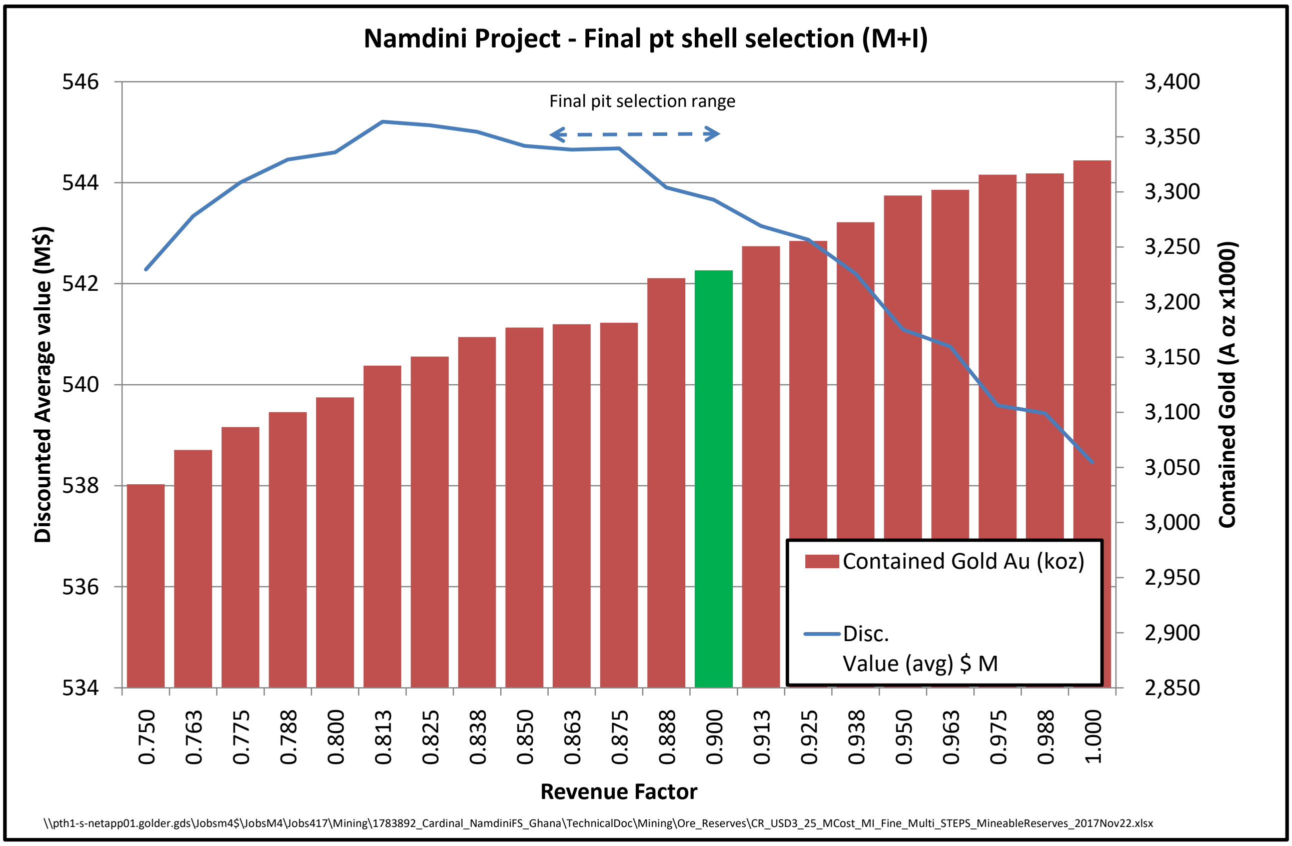


Figure 92: Close-spaced revenue factors for selection of final pit size

The pit Phase selection identified four potential stages including the initial starter pit. The initial starter pit (Phase 1) has targeted an area that contains some 28 Mt of ore feed, with only 9 Mt of waste; this very low strip ratio aids in bringing value forward for the project. The subsequent pushbacks (Phase 2 to Phase 4) have targeted areas that contain 3 years of ore feed per Phase, whilst maximising value. The final pit targets 128 Mt (excluding Inferred Resource) at an average head grade of 1.14 g/t Au containing 4.7 M oz of gold. Table 94 shows the cumulative material quantities at each Phase selected for the pit design process.

Table 94: Cumulative pit Phase selection

RF $_{1300}$	Phase	Rock (Mt)	Ore (Mt)	Waste. (Mt)	Contained Gold (koz)	Strip ratio (W:O as t:t)	Grade (Au g/t)
0.488	Phase 1	37.1	27.8	9.3	1,221	0.33	1.37
0.600	Phase 2	140.5	81.9	58.6	3,033	0.72	1.15
0.738	Phase 3	221.1	106.1	115.0	3,921	1.08	1.15
0.850	Phase 4	313.6	127.8	185.7	4,690	1.45	1.14

Table 95 shows the tonnes, contained gold and grade by Phase and the incremental strip ratio for each Phase and the notably higher average grade achieved from the starter pit compared to the remaining Phases.

Table 95: Pit Phase selection from Whittle output

RF $_{1300}$	Phase	Rock (Mt)	Ore (Mt)	Waste. (Mt)	Contained Gold (koz)	Strip ratio (W:O as t:t)	Grade (Au g/t)
0.488	Phase 1	37.1	27.8	9.3	1,221	0.33	1.37
0.600	Phase 2	103.4	54.1	49.3	1,812	0.91	1.04
0.738	Phase 3	80.6	24.2	56.4	888	2.33	1.14
0.850	Phase 4	92.4	21.7	70.7	768	3.25	1.10
Total		**313.6**	**127.8**	**185.7**	**4,690**	**1.45**	**1.14**

The Phase selection and pit design used only Indicated Resource material with any Inferred Resource material being regarded as waste.

Cardinal anticipates that a drilling and sampling campaign in 2018 focussed on increasing the resource confidence will result in an update to the Mineral Resource model. This may convert some of the Inferred material within the pit design to Indicated Resource, increasing the Ore Reserve.

16.8 Pit design

Pit design criteria were based on geotechnical recommendations, with the deposit broadly broken up into weathered (Oxide), partially weathered (Transition) and Fresh domains, with two distinct domains on the hangingwall and footwall sides of the ore zone (bearing 295°). Table 96 shows the geotechnical design criteria for the Namdini pit.

Table 96: Recommended geotechnical slope configurations for Namdini Pit

State of Weathering	Bench Face Angle	Production Bench Height (m)	Vertical Bench Separation (m)	Berm Width (m)	Inter-Ramp Angle
295° (Footwall) Wall Orientation					
SOX (Saprolites & Saprock)	60°	5	10	6	40.3°
Transition (Moderately Weathered Rock) – Single Benching	60°	10	10	4	45.7°
Slightly Weathered to Fresh Rock	65°	10	20	7.5	49.9°
All Other Wall Directions					
SOX (Saprolites & Saprock)	60°	5	10	6	40.3°
Transition (Moderately Weathered Rock) – Single Benching	60°	10	10	4	45.7°
Slightly Weathered to Fresh Rock	75°	10	20	8	56.3°

For practical pit design purposes, the berm widths were rationalized to an 8 m wide berm to avoid having multiple berm widths required on the same mining bench. Analysis of the block model indicated that the semi-weathered (Transition) material reaches a maximum depth of 160 m RL. Thus, it was deemed prudent to maintain single benches with 6 m berm widths above this level and adopt double-benching (20 m) with 8 m berms below it. Adoption of the 6 m berm in both the Oxide and Transition zones adds a level of increased safety and ease of management in the weathered part of the deposit.

The pit target region sectors were then used to flag the mining block model to allow more detailed design parameters, particularly in the Fresh deeper zone where a higher overall slope angle markedly reduces the required total waste mining. A summary table of the design criteria used in the mining block model is shown in Table 97 The Geotechnical Zone design sectors are shown in Figure 93 with the lithology flag in the block model being used to identify the rock mass type for pit design purposes.

Table 97: Pit design criteria for Cardinal PFS stages

Rock Mass Domain	Design Sector	Bench Height (m)	Bench Face Angle (°)	Spill Berm Width (m)	Limiting Bench Stack Height (m)	Inter-Ramp Angle (°)
Saprolite & Saprock	all	10	60	6	30	40.3
MW Hangingwall MSE	all		60	6	30	40.3
MW DIO	4 only	10	60	6	30	40.3
MW Footwall MSE	5, 6 and 7	10	60	6	30	40.3
Fr Hangingwall MSE	1 and 2 only	20	70	9	80	50.9
Fr Hangingwall MSE	3 only	20	80	9	80	57.9
Fr Footwall MSE	7 only	20	80	9	80	57.9
Fr MVO	all except 5 and 6	20	80	9	80	57.9
Fr DIO	all except 5 and 6	20	80	9	80	57.9

Rock Mass Domain	Design Sector	Bench Height (m)	Bench Face Angle (°)	Spill Berm Width (m)	Limiting Bench Stack Height (m)	Inter-Ramp Angle (°)
Fr Footwall MSE	5 and 6	20	60	9	80	44.2
Fr MVO	5 and 6	20	60	9	80	44.2
Fr DI0	5 and 6	20	60	9	80	44.2

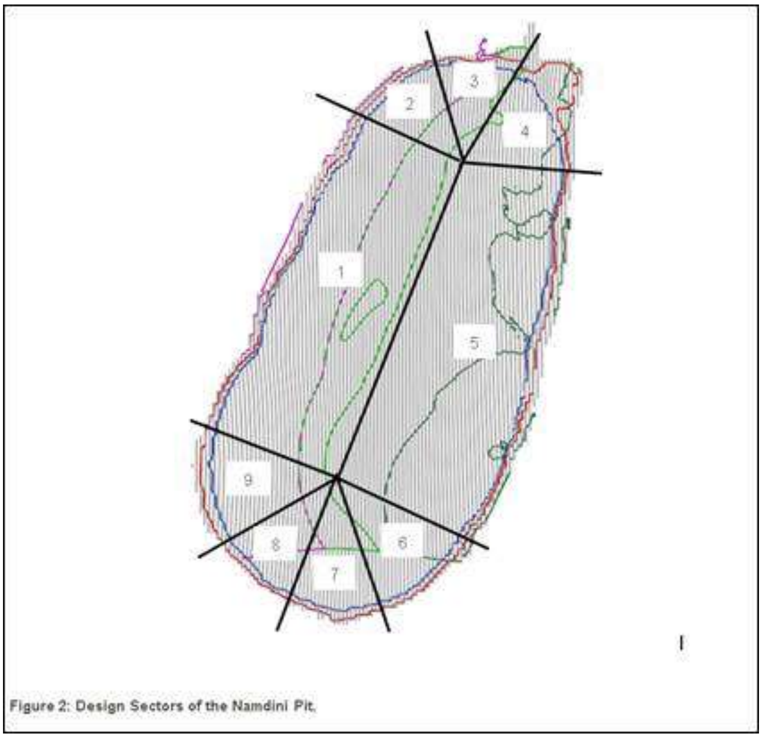


Figure 93: Namdini Geotechnical Zones flagged in mining block model

The overall slope angle in the Fresh at the lower benches equates to some 58 degrees.

The pit was designed with four stages, the initial stage (Starter Pit) being for early access to the higher-grade ore near the surface. The second stage is largely an expansion of the initial stage targeting the ore to a greater depth.

The stage designs were created for optimal ore delivery from the first two stages, due to their low strip ratio, waste rock movement. The third and fourth stages contain a greater proportion of waste rock.

A minimum mining width of 60 m was established between the stages.

16.8.1 Phase design

The pit designs have targeted the maximum discounted value pit shell at a US$1,300/oz gold price. The pit optimization using the Whittle software was used to identify the optimum pit shell with the Inferred Resource material considered as waste rock. The identified pit was then considered for practical staging to minimize waste movement and improve the cashflow for the project. The analysis allowed the selection of four stages with the initial stage targeting a relatively higher-grade area of ore near surface. Access was allowed to the first three stages by a ramp from the northern edge of the pit as the volume of waste rock in the first three stages is considered modest. The final fourth stage has a main access ramp on the western side of the pit to provide a shorter haul to the waste rock dump, given that the final stage has a higher strip ratio than the preceding three stages. Having the primary access on the western side of the pit reduces waste rock haulage costs and thus improves the overall value.

Given limited opportunity outside Phase 1 to target higher grade zones, Phase design was largely focussed on targeting maximum value change points within practical mining constraint limits, such as the minimum mining width for the pushbacks.

The first Phase is a relatively small 'mini-pit' on the eastern side of the deposit (Figure 94). The first Phase contains an estimated 19.9 Mt of Fresh ore with an additional 4.0 Mt of Oxide and Transition ore. This will be stockpiled and processed in campaigns such that a maximum of 10% of available processing time is used for treating the Oxide and Transition ore in any annual period.

The remaining three pit Phases follow a traditional pit expansion with the pits pushing out towards the dip of the ore and the pit deepening with each Phase. The staged progression of all four Phase is shown in Figure 94 to Figure 98.

A summary of the ore by Phase is shown in Table 99 with the gold grade and contained metal in Table 100.

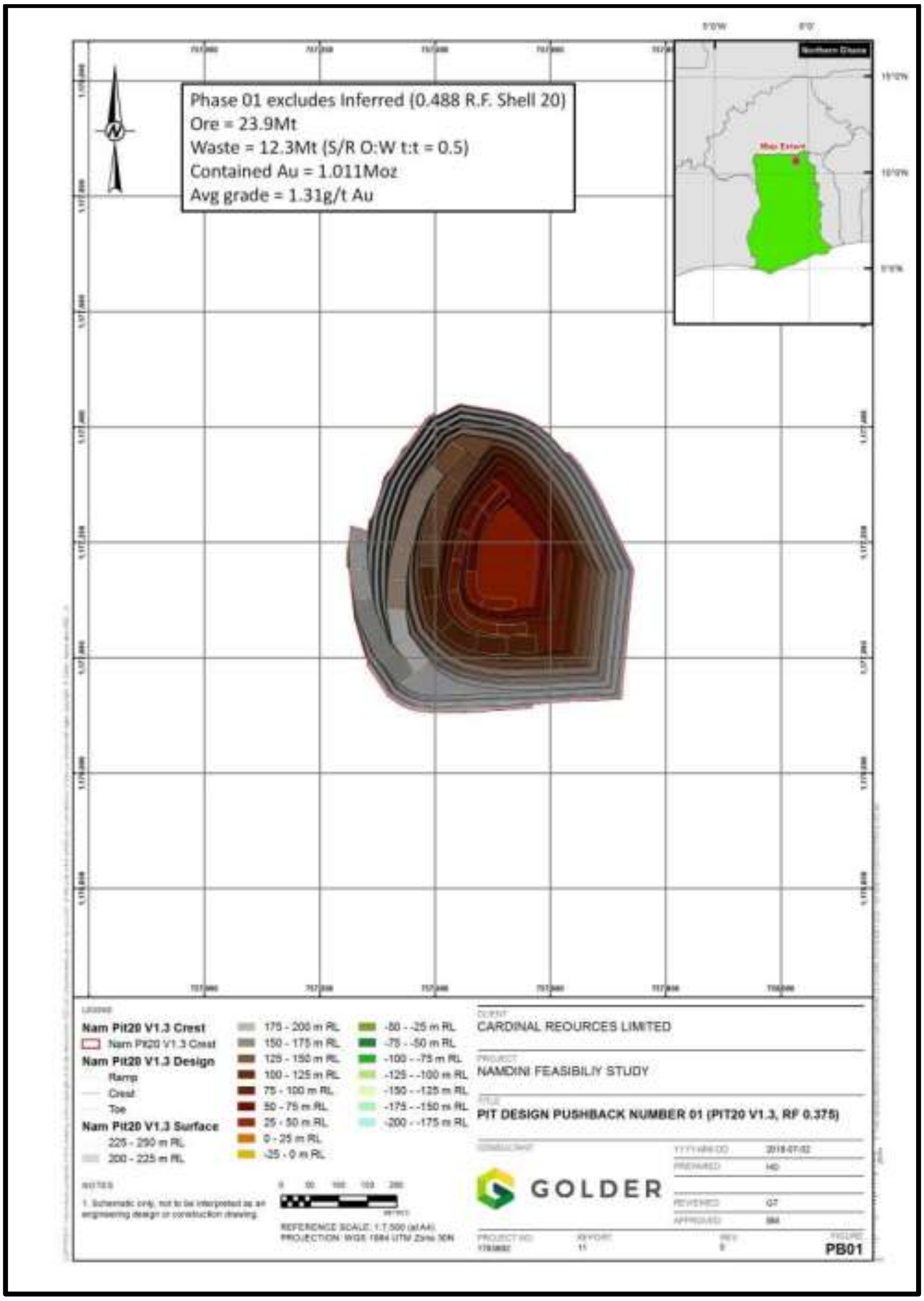


Figure 94: Design Phase 1 (*nam_20pit_v1.3.00t*)

The initial stage (Phase 1) targets a relatively small high-grade zone of near surface mineralization that results in a mini-pit being created to provide early access to the higher-grade portion of the ore to enhance the project cashflow. Phase 1 targets a final pit depth of 160 m from surface with a long axis (N-S) maximum dimension of 530 m and a short axis (E-W) maximum dimension of 480 m.

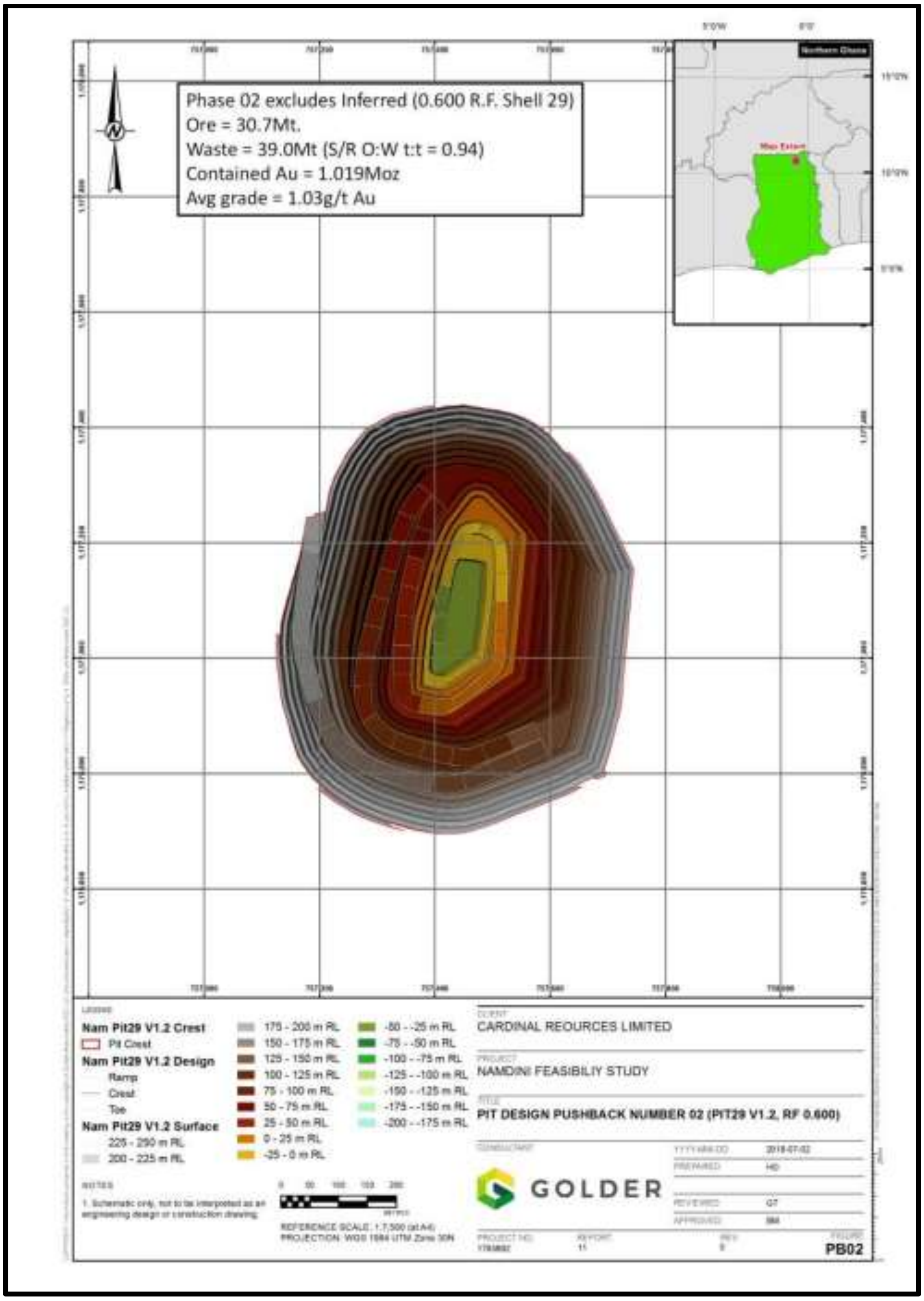


Figure 95: Design Phase 2 (*nam_29pit_v1.2.00t*)

The second stage (Phase 2) effectively expands on the initial first stage, taking and targeting a further 30.7 Mt of ore at an average head grade of 1.0 g/t Au for a contained gold content of some 1,019 koz. Phase 2 targets a final pit depth of 245 m from surface with a long axis (N-S) maximum dimension of 740 m and a short axis (E-W) maximum dimension of 610 m.

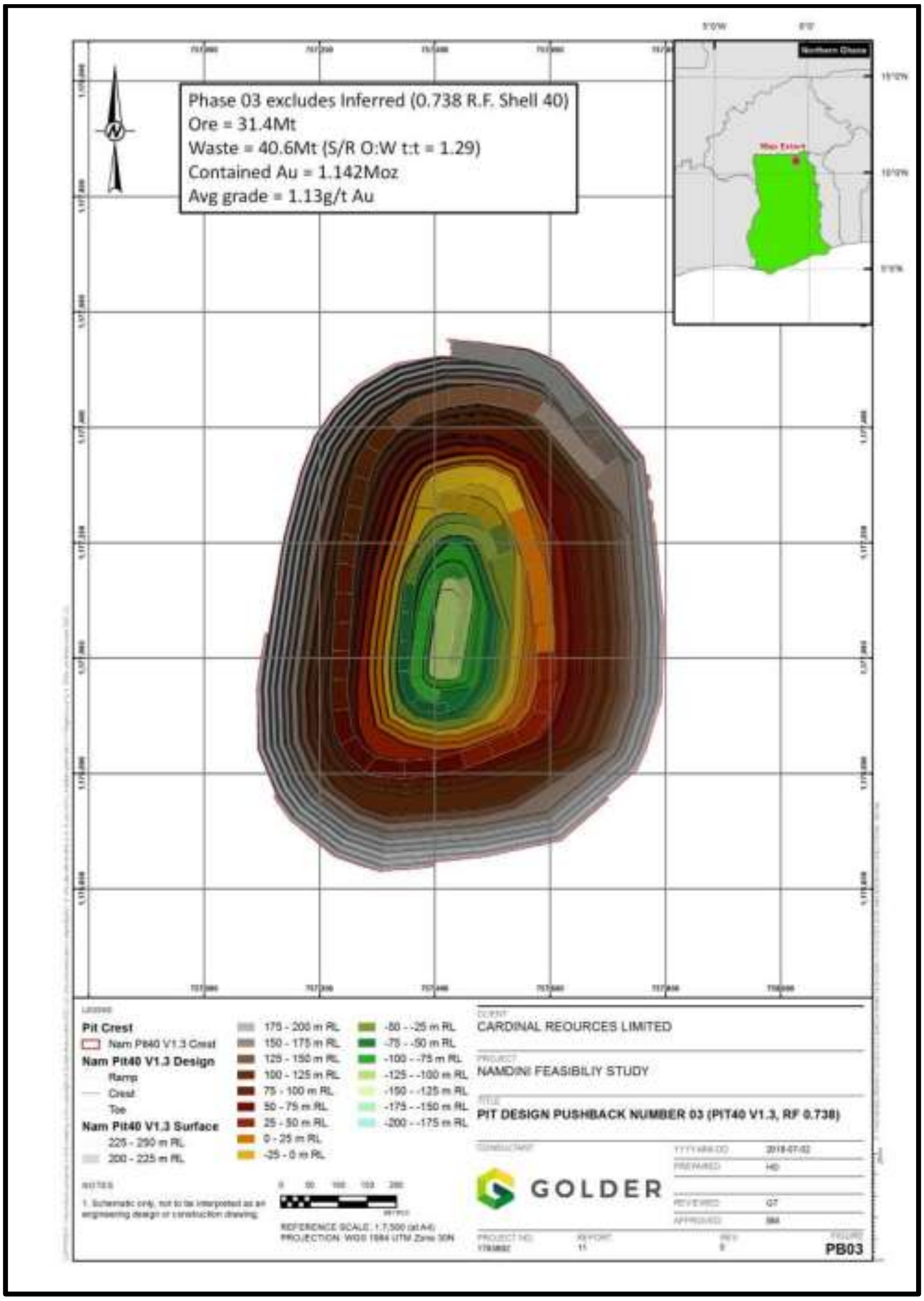


Figure 96: Design Phase 3 (*nam_40pit_v1.3.00t*)

The third stage (Phase 3) also targets a similar volume of ore (31.4 Mt) whilst requiring the movement of 40.6 Mt of waste rock. The contained gold content is some 1,142 koz with an average grade of 1.13 g/t Au. Phase 3 targets a final pit depth of 325 m from surface with a long axis (N-S) maximum dimension of 920 m and a short axis (E-W) maximum dimension of 690 m.

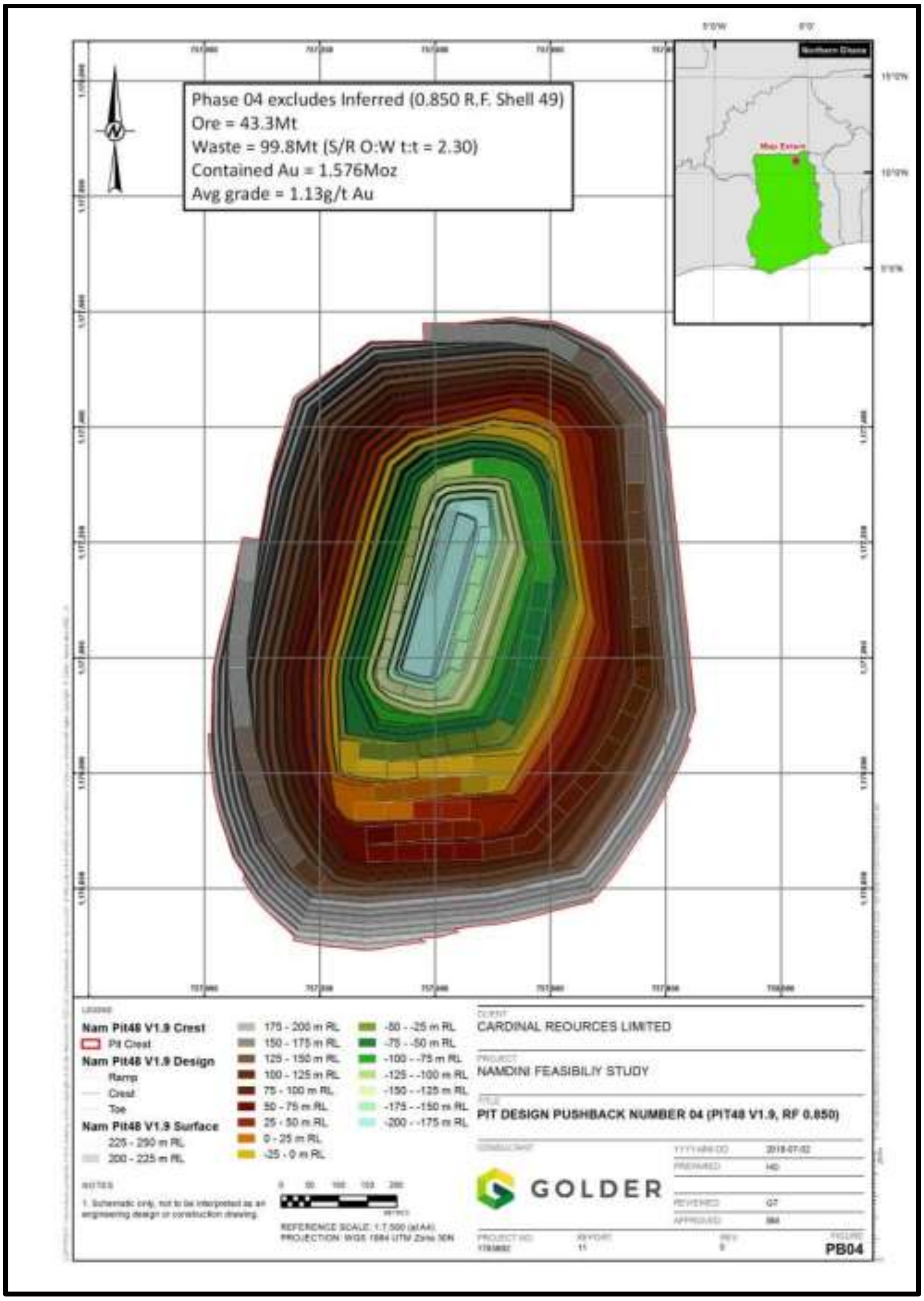


Figure 97: Design Phase 4 (*nam_48pit_v1.9.00t*)

The final stage (Phase 4) targets some 43.3 Mt of ore requiring the movement of an additional 99.8 Mt of waste rock. The increase in strip ratio is to be expected given the overall dip of the ore body to the west. The contained gold content is some 1,576 koz with an average grade of 1.13 g/t Au. Phase 4 targets a final pit depth of 410 m from surface with a long axis (N-S) maximum dimension of 1110 m and a short axis (E-W) maximum dimension of 830 m. The main access ramp for Phase 4 was planned to exit on the western side of the pit closest to the waste rock dump to reduce overall haulage costs.

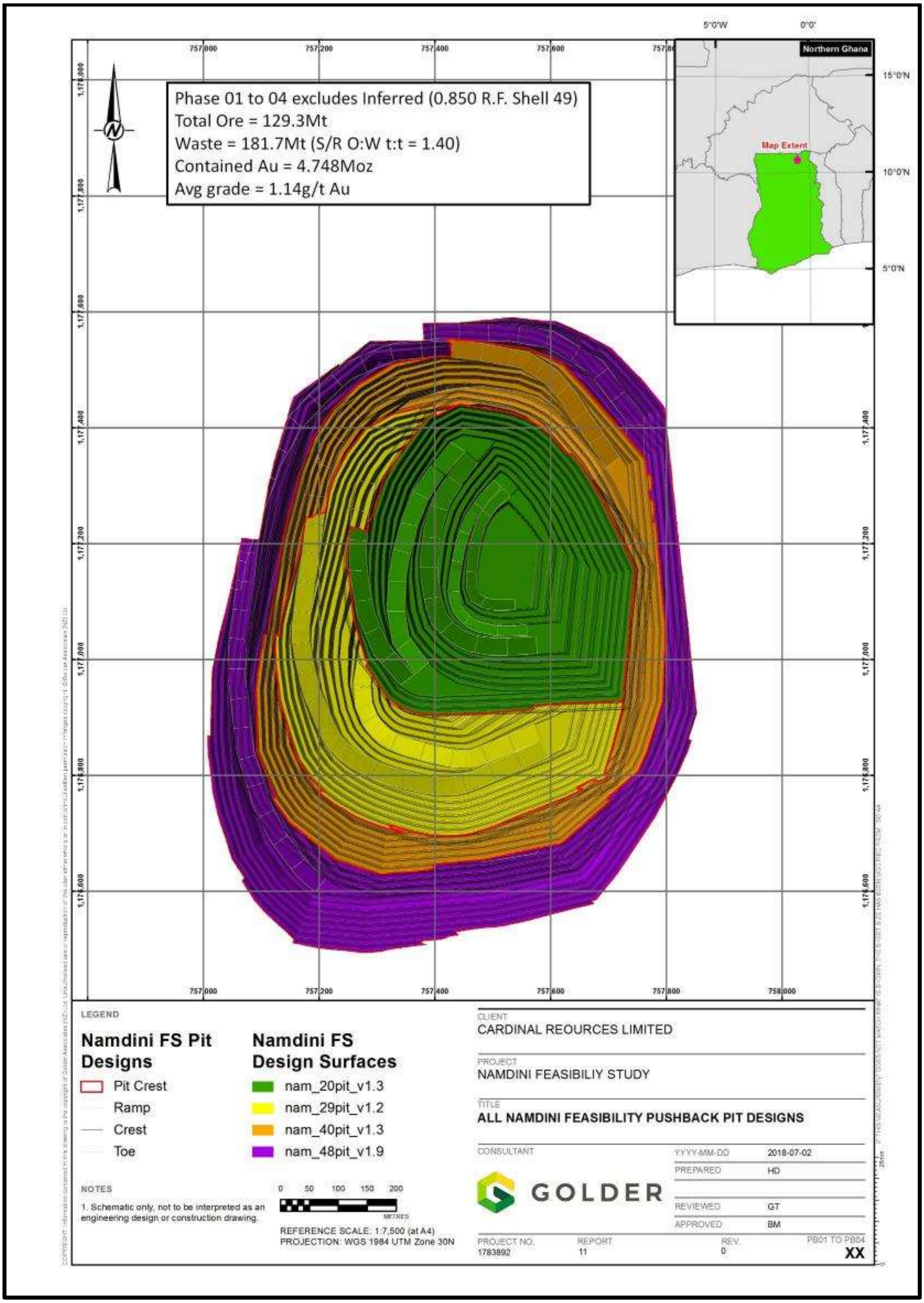


Figure 98: Design of all four pit Phases

The progression of the four planned stages can be seen in Figure 98 with the pits largely moving towards the west following the dip of the orebody.

Figure 99 shows the overlay of the four pit stages and the mineralized ore by gold grade within the pits; the ore is shown for the Indicated Resource above 0.5 g/t Au.

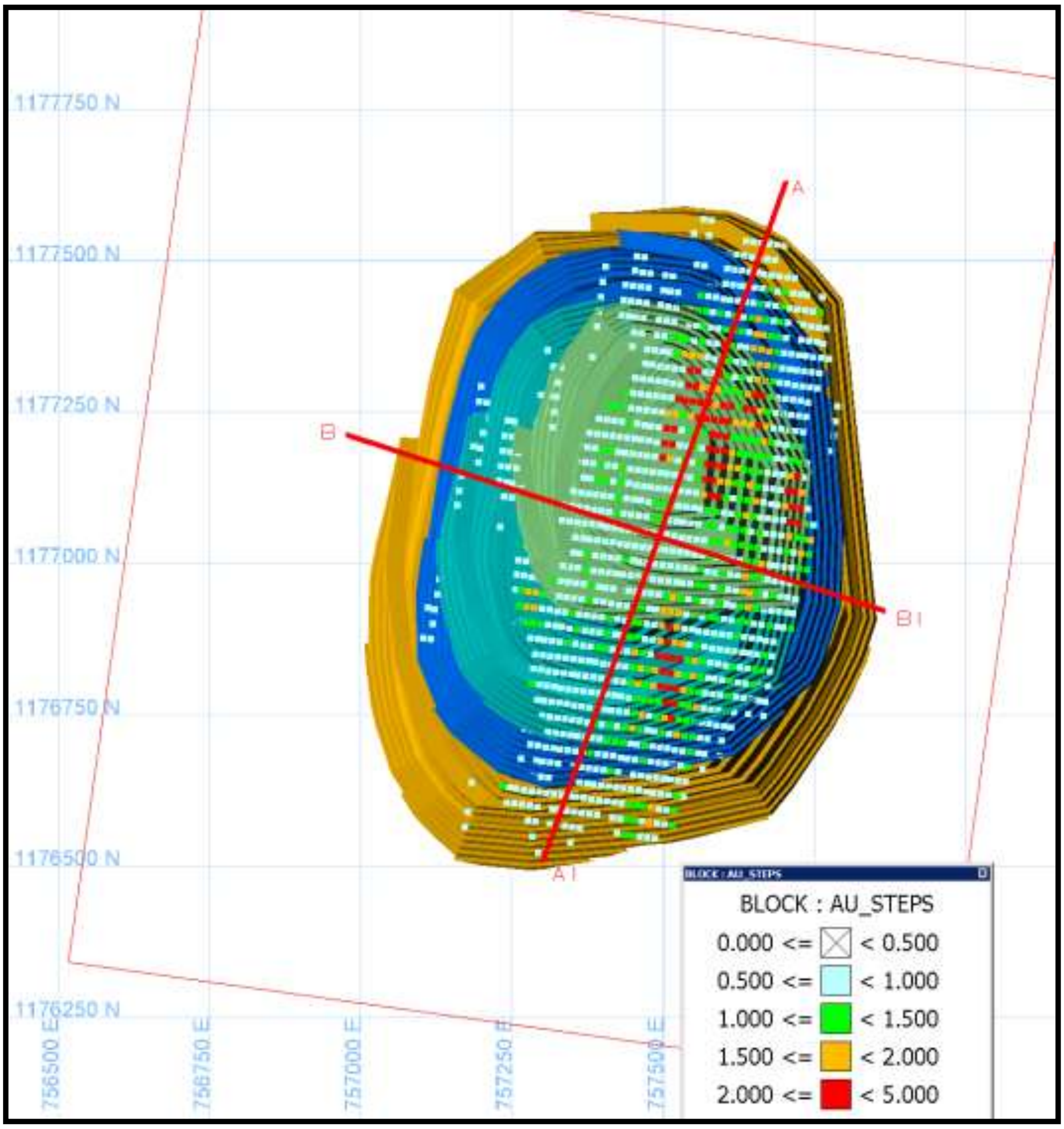


Figure 99: Plan view of pit Phases showing Indicated Resource grade above 0.5 g/t Au

Figure 100 and Figure 101 show the position of the ore within the Phases and their progression along the N-S axis and E-W axis.

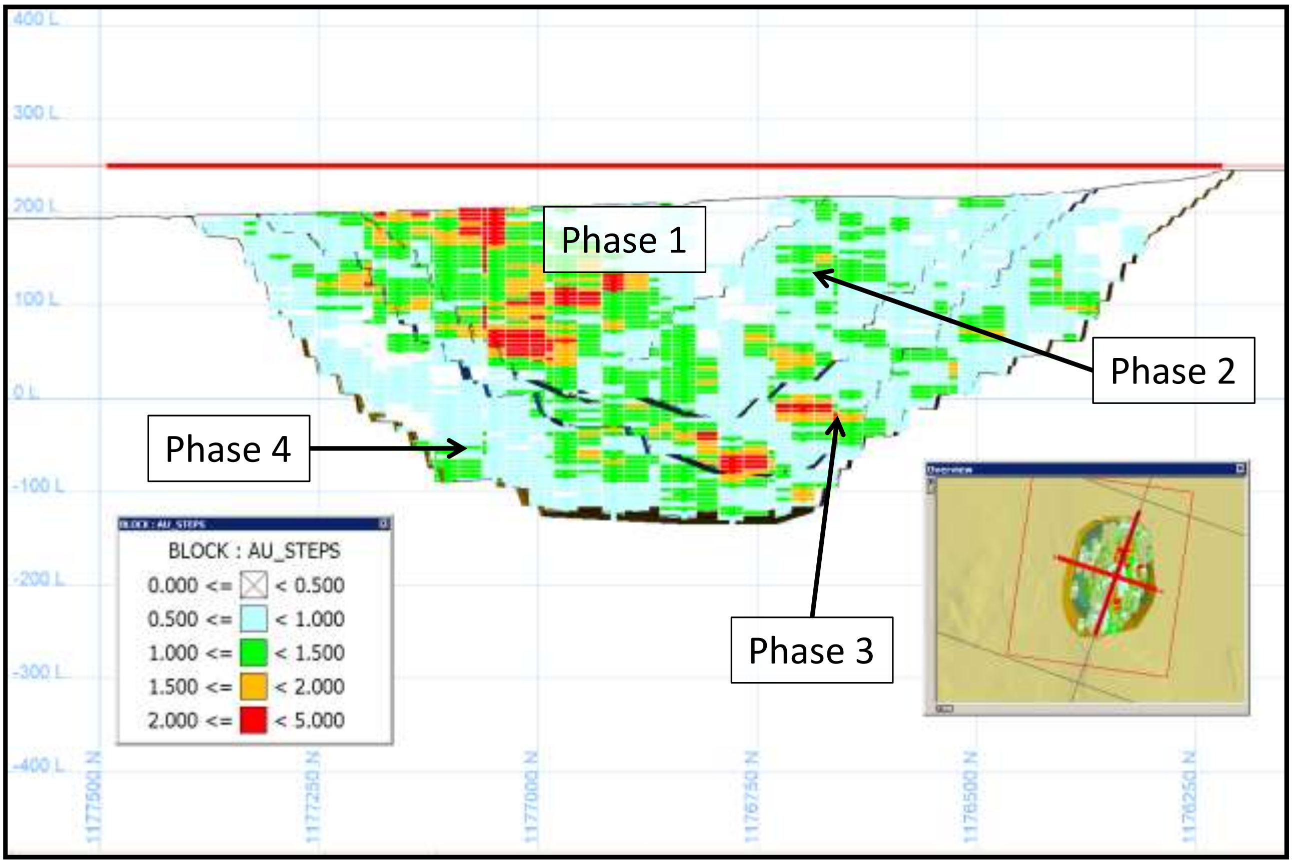


Figure 100: Section view of final pit along N-S line (A-A1) facing east

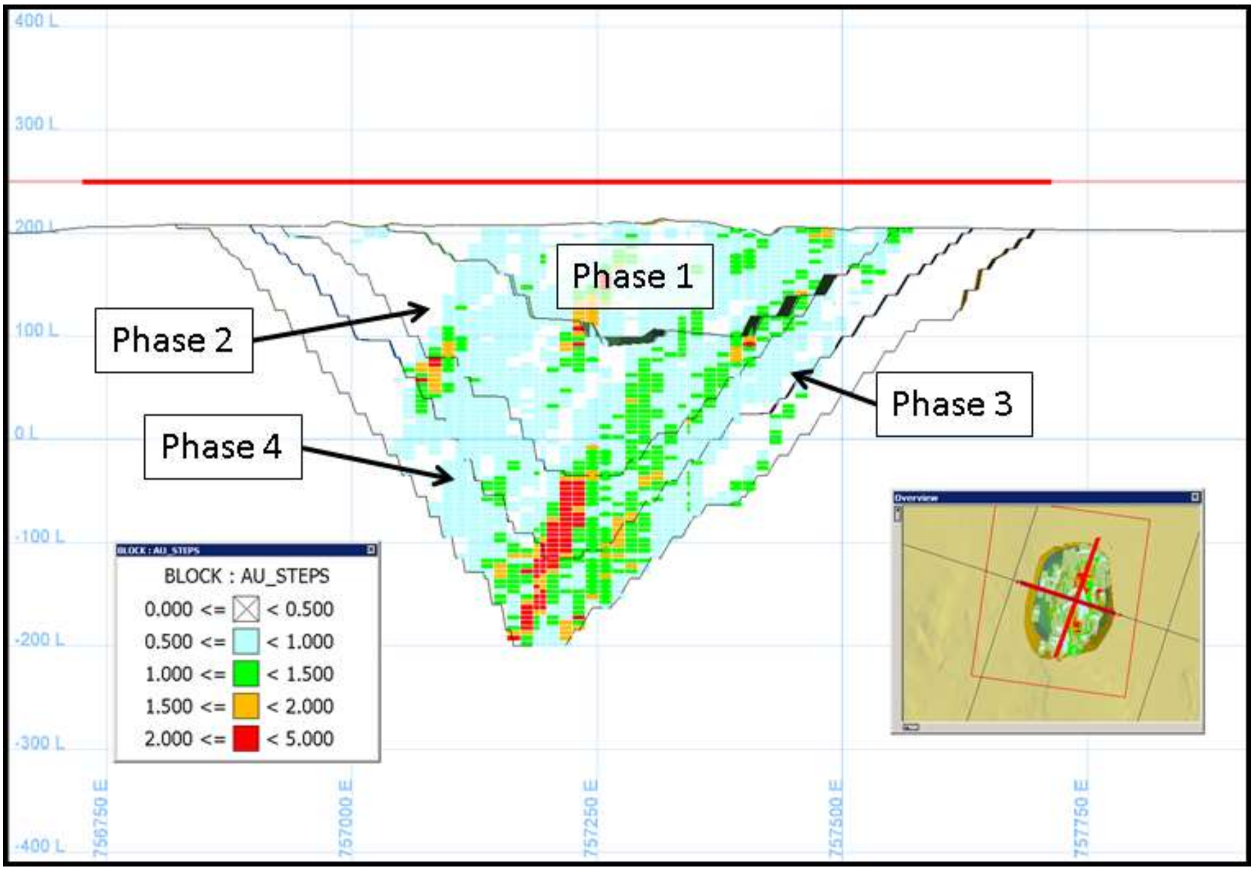


Figure 101: Section view of final pit along W-E line (B-B1) facing North, with each Phase indicated

The total ore and waste within the Ore Reserve by Phase is shown in Table 98.

Table 98: Ore Reserve by pit Phase

RF/Pit Shell	Phase	Ore* (Mt)	Waste (Mt)	S/R (W:O as t:t)	Contained Gold (koz)	Grade (g/t Au)
0.375/20	PB01	23.9	12.3	0.51	1,011	1.31
0.600/29	PB02	30.7	29.0	0.94	1,019	1.03
0.738/40	PB03	31.4	40.6	1.29	1,142	1.13
0.850/49	PB04	43.6	99.5	2.30	1,587	1.13
Total		**129.6**	**181.4**	**1.40**	**4,760**	**1.14**

Source: "Namdini_OreReserves_Scheduler_v004_Results_2018June02f.xlsx"

Notes:*ore is based on M+I only

Table 99 shows the Ore Types by Phase within the Mineral Resource; the ore represents the Indicated Resource within the stage above the 0.5 g/t Au. Any Inferred Resource material within the pit stages is regarded as waste.

The final pit was selected at a revenue factor of 0.85 using a US$1,300/oz gold price and represents the maximum discounted average value pit shell when only the Indicated resource material is considered for processing. This represents a US$1,105/oz gold price optimised pit shell, for LOM pit design.

Table 99: Ore Reserve by Phase and Ore Type

Phase	Rock (Waste + Ore) (Mt)	Ore* (Mt)	Fresh Ore (Mt)	Oxide + Transition Ore (Mt)	Strip Ratio (W:O as t:t)
Phase 1	36.2	23.9	19.9	4.0	0.51
Phase 2	59.7	30.7	28.0	2.6	0.94
Phase 3	72.0	31.4	30.2	1.2	1.29
Phase 4	143.1	43.6	43.0	0.6	2.28
Total	**311.0**	**129.6**	**121.2**	**8.4**	**1.40**

Notes: *ore is based on M+I only

The first mining Phase was selected based upon maximising value by targeting a relatively small mining area of higher average grade. Phase 2 to Phase 4 (the remainder of the pit) were selected to contain a similar volume of ore per Phase, hence a similar life for each stage.

Table 100: Ore Reserve by ore type and pit Phase

Phase	Fresh ore		Oxide/Transition Ore		All Ore		
	(Mt)	Contained gold (koz)	(Mt)	Contained gold (koz)	(Mt)	Contained gold (koz)	Grade (Au g/t)
Phase 1	19.9	856.4	4.0	154.7	23.9	1,011.1	1.31
Phase 2	28.0	926.5	2.6	92.0	30.7	1,018.6	1.03
Phase 3	30.2	1,103.4	1.2	39.0	31.4	1,142.5	1.13
Phase 4	43.0	1,571.7	0.6	16.2	43.6	1,587.9	1.13
Total	**121.2**	**4,458.1**	**8.4**	**301.9**	**129.6**	**4,760.0**	**1.14**

Source: namdini_pfs_adv_reserves_2018_may_22_v3.xlsx

The final pit contains minor Inferred Resource (0.27 Mt) that was considered waste material. No additional drilling is required to attempt to upgrade Inferred to Indicated Resource.

Further delineation and infill drilling will be carried out to improve confidence in the Indicated Resource and convert a portion of it to Measured Resource.

16.8.2 Mine layout

As the initial starter pit is coincident with the highest-grade area, this is also the area where the majority of the artisanal mining has taken place. Some minor preparation (bench levelling and ramp access) will be required as part of the starter pit formation. The existing small pit void can be seen in Figure 102.

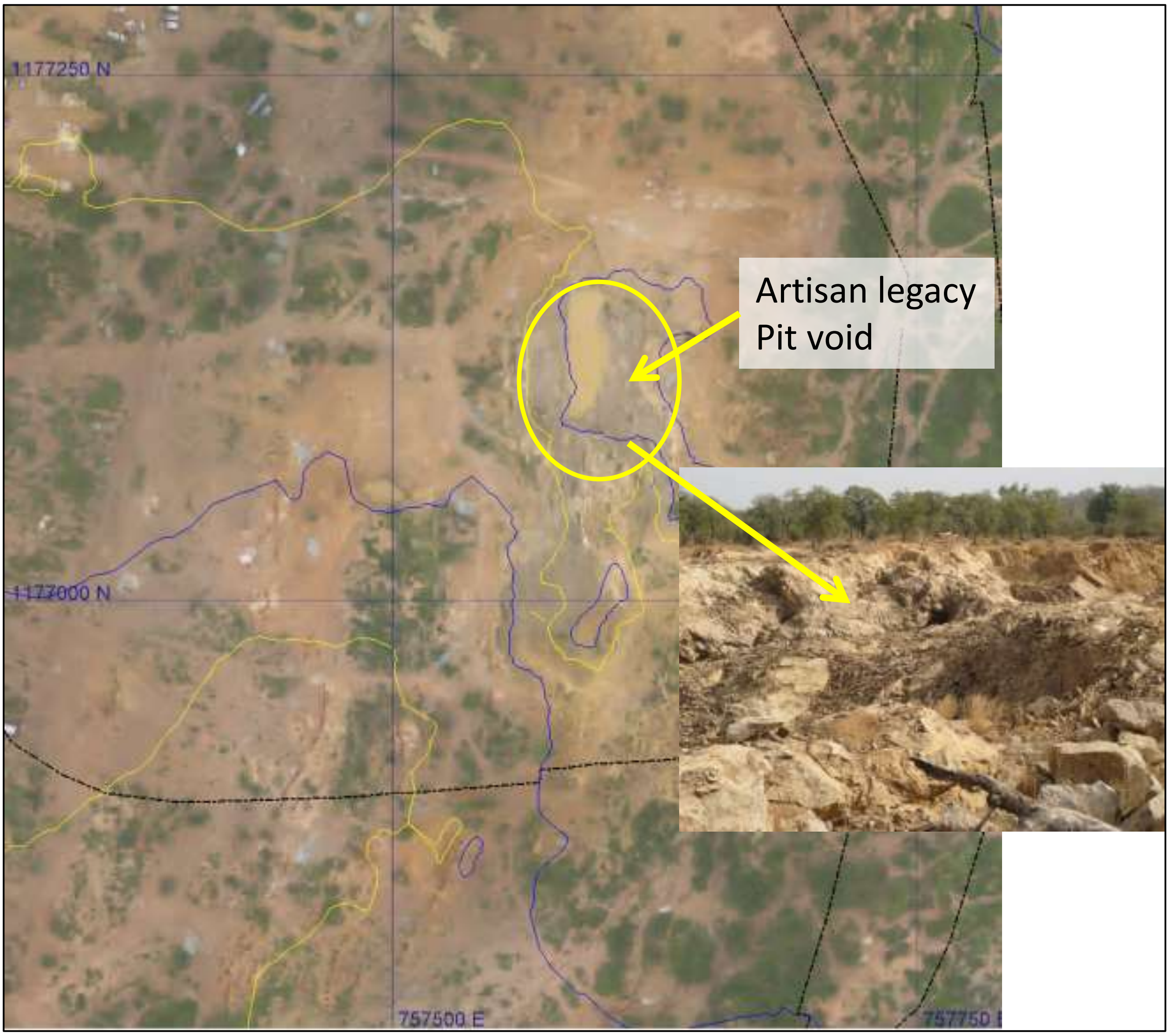


Figure 102: Planned starter pit area and remnant artisanal mining activity in 2017 (source: Golder)

The local topography in the vicinity of the designed open pit is shown in Figure 103.

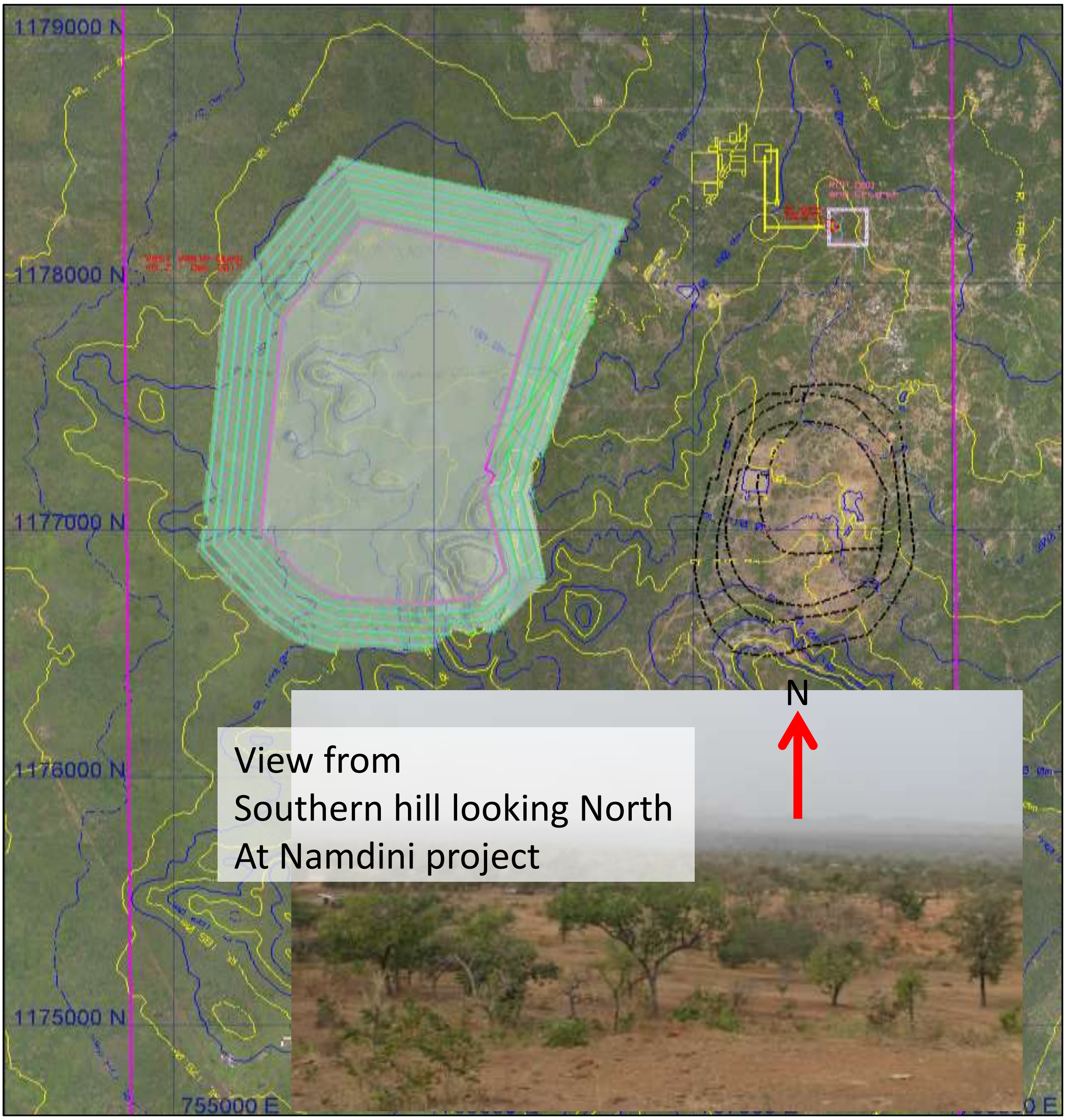


Figure 103: Planned final pit crest and waste dump location with site view at July 2018 (source: Golder)

16.8.3 Ore selectivity

The grade tonnage curve shown in Figure 104 shows the ore within the final selected pit region. The steep gradient of the tonnage indicates that most of the defined ore is within the 0.5 to 1.5 g/t Au grade range.

The grade tonnage profile for the entire Ore Reserve within the final pit designs shows that some 85% of the Ore Reserve is within a gold grade range of 0.5 to 1.5 g/t Au. Of the 129 Mt (Figure 112) of ore within the final pit design, only 20 Mt of the total is above a gold grade of 1.5 g/t Au.

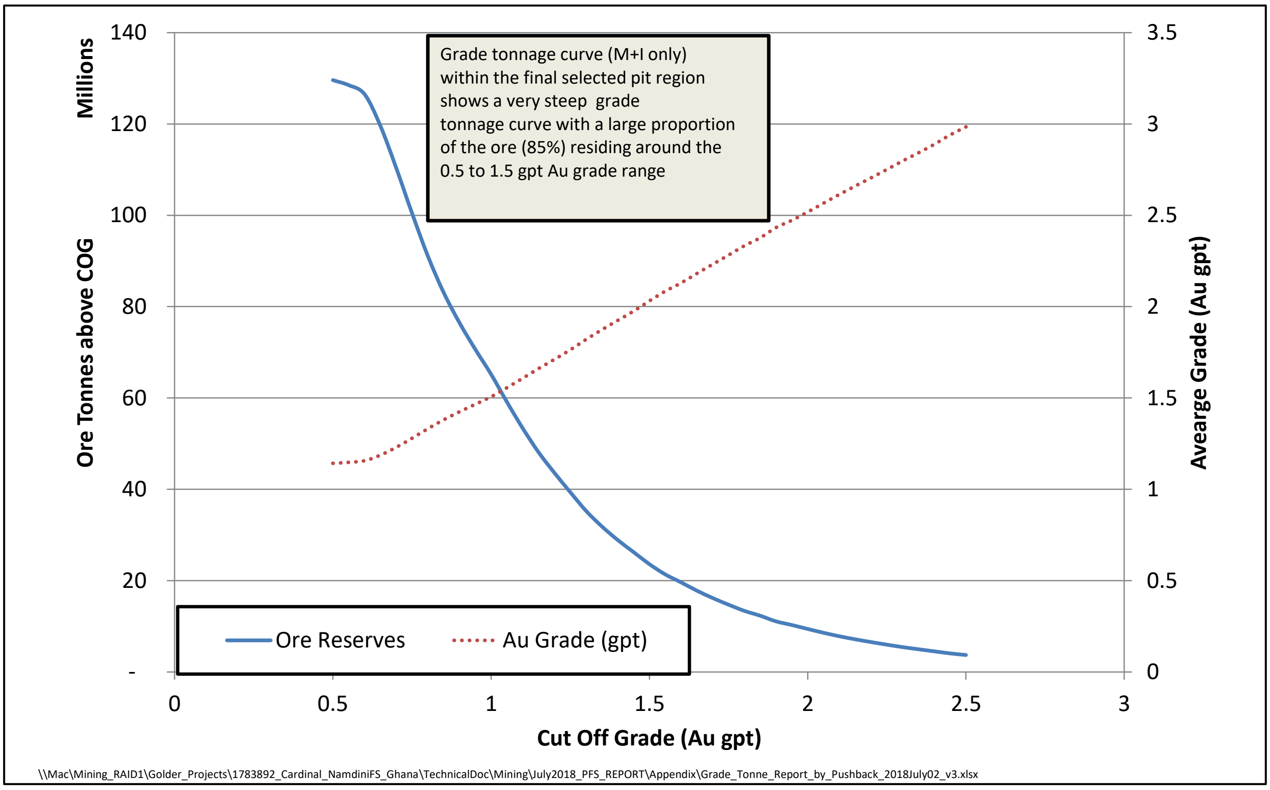


Figure 104: Grade-tonnage curve within Ore Reserve region (M+I only)

The higher-grade mineralization near surface in the initial starter pit is shown in Figure 105.

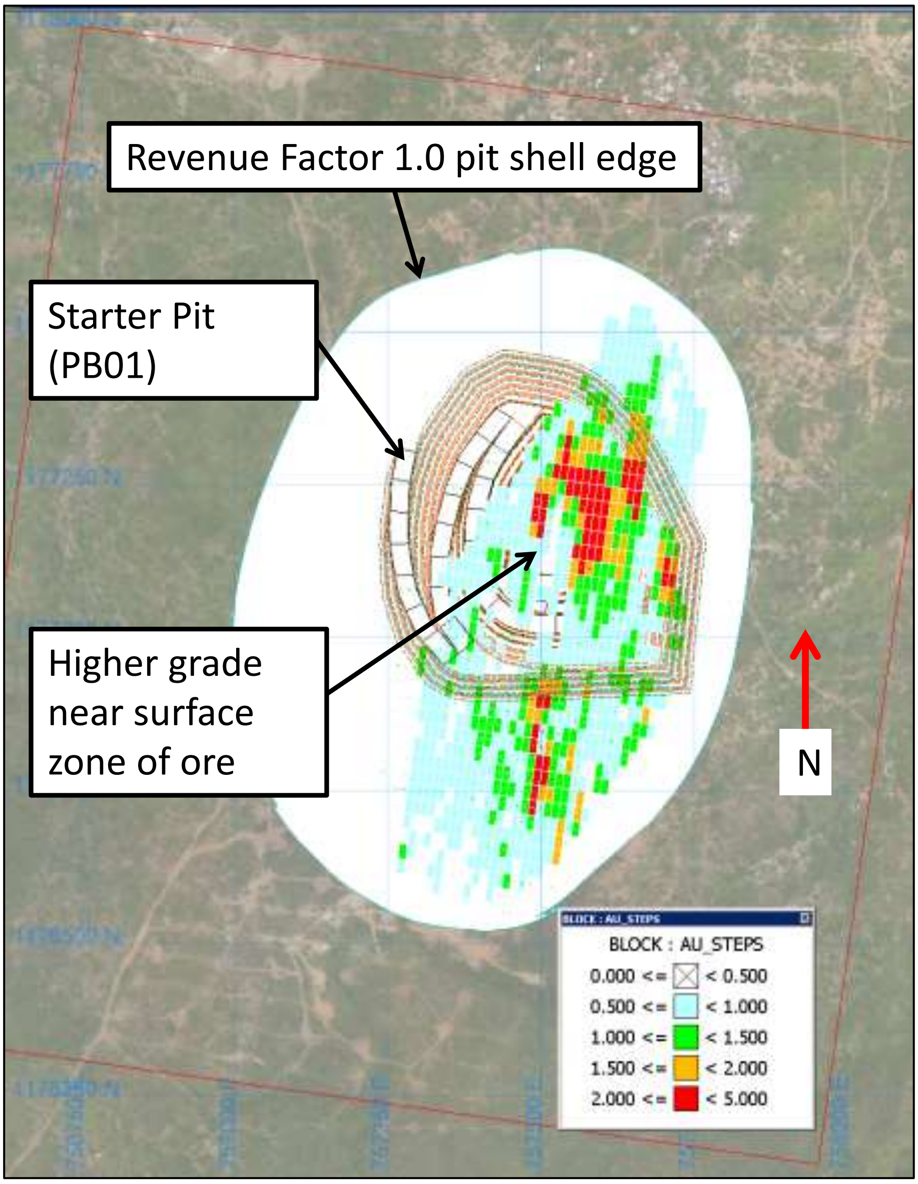


Figure 105: Plan of higher-grade zone with Starter Pit (Phase 1) and RF 1.0 pit shell

The starter pit region was selected to take advantage of one of the higher grade, near-surface zones. An analysis of the grade tonnage curve shown in Figure 106 for the Starter Pit (PB01) compared to the second pushback (PB02) verifies that there is a greater proportion of higher-grade material (i.e. ore above 1 g/t gold) in the starter pit region. The subsequent Phases, PB03 and PB04, are similar to the second Phase in that there are very limited higher-grade ore regions.

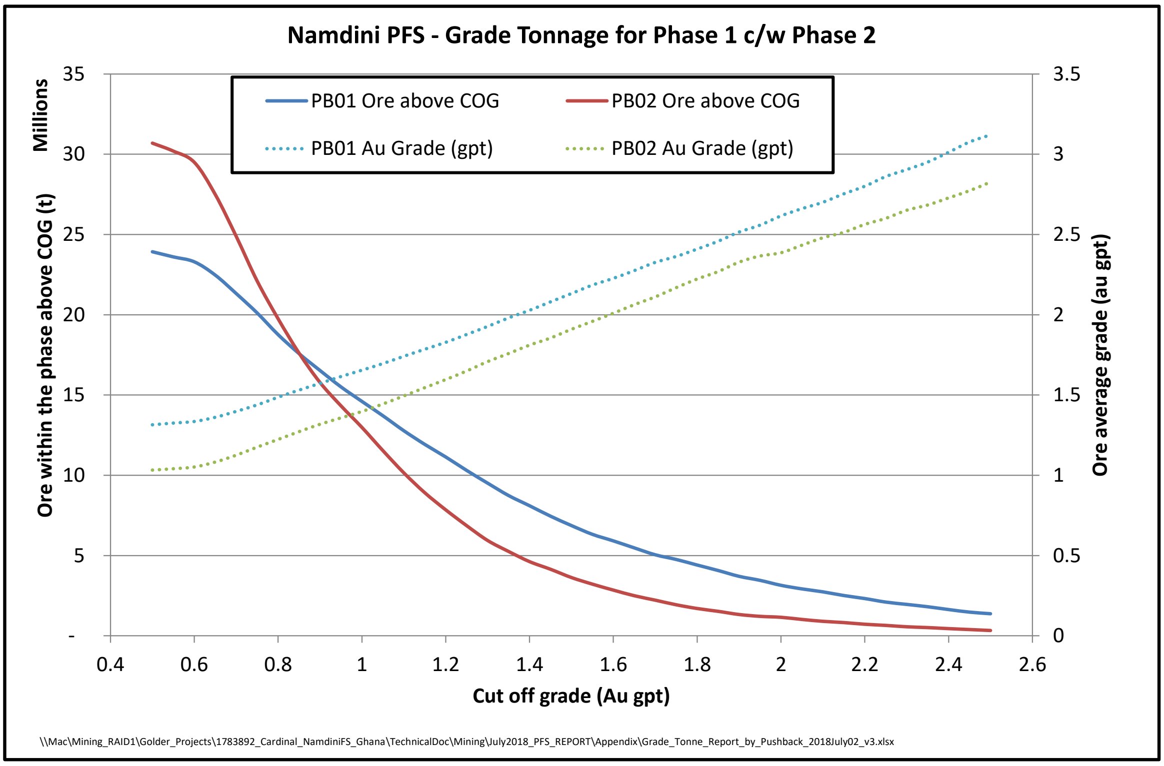


Figure 106: Grade tonnage curve for Phase 1 compared to Phase 2

The proportion of ore by Phase is plotted in Figure 107, showing that the Phase 1 (Starter Pit) region has a much flatter tonnage profile, indicating a greater potential for accessing a relatively higher-grade area within this Phase compared to the other three mining Phases.

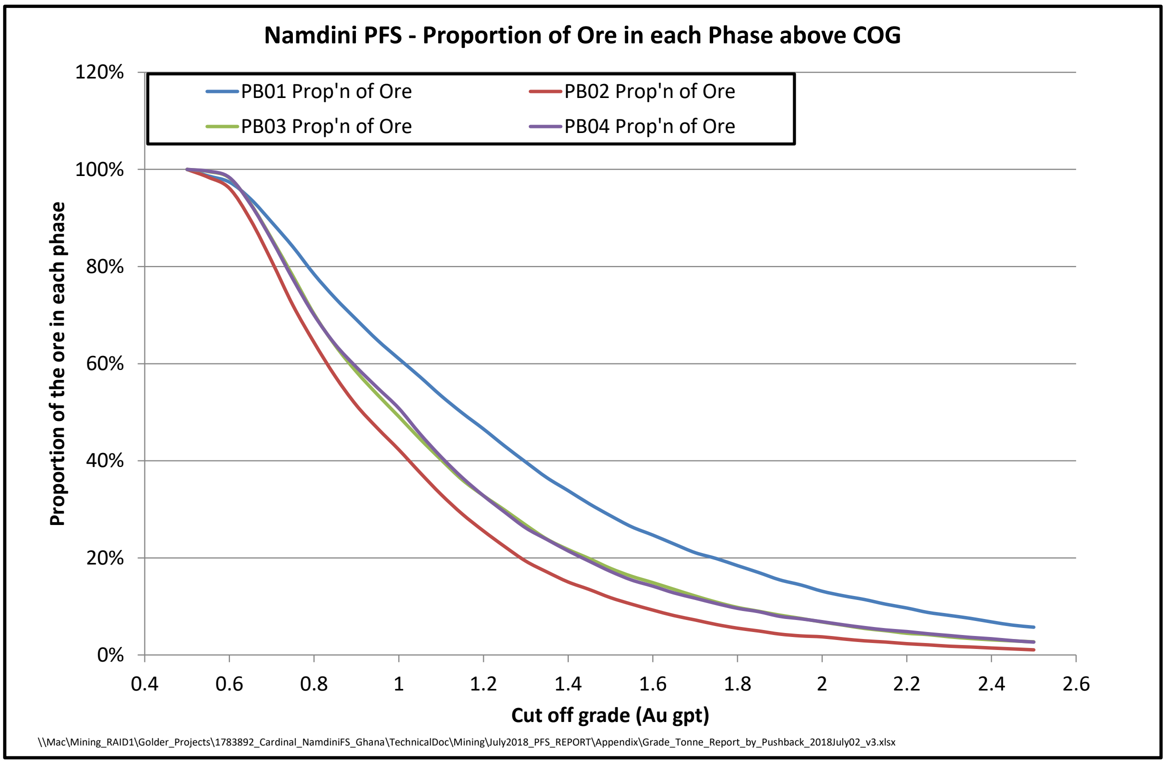


Figure 107: Proportion of ore above cut-off grade by Phase within the final pit

The grade tonnage curves for all four Phases are shown below in Figure 108 to Figure 111 below, the first Phase (PB01) presents a notably flatter tonnage curve than the three subsequent Phases.

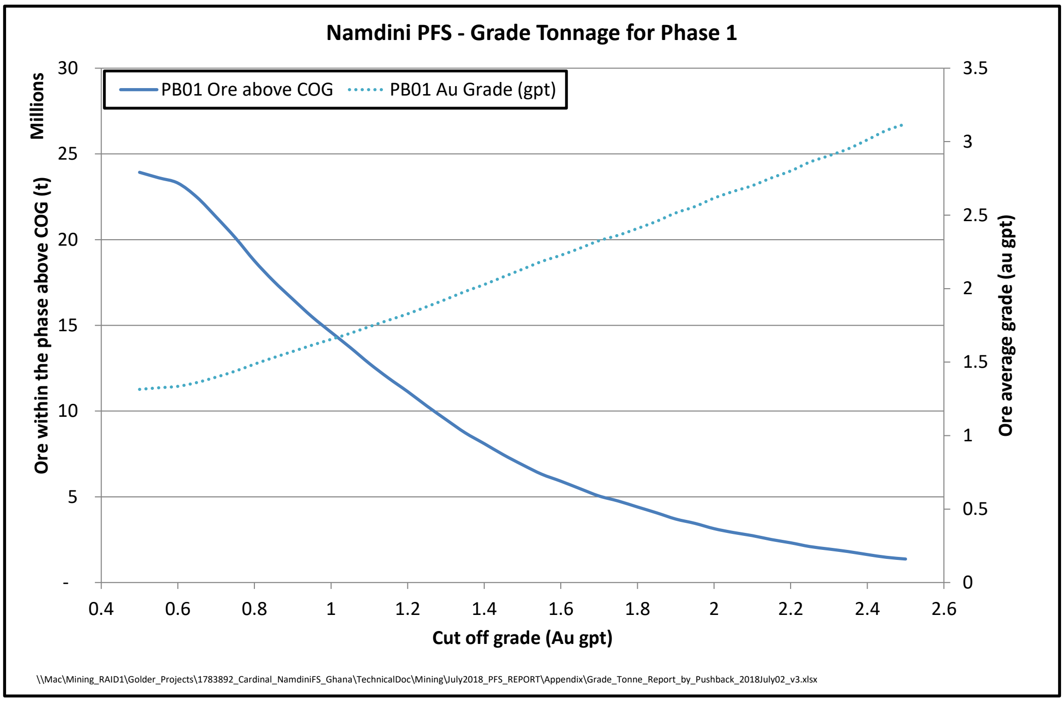


Figure 108: Grade tonnage curve (M+I only) within Phase 1 (PB01, Starter Pit)

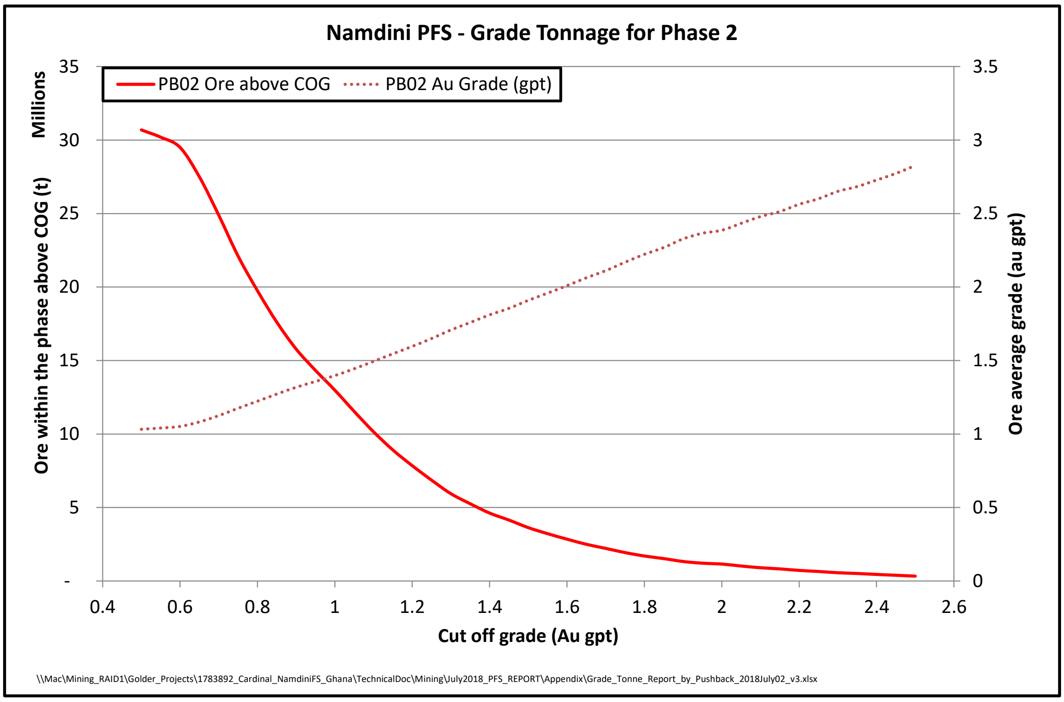


Figure 109: Grade tonnage curve (M+I only) within Phase 2 (PB02)

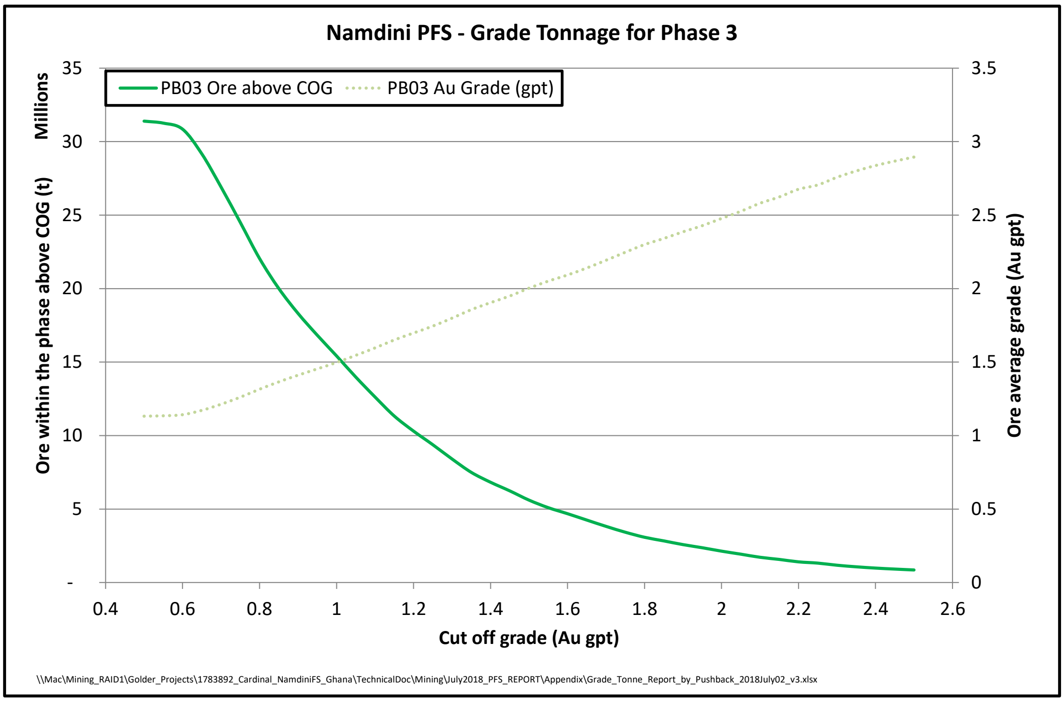


Figure 110: Grade tonnage curve (M+I only) within Phase 3 (PB03)

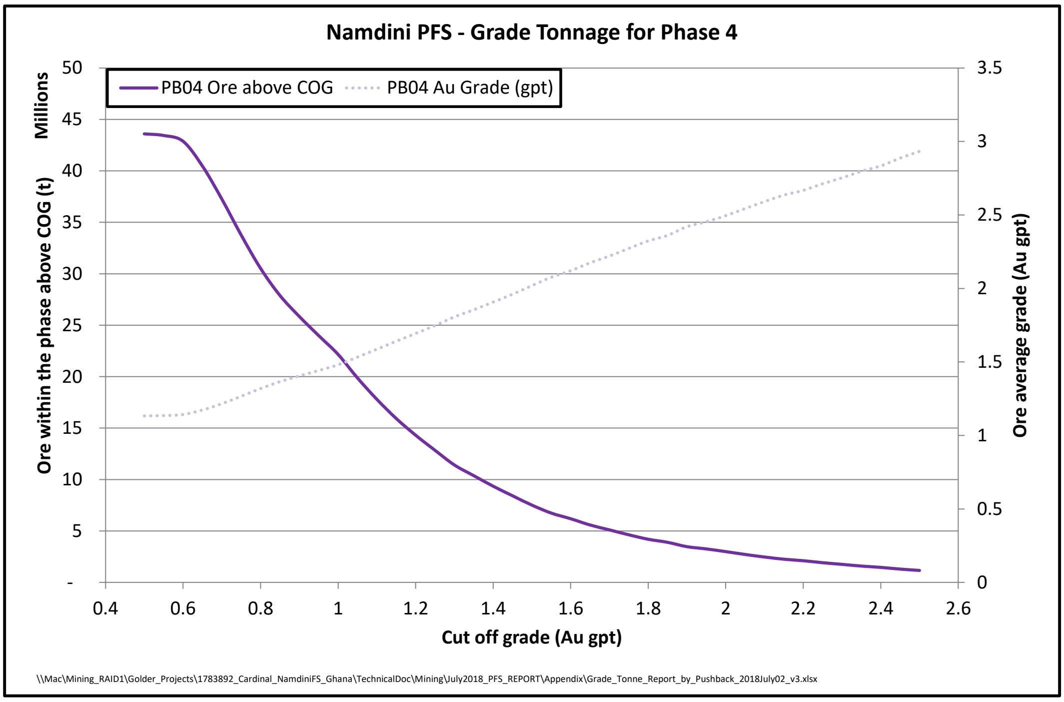


Figure 111: Grade tonnage curve (M+I only) within Phase 4 (PB04)

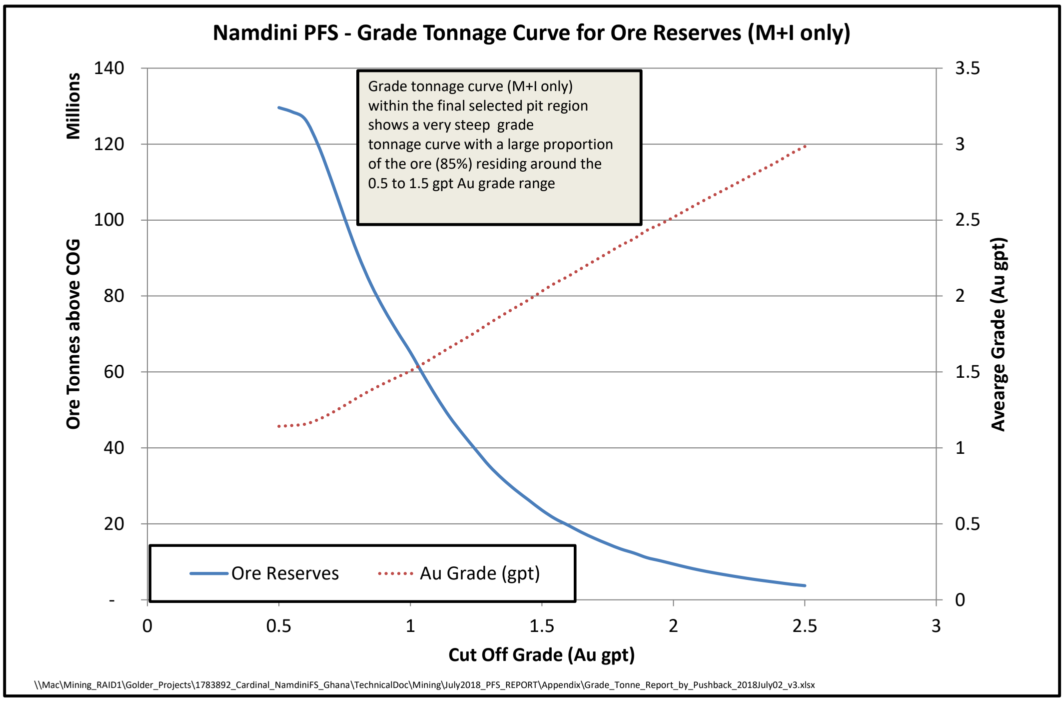


Figure 112: Namdini PFS – Grade-tonnage curve (M+I only) within final pit design

The Namdini deposit is a largely diffuse orebody with limited higher-grade zones.

It is evident that the Namdini Gold Project will be most suited to a bulk mining and processing approach which maximizes the mined tonnage and mill feed above the cut-off grade. Opportunities for increased mining selectivity during grade control will help maximize revenue by prioritising higher grade ore into the mill when there is spare capacity.

Bulk mining will also help minimize dilution and ore loss during mining recovery to ensure that all available metal is recovered. The diffuse nature of the mineralization lends itself to bulk mining with minimal loss expected at the edge of the marked-out ore zones during mining.

In Golder's view the potential for mining selectivity is markedly less than has been seen in some similar massive gold deposits.

The orebody presents limited opportunity for mining selectivity of higher-grade zones outside the targeted Starting Pit area.

The project area is generally flat with a modest rise to the southern extent of the planned pit; a view looking from the south towards the planned pit area can be seen in Figure 103 below.

16.8.4 Haul roads, stockpiles and waste dumps

Haul roads on the Namdini site consist of surface roads, in-pit ramps, and ramps for stockpile and waste storage access. All roads are designed using the same two basic parameters: the road width, which must allow safe passage of two haul trucks travelling in opposite directions on the road; and the gradient, which must be shallow enough to allow the largest of vehicles to traverse them safely.

The operation tonnages appear suited to a 150 t class truck (Cat 785 class or equivalent). All roads on site are designed with a 32 m width to provide a minimum width greater than or equal to 3.5 times the expected maximum size truck width. Roads are also designed with a maximum grade of 10% (1:10).

The position of the proposed waste dump on the western side of the final pit is shown in Figure 113.

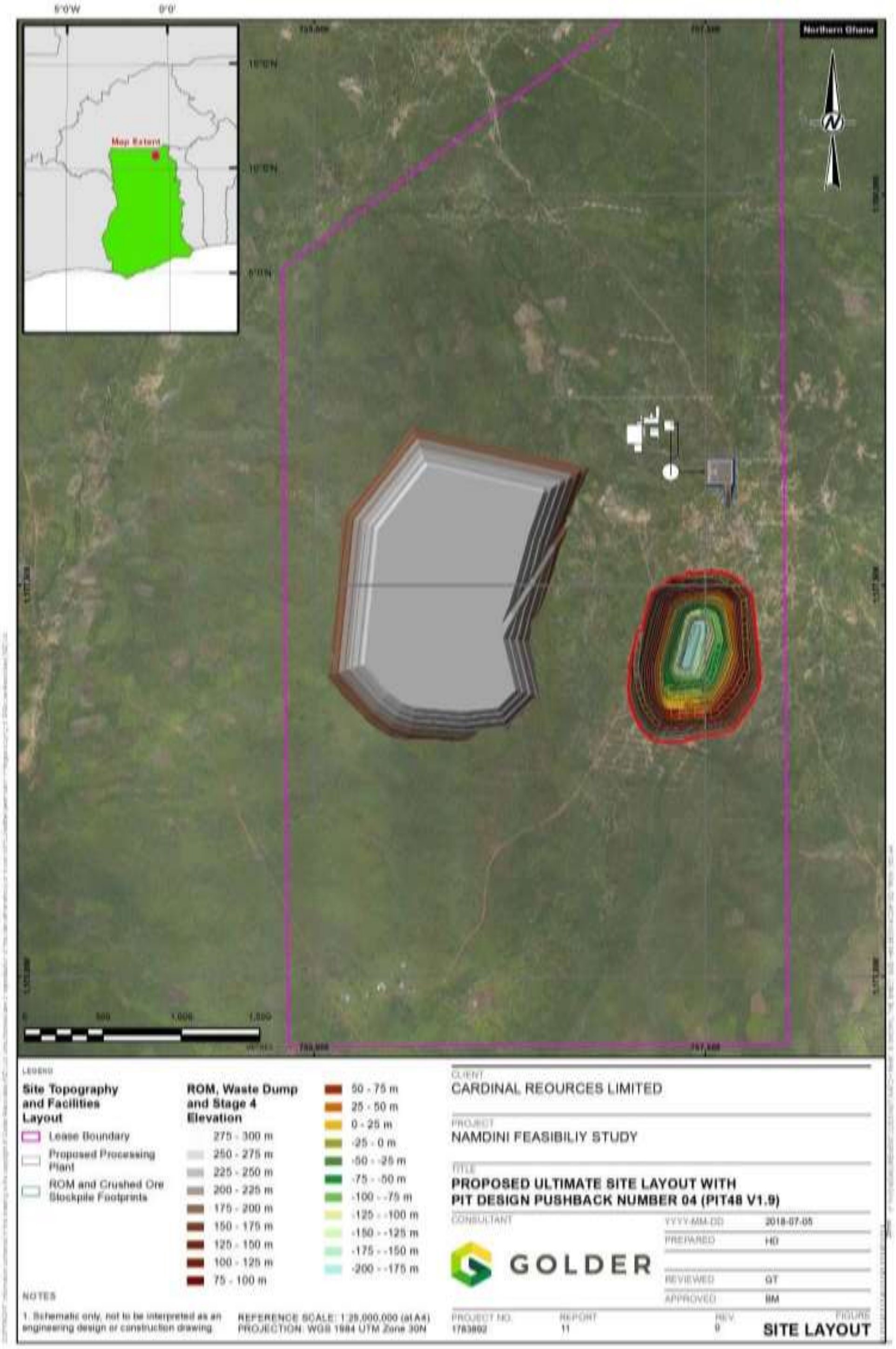


Figure 113: Pit, waste dump and Process Plant location at July 2018

16.8.5 Excavation assumptions

To date no trials have yet been completed to determine the large-scale dig-ability of the material types within the proposed pit. In the absence of specific data, the following assumptions were applied:

- In the Oxide and Transition material a mixture of free-digging, ripping, drilling and blasting methods will be used
- In the Fresh competent material at depth conventional pre-splitting, drilling and blasting will be used to extract the ore and waste.

No studies of topsoil quantities have been completed and for this PFS the volumes and costs for removal are included in those applied to removal of waste materials.

16.9 Grade control trial

The Mineral Resource MIK model contains an estimate at various grade thresholds of the proportion of ore and waste. The estimation method allowed for moderate selective mining by adjusting the variance. It does not however model the expected spatial position of partial ore blocks or the nature of the ore boundaries.

During mine scheduling it is normal practice to add additional factors to account for unavoidable ore loss and dilution during grade control, ore block design and for mining recovery. Modelling expected 'edge dilution' is not appropriate on an MIK model.

In November 2017, Cardinal completed close-spaced drilling at Namdini to assess the representativity of the Mineral Resource Model and to trial the proposed grade control approach being developed for the mine. Close-spaced RC drilling in the initial starter pit area totalled 13,271 m in 317 holes on a 10 by 15 m grid, covering an area of 350 m (E) by 200 m (N) to a vertical depth of approximately 40 m. The holes were drilled at 65º to the east consistent with the resource drilling an orebody geometry.

A summary is provided in Table 101 (Cardinal, 2017). This test grade control drilling programme confirmed that the Mineral Resource model using the MIK estimation method is a valid representation of the mineralization in the upper portions of the deposit.

Table 101: Comparison between Mineral Resource model and Grade Control Model (Cardinal, 2017)

Resource Model (September 2017)				Grade Control Model			Difference
Cut-off (g/t Au)	Tonnes (Mt)	Grade (g/t Au)	Metal (koz Au)	Tonnes (Mt)	Grade (g/t Au)	Metal (koz Au)	Metal (koz Au)
0.3	4.05	1.4	183	4.04	1.4	183	0
0.5	3.44	1.6	176	3.43	1.6	175	1
0.7	3.00	1.7	168	2.89	1.8	165	3

Given the close correlation between the Mineral Resource Model estimate and the trial grade control results, Golder considers the Mineral Resource a valid representation of the expected mineable ore for the near-surface mineralization.

16.10 Mining schedule

The mine scheduling used the Minemax Scheduler software with annual periods. The mining sequence was designed to incorporate the four mining Phases. The initial mining Phase targets a relatively small initial pit of higher-grade ore with minimal waste stripping requirements (the Starter Pit). The overall mining schedule is targeted towards the early generation of cashflow to assist in repayment of the project capital and to maximize project discounted value.

A plant throughput of 7.0 Mtpa (the Mid Case) results in an 18-year mine life including ramp up. The ramp-up was assumed to cover a three-month period within year one when it is planned to treat some 90% of annual throughput. An initial pre-strip period was incorporated within the schedule to allow access to the Fresh ore within the starter pit and the stockpiling of the overlying Oxide portion in the starter pit.

Two additional schedules considered the operation of a Low Case (4.5 Mtpa) plant throughput and a High Case (9.5 Mtpa) plant throughput, which resulted 20 years and 13 years respectively of mine life.

The schedule was constrained to a maximum vertical sink rate of 80 m per year, with the mining Phases set to honour a minimum two bench lead during any period. A summary of the bench sink rate showing the number of 10 m benches per year per Phase is shown Table 102, confirming that the vertical sink rate limit of 80 m a year is not exceeded.

Table 102: Bench sink rate for 10 m benches (Mid Case)

Year	Phase 1	Phase 2	Phase 3	Phase 4
2020	4			
2021	4			
2022	5	3		
2023	7	2		
2024	1	5		
2025		6		
2026		7		
2027		8	7	
2028			8	
2029			8	5
2030			6	4
2031			6	4
2032			4	6
2033				5
2034				7
2035				6
2036				5
2037				7
2038				4

The schedule targeted the ore above the marginal COG (0.5 g/t Au), but used the maximum recoverable value to prioritise process plant feed. Lower grade ore was stockpiled while there was not excess plant capacity.

The scheduler was set to maximize NPV whilst maintaining a reasonable level of smoothing of tonnage movement per year. The peak tonnage requirements over the period 2027 to 2034 are a result of the higher strip ratio in the final mining stage (Phase 4). The initial mining stage (Phase 1) is mined from 2020 to 2024, with the second stage (Phase 2) commencing in the latter part of 2022 to ensure process plant feed.

Figure 114 shows the total Rock movement by Phase over the life of mine for the Mid Case. Similar profiles were evident for the Low and High Cases. The total required peak movement in the High Case was up to 36 Mtpa in 2026. Very high movement tonnages were required over a four-year period (2026 to 2029) at greater than 35 Mtpa, a level that is considered impractical for the size of the pit.

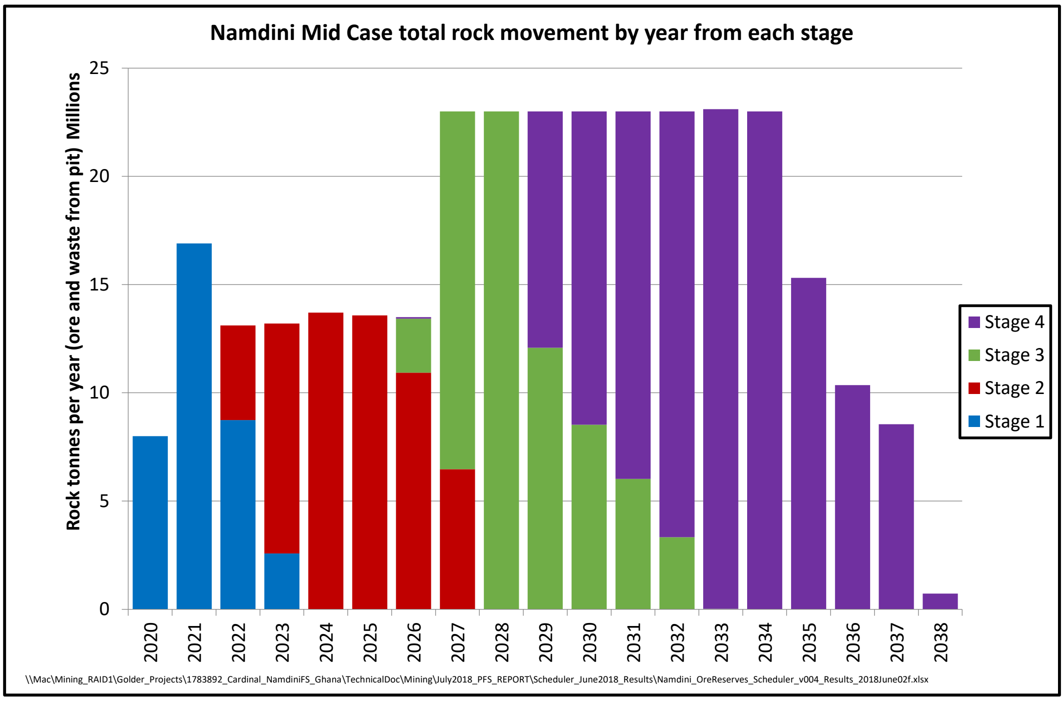


Figure 114: Mid Case total rock movement by year

The material type mined per year (Figure 115) shows that the Oxide material is predominantly mined in the first two mining Phases. Some 80% of the total Oxide material is mined in the first four years and will be stockpiled to allow batch feeding of Oxide ore to the process plant.

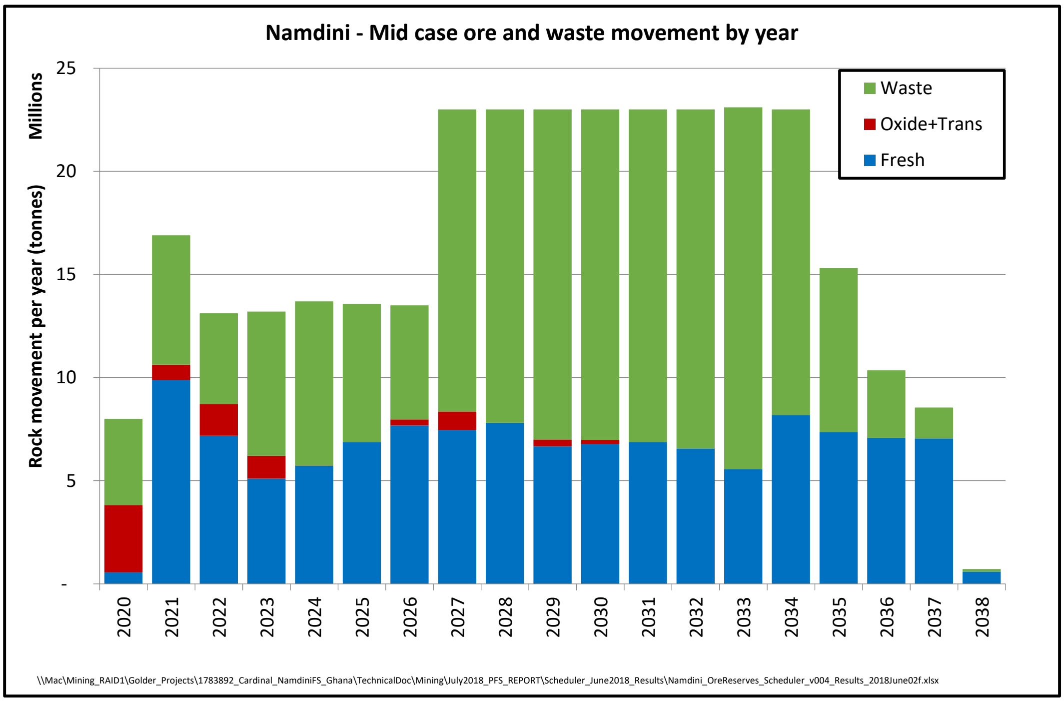


Figure 115: Mid Case ore and waste movement by year

The Oxide feed to the process plant was limited to less than 10% of total feed on an annual basis after the first year of commissioning. Ore throughputs for the process plant were estimated by material type allowing an annual tonnage to exceed the notional 7 Mtpa depending on the ore blend during that year. The ore feed by material type for the Mid Case is shown in Figure 116.

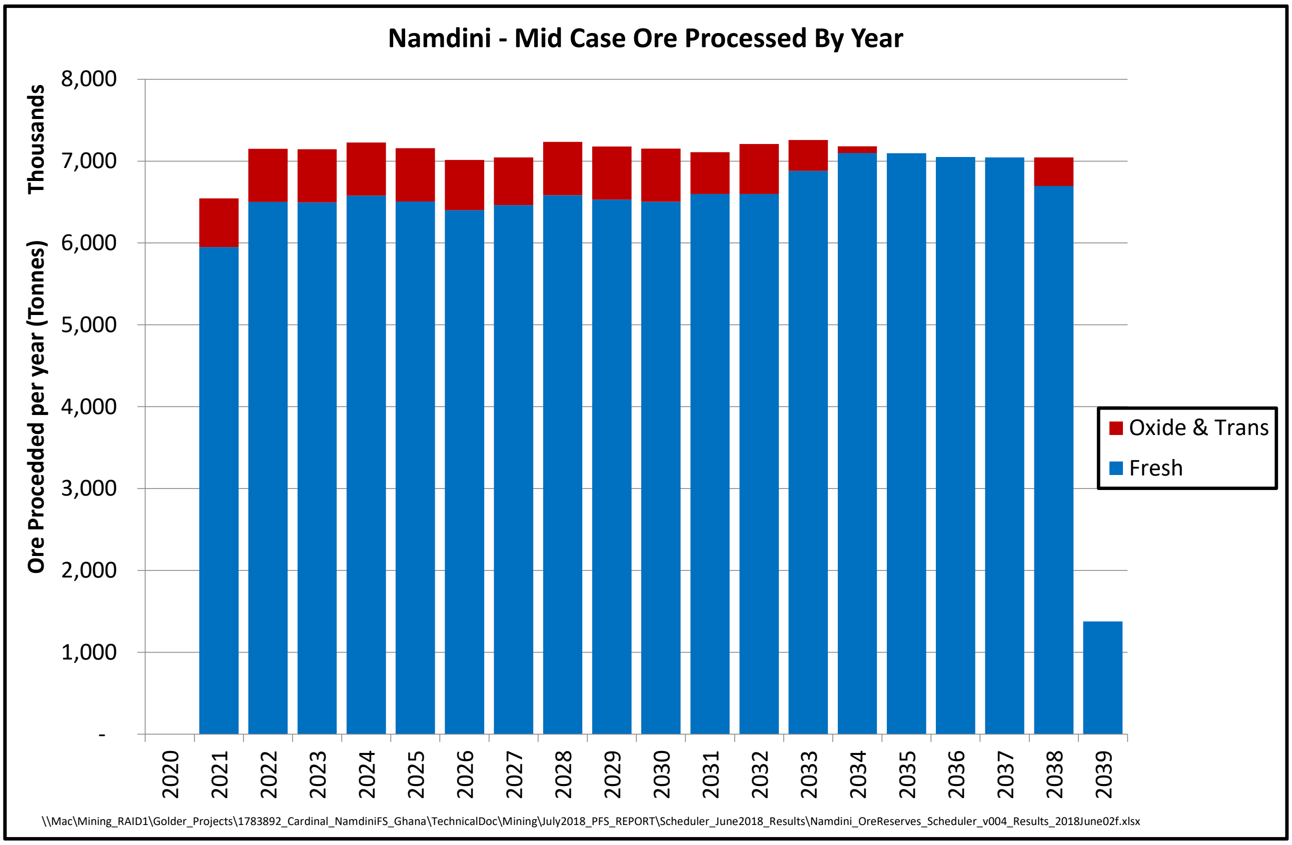


Figure 116: Mid Case ore feed by material type, by year

The pit was also scheduled by lithology to determine plant throughput rates but these results are not presented here.

16.11 Mining sequence and period progress

Period progress plots for the Mid Case were produced showing the general progression of the mining Phases. The initial Phase (PB01) will commence in the region of highest grade near-surface ore. This will require the removal and stockpiling of the Oxide and Transition material to allow access to the Fresh ore for plant feed. The first Phase will last approximately three years, with commencement of the second Phase (PB02) occurring two years after the start of mining operations. Figure 117 shows the first and second Phases for the Mid Case.

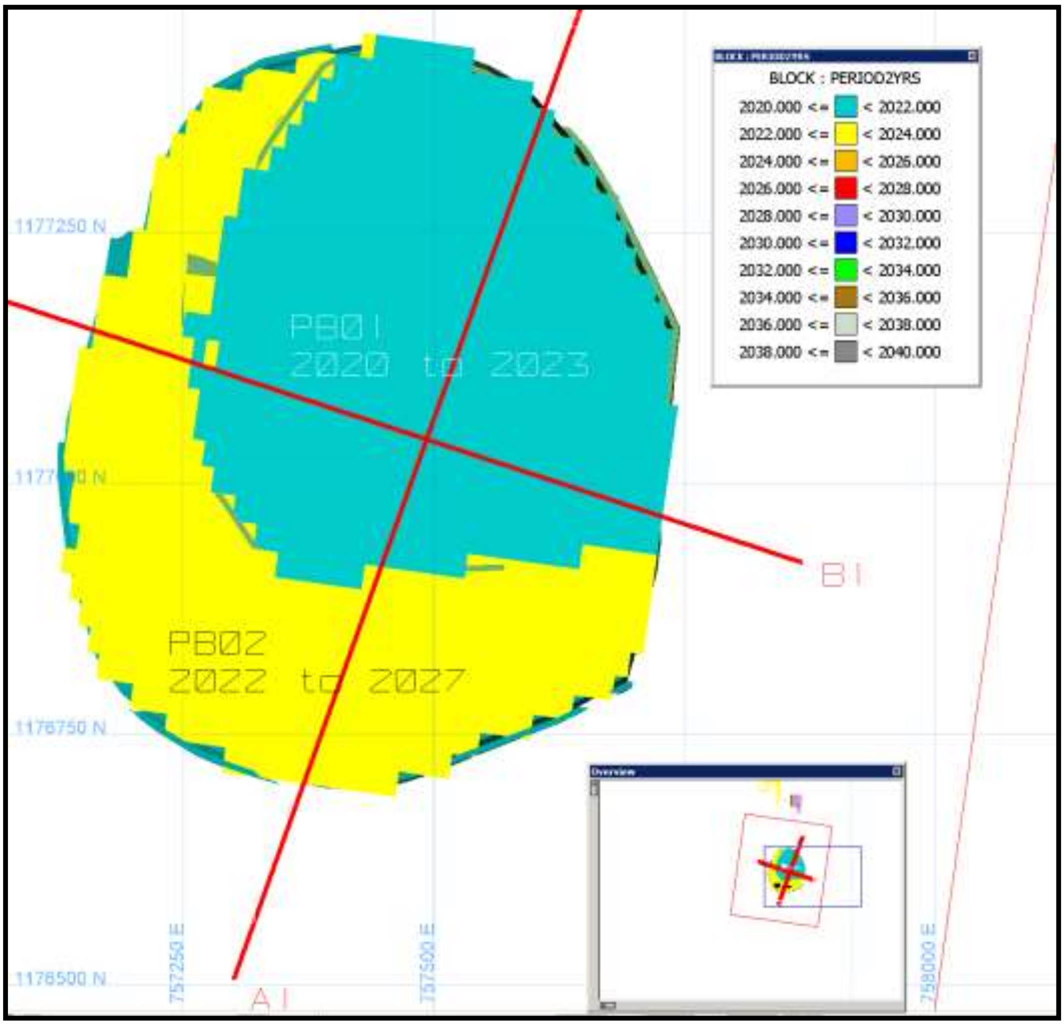


Figure 117: Period progress in Phase 1 (PB01) and Phase 2 (PB02)

Mining of Phase 3 (PB03) will commence 7 years after the start of the mining operations, lasting 5 years. Figure 118 shows the general timing using the Mid Case in the first three mining Phases.

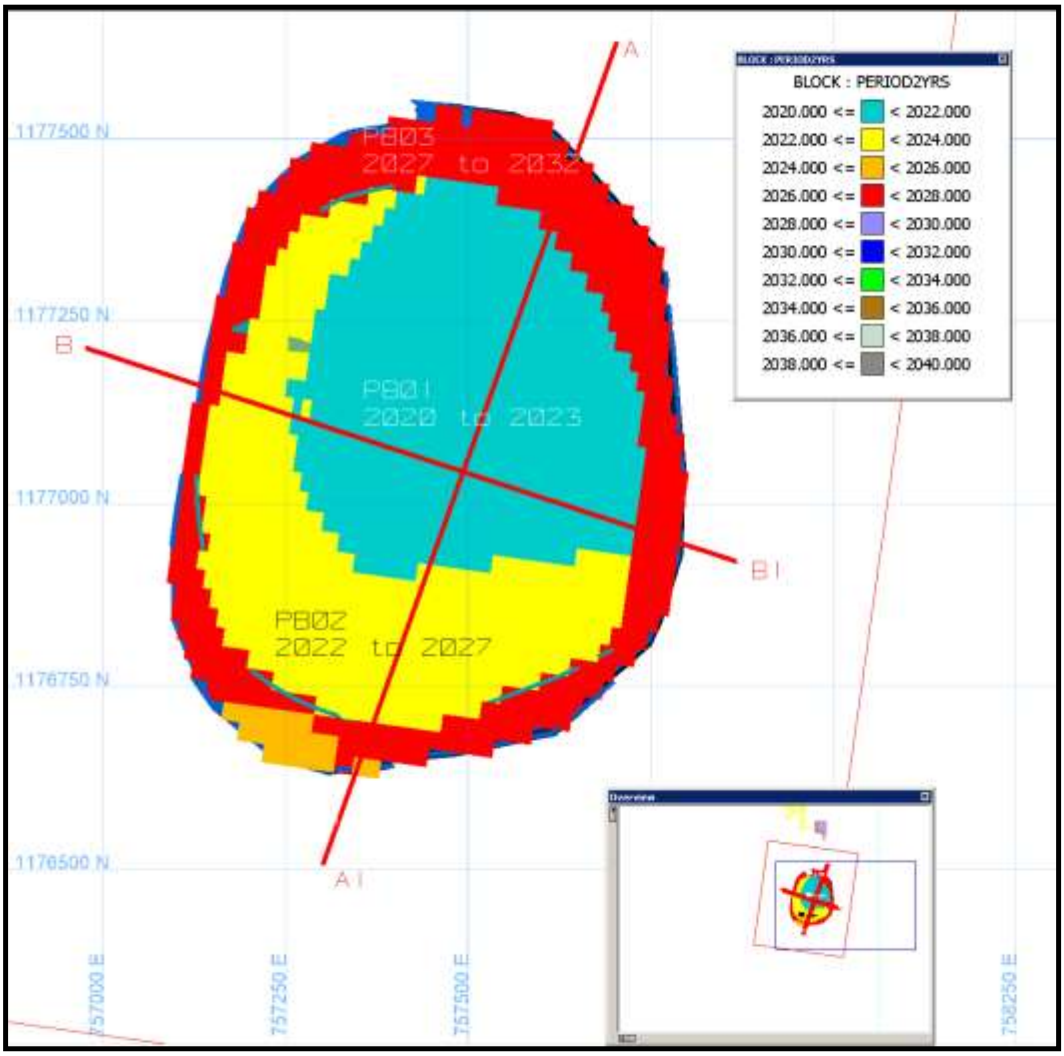


Figure 118: Period progress in Phase 1 (PB01), Phase 2 (PB02) and Phase 3 (PB03)

Mining of Phase 4 (PB04) will commence 9 years after the start of the mining operations, lasting 9 years. Figure 119 shows the general timing for the Mid Case for all the mining stages.

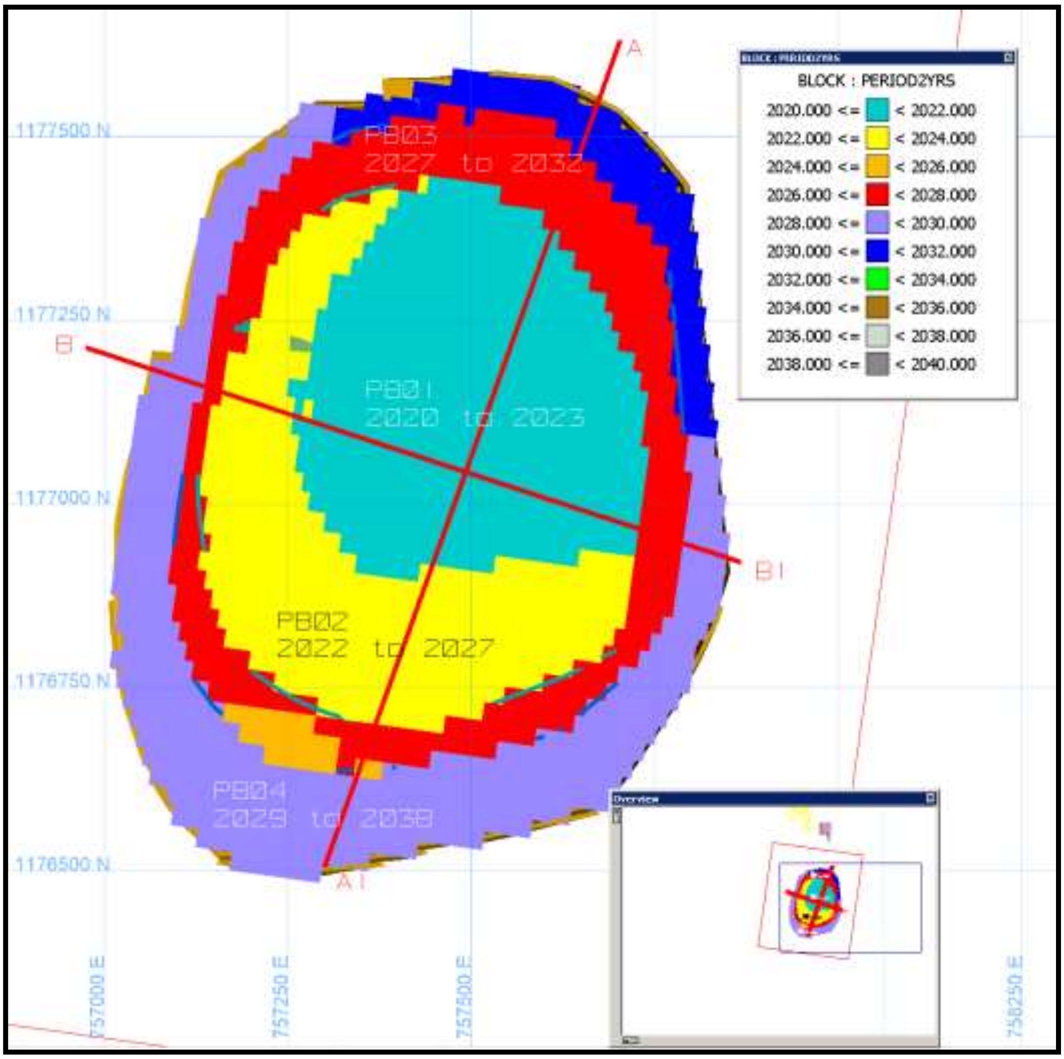


Figure 119: Progress in Phase 1 (PB01), Phase 2 (PB02), Phase 3 (PB03) and Phase 4 (PB04)

A general view of the Phase progress and proposed surface layout is shown in Figure 120 below.

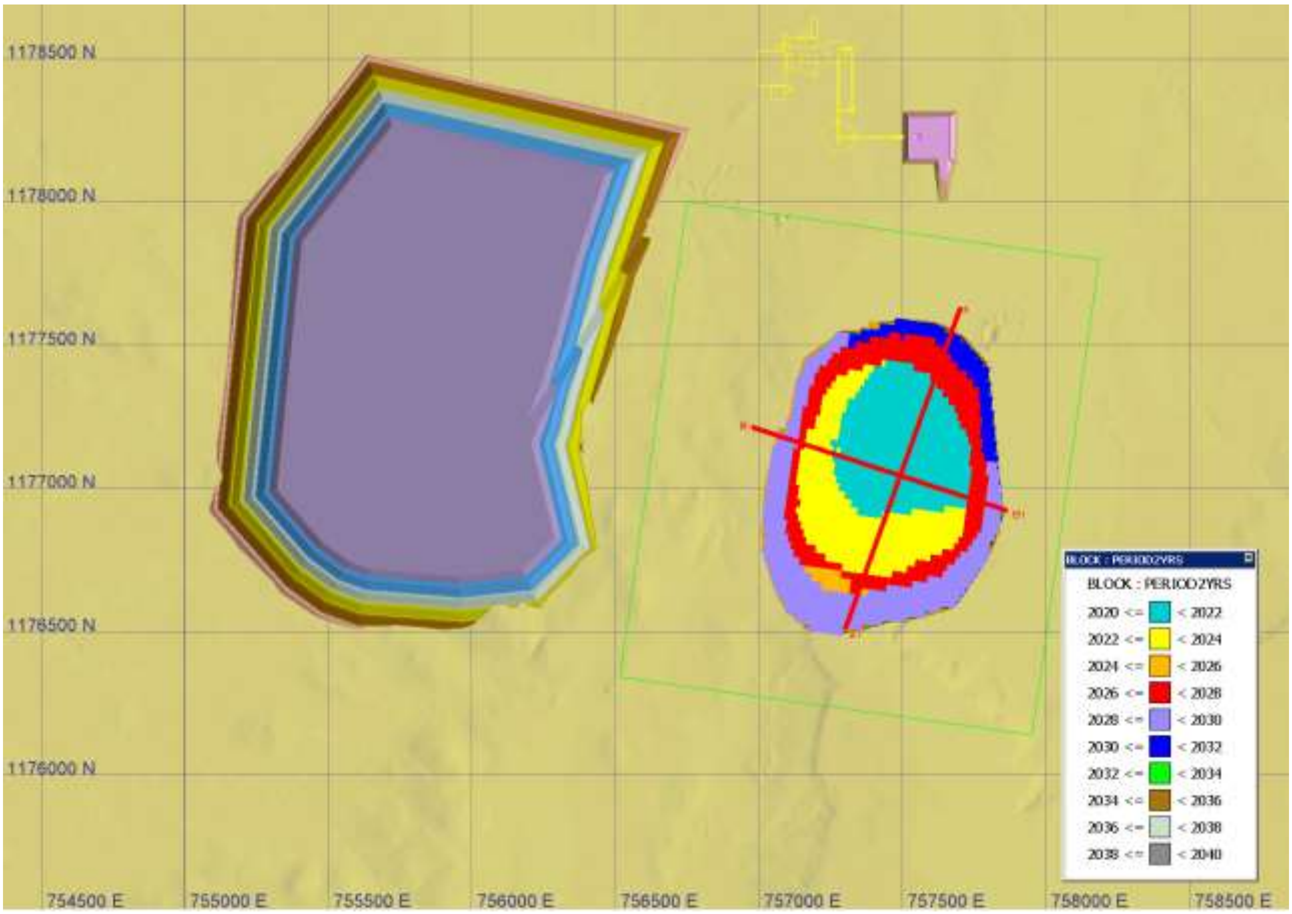


Figure 120: Mining progression and waste dump location

A cross section along the North South axis in Figure 121 shows the general mining sequence through the mining Phases for the Mid Case. The commencement of waste stripping (yellow) in Phase 2 can be seen to commence when the starter pit (Phase 1) is being completed. Overlap of the two mining Phases ensures continuous feed of the highest value material to the process plant and stockpiling of the overlying Oxide material in the Phase 2 pit.

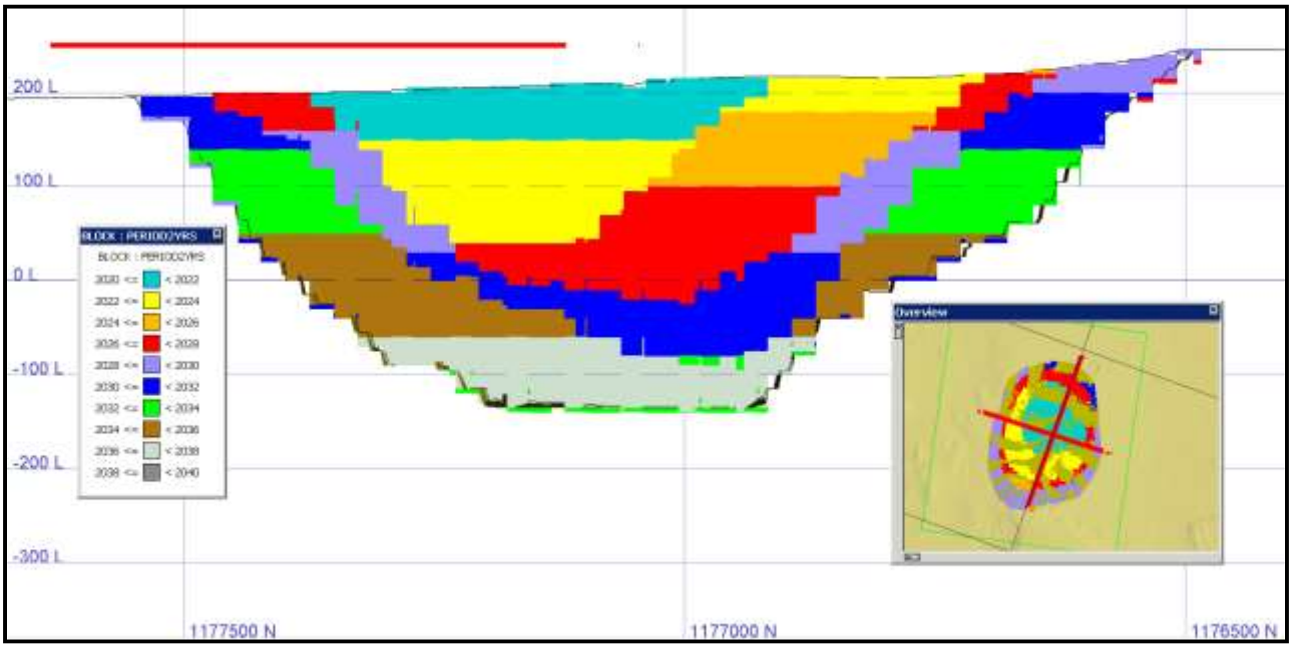


Figure 121: Cross section showing mining sequence, facing east

Figure 122 shows the same mining case with the period progress section being taken across the east-west axis. The final mining periods of stage 4 can be seen in the light and dark grey areas at the base of the final pit.

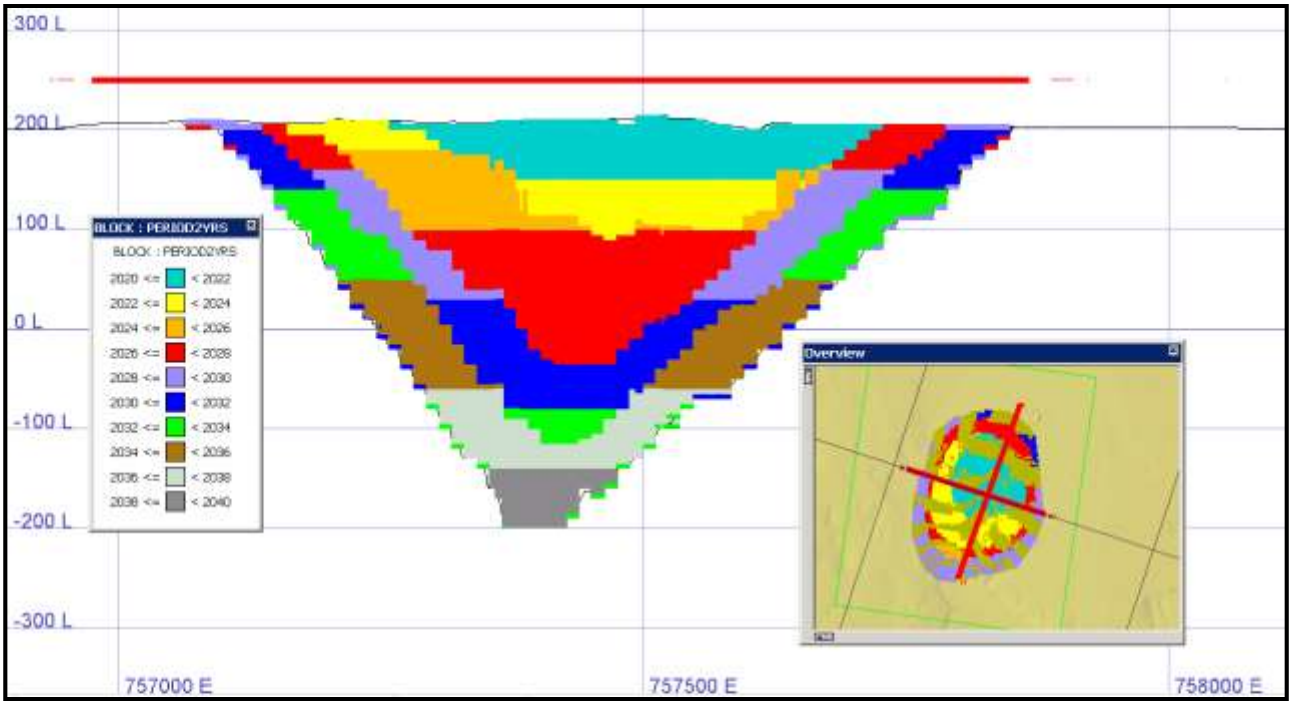


Figure 122: Cross section showing mining sequence, facing north

A smoothed mining profile was maintained whilst ensuring both mill capacity and the highest possible ore grades are fed to the process plant in the early years to maximize the NPV.

16.12 Mining equipment

Using the schedule and an estimate of truck productivity from TALPAC, the annual truck requirements assuming a Cat 785 size haul truck were estimated. The Mid Case option was used to estimate the ore and waste trucking requirements by period based on the tonnes mined by bench for that period. The Mid Case option requires a peak truck requirement of about 23 trucks. The schedules considered only a single size truck, with minimal selective ore mining requirement.

An estimate of the truck profile for ore and waste is shown in Figure 123, the tonnage movement profile was partially smoothed to negate single-year peak requirements. Mining is expected to be carried out on 10 m benches, with two 5 m flitches for ore mining within a 10 m bench.

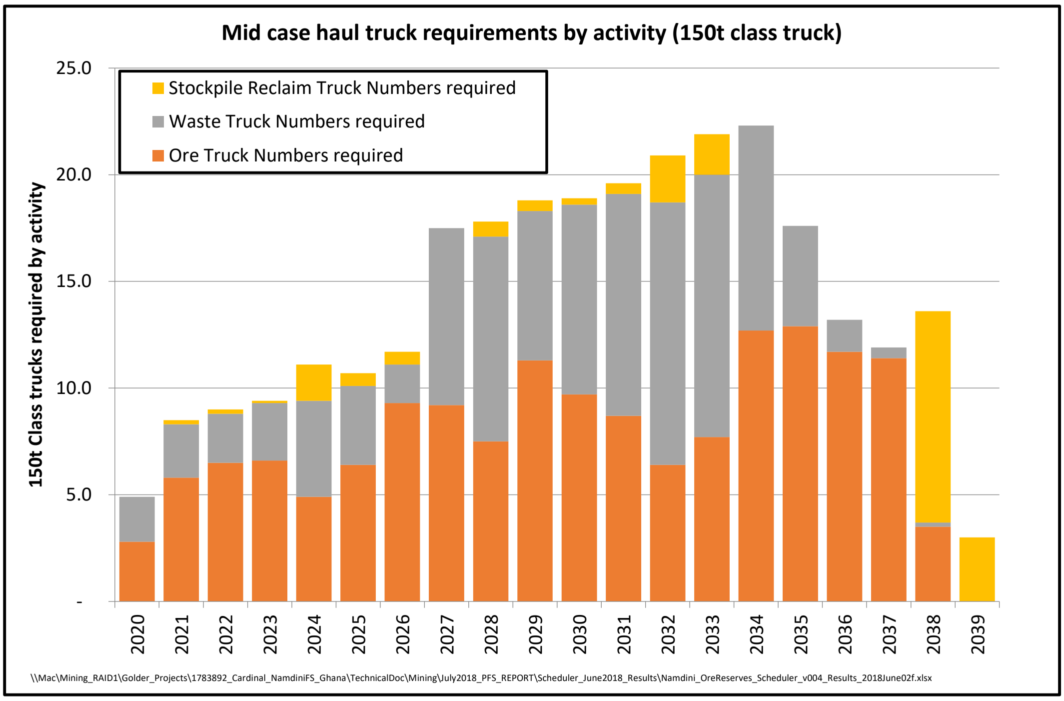


Figure 123: Mid Case annual truck requirements

16.13 Mining capital cost estimates

The open pit will be mined by Cardinal on a contract (outsourced) mining basis with capital costs for the mining infrastructure and mining fleet amortized and charged back to Cardinal as a cost per tonne.

16.13.1 Power supply

Electrical power requirements for the mining contractor operating the mining fleet will be minimal, with all mining equipment being diesel-powered.

Electrical supply for the primary crusher will be sourced as part of the process plant and infrastructure requirements.

16.13.2 Access, roads and bridges

Overall project infrastructure requirements have been estimated by consultants (primarily Lycopodium and Knight Piésold). No other heavy-duty road access or heavy-duty bridge requirements have been identified. Upgrading of access roads and small river bridges is anticipated prior to supply of heavy mining equipment to site. Cost and time for such infrastructure upgrades will need to be accommodated within the Mining Contract cost proposal.

16.13.3 Mining method

The Namdini Gold Project has been evaluated by Golder based on a conventional hard-rock mining equipment fleet. There are no requirements for untested or novel mining techniques.

16.13.4 Pit dewatering

Preliminary pit dewatering requirements were identified as part of the hydrological and hydrogeological studies. This will be further refined as the mine design is finalized.

For the purposes of the mining study, it was assumed that a ‘dry pit’ will be operated and that sufficient dewatering will allow for the geotechnical design for dry slopes.

All tonnage is estimated on a dry basis with expected surface moisture associated with the hauled rock being less than 3% on average.

The requirement to maintain the pits in a de-watered condition during mining is critical for both safety and operational success.

16.13.5 Explosive facilities

The selected mining contractor will be expected to manage and be responsible for the explosive magazine and bulk explosives production facility. This will be outsourced to a specialized explosives supplier. There are several reputable international explosives suppliers currently active in Ghana.

16.13.6 Summary capital costs

Golder expects the total mining capital costs to be borne by the mining contractor under an outsourced mining arrangement and charged back to Cardinal under a cost per tonne mined basis.

The contractor capital cost estimates should include:

- Establishment to site
- Diesel power generators (as required)
- Diesel tank farm
- Miscellaneous site buildings
- Mining excavation equipment
- Haul trucks
- Light vehicles
- Pit de-watering pumps
- Explosive storage facilities
- Bulk explosives storage and mixing facility

- Waste disposal infrastructure
- Fencing of the mining area as required by the Mining Lease (*Section 5c of lease agreement*).

Additional to these and based on the infrastructure and process plant studies, Cardinal may be required to provide the following:

- Connection to the National Grid for power supply
- Additional roads, bridges and river crossings.

16.14 Mining operating cost estimates

Golder estimated a mining cost of US$3.25 per tonne of rock mined based on experience with similar mining operations in the region. An additional allowance of 25 c per tonne allocated for ore mining costs reflects the additional costs of grade control sampling, laboratory assay, analysis and supervision.

A detailed mining movement schedule was supplied to two prospective mining contract companies currently operating in Ghana, to assist with provision of a detailed mining cost estimate. The mining schedule incorporates movement of ore and waste on 10 m mining benches, by year, for each of the four mining Phases. Detailed discussions have been followed with budgetary quotations. These have provided an initial all-in contract mining cost similar to that used in this PFS. Further discussions and negotiations will continue with suitable mining contractors prior to any award of the mining contract.

Detailed mining cost estimates will be the subject of a formal tender process from suitably qualified mining contractors with proven experience in the region, during the Feasibility Study stage of the project.

The budgetary costs quoted by the mining contractors have been used by Cardinal in the economic model.

16.14.1 Load and haul

Golder has estimated haul requirements using the industry standard TALPAC truck and loader productivity software. An incremental cost allowance of 6c per tonne hauled per 10 m bench was allowed as a vertical cost increment. The incremental cost with depth is largely driven by the increased haulage time required for the haul trucks and the increased fuel burn of the trucks on the ramp. It is assumed that the loading equipment will remain fully trucked by the contractor with additional trucks being supplied to the fleet as the average mining depth per cutback increases.

Figure 124 shows the method used by Golder to estimate the expected cost increment with depth for a specific cut-back. The load and haul component within a hard rock open pit mining operation typically accounts for 50 to 65% of the total mining cost, with the remainder being made up of drill and blast and general mining support activities. It is critical that the load and haul fleet is used effectively and that the loading tools are fully trucked to support cost minimization within the mining operation.

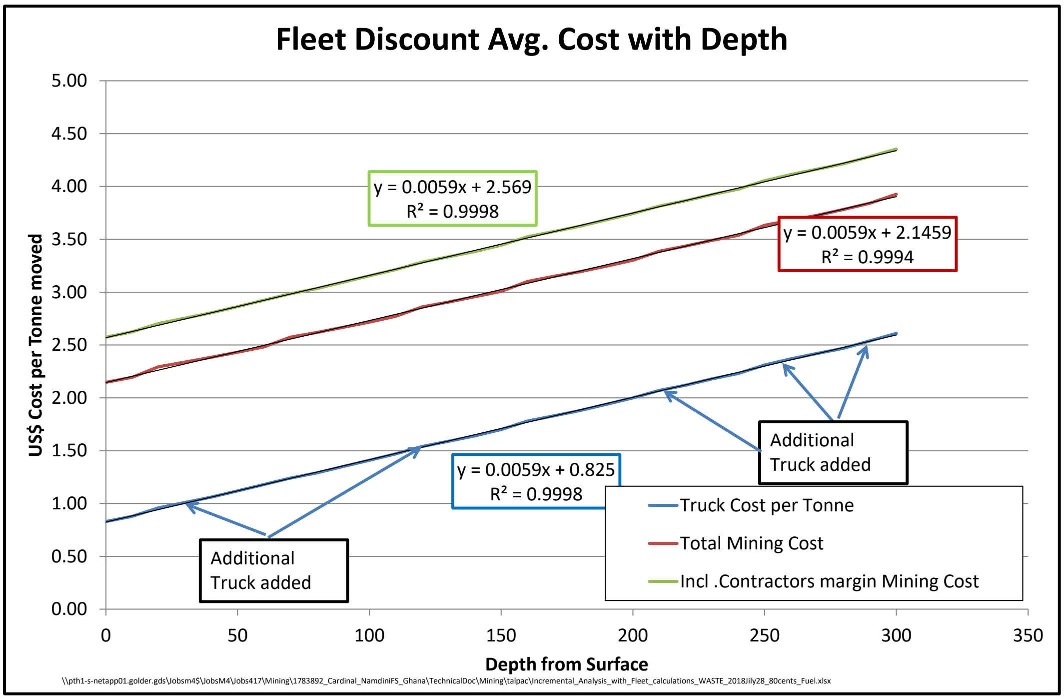


Figure 124: Incremental cost analysis

The cost for ore and waste was estimated by a contractor based on a preliminary schedule. The ore cost and waste cost per bench is plotted for each of the four Phases to provide an estimated mining cost and bench incremental cost. These show a close correlation to the Talpac estimated mining costs and are considered satisfactory for this level of study. The mining costs by Phase and for the Life of Mine ("LOM") by material type are shown in Figure 125 to Figure 129 below.

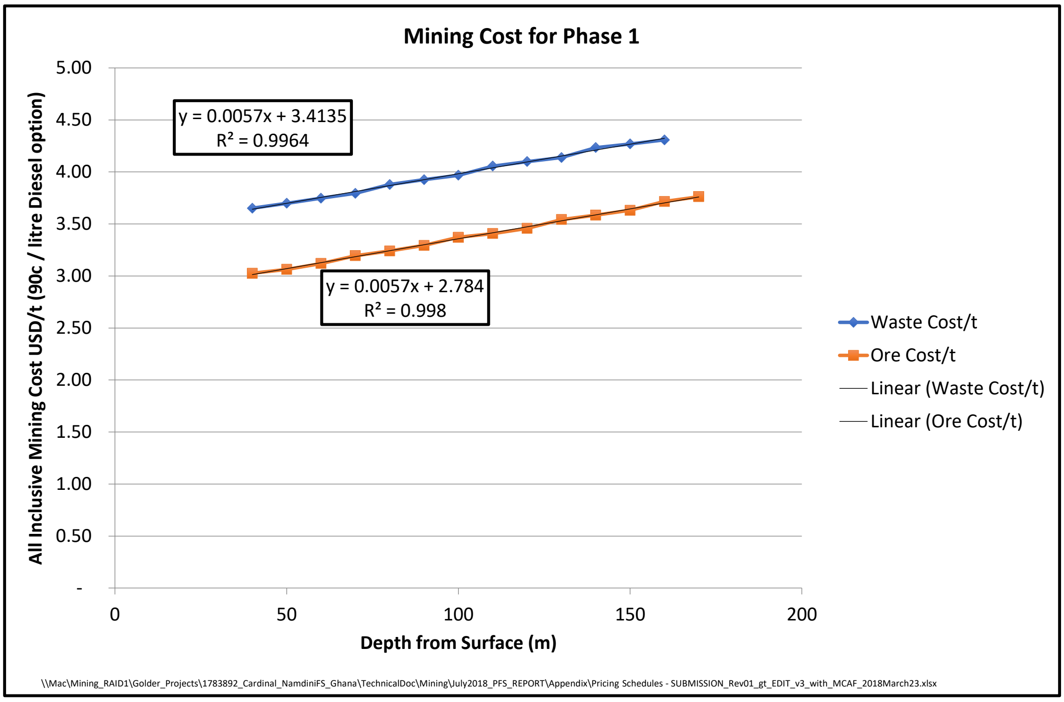


Figure 125: Contractor provided mining costs for Phase 1 (March 2018)

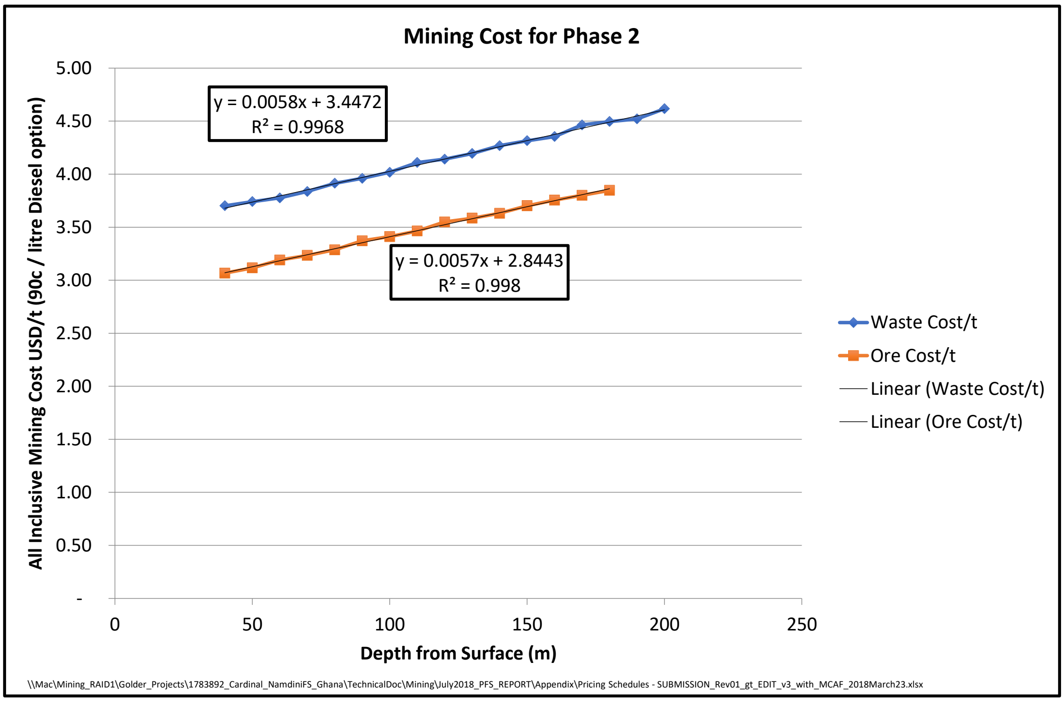


Figure 126: Contractor provided mining costs for Phase 2 (March 2018)

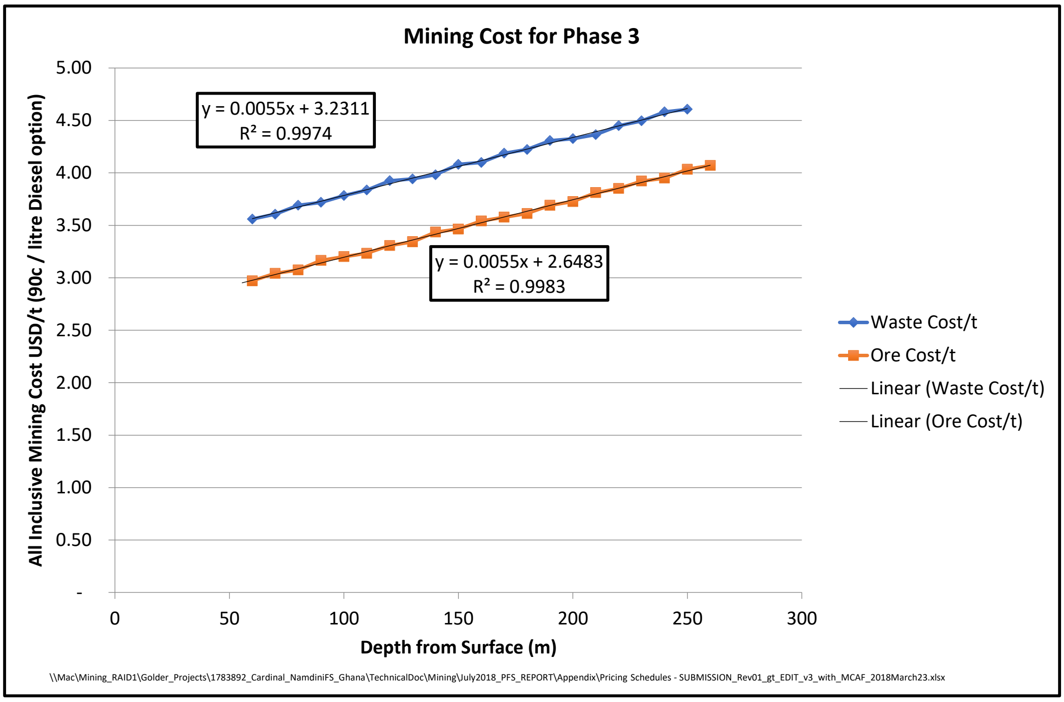


Figure 127: Contractor provided mining costs for Phase 3 (March 2018)

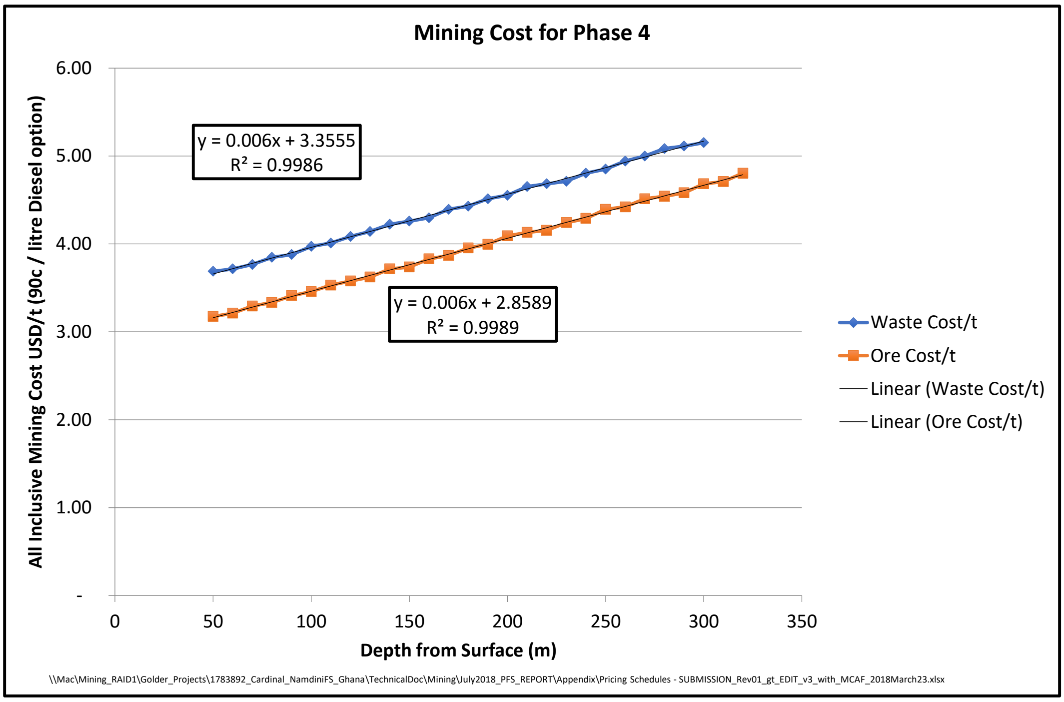


Figure 128: Contractor provided mining costs for Phase 4 (March 2018)

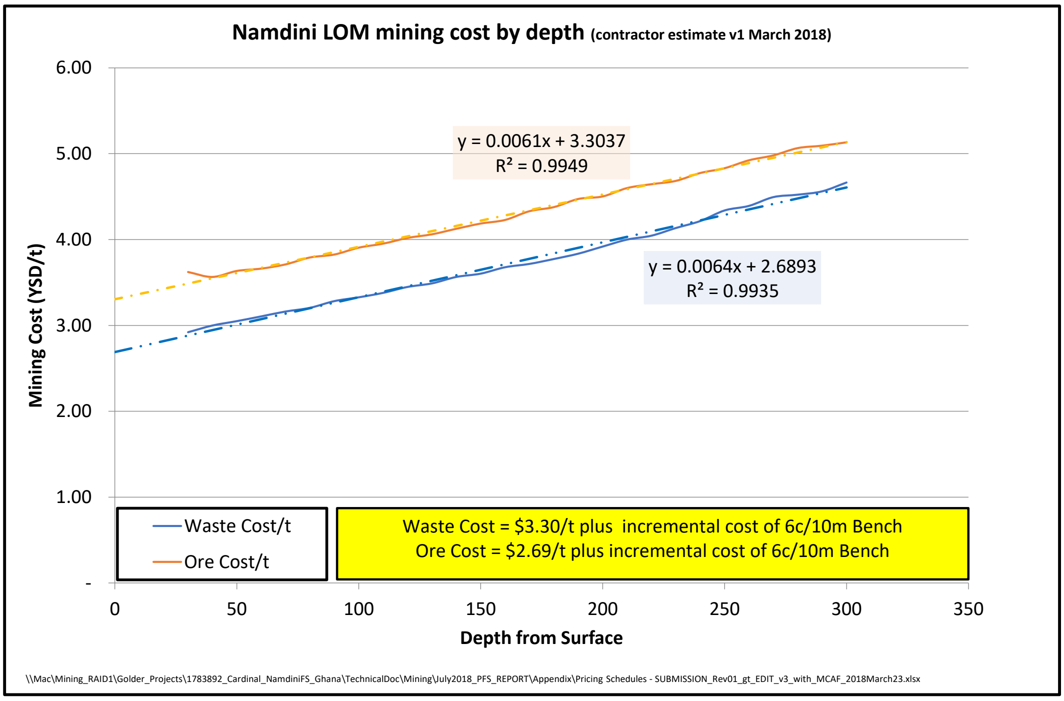


Figure 129: Contractor provided mining costs for LOM aggregate cost (March 2018)

16.14.2 Drill and blast

Specific drill and blast requirements have not been established as part of this study, but it is assumed that all but the surface Oxide ore will require drilling and blasting prior to load and haul. Preliminary drill and blast cost estimates were supplied by a suitable mining contractor (March 2018), however confirmation of the hardness and required drilling patterns will be needed to confirm this estimate.

16.14.3 Operational mining costs

The contract mining cost for the Namdini operation will be inclusive of drill and blast, load and haul and all associated mining costs with the exception of the mine planning, mine scheduling and grade control components which will be retained by Cardinal.

17.0 RECOVERY AND PROCESS PLANT

17.1 Process design

17.1.1 Design philosophy

The process plant for the Namdini Gold Project is based on a robust metallurgical flowsheet designed for optimum recovery with minimum operating costs. The flowsheet is configured with unit operations that are well proven in industry. The key criteria for equipment selection are the suitability for duty and the projected mine life of the operation without unnecessarily compromising reliability and ease of maintenance. The plant layout provides easy access to all equipment for construction, operating and maintenance requirements whilst providing a compact footprint to minimize costs.

Gold mineralization is characterized by disseminated sulfides in sheared Birimian greenstones. As discussed in Section 7.2.6 and Table 11, the main lithologies are Metavolcanics (MVO) intruded by Granite (GRA) and Diorite (DIO). The MVO and GRA ore types will make up the majority of the plant feed with smaller amounts of DIO ore coming later from the life of mine (LOM) pit.

Metallurgical testwork was scoped and managed by Cardinal (Section 12.2). Gold and sulfur deportment studies have identified fine-grained native gold as the primary gold bearing mineral and coarse-grained pyrite as the major carrier of gold. Testwork results indicate that the bulk of the gold in the Namdini ore is associated or hosted by pyrite (~75.4%) whilst another 10.6% is hosted by other sulfides. The remaining gold occurs as free gold and is also hosted by other minor mineral groups. The predominant association of gold with pyrite allows for conventional flotation technology to be applied, thus it was recommended to incorporate a front-end rougher flotation stage followed by re-grind of the concentrate to liberate fine gold particles locked in the pyritic ore.

Three throughputs scenarios were presented in Cardinal's PEA (Golder, 2018) as being 4.5, 7.0 and 9.5 Mtpa. The Mid Case of 7.0 Mtpa was selected for design purposes (Section 21.2).

The key project and ore specific criteria that the plant design will meet are:

- The plant is designed for an initial throughput of 7 Mtpa
- Crushing plant mechanical availability of 75%
- Mechanical availability for the remainder of the plant of 91.3%
- A moderate level of automation to provide control of the key process parameters.

A process design criteria document was prepared incorporating the engineering and key metallurgical design criteria derived from the results of metallurgical testwork and comminution circuit modelling which are still on-going.

17.1.2 Selected process flowsheet

The process plant design incorporates the following unit process operations:

- Single stage primary crushing with a gyratory crusher to produce a crushed product size of 80% passing ("P_{80}") 150 mm.
- Crushed ore feeding a coarse ore stockpile (12 hours live). Ore reclaim by two apron feeders.
- Two stage SAG/Ball milling in closed circuit with cyclones to produce a P_{80} grind size of 106 µm and includes recycle and crush of pebbles from the SAG mill (SABC grinding).
- Gravity recovery includes a scalping screen, a single 70 inch centrifugal concentrator and a CS4000 intensive leach reactor.

- Rougher scavenger flotation to produce a gold-rich sulfide concentrate.
- Thickening of the flotation tails for water recovery prior to disposal in a separate non-cyanide Tailings Storage Facility.
- High Intensity Regrind of the flotation concentrate to a P_{98} grind size of 15 µm followed by thickening to minimize carbon-in-leach (“CIL”) tankage and reduce overall reagent consumption.
- A concentrate CIL circuit incorporating one pre-leach tank and seven CIL tanks for gold and silver adsorption.
- A 3.5 tonne split AngloAmerican Research Laboratory (“AARL”) elution circuit, electrowinning and smelting to recover gold and silver and produce doré.
- CIL tailings treatment incorporating cyanide destruction by sulfur dioxide and oxygen.
- Concentrate CIL Tailings disposal in a lined tailings storage facility.

A process flowsheet showing the unit process operations incorporated in the selected process flowsheet is shown in Figure 130.

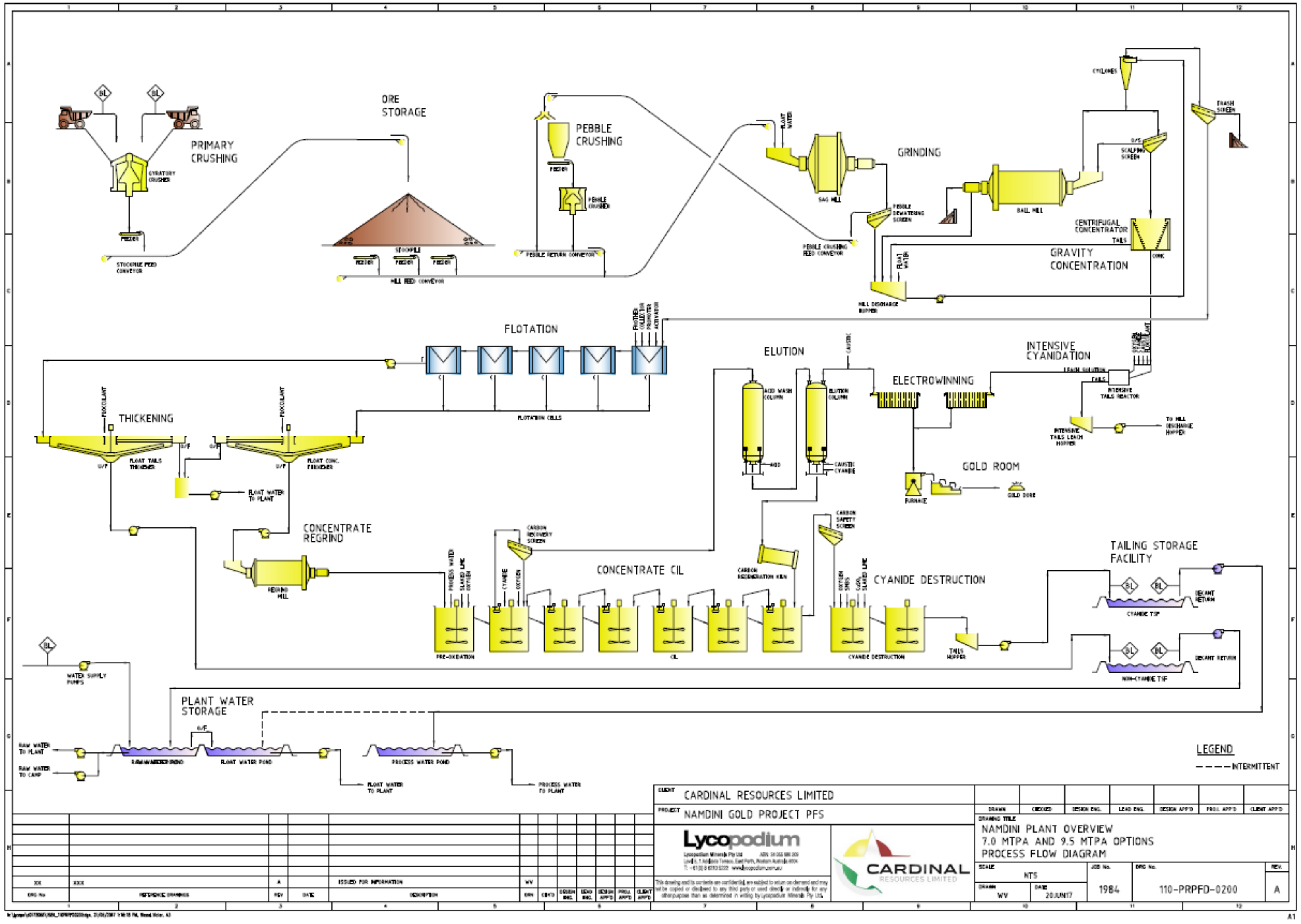


Figure 130: Process flowsheet showing the unit process operations (7.0 and 9.5 Mtpa options shown)

17.1.3 Plant design basis

Process plant

The plant design was based on a nominal capacity of 7 Mtpa. The feed will consist of blended ore from the main LOM pit consisting of 60% MVO, 30% GRA and 10% DIO.

ROM Pad

The Run-Of-Mine mill feed stockpile (ROM Pad) will be used as a buffer between the pit and the plant. The ROM stockpile will allow blending of feedstocks and ensure a consistent feed type and rate to the plant. A fixed rock breaker will be used to break any oversized rocks in the crushing area.

Comminution selection

The initial comminution circuit design was based on evaluation of 2017 testwork. This was followed up by further testwork on a number of separate drill hole samples spread across the orebody. The findings are summarized in the preliminary mill sizing design report compiled by Orway Mineral Consultants (“OMC”).

The conclusions from the report and results were as follows:

- Test ore had high competencies and work indices with MVO being particularly high. While the initial samples were sourced from one drill hole, the follow up testing showed similar trends with competence (as measured by the SMC A*b drop weight value) varying from 29.7 to 40 putting these ores samples in the hard category.
- The use of a jaw crusher is not recommended as a large crusher will not meet the target product size due to the reduction ratio required. A gyratory crusher is recommended for primary crushing. A fixed rock breaker is also recommended.
- The SAG mill power efficiency for this ore is low. Partial secondary crushing of the SAG feed may be considered to improve SAG breakage, should further testing indicate this requirement.
- Pebble crushing is included to manage build-up of critical size material in the SAG mill.
- The 85th percentile design abrasion index range of 0.074 for Metavolcanic to 0.292 for Granite shows relatively low ore abrasiveness and therefore translates into low and favourable wear liner and steel media consumption

Selected comminution circuit

- The circuit option selected was primary gyratory crushing followed by two-stage SAG/Ball milling and recycle crushing (SABC). This was selected as it can accommodate competent ore feed and minimizes operating cost while still producing the relatively fine product size required.
- A summary of the equipment selection and expected consumables is provided in Table 103 and Table 104.

Table 103: Summary of selected milling circuit

Parameters	Units	Value
Cwi	kWh/t	16.4
BWi	kWh/t	16.0
A*b	-	29.6
Abrasion Index	g	0.24
Feed rate	tph	875
Milling feed F_{80}	mm	143
Milling product P_{80}	µm	106
Milling specific energy (total) P_{80}	kWh/t	23.7
SAG Mill		
Mill diameter	m	10.36
Mill EGL	m	6.15
Specific energy	kWh/t	12.3
Ball charge: Duty Max	 % %	 11.0 15.0
Total load: Duty Max	 % %	 25.0 35.0
Mill speed	% Nc	75.0
Drive type	-	Variable speed
Mill pinion power: Nominal Max	 kW kW	 10.762 13,048
Installed power	kW	14,700
Discharge arrangement	-	Grate
Open area	%	8
Grinding media size	mm	125
Ball Mill		
Mill diameter	m	7.31
Mill EGL	m	11.55
Specific energy	kWh/t	11.1
Ball charge: Duty Max	 % %	 27.0 33.0
Mill speed	% Nc	75.0
Drive type	-	Fixed speed
Mill pinion power: Nominal Max	 kW kW	 9,708 11,161
Installed power	kW	11,700
Discharge arrangement	-	Overflow
Grinding media	mm	65
Pebble Crusher		
% Feed	%	20
Feed rate	tph	174
Specific energy	kWh/t	0.3

Table 104: Estimate of consumables

Parameters	Units	Design
Primary Crusher		
Mantle	hours	7,920
Concave	hours	11,880
Gross power	kWh/t	0.24
SAG Mill		
Ball consumption	kg/t milled	0.517
Steel liner consumption	kg/t milled	0.093
Gross power	kWh/t	13.3
Ball Mill		
Ball consumption	kg/t milled	0.637
Steel liner consumption	kg/t milled	0.084
Gross power	kWh/t	12.0
Pebble Crusher		
Mantle/Concave	hours	1,524
Gross power	kWh/t	0.48

Crushing and ore storage

The primary crusher was sized on the basis of achieving 75% utilization. The mine design is based on 82% direct feed to the crusher with minimum rehandle. The crusher will be sized on this basis and to provide capacity for maintenance of the crusher without interrupting mill operation.

Ore will be trucked and direct tipped into the ROM feed pocket for entry into the gyratory crusher. A fixed rock breaker will be used to handle oversize ore. A gyratory crusher was selected to allow direct tipping of ore from the mine and to achieve the target reduction ratio required for mill feed. As the abrasion index of the ore is moderate no significant wear issues are expected.

Crushed ore will feed a 12 hour live capacity stockpile. The live coarse ore stockpile was selected to ensure adequate surge capacity in the event of a crusher breakdown. Crushed ore will be reclaimed by two apron feeders, each capable of providing 100% of the mill feed.

Milling and classification

The milling circuit was sized to reduce the crusher product to the nominal P_{80} size of 106 µm. The SABC facility provides a more uniform product while minimising operating cost. Pebble crushing was included to reduce any critical size build-up. The SAG mill will be equipped with a variable speed drive allowing a full range of speed with a nominal design of 75% of critical speed. The ball mill will be fixed speed. The variability in ore competence can be addressed by varying the speed of the SAG mill.

Three-stage crushing and ball milling was considered; however, the increased complexity of the plant at this scale with multiple crushers and screens was considered to be a disadvantage.

A gravity circuit was included as some of the ore types tested, in the Starter Pit samples in particular, have a high gravity recoverable gold component.

Flotation

Testwork on samples of the ore has indicated that high gold recovery, through its predominant association with pyrite to concentrate, can be achieved by flotation, with a design point of 7.5% of the feed mass being selected. The testwork indicated a simple roughing circuit with a residence time of 30 minutes was adequate without a cleaning stage. Given the significant slurry flow from the grinding facility at 35% solids, tank cells were selected with six cells. All flotation concentrate will report to a single concentrate hopper feeding the regrind circuit.

Concentrate regrind

Whole of ore leach testwork results indicated that regrinding of flotation concentrate would be necessary to significantly improve leach extraction. Grind optimization showed that a regrind to a P_{98} grind size of 15 µm gave acceptable extraction. As the primary grinding will produce some final product size material, a classification stage was included ahead of the regrind mill with 30% of cyclone feed expected to report to overflow. The remaining 70% will be reground.

A High Intensity Grinding ("HIG") mill was selected for the regrind duty. This equipment minimizes footprint, provides internal classification and has minimal overgrind issues, while providing easy access for maintenance compared to other stirred grinding mills. The mill was sized based on typical power demand for concentrate of this type. The reground concentrate together with cyclone overflow will be combined and thickened prior to reporting to the CIL circuit.

Flotation concentrate thickening

Inclusion of a flotation concentrate thickener before the CIL facility will achieve a higher and more consistent leach feed density. The higher leach feed density will reduce the overall tankage volume required to achieve the target residence mixing time. It will also improve slurry-mixing characteristics and reduce reagent costs. Thickening also recycles float water for re-use in the process plant. The thickener was sized to allow for potential froth management.

Leach and adsorption circuit

Metallurgical testwork indicated that:

- The effect of pre-oxidation improved both the gold leaching kinetics and overall dissolution by gold cyanidation. As a result, a pre-leach tank was included in the design
- The initial leach kinetics for most of the types of ore were rapid with GRA and MVO reaching over 80% recovery in 4 hours. However, a longer leach time of 72 hours was identified as optimal due to MVO and GRA reaching higher recoveries after extended leach.

On this basis, a configuration of one pre-leach tank and seven CIL tanks was selected to achieve optimal gold recovery. The tanks will be identical in size with staged addition of cyanide as required. Oxygen will be added to all tanks to oxidize any cyanicides and maintain an adequate dissolved oxygen level in the CIL tanks.

Elution and gold room

The average daily movement of carbon was calculated based on the design feed grade and maximum gold and silver extraction. On this basis a 3.5 t capacity split AARL elution circuit was selected requiring just under six strips per week. The AARL circuit was selected as it offers the flexibility to run more than one elution per day.

Flotation tails thickening

Tails from the flotation circuit will be thickened on site to recover cyanide-free water for re-use in the grinding circuit. Approximately half the water will be recovered in the thickener.

Cyanide destruction

Concentrate leach generally requires a higher cyanide concentration to achieve efficient extraction. As a result, tailings from the CIL circuit may contain up to 500 ppm weak acid dissociable cyanide ("CNWAD"). A cyanide destruction circuit incorporating sodium metabisulfite and oxygen was included to reduce the cyanide level to below 50 ppm CNWAD.

Tailings disposal

Two separate tailings streams will be produced. Tailings from the cyanide destruction circuit will be pumped to a plastic-lined tailings storage facility. Tailings from the flotation circuit will be pumped to a separate storage facility without the need for plastic lining. This will allow the two decant water streams to be recycled separately.

Water and air services

Three major water systems will be used:

- Flotation water – recovered from the flotation tails thickener and tailings storage facility will be recycled in the grinding and flotation areas.
- Process water – recovered from the cyanide tailings storage facility will be recycled in the CIL area.
- Raw water – sourced from the Volta River to a thirty day capacity raw water storage dam will be used for reagent mixing and will feed a water treatment plant.

Three air systems will be used:

- Low pressure air – will be used in the flotation cells.
- Oxygen – will be used in the CIL and cyanide destruction areas.
- Compressed air – will be used for instruments and plant services.

17.1.4 Key process design criteria

Table 105 presents the key process and mechanical design criteria.

Table 105: Summary of key process design criteria

Parameter	Units	Primary	Source
Plant capacity	Mtpa	7	Cardinal
Gold head grade – design	g/t Au	1.20	Cardinal
Crushing plant availability	%	75	Lycopodium
Plant availability ROM ore top size Crushing Work Index (CWi)	% mm kWh/t	91.3 856 16.4	Lycopodium OMC Testwork – Ore Blend
Bond Rod Mill Work Index (RWi)	kWh/t	27.0	Testwork – Ore Blend
Bond Ball Mill Work Index (BWi) Abrasion Index Drop Weight (SMC, A*b)	kWh/t g	16.9 0.24 29.6	Testwork – Ore Blend Testwork – Ore Blend Testwork – Ore Blend
SG Comminution circuit Crush size (P_{80})	 mm	2.78 SABC 150	Testwork OMC OMC
Grind size (P_{80}) Regrind size (P_{98}) Regrind Specific Energy	µm µm kWh/t	106 15 50	Testwork Testwork Vendor
HIG Mill grinding media consumption	kg/t	0.26	Vendor
Flotation mass recovery	%	7.5	Testwork
Concentrate regrind Size (P_{98})	µm	15	Testwork
Pre-leach thickener solids loading	$t/m^2.h$	0.25	Vendor
Leach circuit residence time	hours	74	Prelim testwork
Leach slurry density	kg/m^3	1.43	Lycopodium
Number of pre-leach tanks		1	Lycopodium
Number of CIL tanks		7	Lycopodium

Parameter	Units	Primary	Source
Cyanide consumption	kg/t ore	0.57	Prelim testwork
Hydrated lime consumption	kg/t ore	1.24	Prelim testwork
Elution circuit type		AARL	Lycopodium
Elution circuit size	t	3.5	Lycopodium
Frequency of Elution	strips/week	5.9	Lycopodium
Cyanide destruction process		SO_2/O_2 process	Lycopodium

17.2 Process and plant description

17.2.1 ROM Pad

Ore will be trucked and either direct tipped or dumped onto the ROM Pad for blending and rehandling into the ROM feed pocket. Ore will be blended under the guidance of process personnel to maintain a relatively constant feed grade and ore hardness to the plant.

17.2.2 Crushing circuit

ROM ore will generally be deposited into the feed pocket by direct tip with front-end loader (FEL) reclaim where required. A fixed rock breaker will be used to break up any oversize material. The crusher selected is a 45" by 65" gyratory crusher with a 375 kW motor.

The gyratory crusher product will discharge into a chamber with two trucks live capacity. From the discharge chamber, the primary apron feeder will transfer crushed ore onto the primary discharge conveyor in turn feeding the stockpile feed conveyor onto the coarse ore stockpile. A weightometer on the stockpile feed conveyor will measure crusher product. Dust suppression sprays will be installed at the ROM feed pocket and stockpile feed conveyor. A dust collector will draw dust from key transfer points and will deposit onto the stockpile feed conveyor.

A 5 t capacity maintenance crane will be provided for the crusher and a 2 t hoist for the apron feeder.

17.2.3 Ore storage and reclaim

The stockpile feed conveyor will discharge onto a 10,500 tonne live capacity stockpile providing 12 hours milling feed to allow for maintenance on the crusher. The total stockpile capacity will be 52,500 tonnes or 60 hours of plant feed and will be accessed by pushing in with a dozer. This will provide capacity for extended crusher shutdowns.

Primary ore will be reclaimed from the live stockpile by two variable speed apron feeders. Each of the feeders will be rated to reclaim at the full mill feed rate. The feeders will be located in a concrete tunnel above the mill feed conveyor. A forced air fan will ventilate the reclaim tunnel and a pulsed jet dust scrubber will reduce airborne dust generation. A weightometer on the mill feed conveyor will measure mill feed weight.

A reclaim sump will recover any spillage from clean-up.

17.2.4 Grinding and classification circuit

The grinding circuit will consist of an SABC circuit with hydrocyclones.

Feed from the coarse ore stockpile will be conveyed to the SAG mill feed chute where it will be diluted with water. A 10.36 m diameter × 6.15 m EGL SAG mill will grind crushed ore from a F_{80} of 150 mm to produce the target transfer P_{80} size of 1,000 to 2,500 µm. The SAG mill will be equipped with two 7.35 MW variable speed drives to provide a total of 14,700 kW installed power. The variable speed drives will allow the mill speed to be adjusted to address variable ore competence.

SAG mill grinding media typically 125 mm diameter balls will be loaded by FEL into a SAG mill ball hopper for transfer to the mill feed conveyor.

Product from the SAG mill will discharge to a vibrating single deck pebble dewatering screen fitted with a nominal 12 mm slotted aperture screen. Pebble screen undersize will flow to the cyclone feed hopper.

Pebble screen oversize, consisting of pebbles and worn steel grinding media will report to the pebble discharge conveyor. If required, screen oversize will be diverted to the milling area drive in sump. A magnet mounted above the pebble conveyor will remove tramp steel grinding media. Any tramp metal not removed will be detected by a metal detector located above the pebble conveyor prior to the pebble crusher. The metal detector will activate a diverter gate to prevent steel from reporting to the pebble crusher. This diverter gate will also allow bypass of pebbles for recycle to the SAG mill or to enable maintenance to be carried out on the pebble crusher without the need to shut down the milling circuit. A weightometer on the pebble transfer conveyor will indicate the mass of pebbles being recycled.

Pebbles will be crushed in the pebble crusher and will discharge to the crushed pebble transfer conveyor which will report to the mill feed conveyor.

The SAG and ball mill discharge will feed a combined cyclone feed hopper and will be diluted with water prior to classification. The mill discharge slurry will be pumped using duty/stand-by pumps to the classifying cyclones. The cyclone cluster will comprise 11 × 650 mm diameter cyclones operating at 80 kPa. The cluster will have eight cyclones in duty and three in stand-by for maintenance purposes. Cyclone overflow will report to trash screening and cyclone underflow will feed the ball mill with a portion reporting to the gravity circuit.

The ball mill will be a 7.31 m diameter × 11.55 m EGL overflow mill with a target grind P_{80} size of 106 µm. The ball mill will be equipped with two 5.85 MW fixed speed drives giving a total installed power of 11,700 kW. The design mill rotation speed will be 75% of critical speed.

Ball mill grinding media consisting of 65 mm diameter balls will be charged by a dedicated ball charging hoist.

A drive-in sump and feed end sump will be provided to recover spillage from the circuit.

Cyclone overflow at 35% w/w solids will flow by gravity to the vibrating trash screen with a nominal 0.63 mm aperture. Trash screen oversize will pass to a trash collection bin. Trash screen undersize will report to the conditioning tank ahead of the flotation circuit.

17.2.5 Gravity circuit

A bleed from the classification cyclone underflow will feed a 3.7 by 7.0 m scalping screen with 2.4 mm apertures. The screen oversize will flow to the ball mill feed while the screen underflow will feed the gravity concentrator. The concentrator will be a 70 inch diameter centrifugal unit complete with variable speed drive and a dedicated fluidising water system.

The unit will operate on a 60 minute (nominal) cycle time. After 60 minutes it will bypass feed to ball mill feed and start a flush cycle. Gravity concentrate will gravitate to an intensive leach reactor. Once the flush cycle is complete the unit will start up again and proceed to treat screen underflow.

The intensive leach unit will consist of a feed cone, a fluidized bed reactor and a solution cone. After desliming the feed, concentrate will be leached with cyanide and a specialized oxidant. After completion of the leach cycle the solution will be pumped to the gravity pregnant liquor tank for electrowinning. The tail from the leach reactor will be pumped to the cyclone feed hopper.

17.2.6 Flotation and regrind

Slurry from the trash screen will flow to the flotation conditioning tank where frother and collector will be added.

The conditioned slurry will then flow to the first of six 200 m^3 capacity forced air flotation cells arranged in a straight configuration with total residence time of 32 minutes. Flotation concentrate from all the cells will report to a concentrate hopper from which it will be pumped to the regrind mill. Flotation tails will discharge from the last cell to the flotation tails hopper from which it will be pumped to the float tails thickener. The design assumes a mass pull of 7.5%. Two sump pumps are provided for spillage handling in this area.

Flotation concentrate will be pumped to the regrind classifying cyclones for classification prior to regrind. The regrind cyclone cluster will consist of 6 × 250 mm diameter cyclones operating at 80 kPa. The cluster will have four cyclones in duty and two in stand-by for maintenance purposes. The cyclones will cut at a P_{98} of 15 µm to produce an overflow containing 30% of the feed with the overflow passing to the flotation concentrate thickener.

The cyclone underflow will flow to a HIG Mill equipped with a 2,300 kW variable speed drive. The regrind mill product with a P_{98} size of 15 µm will then flow to the float concentrate thickener.

The HIG mill will be charged with 3 mm ceramic media, which will be loaded by a media hopper and screw feeder. A single sump pump will be provided for this area.

17.2.7 Flotation concentrate and tails thickening

Concentrate will be thickened in an 18 m diameter high rate thickener ahead of the CIL circuit to provide a higher slurry density to the CIL. Flocculant will be added to the feed and the underflow at 55% w/w solids will be pumped to the CIL feed box.

Flotation tailings will be thickened in a 30 m diameter high rate thickener ahead of the tailings disposal tank to maximize recovery of water. Flocculant will be added to the feed and the underflow at 55% w/w solids will be pumped to the tailings tank prior to disposal.

Overflow from both thickeners will report to a single overflow tank for re-use in flotation as process water. A single sump pump for each thickener will be provided.

17.2.8 Leach and carbon adsorption circuit

The leach process will include one pre-leach tank and seven CIL tanks. The pre-leach tank and the CIL tanks will have a volume of 1,000 m^3. Each tank will be fitted with a dual impeller mechanical agitator to ensure uniform mixing. The CIL tanks will have a mechanically swept wedge wire inter-tank screen to retain carbon and facilitate controlled transfer. The CIL process will have a residence time of 72 hours at 66 t/h solids at 50% w/w density to provide optimal recovery of gold.

The slurry pH will be adjusted with hydrated lime added to the vibrating trash screen feed box. Sodium cyanide solution will be metered into the first four tanks as required to maintain the desired cyanide concentration in the slurry. Oxygen will be sparged to the pre-leach and CIL tanks to maintain a high dissolved oxygen profile.

Fresh and regenerated carbon will be added to the circuit at the last CIL tank and advanced by recessed impeller pumps counter current to the slurry flow. Inter-tank screens in each adsorption tank will retain the carbon. Loaded carbon from the first CIL stage will be pumped to the loaded carbon recovery screen mounted above the CIL tanks. The carbon will be washed and dewatered on the recovery screen and report to the acid wash column while the slurry returns to the CIL tank.

Leach tails from CIL will flow to the carbon safety screen to recover any carbon leaking from worn screens or overflowing tanks. Screen underflow will gravitate to the cyanide destruction circuit. Screen oversize (recovered carbon) will be collected in the fine carbon bin.

Barren carbon returning to the adsorption circuit from the carbon regeneration kiln or elution circuit will be screened on the carbon sizing screen to remove fine carbon and quench water. The sized and regenerated carbon will report directly to Tank 7.

The CIL area will include an overhead gantry crane for operations and maintenance use and two sump pumps for spillage handling.

17.2.9 Stripping plant and gold room

The following operations will be carried out in the stripping and gold room areas:

- Acid washing of carbon
- Stripping of gold from loaded carbon using the split AARL method
- Electrowinning of gold from pregnant solution
- Smelting of electrowinning products.

The spilt AARL stripping plant will be automated and contain a separate acid wash and elution column. The total carbon movement around the elution plant on a daily basis will be approximately 3 tonnes with a solution flow rate in the elution column of two bed-volumes/h (14.9 m^3/h). There will be a nominal 5.9 strips/week. The circuit was designed at 3.5 tonne to accommodate fluctuations in feed grade and consequently carbon movement.

Acid Wash

Loaded carbon will be received into the 3.5 tonne capacity acid wash column. During acid washing a dilute 3% v/v hydrochloric acid (HCL) solution will be pumped into the bottom of the column to remove contaminants such as calcium, copper and magnesium from the carbon which can interfere with elution. After the soak period of 30 minutes has elapsed the loaded carbon will be rinsed with water. Dilute acid and rinse water will be pumped to the flotation tailings hopper for disposal. Washed carbon from the acid column will be pressure transferred from the acid wash column to the elution column and the water drained out.

Pre-soak and Elution

The split AARL elution process will be used to recover gold adsorbed onto the carbon recovered from the CIL circuit. Initially lean eluate from the lean eluate tank will be heated to approximately 95°C and pumped into the base of the elution column using the strip solution pump. A 3% sodium hydroxide (NaOH) and 2% sodium cyanide (NaCN) solution will be pumped into the elution column by the strip solution pump to pre-soak the loaded carbon for 30 minutes to elute gold.

The pregnant solution will then be rinsed from the carbon by up to 10 bed volumes (BV) of solution heated to approximately 135°C. The first five bed-volumes of the elution will be drawn from the lean eluate tank and directed to the pregnant solution tank for recovery of gold by electrowinning. The last five bed volumes of the elution will be drawn from the treated water tank and will be directed to the lean eluate tank for re-use during the next elution cycle.

Once elution is complete the eluted carbon will be cooled with water prior to being transferred to the hopper above the carbon regeneration kiln.

Strip solutions will be indirectly heated by a diesel fired oil heater and heat exchanger. Heat recovered from solution exiting the column will be used to pre-heat solution prior to the stripping column input.

Solution samplers will be provided to collect pregnant and stripped eluent for assay.

Electrowinning and Gold room

Gold will be extracted by electrowinning from the pregnant solution within the security area of the gold room. As the pregnant solution has a high gold tenor, it will be circulated through two parallel cells of 22 cathodes each in multiple passes to achieve electrowinning and ensure a minimum gold tenor in the barren eluate. Direct current will be passed through stainless steel anodes and stainless steel wool mesh cathodes to deposit gold and silver sludge on the cathodes. A sludging cell design with in-tank high pressure washing of the cathodes was selected to simplify the cathode handling process.

Rectifiers, one per cell, will be located in a non-secure area below the cells allowing maintenance access without breaching gold room security. Rectifier remote indication and controls will be located adjacent to the electrowinning cells. In normal operations electrowinning will be completed in less than a 12 hour shift.

The system was configured to allow multiple pass electrowinning for operation when two strips per day are required. The silver and gold sludge will be filtered and dried before smelting with fluxes to produce doré bars. Slag from smelting operations will be returned to the milling circuit. Fume extraction equipment will remove gases from the cells, sludge drying oven and smelting furnace.

A separate electrowinning cell will be used for the gravity circuit pregnant liquor. This will be fed by a dedicated pump.

Site Security

The gold room design is based on full security surveillance by a security guard and a second level surveillance by remote control CCTV cameras with remote viewing and recording facilities. Additional cameras will be located at key locations to maintain surveillance particularly in regard to future gravity gold processing and to assist with operational monitoring. An in-depth description of security methods and practices is intentionally excluded from this report and is generally implemented separately in the design and construction cycle.

Carbon Regeneration

After elution, barren carbon will be transferred from the elution column to the carbon dewatering screen prior to entering the feed hopper of the horizontal carbon regeneration kiln. Any residual water in the feed hopper will also drain before the carbon enters the kiln. In the kiln the carbon will be heated to 650-750°C and held at this temperature for 15 minutes to allow regeneration to occur. Regenerated carbon exiting the kiln will be quenched by spray water on the carbon sizing screen. Screen oversize (regenerated, sized carbon) will be returned to the CIL process (usually Tank 7). The quench water and fine carbon will report to the carbon safety screen.

17.2.10 Cyanide destruction circuit

The SO_2/Oxygen process was selected to reduce weak acid dissociable cyanide (CNWAD) in the plant tailings stream to less than 50 ppm. After passing through the carbon safety screen the CIL tailings slurry will flow to the cyanide destruction process consisting of two agitated tanks which can operate in either series (normal operation) or parallel configuration; each tank will provide a one hour residence time.

The SO_2 source for detoxification will be sodium metabisulfite (“SMBS” $Na_2S_2O_5$). Oxygen will be supplied from the onsite oxygen generating plant. The detoxification process requires the presence of a soluble copper catalyst which will be supplied by metering copper sulfate solution directly from a mixing tank. Hydrated lime slurry from the CIL distribution ring main will be added to maintain pH in the range of 8.0 to 9.0. Cyanide analysers in the CIL circuit and the destruction circuit will monitor free and CNWAD cyanide respectively and can be used to adjust the flow of SMBS and copper sulfate ($CuSO_4$) to control CNWAD cyanide discharge. Probes measuring Eh and pH will be will also be included.

17.2.11 Tailings disposal and decant return

Tailings from the cyanide destruction process will flow to the tailings disposal tank and will be pumped to the cyanide tailings storage facility (TSF) by multi-stage pumps. Decant return water from the cyanide TSF will be pumped to the process water tank for reuse in the CIL circuit. Additional detoxification of TSF return water will be provided by an IBC containing hydrogen peroxide on an as required basis.

Flotation tails from the float tails thickener will be pumped to a tails storage tank. From here, multi-stage pumps will transfer the tailings to the non-cyanide TSF. Decant return water from the non-cyanide TSF will be pumped to the flotation water pond for reuse.

17.2.12 Reagents and consumables

Table 106 summarises the reagents and consumables required for the Namdini Gold Project. Sufficient stocks will be maintained on site (2 to 4 weeks) to ensure that supply interruptions due to transport or weather do not impact production. Cyanide will be an exception, where stocks for three months will be kept on site.

Table 106: Reagents and consumables summary

Description	Application	Mode of Delivery	Method of Handling
Hydrated lime	pH control in CIL and Detox circuits	1 tonne bulk bags	Mixed to 25% w/w, distributed by ring main
Potassium Amyl Xanthate (PAX)	Sulfide flotation collector.	Drums	Mixed to 10% solution, meter pumped to flotation
A7249	Sulfide flotation collector.	Drums	Mixed to 4% solution, meter pumped to flotation
Copper sulfate	Flotation conditioning agent Catalyst in detox	25 kg bags	Mixed to 20% w/w, metered to flotation and detox.
Sodium cyanide	Leach reagent for CIL and elution	Dustless pellets in 1 tonne boxed bulk bags	Mixed to 20%w/v, metered to CIL and elution
Caustic soda	pH modifier in elution	25 kg bags of pearl pellets	Mixed to 20%w/w, metered to elution circuit
Hydrochloric acid	Carbon acid wash reagent	32% concentrated acid delivered in 1,000 L isotainers	Metered to acid wash feed line to obtain a 3%w/w solution
Activated carbon	Gold adsorption in CIL	500 kg bulk bags	To carbon sizing screen (vendor package)
Sodium metabisulfite	Reagent for Detox	1 tonne bulk bags	Mixed to 20% w/w, metered to detox circuit
Flocculant	Flocculant	25 kg bags	Mixed and diluted in vendor package, dosed to pre-leach thickener, float tails thickener
Grinding media	SAG mill balls	200 L drums	Charged by FEL to mill feed conveyor.
	Ball mill balls	200 L drums	Charged by overhead crane
	HIG Mill Media	2 tonne bags	Charged by overhead crane
Diesel	Heat source	Trucked and pumped into site storage tank	Reticulated to elution heater, carbon kiln and smelting furnace
Anti-scalant	Grinding water circuit	1,000 L isotainers	Dosed by metering pump at 100% strength

17.2.13 Services

Raw water supply and distribution

Raw water for the project will be extracted from the Volta river and pumped to a thirty day capacity raw water storage dam located east of the plant site. In addition, seasonal precipitation plus water runoff from the waste rock dumps will be collected in the Pollution Control Dam, which will be pumped back to the process water storage tank at the plant. The dam will feed an onsite raw water tank.

Raw water will be used to feed the potable and stripping water treatment plants as well as reagent mixing, and gland water.

Fire water

Fire water for the process plant will be drawn from the bottom of the raw water tank (the reserve is based on four hours of continuous firefighting).

The fire water pumping system will consist of:

- an electric delivery pump to supply fire water at the required pressure and flowrate
- a diesel driven pump that will automatically start in the event that power is not available for the electric pump and
- an electric jockey pump to maintain fire ring main pressure.

Fire hydrants and hose reels will be placed throughout the process plant, fuel storage and plant offices at intervals that ensure complete coverage in areas where flammable materials are present.

Potable water

Water from the raw water tank will supply the potable water treatment plant. Water treatment will include filtration, carbon contacting, and chlorination. Additional ultra-violet sterilization units will be installed on outgoing potable water distribution headers if required. Potable water will be reticulated from the potable water storage tank to the site ablutions, safety showers and other potable water outlets. Potable water will also be pumped to the mine services facilities.

Process water

Process water will be separated into two water management systems. The flotation water circuit with water recovered from the flotation circuit will be returned to the grinding circuit, and the process water circuit with water recovered from the CIL tails thickener will be returned to the CIL circuit. The two circuits are designed to operate separately to avoid cyanide and high pH water being returned to the flotation circuit. The presence of cyanide and/or high pH in the flotation circuit will have a detrimental effect on gold recovery from flotation. Separate storage dams/tanks will be provided for each water management system.

Flotation water will consist primarily of recycle from the flotation thickener overflows with make-up from raw water and any surplus mine discharge water. The flotation water will be fed to the grinding circuit, flotation circuit, regrind circuit, and flotation concentrate thickener feed dilution.

Process water will consist primarily of the CIL tailings decant return. Seepage pond, monitoring and event pond water can also be returned to the process water tank.

Duty and standby pumps will be provided for raw water, grinding water and process water. The opportunity will exist for anti-scalant to be added to condition the grinding water and reduce fouling of pipelines, spray nozzles and screen decks.

Air services

Plant and instrument air will be supplied from two high pressure screw compressors. The air will be dried before distribution with separate air receivers supplying plant and instrument air and an auxiliary plant air receiver located at the crusher.

Low pressure blowers will be dedicated to the flotation circuit with one duty and one stand-by.

An oxygen plant producing at least 90% purity and 1,200 Nm^3/h will be installed to supply the oxygen requirements of the pre-leach and CIL as well as the cyanide destruction circuit.

17.3 Plant area design

17.3.1 Plant layout and design considerations

The civil, mechanical and electrical design of the plant facilities is based on standards deemed appropriate for the local climatic conditions and reflective of the local topography. The plant layout and equipment selection considers the requirements necessary for Project implementation.

17.3.2 Site selection

The plant site geotechnical investigation conducted by Knight Piésold was undertaken on a previous plant layout. For the initial layout, the ground conditions are within a reasonably consistent geological profile. The new process plant location is in close proximity to the original and it is unlikely that the new site will require improved ground conditions.

A further Process Plant geotechnical site investigation will be required in the future.

The Process Plant site location is shown in Figure 5. The plant is to be located on relatively flat terrain to minimize earthworks and is to the north west of the open pit and directly north east of the waste dump. The tailings storage facility is to the south and south west of the open pit.

The proposed plant layout is shown in Figure 131. An allowance for the Mine Services Area is included in the layout.

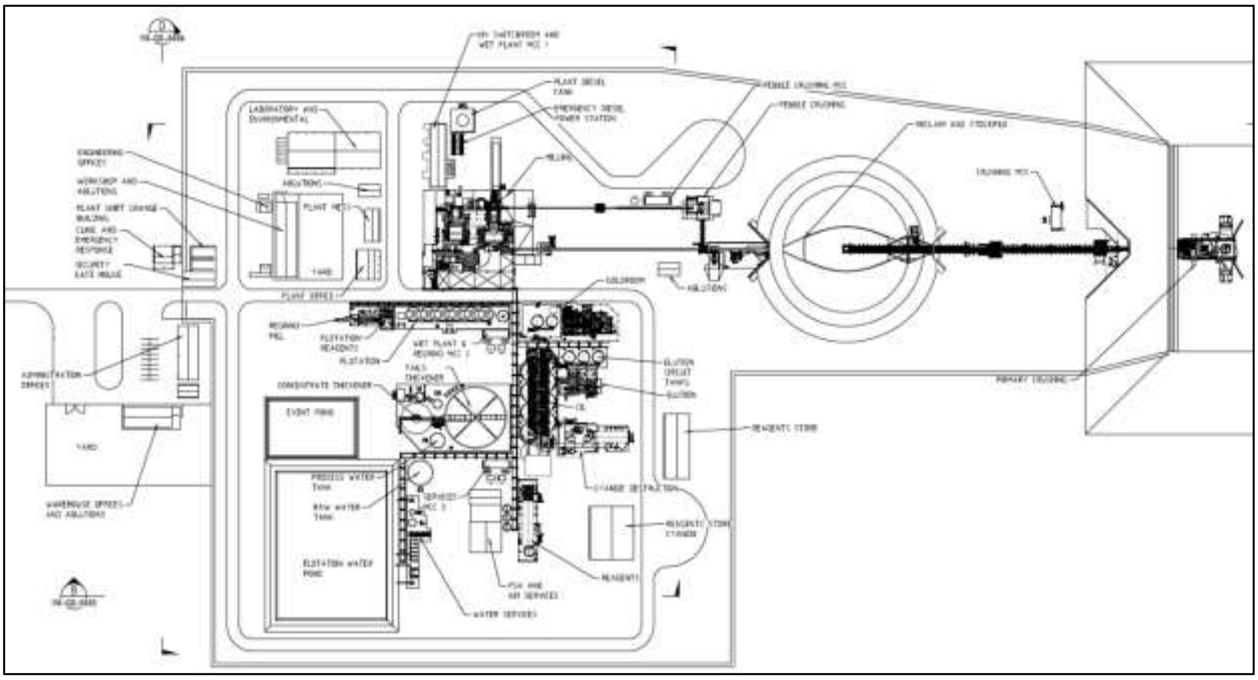


Figure 131: Proposed plant layout (source: Lycopodium, 2018, 2006-110-GD-004_A)

General design considerations

Process equipment was selected to meet the process duty defined by the process design criteria. A suitable design factor was added to the duty requirements to account for the minor process interruptions and the inevitable variability that occurs in an operating plant.

Typically, manufacturers' standard equipment was adopted for design and costing purposes. The equipment was selected on the basis of its criticality to the process and suitability for use within the operating environment.

For the management of cyanide, the design addresses reagent unloading, storage, handling, containment and detoxification of cyanide containing process streams. The cyanide handling area is to be located remote from offices and workshops and packaging will be disposed of by incineration. This design approach aligns with the requirements of the International Cyanide Management Code as well as local regulatory requirements for dangerous goods.

The design also addresses the environmental requirements for the Project. This includes the provision of suitable dust suppression, fume extraction and treatment, discharge stream treatment and bunding of relevant areas.

Plant layout

The Process Plant and infrastructure layout was developed with consideration given to site access, proximity to the pit, waste storage areas and optimization of site preparation effort.

Layout of plant equipment was arranged to satisfy the following criteria:

- Grouping of process equipment by unit operation circuit or similar equipment type, to facilitate monitoring, control, operability, containment and recovery of any spillage
- Ease of access for maintenance
- Use of gravity, where possible, to minimize transfer equipment, especially for solid materials.
- Logical flow of material and activity from one end of the plant to the other whilst being cognisant of process flow requirements.

Maintenance access

The key criteria for equipment selection are the suitability for duty, reliability and ease of maintenance. The plant layout provides ease of access to all equipment for operating and maintenance requirements while maintaining a cohesive footprint to minimize construction costs.

Process Plant equipment will be maintained in the workshop located within the Process Plant. This facility will be equipped with an appropriately sized overhead crane.

Spillage handling

Throughout the plant and as determined by the process requirements, there will be a number of self-bunded areas.

Each of the bunded areas will have a dedicated sump pump for spillage control.

An event pond was included to contain flows from a storm event and 110% of the largest process tank. The location of the event pond had been determined by contours and is near the milling area.

Spillage within the reagents area will disposed to locations as nominated in Table 107 below.

Table 107: Spillage plan within reagents area

Reagent	Spillage Directed To
Hydrated lime	Concentrate Thickener
Sodium cyanide	CIL Feed Box
Caustic	Caustic Mixing Tank
Flocculant, Flotation Collector	Flotation Tails Thickener
Copper sulfate	Rougher Conditioning Tank
Sodium metabisulfite, hydrochloric acid	Cyanide Destruction Tank

Pipe racks

The plant was designed with pipe racks to facilitate process piping routes. The pipe racks will also accommodate electrical and instrumentation cable trays.

A central multi-level pipe rack was included from the grinding area through to CIL and detoxification. To service flotation and air services a similar concept was adopted at right angles to the central pipe rack. Pipe off-take from this elevated structure enables piping to be distributed to the mechanical equipment. The distance between levels is cognisant of maintenance requirements. Sufficient space allowance for light vehicle and maintenance access was provided.

Low level pipe supports for water services and other minor services is also included in the layout.

Plant fencing and security

This is addressed in Section 18.15.

17.4 Instrumentation and control

The level of instrumentation and control was selected to provide basic regulatory control to maintain steady operation with minimal process excursions. Following industry practice for similar size plants, a supervisory control and data acquisition (“SCADA”) and programmable logic controller (“PLC”) architecture was selected for the plant wide process control system as a reliable and low-cost approach.

The SCADA/PLC will integrate original equipment manufacturer controllers in the field into a uniform operator interface located in the main control room. Where vendor packaged process control systems are not available, logic will be developed at the SCADA level for process control and monitoring.

The control room will house two PC based operator terminals which both act as the control system (SCADA) servers as well as engineering/operator stations. The system will include a historian capability for data analysis and reporting.

In general, the status of process plant drives will be reported to the SCADA and displayed in the control room. Local control stations will be located in the field in proximity to the relevant equipment. These will, as a minimum, contain Start and Lock-Off-Stop pushbuttons hard-wired to the drive starter. Local selection will allow each drive to be started and stopped by the operator in the field by pushbutton. Plant drives will generally be started remotely from the control room. Remote selection will allow the equipment to be started from the control room. Status indication of process interlocks as well as the selected mode of operation will be displayed in the control room.

Safety interlocks such as emergency stops and thermal protection will be hardwired and will apply in all modes of operation. All software process interlocks will also apply in both Local and Remote modes.

17.5 Metallurgical accounting

A weightometer on the primary crusher discharge conveyor will measure the primary crushed ore tonnage. A second weightometer on the SAG mill feed conveyor will determine mill feed tonnes. A weightometer on the pebble return conveyor will indicate the pebble recycle load.

Cross-cut slurry samplers will be located on the flotation feed, flotation tailings, CIL feed and CIL tailings streams to provide representative samples for metallurgical accounting and process control.

On stream analysis will be provided to assess grind performance and metallurgical efficiency.

Density and flow meters on the flotation and leach feed streams will allow the dry tonnage of solids to be determined. In conjunction with the feed and tails slurry samples, the mass flow measurements will allow the gold recovered in the CIL to be calculated.

Regular ‘in circuit' surveys will allow reconciliation of precious metals in feed compared to doré production. Weights and assays of the doré bars will be reconciled against the calculated gold recovery.

Reconciliation of the amount of reagents used over relatively long periods will be achieved by delivery receipts and stock takes. On an instantaneous basis, reagent usage rates of cyanide, elution and detoxification reagents to unit operations will be measured (L/min) and accumulated (m^3) using flow meters.

18.0 PROJECT INFRASTRUCTURE

18.1 Hydrogeology and hydrology

Comprehensive field programmes and reports were completed by Golder on the Namdini Gold Project for the hydrology (Golder Africa, 2018a) and the hydrogeology (Golder Africa, 2018b).

A hydrogeological fieldwork program was undertaken comprising a hydro-census of surrounding properties to identify groundwater users. Groundwater exploration drilling of five pairs of boreholes converted to deep and shallow monitoring wells was completed. Characterization of groundwater quality by sampling and laboratory analysis, groundwater monitoring and hydraulic testing was completed. In support of the mine design, a conceptual model was developed for assessment of pit inflows and the potential impacts of mine dewatering on local groundwater, regional groundwater and surface water systems.

A hydrology program including the development of a stormwater plan and overall site water balance was completed. Hydrological design criteria are being developed, largely based on International Finance Corporation requirements.

The water table level around the pit and potential ingress is summarized as follows:

- Groundwater levels show variability across the site but are generally 7 to 15 m below ground level for shallow bores targeted in the saprolite and saprock. Deeper exploration holes show depressed water levels up to 85 m below ground level with an average of 42 m below ground level. Water level fluctuations may differ up to 4 m during the dry season in shallow bores.
- Groundwater recharge to the aquifer systems is mainly by direct infiltration of precipitation through fractured and fault zones along the highland fronts, and through the sandy portions of weathered zones.
- Expected groundwater yields for bore holes will be highly variable due to the anisotropy of the aquifer systems, with the majority of water supply bore holes having low yields between <0.1 to 0.5 l/s.

18.2 Seismicity

The seismicity of much of West Africa is typical of an intra-plate region, characterized by low levels of seismic activity and earthquakes apparently randomly distributed in location and time. The correlation between recorded earthquakes and geological features is typically not well understood. There are no historical earthquakes recorded near the site. A total of 13 earthquakes have been recorded within 500 km of the site, although these comprise small events ranging from magnitude 2.9 to 5.2. The closest earthquake to the site occurred in 1993 and was located approximately 240 km north-east of the site with a magnitude of 4.4.

Based on this information, the following design bases were developed:

- Tailings Storage Facilities (ANCOLD) – 0.15g Peak Ground Acceleration
- Buildings (IBC) – 0.039g Peak Ground Acceleration (max 0.083g).

18.3 Site layout

General

A site layout was developed based on the following information:

- Total ore tonnage – 129.6 Mt
- Process throughput – 9.5 Mtpa
- Flotation/Concentrate split – 92.5%:7.5%
- Tailings to Flotation TSF – 120 Mt

- Tailings to CIL TSF – 9.6 Mt
- Life of Mine pit extent
- Life of Mine waste dump footprint
- 1 m contour topography over approximately 9 km by 6 km plan area broadly encompassing the project area, together with a preliminary site access corridor from the west and north-west
- Mining lease boundary.

Waste dump

Approximately 180 Mt (80 Mm^3) of waste will be generated from the life of mine open pit development. The waste dump will be located directly to the west and north-west of the open pit, and is bounded to the east by the project mining lease. The location of the plant, pit and waste dump is shown in Figure 132.

Process plant and mine services area

The process plant (and mine services area) will be located on relatively flat terrain to the north-north-west of the open pit and directly to the north-east of the waste dump.

Tailings storage facilities

The lease boundary, pit outline and waste dump footprint were provided. In addition, Knight Piésold was advised of a number of grade anomalies to the north and north-west of the pit which were not yet sterilized. On this basis the available area for siting of tailings storage facilities was limited to south and south-west of the open pit.

For the PEA study the design concepts for the facilities were to site the Flotation TSF to the west of the plateau using the escarpment as the eastern perimeter of the facility and to site the CIL TSF on the plateau to the south of the open pit. For the Pre-feasibility Study the TSF designs were further optimized using these locations and incorporating more recent topography for the site area together with the updated design parameters.

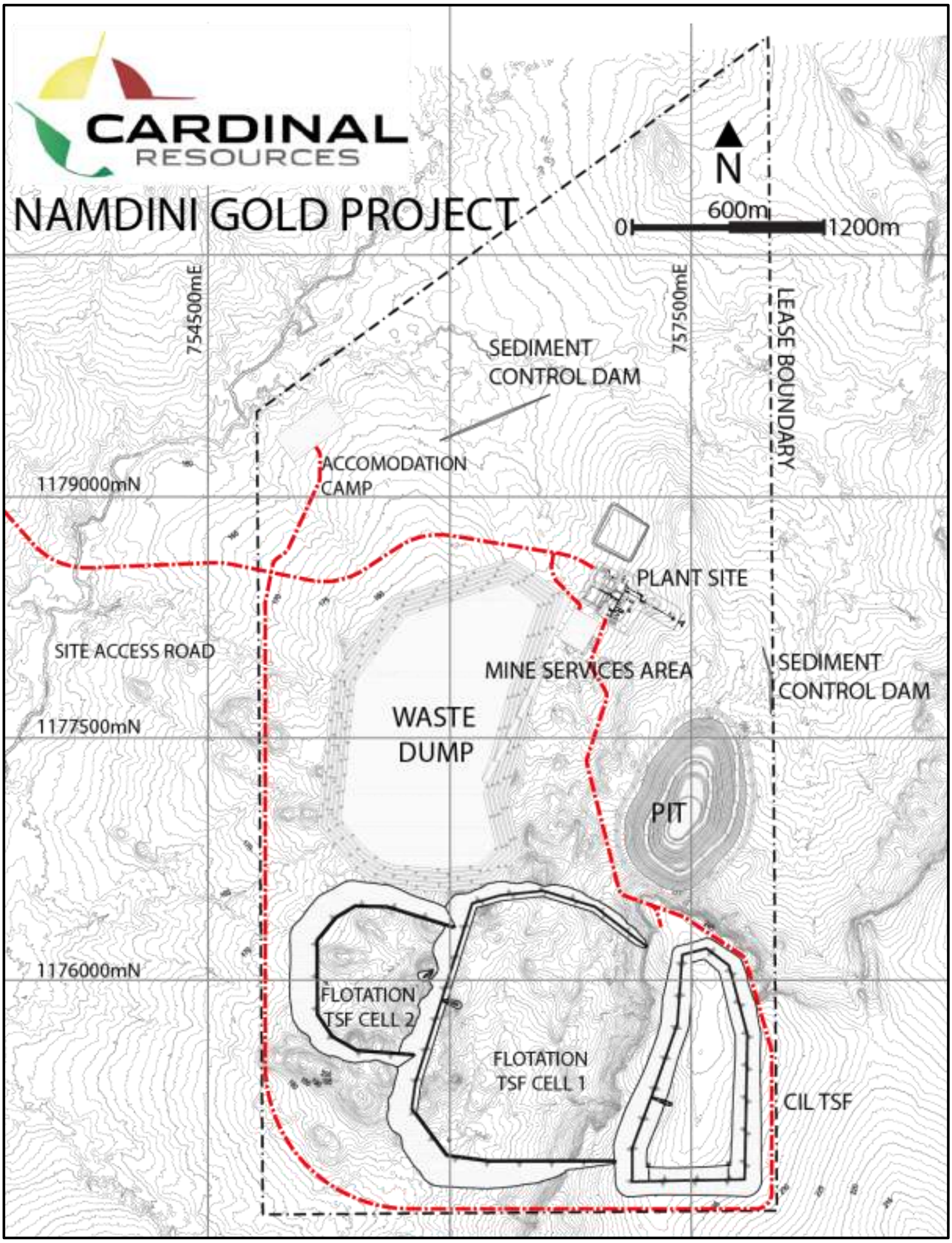


Figure 132: Site infrastructure layout

18.4 Geotechnical investigation

Geotechnical investigations of the Flotation TSF, CIL TSF and process plant sites were carried out by Knight Piésold as part of the Pre-feasibility Study to investigate the sub-surface conditions and to provide geotechnical parameters for design. The scope of the site investigation comprised the following:

- Four diamond cored drill holes within the Flotation TSF footprint (BH-NM-01 to BH-NM-04)
- 73 test pits within the Flotation TSF footprint (TP-NM-01 to TP-NM-073)
- Two diamond cored drill holes within the CIL TSF footprint (BH-NM-05 and BH-NM-06)
- 24 test pits within the CIL TSF footprint (TP-NM-074 to TP-NM-097)
- Four diamond cored drill holes within the process plant footprint (BH-NM-07 to BH-NM-10)
- 18 test pits within the process plant footprint (TP-NM-098 to TP-NM-115)
- Standard Penetration Testing in the drill holes
- Disturbed sampling from the test pits
- Disturbed sampling from potential drainage medium borrow sources
- Laboratory testing of selected samples.

A summary of the findings is as follows:

- The sub-surface profiles at each of the proposed infrastructure locations generally comprised a variable thickness of alluvial/colluvial and/or residual/saprolite soils overlying bedrock. With depth the residual/saprolite soils increasingly exhibited parent rock structure and generally caused excavation refusal. Metavolcanic, metasediment or granite bedrock was encountered below the residual/saprolite horizon at each infrastructure location and at relatively shallow depth. The rocks exhibited extremely to very low strength becoming medium to high strength with depth. Details relating to individual horizons and depths at each infrastructure location are available.
- Groundwater was not encountered during this investigation. It is understood that groundwater resides at an average depth of 42 m within the open pit footprint.
- A detailed laboratory testing programme is in progress.
- Recommended geotechnical parameters for analysis and design purposes were estimated and are available.
- Substantial quantities of borrow material may be sourced from the alluvial, colluvial and residual/saprolite horizons which are present across the site. However, a detailed borrow materials assessment will be required as part of the next stage of project development to delineate and quantify specific sources of material for the earthworks construction.

18.5 Roads

18.5.1 Site access

A new site access road is to be constructed between the main N10 route and the project site. The access corridor is aligned approximately west-south-west to east-north-east and will originate at a junction with N10 approximately 16 km south-southeast of Bolgatanga and terminate at the process plant. The corridor will be shared with the mains power supply to the project over the whole alignment and with the raw water supply from the White Volta River over the last 9 km. The site access road will connect with a network of internal roads which link the various site facilities.

The N10 provides good access to the major cities and ports in southern Ghana and no upgrades of the N10 will be undertaken. The national road N10 will be the most likely route used for transporting construction materials and operating supplies to the site from the ports in southern Ghana.

18.5.2 Site roads

Site roads will be 'fit for purpose' and will comprise haul roads for mining use, full width gravel roads for frequent traffic by site light and heavy vehicles and basic access tracks for infrequent access by light vehicles to site infrastructure.

18.6 Power

18.6.1 Power supply

The design is based on establishing a grid power supply to the process plant. This is achieved by constructing a new GRIDCo switchyard near the process plant to step-down the incoming 161 kV supply to 11.5 kV for plant power distribution. The preliminary scope includes:

- Augmentation to the existing GRIDCo’s Bolgatanga substation
- Construction of a new 161 kV power line between Bolgatanga and new GRIDCo substation near the process plant
- Construction of a new 161/11.5 kV, 45/50 MVA, switchyard, near the process plant.

No allowance was made for Reactive Compensators to stabilise the grid power supply during start-up of mill drives, assumed the grid supply is steady and healthy, based on the previous observations and measurements carried-out by Cardinal.

The estimated power consumption details are shown in Table 108.

Table 108: Estimated power consumption

Installed Power MW	Maximum Demand MW	Plant Throughput
59.588	44.828	7.0 Mtpa

18.6.2 Power distribution

The plant power distribution voltage will be maintained at 11 kV. The 11 kV supply feeder from the GRIDCo switchyard will feed the plant main 11 kV distribution board located near the milling area switch room for distribution of 11 kV power supply to various sections of the process plant.

For process plant reticulation, the 11 kV supply will be stepped down to 415 V at the switch rooms using 11 kV/415 V distribution transformers. These transformers will be fed from the main 11 kV switch room/switchboard. These low voltage (“LV”) switch rooms will house 415 V motor control centers (“MCC”s). These LV MCC/switch rooms will be located at various load centers of the process plant for LV power distribution.

The MCCs will be double sided (back to back), demountable switchgear panel design with Form 4 segregation and Type 2 coordination as per Australian Standards.

18.6.3 Electrical switch rooms

Electrical equipment such as switchgear, secondary substations, MCCs, panel boards, UPS, process control system, I/O cabinets, etc. will be housed in designated electrical switch rooms within and around the process plant in a climate-controlled environment.

Air-conditioning units will be provided to control the humidity and room temperature. Electrical switch rooms will be raised 1.8 m above standard floor levels to prevent flooding and provide access for cables.

Man-doors and double doors will be equipped with panic bars and will open outwards. The electrical switch rooms will be sealed to provide the required fire rating.

18.6.4 Transformers and compound

All plant distribution transformers, 11/0.415 kV, will be of ONAN cooling configuration and vector group Dyn11. Fire rated walls will be constructed around pad mounted transformers. Standard oil containment facility will be provided to comply with the design standards.

18.6.5 Emergency power supply

Emergency power for process plant essential drives/facilities during grid power supply outages will be supplied from 2 × 2.5 MVA, High Speed (HS) diesel generator units operating on diesel fuel oil. These emergency power generators are connected to the process plant main 11 kV distribution switchboard to be operated during grid power outages.

Based on the power supply reliability study done by Cardinal, the grid supply is assumed to be reliable and the expected number of power outages per annum should be low. The utilization of the emergency power station should be relatively low.

The changeover from grid supply to emergency power will be done manually. No automatic changeover has been allowed in the study.

18.6.6 Power demand

Process plant power demand

The estimated load for the Namdini Process Plant is as follows:

- Connected Load 59.588 MW
- Nameplate Load 56.792 MW
- Maximum Demand 44.828 MW
- Average(operating) Load 41.704 MW
- Largest Size Motors 2 × 7.580 MW SAG MILL dual drives on VSDs
- Overall Power Factor – corrected at the 11 kV bus 0.95 or better

A large proportion of the electrical load will be due to the process plant. The process plant is expected to run continuously for 24 hours a day, 7 days a week, with +91% availability. The load list summary and the process plant power utilization by process have been developed to an appropriate PFS level, as have been the voltage selection and power system earth design.

18.6.7 Construction power

The construction power demand is estimated to be about 1,500 kVA during peak construction activities. One 2,500 kVA diesel (natural gas) generator set will supply power to the different areas of the process plant and mine site during construction.

This diesel generator set will be used as part of the mine emergency power supply for the process plant and mine site upon the completion of the construction. All mining and construction contractors have based their quotes on construction site power supplied by generator sets.

18.6.8 Power quality

Power factor correction capacitors will be used to improve process plant overall power factor. Multi-stage capacitor banks in an outdoor containerized switch room will be provided to improve the overall site power factor. The capacitor banks will be connected to the process plant main 11 kV switchboard.

18.6.9 Lightning protection system

The lightning protection system will use the most suitable location(s) for the air terminal(s) to provide the zones of protection to the structure/building against direct lightning strikes. The method to determine the zones of protection will be based on the 'Rolling Sphere' technique or equivalent method.

Air terminal(s) shall be installed at any salient point (e.g. corners and highest point) on the structure/building/roof to achieve the high probability interception point.

In addition, all building columns and lighting poles will be grounded. This provides extra risk mitigation on top of the air terminals installed for the lightning protection system.

18.7 Plant control and instrumentation

The general control philosophy for the plant will be one with a moderate level of automation and control facilities to allow process critical functions to be carried out with minimal operator intervention. Instrumentation will be provided within the plant to measure and control key process parameters. On stream analysis will be provided to assess grid performance and metallurgical efficiency.

The main control room, located near the laboratory and admin building area, will house three PC based operator interface terminals (“OIT”s), two plant OITs and one engineering workstation. These workstations will act as the control system supervisory and data acquisition (SCADA) terminals. All key process and engineering parameters can be monitored and trended through the process control system (PCS).

The process control system will be a programmable logic controller (PLC) and SCADA based system. The PCS will control the process interlocks and PID control loops for non-packaged equipment.

In general, the process plant drives can be controlled from the plant operator station in the main control room. Local Control Station (“LCS”) units are in the field in proximity of the relevant equipment. These LCS units have Lock-Off-Stop pushbuttons which are hard-wired to the drive starters. Local and Remote drive selection can only be done by the OIT, no field selection will be allowed.

The OITs will allow drives to be selected to Auto, Remote, Maintenance or Out-of-Service modes by the drive control popup/faceplate.

Safety and statutory interlocks such as emergency stops, pull wire switches are hardwired and will apply in all modes of operation. All PLC generated process interlocks will apply in Auto, Local, and Remote modes of operation.

Local selection will allow each drive to be operated by the operator in the field by the local start/stop pushbutton which is connected to a PLC input. In Remote mode equipment can be operated from the central OIT in the control room by the drive control popup.

Vendor packages will use vendor standard control system as required throughout the plant. Vendor packages will generally be operated locally with limited controls from the PCS. General equipment operating status, fault alarms, etc. will be monitored by the PCS.

18.8 Water supply

A river abstraction system will be installed to provide any shortfall in process water requirements during the operation. An abstraction tower will be constructed on the northern bank of the White Volta River approximately 8.5 km to the west of the process plant. This will comprise submersible pumps situated within an intake tower located within a trench excavated into the northern bank of the White Volta River. A water storage facility will store 30 days' supply of process water to account for periods during which pumping from the river is not permitted. The facility will comprise a lined turkey's nest pond located directly to the north of the process plant.

A pipe branch from the main raw water pipeline will supply the potable water treatment plant located at the camp that will purify the water after which it will be reticulated across the site.

A vendor packaged modular potable water treatment plant including filtration, ultra-violet sterilization and chlorination will be installed at the accommodation camp with the treated water reticulated to the site buildings, ablutions, safety showers and other potable water outlets.

18.9 Sewage and waste management

18.9.1 General

Grey water and effluent from all water fixtures will drain to gravity sewerage systems at the camp and plant site. Where gravity flow is not practicable suitable macerator pumps will be used.

Effluent will be treated in a sewage treatment plant located adjacent to the camp. The effluent treatment demand for the plant was estimated at 55 m^3/day with the sewage treatment plant having been sized accordingly.

Treated effluent will be discharged into leach drains. Treatment plant sludge, following chlorination, will be suitable for direct landfill burial in unlined pits.

18.9.2 Solid waste

Solid waste will be sorted and reused or recycled as far as the limited access to recycling facilities allows.

General solid wastes will be deposited into a landfill and promptly covered to deter vermin and scavengers.

Materials such as cyanide packaging will be burned and the ashes buried, under supervision, on-site beneath mine waste to prevent unintended use.

18.9.3 Hydrocarbon waste

Waste lubricating oils will be returned to the supplier for recycling.

Hydrocarbon contaminated materials will be spread on volatilization pads for decontamination before disposal in landfill sites.

18.10 Water management

A water management model was developed to understand the hydrology, TSF water balance and the TSF/plant interaction so as to determine the TSF water demand, and to generate design embankment crest levels to maintain containment throughout the operation. The model was developed to calculate process water shortfall and hence to quantify the volume of water required from external sources.

The main components of the water management model are:

- tailings storage facilities consisting of one CIL cell and two flotation cells
- rainfall run-off from TSF catchment area
- process plant balance.

The model was setup as a monthly time-stepped model to cover the whole operational life of each facility. A detailed water balance was carried out for both average conditions and a range of extreme climatic conditions, as follows:

- Average rainfall and evaporation conditions
- 100 year average recurrence interval (ARI) 72 hour storm events with no evaporation or decant return
- 100 year ARI wet precipitation at the middle and towards the end of TSF operating life.

Results indicate the following:

- The CIL plant only requires a small quantity of the CIL decant return, hence no external make-up other than a nominal raw water requirement is needed. As a result, a large proportion of the CIL decant return needs to feed the flotation plant as recycle.
- The flotation plant requires some additional water make-up. This can be sourced either from the Flotation TSF in the wet season if available (currently limited to 90% of water in slurry, however higher values are possible) or the river abstraction system.

18.11 Workforce accommodation

18.11.1 Construction accommodation

An area adjacent to the permanent camp and the contractor laydown areas will be made available to be used by the early earthworks and accommodation camp installation contractors. All contractors will provide their own temporary accommodation and will not be accommodated in the permanent camp. The permanent camp will be used for the Owner, EPCM contractor staff and senior contractor personnel subject to availability.

18.11.2 Permanent accommodation camp

Where possible, employment will be offered to suitably qualified and experienced Ghanaians. All unskilled and semi-skilled positions will be filled by residents of local towns and villages. A bus service will be provided to and from local population centers for workers. It is anticipated that a significant number of skilled Ghanaians from outside the immediate area will be allocated their own accommodation in local towns such as Bolgatanga.

Expatriate and key Ghanaian employees from outside the local area will be provided with accommodation. The project is based on accommodating 200 persons in a permanent camp. The camp will include a dry mess/kitchens, laundry, gymnasium, wet mess and recreational facilities.

The cost estimate is based on a fully modular camp facility. However, it is likely that the camp will be a mix of imported, modular, prefabricated buildings and blockwork construction. Experience is that costs are similar but that modular units can be brought on site, be ready to use in a shorter timeframe and require a smaller site labour force for erection. A commitment to local content will drive the use of local blockwork construction where the building is not schedule critical.

18.12 Communication system infrastructure

Internal communications and IT services will be by a site wide fibre optic network.

A local mobile phone provider will be contracted to upgrade existing facilities on site and provide a link into the local, national and international telecommunication network.

A radio network will be established with dedicated operational, security and emergency channels.

A local ground station will be installed to provide global satellite voice and data connection.

Satellite TV and internet connection will be provided at the accommodation camp.

18.13 Fuel and lubricant supply

Diesel fuel will be stored on site. Allowance was made for six 62.5 m^3 self-bunded fuel storage tanks. These will be mounted on concrete plinths and will include piping, pumps, meters and an electrical fuel management system. A small office will be included.

Fire water and other services will be provided to the fuel depot.

18.14 Explosive storage and handling

It is anticipated that a contract will be entered into with a recognized supplier of mining explosives to establish their own facilities on site and supply emulsion explosives, initiators, detonators and other blasting consumables as needed.

18.15 Security and fencing

Site security is based on concentric lines of fencing and control.

Areas of the lease where operations are actively taking place or where items of decentralized infrastructure are located will be patrolled by the security team.

The Process Plant, Mine Services and General Administration areas will be enclosed within a patrolled 2 m chain link fence line to discourage casual entry. The main point of entry will be where the main access road enters the site. This point of entry will be provided with a gate and manned security post. Access from the mine haul road through the mine services area will also be monitored by a manned security post. Entry into the fenced areas will require a mine identity card and/or proof of legitimate business beyond that point.

The process plant itself will be enclosed by a double line of security fencing monitored by closed circuit cameras. The fence line will be provided with perimeter lighting. Entry will be by a single monitored security post and will be strictly controlled. Exit from the plant area will be subject to a search of vehicles, toolboxes and a 'pat down' and/or metal detector search of all persons.

Access to the gold room within the plant will be restricted and strictly controlled. Extensive camera surveillance will be installed and entry points will be monitored and alarmed. All personnel allowed into the area will be accompanied and monitored by members of the security team. Persons leaving the area will be subject to a comprehensive search of themselves and any tools or equipment leaving the building.

The accommodation camp will be fenced and provided with a manned entry gate to prevent unauthorized access.

The TSF will be provided with a perimeter stock fence comprising three strands of barbed wire to prevent wildlife access to the facility. Active landfill areas will be fenced to prevent wildlife and vermin access.

18.16 Operational facilities

Appropriate buildings and facilities have been allowed for in the PFS estimates as follows:

18.16.1 Plant area

Workshops, warehouses and the like will be of structural steel frame and metal cladding construction on concrete slabs. Office and amenity areas associated with the main structures will generally be of transportable/prefabricated style construction although concrete blockwork construction will be considered to provide additional local content if the schedule allows. The facilities are summarized below:

- Plant gatehouse for access control to the plant security area and all security monitoring functions; the facility will include change rooms, washrooms and laundry for plant staff (214 m^2)
- Plant offices and control room (225 m^2)

- Plant staff mess (140 m^2)
- Shift change (200 m^2)
- Laboratory/environmental, including sample preparation area, fire assay facilities and wet laboratory (1,100 m^2)
- Electrical switch rooms, of prefabricated construction, mounted on plinths for bottom cable entry
- Reagent stores (1,400 m^2)
- Plant workshop/warehouse with offices and ablution (360 m^2).

18.16.2 Administration

- Administration Office (347 m^2).
- Site Warehouse (300 m^2).
- Clinic/Emergency Response (143 m^2).

18.16.3 Mine Services Area facilities

An area will be provided for the mining contractor to establish their offices, workshops and other facilities. Power, potable water and connection to the site sewerage facilities will be provided. The area will also have an office for the Owner's geology/mining technical team who will share the contractor’s facilities such as changerooms to avoid duplication.

The Mine Services Area facilities, based on a quotation, are summarized below:

- Vehicle workshop, tire store and warehouse
- Mine vehicle washdown bay
- Mine contractor’s offices
- Explosives contractor facilities
- Lube storage
- Crib/Training rooms
- Ablutions
- Owner’s team offices
- Site perimeter fencing.

18.17 Tailings storage facilities (TSFs)

18.17.1 Tailings testing

Two TSFs are required, to accommodate tailings with different properties from the Flotation and CIL operations.

Tailings were subject to physical testing. Results indicate that flotation tails will have a rapid rate of supernatant release of 46% of contained water excluding rainfall. CIL tails would be similar but at a slower rate. Ultimate settled density (air dried) was 1.47 t/m^3 for CIL tails and 1.67 t/m^3 for flotation tails.

Geochemical testing indicated the following:

- The flotation tailings samples recorded negative net acid producing potential (“NAPP”) values and weakly alkaline net acid generating (“NAG”) pH values. Therefore, the diorite and metavolcanic flotation tailings are classified as Acid Consuming (“AC”) and the granite rougher tailings as Non-Acid Forming (“NAF”).
- The CIL tailings sample recorded a positive NAPP and a low NAG pH, resulting in a classification of Potentially Acid Forming (“PAF”).
- On the basis of the multi-element results, both the Flotation and CIL TSF’s should be designed to prevent the loss of solids. The Flotation TSF will require a basic cover system on closure. The cover system for the CIL TSF will be driven by the need to control acid generation by precluding oxygen and water ingress to limit on-going oxidation of the tailings and seepage.

Based on supernatant analysis, the flotation tailings facility will require a compacted soil liner to limit seepage. In addition, the facility should have an underdrainage system to limit the hydraulic head acting on the soil liner. The CIL tailings facility will require a robust engineered liner system, likely comprising of a compacted soil liner with overlying HDPE liner and underdrainage system.

18.17.2 TSF design

Knight Piésold Pty Ltd (“KP”) was engaged by Lycopodium Pty Ltd to complete a pre-feasibility study for the following aspects of the project:

- Tailings Storage Facility (TSF)
- Site Water Management
- Surface Water Management
- Site Access Road
- Airstrip.

The work comprised risk assessment, site characterization (including climate analysis and seismic studies), review of the design concept, geotechnical site investigations, tailings characterization, water management, TSF design, infrastructure design (roads and airstrip), design, costing and consideration of monitoring and rehabilitation requirements.

18.17.3 Risk assessment

A consequence assessment was completed for the Flotation and CIL TSFs in accordance with the requirements of ANCOLD (2012a).

The hazard rating of a facility is derived by considering the potential impacts of a significant embankment breach and resulting release of tailings slurry in terms of safety, environmental and economic factors. The assessment was an Initial Assessment based on limited information relating to the population and communities close to and downstream of the project site. As such it provided order of magnitude impacts to define the consequence category for each facility for preliminary design.

A dam breach assessment was conducted for potential dam break scenarios of each TSF assuming a massive loss of containment. Dam break modelling assumes that a dam fails and does not consider the likelihood of such a failure occurring. The identified flow paths were used to determine the Population at Risk (“PAR”), the severity of damage and loss, and hence the consequence category of the facility. A significant failure of any of the TSF embankments would result in a release of tailings and/or water, though the extent and magnitude of the release would depend on the location of the breach, its size and the cause. For this assessment, the possible breach flow paths for each embankment at final height (when the facility is at its maximum tailings storage volume) were assessed.

18.17.3.1 Flotation TSF – Cell 1

A breach of the Cell 1 embankment could result in a tailings flow slide to the North. If this were to occur the tailings breach would likely impact the Process Plant. The flow slide could then continue to the North across the site access road, looping around to the west past the accommodation camp and then to the south, eventually flowing into the White Volta River. A breach to the South or West would be unlikely to impact any infrastructure directly but could flow into the White Volta River. A PAR of 10 to 100 was adopted for this scenario due to the possible impacts to the Process Plant.

18.17.3.2 Flotation TSF – Cell 2

A breach of the Cell 2 embankment would result in a tailings flow slide predominantly to the West. The breach would be unlikely to impact any infrastructure directly but could eventually flow into the White Volta River. A PAR of 1 to 10 was adopted for this scenario.

18.17.3.3 CIL TSF

A breach of the CIL TSF embankment could flow to the North and into the open pit. It is also possible that a breach could flow to the North, past the pit, looping around to the West and South, impacting the site access road and eventually flowing into the White Volta River. A breach to the South or East would flow down the valley and possibly into the White Volta River. It is not expected to impact directly on any of the project infrastructure. A PAR of 10 to 100 was adopted for this scenario due to the possible impact to the open pit.

18.17.3.4 Hazard assessment

The hazard assessments for the Flotation and CIL TSFs resulted in a consequence rating of “Catastrophic”. This rating was adopted as a flow slide resulting from any significant breach of either facility would be expected to flow into the White Volta River, which could then carry a portion of the released tailings approximately 440 km downstream into Lake Volta. As a consequence, the impacted area for breaches of either facility is in excess of 20 km^2. It is likely that there are a considerable number of communities downstream of the project site that rely on the White Volta River for water supply, agriculture, and other activities, and that could be severely impacted due to contamination of the river resulting from a significant breach of either TSF.

Based on a PAR of 1 to 10 and Catastrophic severity of impact, a consequence category of ‘High B’ was adopted for the design of Flotation TSF Cell 2. In accordance with ANCOLD (2012b) the minimum design criteria for this consequence category were applied to develop the design, construction and verification requirements for Cell 2 of the Flotation TS, including:

- Storm Allowance
- Contingency Freeboard Wave Run-up
- Spillway Capacity
- Earthquake Loading
- Stability Minimum Factor of Safety
- Dam Safety Inspection Frequency.

Based on a PAR of 10 to 100 and Catastrophic severity of impact, a consequence category of ‘High A’ was adopted for design of the Flotation TSF Cell 1 and the CIL TSF. Similar appropriate design criteria were then used to develop the design, construction and verification requirements.

The process criteria are shown in Table 109, and the design criteria are shown in Table 110 and Table 111.

Table 109: Flotation and CIL TSFs process design criteria

Design Component	Design Value		
	Option 1 – 4.5 Mtpa	Option 1 – 7.0 Mtpa	Option 3 – 9.5 Mtpa
Plant Operations – Processing Data			
	Total Ore Tonnage – 130 Mt Flotation/Concentrate Split – 9.25:0.75		
	Flotation Tailings – 4.16 Mtpa CIL Tailings – 0.34 Mtpa	Flotation Tailings – 6.48 Mtpa CIL Tailings – 0.52 Mtpa	Flotation Tailings – 8.79 Mtpa CIL Tailings – 0.71 Mtpa
Plant Operations – Flotation TSF			
TSF Storage Capacity – Final – Starter	120.25 Mt of dry tails over 25.1 years. 4.16 Mt of dry tails – 12 months initial capacity.	120.25 Mt of dry tails over 16.1 years. 6.48 Mt of dry tails – 12 months initial capacity.	120.25 Mt of dry tails over 11.9 years. 8.79 Mt of dry tails – 12 months initial capacity.
Production Rate	478 tonnes/hour of dry tails (average, based on 8,000 hours/yr).	744 tonnes/hour of dry tails (average, based on 8,000 hours/yr).	1,009 tonnes/hour of dry tails (average, based on 8,000 hours/yr).
Plant Operations – CIL TSF			
TSF Storage Capacity – Final – Starter	9.35 Mt of dry tails over 25.1 years. 0.34 Mt of dry tails – 12 months initial capacity.	9.35 Mt of dry tails over 16.1 years. 0.52 Mt of dry tails – 12 months initial capacity.	9.35 Mt of dry tails over 11.9 years. 0.71 Mt of dry tails – 12 months initial capacity.
Production Rate	84 tonnes/hour of dry tails (average, based on 8,000 hours/yr).	131 tonnes/hour of dry tails (average, based on 8,000 hours/yr).	178 tonnes/hour of dry tails (average, based on 8,000 hours/yr).

Table 110: Flotation TSF design criteria

Project Operations	
Capacity – Final – Starter	120.25 Mt. Throughput option dependent.
Slurry Characteristics	60% solids by weight. Slurry settled density = 1.40 to 1.62 t/m3. Supernatant release – 46% of water in slurry. Underdrainage release – ~15-20% of water in slurry. Non-Acid Forming.
Fluid Management	Partial basin drainage system drains by gravity to sumps in each cell and is then pumped into the supernatant pond. Decant tower removal of supernatant solution via a pumping system and pressure pipeline back to the plant.
Hydraulic Design	
TSF storm storage capacity	The more onerous of the following scenarios apply: ■ 1 in 100 notional AEP wet season run-off ■ 1:10,000 AEP (Cell 1)/1:1,000 AEP (Cell 2), 72 hour flood
TSF emergency spillway	PMF
Embankment Stability/Earthquake Criteria	
Earthquake Loading ■ Operating Basis Earthquake (OBE) ■ Maximum Design Earthquake (MDE)	 ■ 1 in 1,000-year ARI ■ 1 in 10,000-year ARI

Stability Factors of Safety: ■ Long term drained ■ Short term undrained (potential loss of containment) ■ Short term undrained (no potential loss of containment) ■ Post seismic	 ■ 1.5 ■ 1.5 ■ 1.3 ■ 1.0-1.2
Facility Construction and Operation	
General	Deposition from north and south embankment crest and east perimeter (Cell 1). Deposition from north, south and west embankment crest (Cell 2). Minimum tailings freeboard of 0.5 m. The supernatant pond will form midway along the west embankment (Cell 1)/east embankment (Cell 2) of the facility. Decant facilities will be provided at all stages to enable removal of water from the ponds.
Construction	Upstream cut-off trench and toe drain. Zoned starter embankment constructed from mine waste and/or local borrow, comprising an upstream low permeability zone and downstream structural zone. 8 m crest width.
Materials	Remove unsuitable foundation soils from embankment footprint. Structural fill won from mine waste and/or local borrow. Low permeability material won from selected local borrow areas within and/or near to the basin.
Tailings Basin	
Basin Lining	*In situ* or imported soils, scarified, moisture conditioned and compacted to form a soil liner.
Basin Underdrainage	Partial basin underdrainage system comprising main collector drains along the basin spine, and branch/finger drains across the basin area.

Table 111: CIL TSF design criteria

Project Operations – Phase 1	
Capacity – Final – Starter	9.35 Mt. Throughput option dependent.
Slurry Characteristics	45% solids by weight. Slurry settled density = 1.10 to 1.42 t/m^3. Supernatant release – 46% of water in slurry. Underdrainage release – ~15-20% of water in slurry. Low pH, Acid Forming.
Fluid Management	Partial basin drainage system drains by gravity to two sumps and is then pumped into the supernatant pond. Decant tower removal of supernatant solution via a pumping system and pressure pipeline back to the plant.
Hydraulic Design	
TSF storm storage capacity	The more onerous of the following scenarios apply: ■ 1 in 100 notional AEP wet season run-off. ■ 1:10,000 AEP, 72 hour flood.
TSF emergency spillway	PMF
Embankment Stability/Earthquake Criteria	
Earthquake Loading ■ Operating Basis Earthquake (OBE) ■ Maximum Design Earthquake (MDE)	 ■ 1 in 1,000-year ARI ■ 1 in 10,000-year ARI
Stability Factors of Safety ■ Long term drained ■ Short term undrained (potential loss of containment)	 ■ 1.5 ■ 1.5

■ Short term undrained (no potential loss of containment) ■ Post seismic	■ 1.3 ■ 1.0-1.2
Facility Construction and Operation	
General	Deposition from perimeter embankment crest. Minimum tailings freeboard of 0.5 m. The supernatant pond will form along the western perimeter embankment of the facility. Decant facilities will be provided at all stages to enable removal of water from the pond.
Construction	Upstream cut-off trench and toe drain. Zoned starter embankment constructed from mine waste and/or local borrow, comprising an upstream low permeability zone and downstream structural zone. 8 m crest width.
Materials	Remove unsuitable foundation soils from embankment footprint. Structural fill won from mine waste and/or local borrow. Low permeability material won from selected local borrow areas within and/or near to the basin.
Tailings Basin	
Basin Lining	*In situ* or imported soils, scarified, moisture conditioned and compacted to form a soil liner. Composite liner (compacted *in situ*/imported soil plus 1.5 mm smooth HDPE liner).
Basin Underdrainage	Partial basin underdrainage system comprising main collector drains along the basin spine, and branch/finger drains across the basin area.

18.17.4 Site characterization

18.17.4.1 Climate

A detailed climatology assessment was completed and the findings are summarised here.

August is the wettest month of the year and appreciable rainfall begins (on average) in April and finishes in October. Accordingly, the wet season typically lasts seven months out of each year, with the remaining five months experiencing low to negligible rainfall. The average annual rainfall for the project area is 1,069 mm and the average annual evaporation for the project area is 1,872 mm.

The rainfall resulting from a range of extreme short duration rainfall events is summarised in Table 112.

Table 112: Extreme daily (24-h) design rainfall

Annual Exceedance Probability (AEP)	Annual Recurrence Interval (ARI) (year)	24-h Duration Precipitation Depth (mm)
20%	5	121
10%	10	145
5%	20	170
2%	50	203
1%	100	229

18.17.4.2 Seismic assessment for TSFs

A detailed seismic hazard assessment was completed and the main findings are summarised here.

The seismicity of much of West Africa is typical of an intra-plate region, characterised by low levels of seismic activity and earthquakes apparently randomly distributed in location and time. The correlation between recorded earthquakes and geological features is typically not well known or understood. There are no historical earthquakes recorded in close proximity to the site. However, 13 earthquakes have been recorded within 500 km of the site, although these comprise small events ranging from M2.9 to M5.2. The closest earthquake to the site occurred in 1993 and was located approximately 240 km north-east of the site with a magnitude of M4.4.

The computer program EZ-FRISK was used to develop a probabilistic seismic hazard model for the site. The seismic hazard analysis module available with the software includes a database of faults and areal seismic sources for Ghana and surrounding West African countries.

The results of the probabilistic analysis were used to determine the relationship between peak ground acceleration and return period for the site. Mean hazard values of peak ground acceleration have been determined for return periods ranging up to 20,000 years. Predicted values for the site in terms of earthquake return period, probability of exceedance for a range of design lives (10, 20, 30, and 50 years) and peak ground acceleration were developed.

Reference to a seismic hazard map that includes West Africa indicated that the site is in a low seismic hazard zone, with a peak ground acceleration of less than 0.02g, for a return period of 475 years (10% chance of exceedance in 50 years). This is in reasonable agreement with the 1 in 475 year PGA of 0.012g calculated in this study.

Consistent with ANCOLD (2012b) guidelines on tailings dams, three levels of design earthquake are typically considered; the Operating Basis Earthquake (“OBE”) for normal operations, the Maximum Design Earthquake (“MDE”) for extreme (dam safety) conditions during operations, and the Maximum Credible Earthquake (“MCE”) for post closure stability.

The 1 in 1,000 year earthquake was adopted for the OBE, based on a high consequence category. The estimated mean value of PGA for a 1 in 1,000 year return period earthquake is 0.021g. A design earthquake of magnitude of M5.35 located at an epicentral distance of approximately 70 km and a focal depth of 15 km was selected for the 1 in 1,000 year OBE.

The 1 in 10,000 year earthquake was adopted for the MDE, based on a high consequence category. The estimated mean value of PGA for a 1 in 10,000 year return period earthquake is 0.10g. A design earthquake of magnitude of M6.75 located at an epicentral distance of approximately 50 km and a focal depth of 15 km was selected for the 1 in 10,000 year MDE.

The MCE scenario assumes that a M7.0 shallow crustal earthquake occurs within 50 km of the site, causing a PGA of 0.15g. Comparison with the probabilistic analysis results indicated that this acceleration is similar to the PGA calculated for the 1 in 20,000 year return interval.

18.17.4.3 Seismic assessment for Process Plant

Building structures for the project should be designed to an appropriate seismic design code, such as the International Building Code (IBC, 2015). Seismic design in accordance with IBC requires determination of seismic coefficients, SS and S1, defined as follows:

- Seismic coefficient, SS: maximum considered earthquake ground motion of 0.2 seconds spectral response acceleration (5% of critical damping)
- Seismic coefficient, S1: maximum considered earthquake ground motion of 1.0 second spectral response acceleration (5% of critical damping).

In accordance with the IBC, the maximum considered earthquake ground motion has been defined as the ground motion with a 2% probability of exceedance in 50 years (return period of 2,500 years). Specific seismic design parameters for use with the IBC are provided below:

- Peak Ground Acceleration = 0.039g

- Seismic coefficient, SS = 0.083g
- Seismic coefficient, S1 = 0.026g.

These values correspond to assumed ground conditions (SEI, 2013) of Site Class B/C as very dense soil and soft rock to rock, with an average shear wave velocity of 760 m/s in the top 30 m.

18.17.5 Review of the design concept

A concept design was developed for the three nominated throughputs (9.5 Mtpa, 7 Mtpa and 4.5 Mtpa) based on the following criteria:

- Tailings facility restricted to the south side of the pit area
- Tailings split – Flotation CIL = 9.25:0.75
- Total tonnage – 120 Mt
- Embankments – Flotation partly downstream, modified centerline and upstream
- Liners Flotation – 10% of area HDPE lined remainder soil liner
- CIL – Fully HDPE lined
- Location:
  - Flotation – Southwest area of lease
  - CIL – Plateau southeast area of lease.

On the basis of these design criteria the concept designs of both facilities for each option were developed. Two of the controlling factors in the design process were:

- Active Tailings Area – to achieve good drying of the deposited tailings a minimum operating area is required.
- Total Tonnage Storage – the total tonnage needs to be stored; to minimize embankment heights and thus earthworks volumes and costs the area required over the life of the facility may need to be larger than the minimum operating area.

In the concept design these controlling factors resulted in the 4.5 Mtpa option requiring two cells for operating purposes but three cells for long term storage requirements. For the 9.5 Mtpa option the minimum operating area was larger than the area required for efficient storage and as a result the overall embankment height was lower.

For the pre-feasibility design a number of changes to the design data were made as follows:

- Total tonnage increased to 129.6 M
- Tailings split – Flotation:CIL modified to 9.25:0.75
- CIL tailings discharge %solids increased from 35%solids to 45%solids

In addition, updated information was available which influenced the design as follows:

- Tailings physical/geochemical testing was completed – the results of this testwork was incorporated into the modelling and was utilised for liner selection.
- The final waste dump profile was expanded and shifted southwards impacting the northern end of the TSF facility area.

- Updated LIDAR topography was produced which showed a significant ridgeline running from north-east to south-west through the Flotation facility footprint – this resulted in a modification of the orientation and configuration of the Flotation facility cells.
- Preliminary dam break modelling was undertaken for the facilities – on the northern side of the facility a dam break could potentially flow into the open pit.

As a result, two design decisions were made:

1) The northern embankment of the Flotation facility would remain on the south side of the local saddle between the facility area and the pit.
2) The embankments of the northern side for both the CIL and the Flotation facility would be fully downstream.

On the basis of the above factors a nominal 3 cell configuration for both the Flotation and CIL facilities was generated (Figure 133). The initial layout was based on storing the total tonnage assuming full downstream construction on all embankments. The design was later adjusted based on defined criteria to determine the transition from downstream to modified centerline to upstream construction. The embankment alignment remained in its current configuration so that there was sufficient space on the downstream side to construct a fully downstream embankment if required.

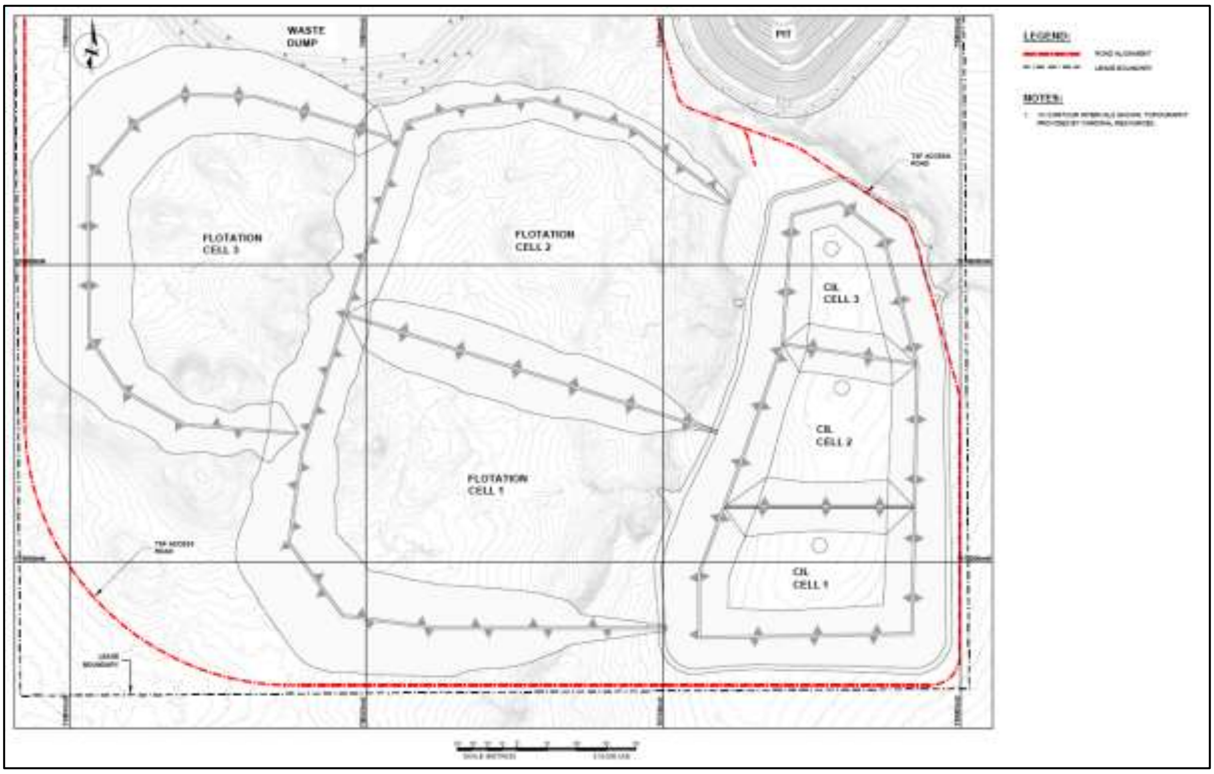


Figure 133: Initial Prefeasibility TSF layout

The design of the CIL facility for each option was assessed as follows:

- The combined area of the CIL cells on top of the plateau was expanded to the maximum area available as a result of the increase in the total CIL tailings tonnage.
- The full area of all three CIL cells is required to store the required total tonnage for all options.
- A density model was run for each of the throughput options, e.g. for the 4.5 Mtpa option the three cells were run in sequence (Cell 1, then 2, then 3) and the resulting densities assessed.

Based on the modelling results and given the fact the CIL facility will be fully HDPE lined with downstream embankments it was decided that a single cell configuration covering the area of all three initial cells would be used for all three throughput options. The cost savings resulting from the removal of the internal embankments and the overall lowering of the embankments due to the improved densities were considered to offset any management issues relating to potential acid generation impacts during operation.

The design and operating of the three Flotation cells for each throughput option was similarly assessed, including the consequence of delays in commissioning Cells 2 and 3.

18.17.5.1 Final assumptions

Based on the review of the various modelled options the following operational assumptions were incorporated in the design and quantity calculations:

- Flotation Cells 1 and 2 combined into a single cell.
- The split in tonnage between Cell 1/2 and Cell 3 fixed at 70:30.
- Cell 3 commissioning delayed by 5 years for the 4.5 Mtpa throughput and by 2 years for the 7.0 Mtpa option.

18.17.5.2 TSF embankment configuration

The Flotation facility embankments will be constructed as a combined downstream/modified centerline/upstream construction embankment.

The following nominal criteria were used to select the transition points from downstream to modified centerline to upstream construction:

- Rate of rise > 3.5 m/year requires downstream construction
- A minimum of 2 downstream stages is required
- A total of 1/3 of the final embankment height is required to be downstream
- A rate of rise of between 2.5 and 3.5 m/year can be constructed using a modified centerline configuration
- A minimum of 2 modified centerline stages is required
- The entire north wall of the Flotation facilities (Cell 1/2) is downstream
- The dividing wall between Cells 1/2 and Cell 3 is fully downstream.

As a result of combining Cells 1 and 2 of the Flotation facility the cell numbering was amended for the Prefeasibility Study with Cell 3 becoming Cell 2.

All of the embankments will be assessed and the configuration confirmed using appropriate stability analysis methods during the next design phase.

18.17.6 Geotechnical site investigations

These are discussed in 18.4.

18.17.7 Tailings characterization

Samples of the Flotation and CIL tailings were tested as part of the design scope to assess their physical and geochemical properties. The results of the testing are summarised below.

18.17.7.1 Physical testing

Four tailings samples were generated by Suntech Geometallurgical Laboratories in Johannesburg, South Africa on behalf of Cardinal Resources from bench scale pilot plant testing and delivered to the KP laboratory in Perth in December 2017.

One sample was labelled “CIL Tails” and the other three samples were labelled “MVO Rougher Tails”, “GRA Rougher Tails” and “DIO Rougher Tails”. The three Flotation tailings samples represent the three major lithologies which will be encountered within the open pit (meta-volcanic, granite and diorite). All four samples were received in dry powder form. The samples were converted to slurry form using Perth tap water in order to achieve a target of 45% solids w/w for the CIL tails and 60% solids w/w for all of the rougher tails samples. The samples were allowed to stand overnight prior to commencement of testing.

Initially, basic classification and undrained testing was carried out on each rougher tails sample in order to assess variability across the different ore types. The three samples were then blended in the ratio 6:3:1 (MVO:GRA:DIO) in accordance with the proportions in which they are expected to be produced to form a composite Flotation sample.

The following tests were carried out on the tailings samples:

i) Classification tests to determine:

- Particle size distribution of the tailings (CIL, MVO, GRA, DIO)
- Supernatant liquor density and pH (CIL, MVO, GRA, DIO)
- Tailings solids particle density (CIL, MVO, GRA, DIO), and
- Atterberg limits of the tailings solids (CIL, MVO).

ii) Undrained (All) and drained (CIL, Flotation) sedimentation tests

iii) Air drying tests (CIL, Flotation)

iv) Permeability tests (CIL, Flotation), and

v) High strain consolidation tests (CIL, Flotation).

The results of the classification tests for the samples and relevant Australian Standards are summarised in Table 113 and Table 114.

Table 113: Classification testing – results and relevant standards

Test	CIL Tails	MVO Rougher Tails	GRA Rougher Tails	DIO Rougher Tails	AS1289
Solids Particle Density (t/m^3)	2.94	2.79	2.70	2.79	3.5.1
Supernatant Density (t/m^3)	1.00	1.00	1.00	1.00	(hydrometer)
Supernatant pH	7.9	7.8	8.2	8.0	(pH meter)
Liquid Limit (%)	41	N.O.	-	-	3.9.2
Plastic Limit (%)	37	N.O.	-	-	3.2.1

Test	CIL Tails	MVO Rougher Tails	GRA Rougher Tails	DIO Rougher Tails	AS1289
Plasticity Index (%)	4	N.P.	-	-	3.3.2
Linear Shrinkage (%)	2	N.O.	-	-	3.4.1

Notes: N.O. = Not obtainable

N.P. = Non-plastic

Table 114: Particle size distribution

Fraction	Particle Size (µm)	CIL Tails	MVO Rougher Tails	GRA Rougher Tails	DIO Rougher Tails
		Percent	Passing (%)		
Sand	600	100	100	100	100
	200	100	93	99	96
Silt	75	98	65	67	66
	20	88	3839	28	39
	6	61	18	13	14
Clay	2	21	7	6	6

The particle size distribution analyses for the four samples were completed in accordance with AS1289 3.6.1 and AS1289 3.6.3.

The CIL Tails consisted of 2% sand, 77% silt and 21% clay sized material. The testing indicates that the material is low plasticity clayey SILT with trace sand, and would be classified as ML in accordance with the Unified Soil Classification System (USCS) (Geotechnical Site Investigation, AS1726-2017). The sample P_{80} is approximately 12 µm.

The MVO Rougher Tails consisted of 35% sand, 58% silt and 7% clay sized material. The testing indicates that the material is non-plastic sandy SILT with trace clay and would be classified as ML in accordance with the USCS. The sample P80 is approximately 120 µm.

The GRA Rougher Tails consisted of 33% sand, 61% silt and 6% clay with a P80 of approximately 106 µm. The DIO Rougher Tails consisted of 34% sand, 60% silt and 6% clay with a P_{80} of approximately 113 µm.

All of the Flotation Tails samples fall within the particle size distribution envelope for potential liquefaction of tailings and thus liquefaction should be assessed as part of the design. The CIL tailings was finer than the typical PSD envelope for susceptibility to liquefaction and thus is considered less likely to liquefy under earthquake loading.

The testing indicated that the rate of supernatant release for the Flotation tails was relatively quick, with the majority of water released in a few hours. The expected water release would be around 46% of the water in slurry, not accounting for rainfall and evaporation but considering the loss of water to re-saturate lower tailings layers. The CIL tails is expected to release a similar percent of water in slurry but at a slower rate, not accounting for rainfall and evaporation but considering the loss of water to re-saturate lower tailings layers. Due to the lower release rate, higher evaporation losses across the beach are expected for the CIL tailings.

The settled dry density deposited into a tailings storage facility can be predicted from the laboratory test work, facility design and site climatic conditions. It has been observed over a number of years that densities achieved in the field are generally lower than those obtained in the laboratory. In addition, field densities achieved are dependent on the area available for drying and the thickness of the deposited layers.

Three tests provided final dry density values as follows:

18.17.7.2 CIL Tails

- Undrained test – 0.99 t/m^3
- Drained test – 1.13 t/m^3
- Air drying test – 1.47 t/m^3.

18.17.7.3 Flotation Tails

- Undrained test – 1.32 t/m^3
- Drained test – 1.44 t/m^3
- Air drying test – 1.67 t/m^3.

The test work indicated that for the CIL tails there is moderately significant difference in achieved density between tailings based on settlement and tailings exposed to air drying. Initially, a low settled dry density of around 1.1 to 1.2 t/m^3 would be expected. With suitable air drying of the tailings slurry a settled density of approximately 1.38 to 1.42 t/m^3 is expected in the facility.

For the Flotation tails there is a smaller difference in achieved density between tailings based on settlement and tailings exposed to air drying. Initially, a moderate dry density of around 1.4 to 1.45 t/m^3 would be expected. With suitable air drying of the tailings slurry a dry density of approximately 1.55 to 1.62 t/m^3 is expected in the facility.

The CIL tails has a moderately high vertical permeability of approximately 1×10^{-7} m/s in the range of expected settled densities. The Flotation tails has a higher vertical permeability of approximately 1×10^{-6} m/s in the range of expected settled densities. Because of the rapid settlement of the Flotation tails it is possible that the horizontal permeability may be an order or two magnitude higher in permeability than the vertical permeability. Over time, both materials may decrease in permeability by an order of magnitude with further consolidation and/or desiccation.

18.17.7.4 Geochemical testing

Geochemical testing of the CIL and rougher tailings solids, supernatant (CIL tails), and distilled water extract (rougher tails) was carried out to assess the acid generation potential, element enrichment and supernatant/seepage water quality against reference standards.

18.17.7.5 Acid Forming Potential

This is discussed in Section 18.17.1.

18.17.7.6 Multi-element enrichment

The rougher tailings samples recorded a low number of element enrichments, with the level of enrichment ranging from slight to significant. The rougher tailings recorded slightly to significantly enriched levels for arsenic, molybdenum and antimony. In contrast the CIL tailings recorded a higher number of element enrichments with eight elements found to be significantly to highly enriched. The CIL tailings recorded highly enriched levels of arsenic, bismuth, molybdenum, sulfur, antimony and selenium. Silver and tin were recorded to be significantly enriched, and cadmium, cobalt, copper and mercury slightly enriched.

Comparison of the multi-element results to soil quality screening guidelines indicated that the rougher tailings samples met the human health criteria, whilst the CIL tailings exceeded the threshold for arsenic. A number of metals and metalloid concentrations exceeded the ecological guidelines in all four tailings samples, with antimony, chromium and nickel the most commonly elevated metals. Additionally, arsenic was elevated above the ecological threshold in the metavolcanic rougher tailings and CIL tailings, with zinc elevated in the diorite rougher tailings and CIL tailings. Overall the CIL tailings had a greater number of elevated metals, with cadmium, copper, lead, selenium and sulfur also exceeding the ecological criteria.

The rougher tailings exceeded soil intervention values only for arsenic in the metavolcanic sample. However, based on the approximate blending ratios of the rougher tailings, the overall average results are indicated to meet the soil intervention criteria. The CIL tailings exceeded soil intervention values for arsenic, chromium, copper and nickel.

On the basis of the multi-element results, both the Flotation and CIL TSFs should be designed to prevent the loss of solids. The Flotation TSF will require a basic cover system on closure. The cover system for the CIL TSF will be driven by the need to control acid generation by precluding oxygen and water ingress to limit on-going oxidation of the tailings and seepage. The CIL facility would also benefit from co-disposing the rougher and CIL tailings together within the CIL facility towards the end of the mine life and prior to construction of the engineered low permeability capping layer in order to provide a top surface of tailings which is NAF and contains excess alkalinity.

18.17.7.7 Supernatant and Leachate Water Quality

The rougher tailings sample generally met the Ghanaian and international reference water quality standards for effluents from mining operations, although the metavolcanic sample exceeded the guideline for arsenic. Based on the approximate blending ratios of the rougher tailings, the average results indicate that the concentration of arsenic in the rougher tailings may be similar to the guideline value. Further, all the rougher tailings samples exceeded Ghanaian drinking water guidelines for antimony and arsenic. Additionally, all rougher tailings samples exceeded arsenic international drinking water guidelines, whilst the granite rougher tailings sample also exceeded international drinking water guidelines for pH (i.e. alkaline), aluminium and antimony. Based on the approximate blending ratios of the rougher tailings, the average results indicate that the rougher tailings may have elevated levels of aluminium, antimony and arsenic.

The CIL tailings were found to exceed Ghanaian water quality standards for effluents from mining operations for arsenic and TDS. The CIL tailings also did not meet international water quality standards for release due to elevated arsenic, TDS, copper, mercury, molybdenum and sulfate. Comparison of the CIL supernatant results with Ghanaian drinking water guidelines showed that the sample did not meet the thresholds for arsenic, TDS and copper. It was not possible to determine whether the free cyanide in the CIL tailings exceeds the guideline due to the high detection limit and concentrations of total and WAD cyanide in the sample. The CIL tailings exceeded international drinking water standards due to a range of elements, namely antimony, arsenic, TDS, chloride, copper, mercury, molybdenum, sulfate and total cyanide.

Based on these results, the rougher tailings facility will require a compacted soil liner to limit seepage. In addition, the facility should have an underdrainage system to limit the hydraulic head acting on the soil liner. The CIL tailings facility will require a robust engineered liner system, likely comprising of a compacted soil liner with overlying HDPE liner and underdrainage system. Furthermore, a seepage collection system between the soil and HDPE liners is recommended. Both facilities should be designed and operated with adequate freeboard allowance in order to reduce the risk of uncontrolled supernatant or tailings release.

18.17.8 Water management

A water management assessment was carried out for the Flotation and CIL TSFs.

The water management model for the TSF and the TSF/plant interaction was developed to determine the TSF water demand and to generate design embankment crest levels to maintain containment throughout the operation. The model was to calculate process water shortfall and hence to quantify the volume of water required from external sources.

The main components of the water management model were:

- Tailings storage facilities consisting of one CIL cell and two Flotation cells
- Rainfall run-off from TSF catchment area, and
- Process plant balance.

The model was setup as a monthly time-stepped model to cover the whole operational life of each facility. As a result of the three throughput options and two tailings streams, a total of six scenarios were modelled. The water balance for the TSFs was based on the following design data and/or assumptions:

The modelling indicated:

- The CIL Plant only requires a small quantity of the CIL decant return, hence no external make up other than a nominal raw water requirement is needed.
- As a result, a large proportion of the CIL decant return needs to feed the Flotation Plant as recycle. How this water is treated and managed internally will need to be determined within the plant.
- All options indicate that the Flotation Plant requires some additional water make up. This can be sourced either from the Flotation TSF in the wet season if available (currently limited to 90% of water in slurry, however higher values are possible) or the river abstraction system.
- It is recommended that CIL decant water is used to slurry the Flotation concentrate process.

The solids and water balance can be summarised as follows:

- The CIL facility is appropriately sized to achieve high air dry densities in the dry season when the pond is small. All options show an increase in pond volume during the wet season resulting in a reduction in dry density.
- The different throughput options are generally similar in achieved overall dry densities throughout the project life. This is despite the rate of rise ranging from 0.9 m/year for Option 1 to 1.8 m/year for Option 3, as Option 3 has a smaller pond, exposing more drying beach and compensating for the higher rate of rise.
- The CIL decant return should be maximized to prevent accumulation of water between wet seasons for all options. Water will be directed to both the CIL plant preferentially and excess to the Flotation plant following treatment.
- The Flotation facilities are appropriately sized to achieve high air dry densities in the dry season based on a throughput split of 70:30 for Cell 1: Cell 2. All options show an increase in pond volume during the wet season resulting in a reduction in dry density, with Option 1 accumulating volume slightly year on year.

The rate of rise varies between the different options and cells so that transition between downstream, modified centerline and upstream occur at different times and levels. A rate of rise of less than 3.5 m/year indicates modified centerline should be possible the following year and less than 2.5 m/year indicates upstream raising should be possible shortly after.

- The Flotation decant return should be maximized to prevent accumulation of water between wet seasons for all options. Water will be directed to only the Flotation Plant in combination with additional treated CIL water.
- Overall the two process plants are in water shortfall and require additional Flotation TSF decant return in the wet season if available (currently limited to 90% of water in slurry) or make-up water from the river abstraction system.

Plant modelling for the 3 throughput options is summarised in Table 115.

Table 115: Estimated average makeup water requirements

Option	Throughput (Mtpa)	Minimum Raw Water (t/h)	Average Annual Makeup Water (t/h)	Wet Season Makeup Water (t/h)	Dry Season Makeup Water (t/h)	Typical CIL to Flotation Plant (t/h)
1	4.5	35	20	0	139	71
2	7.0	50	84	0	403	90
3	9.5	60	210	0	549	107

18.17.9 TSF design

18.17.9.1 General description

Flotation TSF

qqqThe Flotation TSF will be constructed as a side valley-type storage facility to the south-west of the open pit. The facility will be constructed as two cells with zoned earthfill perimeter embankments and will be lined with a low permeability compacted soil liner. The total basin area will be 311 Ha and is designed to accommodate 120 Mt of tailings. The TSF embankments will be constructed in stages to suit storage requirements with Stage 1 constructed initially to provide capacity for the first 12 months of operation and subsequent stages constructed using downstream, modified centreline and upstream raise construction methods. The final embankment elevations will vary by cell and option, as shown in Table 116.

Table 116: Flotation TSF embankment – final elevation

Throughput Option (Mtpa)	Elevation (RL m)	
	Cell 1	Cell 2
4.5	231.5	214.0
7.0	230.5	215.5
9.5	230.0	219.0

The TSF basin area will be cleared, grubbed and topsoil stripped, and a 300 mm depth compacted soil liner will be constructed over the entire TSF basin area as either re-worked *in-situ* material (assumed 70%) or imported Zone A (30%) material.

The TSF design incorporates an underdrainage system comprising a network of branch and collector drains in each cell. The underdrainage system drains by gravity to a collection sump located at the lowest point in each cell.

Supernatant water will be removed from the TSF via a submersible pump (designed by others) mounted in a decant tower. Temporary decants will be provided to suit the tailings deposition schedule in each cell. The final decants will be located along the divider embankment between the two cells.

CIL TSF

The CIL TSF will be constructed as a paddock-type storage facility to the south of the open pit. The facility will be constructed as a single cell with zoned earthfill perimeter embankments and will be lined with compacted soil liner overlain by a synthetic HDPE geomembrane. The total basin area will be approximately 45 ha and is designed to accommodate 9.6 Mt of tailings. The TSF embankments will be constructed in stages to suit storage requirements with Stage 1 constructed initially to provide capacity for the first 12 months of operation and subsequent stages constructed using downstream raise construction methods to a final elevation of RL 266.0 m (all throughput options). Staged embankment crest elevations will vary between throughput options.

The TSF basin area will be cleared, grubbed and topsoil stripped, and a 200 mm depth compacted soil liner will be constructed over the entire TSF basin area as either re-worked *in-situ* material (assumed 30%) or imported Zone A (70%) material. This will be overlain by a 1.5 mm thick smooth HDPE geomembrane liner.

The TSF design incorporates an underdrainage system comprising a network of branch and collector drains. The underdrainage system drains by gravity to two collection sumps located at the lowest points in the cell at the south-east and south-west corners.

Supernatant water will be removed from the TSF via a submersible pump (designed by others) mounted in a decant tower located along the western embankment of the facility.

18.17.9.2 Embankment construction

The Stage 1 embankments will have upstream and downstream slopes of 1V:3H and a crest width of 8 m. The same crest width will be adopted for subsequent stages. The final stage raises for each facility will have a downstream slope of 1V:3H with 5 m benches every 10 m in vertical height. It is expected that the tailings will not be suitable as a construction material and the design is based on all lifts being constructed using mine waste and/or local borrow. Cut-off trenches will be excavated to a depth of approximately 2 m into the natural subgrade in each cell and backfilled with low permeability fill in order to tie into the embankment Zone A.

The TSF embankment will comprise a zoned embankment constructed of selected mine waste/local borrow. The embankment consists of an upstream low permeability zone (Zone A) and a downstream structural zone (Zone C). If required, a transition material (Zone B) will be introduced between the Zone A and Zone C material to ensure filter compatibility.

Typical material specifications for the embankment are summarised below:

- Zone A material will be selected mine waste/local borrow with an average hydraulic conductivity not greater than 3×10^{-8} m/s after compaction.
- Zone C material for Stage 1 and subsequent stages will be sourced from mine waste.

Construction of the staged raises will commence before the operating stage is full so that there is adequate storage volume available throughout the life of mine and to minimize construction delays. Construction will be scheduled to take place during the dry season to minimize the potential for adverse weather impacting the construction.

18.17.9.3 Seepage control

In order to mitigate seepage losses through the basin area, minimize the phreatic surface in the embankments, and increase the settled density of the deposited tailings a number of seepage control and underdrainage collection features have been integrated into the design of each facility. The seepage control and underdrainage collection systems will consist of the following components:

- Cut-off trench.
- Low permeability soil liner.
- Synthetic HDPE geomembrane
- Basin underdrainage collection system.
- Underdrainage collection sump.
- Leak collection system.
- Upstream toe drain.

- Cut-Off Trench.

Primary seepage control from each tailings facility will comprise the construction of a cut-off trench excavated into the foundation soils and backfilled with low permeability fill to reduce seepage loss through the embankment foundation.

The cut-off trench will be located beneath the upstream toe of the embankment and will be cut to a depth of approximately 2.0 m (depending on ground conditions). The cut-off trench will be constructed continuously along the upstream toe of the embankment to the full deposition elevation to limit potential seepage at any level. If the cut material is suitable as Zone A fill it may be replaced in the excavation in compacted layers; alternatively, suitable low permeability material will be won, conditioned, placed and compacted in the trench.

Low permeability soil liner

The basin area of each facility will be cleared and grubbed of vegetation and the topsoil removed. A compacted soil liner of either 300 mm thickness (Flotation TSF) or 200 mm thickness (CIL TSF) will be constructed either by scarifying and compacting the *in-situ* soils or, if they are not suitable, by importing suitable borrow material. The soil liner will tie into the low permeability zone of the embankment and will have a permeability of less than 1×10^{-8} m/s.

Synthetic HDPE geomembrane

In the CIL TSF the complete basin area and upstream embankment face will be lined with a 1.5 mm HDPE geomembrane liner (nominal permeability of 1×10^{-11} m/s) to final stage level in order to reduce any seepage from the tailings and the supernatant pond. The HDPE liner will be placed on top of the compacted soil liner forming a composite liner system. Smooth geomembrane will be utilised, except at the location of the decant tower where a textured geomembrane will be placed to provide additional stability to the causeways and tower.

Basin underdrainage collection system

The underdrainage collection system is designed to reduce the phreatic surface on the compacted soil liner. The underdrainage has several benefits as follows:

- Reduces seepage through the basin and under/through the embankment.
- Drains the tailings mass, thus increasing the density of the tailings and providing a more efficient facility in terms of storage.
- Increases the strength of the tailings mass immediately adjacent to the embankment.

The design of the underdrainage system takes advantage of the natural fall of the terrain to minimize re-shaping of the basin. The underdrainage system will consist of a drainage networks comprising a series of collector and finger drains.

The collector drains will have a 100 mm draincoil pipe along the centreline and a sand surround (Zone F) wrapped in geotextile. The collector drains will feed directly into the underdrainage collection sumps.

A network of finger drains will be constructed across the basin of each cell. These comprise 63 mm draincoil pipes (respectively) along the centreline with a sand surround (Zone F) wrapped in geotextile. The finger drains join into the collector drains.

Underdrainage collection sump

Underdrainage collection sumps will be constructed against the upstream toe of the embankment in each facility. The Flotation TSF incorporates a single sump in each cell, whilst the CIL TSF incorporates two sumps. The sumps are located at the topographical low points in each cell to facilitate collection of solution from the toe drains and underdrainage system, and gravity drainage to the sumps. The underdrainage sump consists of the following components:

- An excavated sump, filled with clean gravel wrapped in geotextile.
- A 450 mm diameter HDPE (SDR7.4) solid riser pipe, slotted only at the base. The pipe is located on top of the embankment face/geomembrane liner (protected with a wearsheet) and runs up the upstream embankment face.
- A submersible pump.
- A hoist and pulley to raise and lower the pump.

Leak Collection System

A leak collection drain will be constructed along the spine of the CIL TSF within the basin area to intercept seepage through the basin liner during operation. The drain will comprise a 100 mm diameter draincoil pipe situated at the base of a 1 m deep trench, backfilled with clean sand/gravel (Zone F) to 700 mm depth, wrapped with geotextile, and overlain by a 300 mm thick low permeability material cap. The leak collection drain will feed directly into the leak collection sump which will incorporate the following components:

- An excavated sump, filled with clean gravel wrapped in geotextile. The sump will be located below the geomembrane liner.
- A 450 mm diameter HDPE (SDR7.4) solid riser pipe, slotted only at the base. The pipe is located below the geomembrane liner in a trench that runs up the upstream embankment face.
- A submersible pump.
- A hoist and pulley to raise and lower the pump.

Collected solution will be pumped back onto the tailings beach, with flows reporting to the supernatant pond for recycling back to the process plant.

Upstream Toe Drain

In addition to the basin underdrainage system in each facility, a toe drain will be constructed along the upstream toe of the embankment. The main purpose of the toe drain is to increase the stability of the embankment by providing drainage of the tailings, hence lowering the phreatic surface adjacent to the embankment. The second purpose of the toe drain is to act as an additional underdrainage collection pipe.

The toe drain will be similar in design to the collector drains and will comprise a 100 mm draincoil pipe laid at the base of the drain within 300 mm of drainage material (Zone F) wrapped in geotextile. The toe drain will drain directly into the underdrainage collection sump/s.

18.17.9.4 Decant return water system

Each cell of the Flotation TSF will operate with a series of three decant towers which will be constructed, operated and subsequently de-commissioned to suit the staged development of the facility and of the tailings beaches in each cell. The CIL TSF will operate with a single decant tower throughout the life of the facility.

The decant towers will be raised as required with each embankment lift and will consist of the following components:

- An access causeway constructed of Zone C material.
- A slotted concrete decant tower consisting of 1.8 m square slotted precast concrete sections surrounded by clean waste rock (Zone G) with a minimum size of 100 mm.
- A submersible pump with float control switches mounted on a lifting hoist (designed by others).

The decant pump in each tower will be raised on a regular basis to ensure that no tailings enters the pump intake.

18.17.9.5 Emergency spillway

The tailings storage facilities have been designed to completely contain storm events during operation up to and including an annual exceedance probability (AEP) of 1 in 1,000 (Flotation TSF Cell 2) or 1 in 10,000 (Flotation TSF Cell 1 and CIL TSF) on top of the predicted maximum pond level under average climatic conditions, without the emergency spillways operating. Consequently, exceeding the storm storage capacity of the facilities at any stage of operation is unlikely.

Regardless, in the event that the storage capacity of a facility is exceeded, water which cannot be stored within the facility will discharge via an engineered spillway. Each emergency spillway during operation is designed to convey run-off from a PMF storm, assuming that the decant pond level is at the spillway invert level at commencement of the storm event. A new spillway will be constructed at each stage of construction and will be excavated along the south-western (Flotation TSF Cell 1), western (Flotation TSF Cell 2), and southern (CIL TSF) embankments.

18.17.9.6 Tailings and decant return trench

The tailings delivery and decant return pipelines will be located within a bunded and HDPE lined corridor between the process plant and the TSFs to provide secondary containment in case of leakage of the pipelines. The corridor is designed so it is free draining either to the process plant, to dedicated containment ponds, or into the TSF.

18.17.9.7 Seepage assessment

It is expected that seepage from the TSFs will be limited by the seepage control measures included in the facility design. A detailed seepage assessment will be carried out for each facility during the next design phase when additional geotechnical information is available.

18.17.9.8 Stability assessment

A detailed stability assessment will be carried out during the next design phase in order to confirm the proposed embankment raise configurations for each facility.

18.17.9.9 Tailings storage facility operation

General

A dedicated operating manual will be prepared for the facility prior to commissioning. A summary of the facility operation is detailed in the following sections.

Tailings Deposition System

The tailings delivery pipelines will be routed from the process plant onto the embankment crest of each TSF. The tailings distribution pipeline will be located on the upstream embankment crest and will be raised with each stage.

Deposition of tailings into the storage facilities will occur from offtakes inserted along the tailings distribution pipeline. The deposition location will be moved on a daily basis, or as required to control the location of the supernatant pond.

Deposition Technique

Tailings deposition will be carried out using the sub-aerial technique in order to promote the maximum amount of water removal from the facility by the formation of a large beach for drying and draining. Together with keeping the pond size to a minimum, sub-aerial deposition will increase the settled density of the tailings and hence maximize the storage potential and efficiency of the facility.

The tailings will be deposited into the facility from the embankment in such a way as to encourage the formation of beaches over which the slurry will flow in a laminar non-turbulent manner. Limited settlement and water release will occur. The released water will form a thin film on the surface of the tailings. This water will flow to the supernatant pond from where it will be removed from the storage area via the decant.

Deposition of the tailings will be carried out on a cyclic basis with the tailings being deposited over one area of the storage until the required layer thickness has been built up. Deposition will then be moved to an adjacent part of the storage to allow the deposition layer to dry and consolidate. This will facilitate maximum storage to be achieved across the whole facility.

After deposition on a particular area of beach ceases and settling of the tailings has been completed, further de-watering will take place due partly to drainage into the underdrainage system, but mainly due to evaporation. As water evaporates and the moisture content drops, the volume of tailings will reduce to maintain a condition of full saturation within the tailings. This process will continue until interaction between the tailings particles negates volume reduction.

18.17.10 River abstraction system

Raw water for the project will be sourced from the White Volta River by means of a river abstraction system comprising a water abstraction tower and associated pumps and pipeline. The water abstraction tower will be located along the northern bank of the White Volta River approximately 8.5 km to the west of the process plant. Preliminary design parameters for the river abstraction system are provided in Table 117.

Table 117: Design parameters – river abstraction system

Parameter	Detail
Causeway elevation	1 in 100 year recurrence interval, peak river flow elevation plus 0.3 m freeboard
Maximum river abstraction	5% of total river flow
Restricted offtake period*	30 days
Construction Materials ■ Structural Fill (Zone C) ■ Erosion Protection (Zone E) ■ Coarse Rockfill (Zone G)	 ■ Won from local borrow areas established by the civil contractor, or supplied by the mining operation to the embankment, spread and compacted by the civil contractor ■ Imported from off site, supplied to local stockpile for the civil contractor ■ Imported from off site, supplied to local stockpile for the civil contractor
Construction Description	Slotted steel tower, situated within inlet trench excavated perpendicular to river flow direction, surrounded by clean coarse rockfill, with general fill access causeway. Rockfill erosion protection as required

Notes: *no pumping allowed from river due to insufficient flow and regulated offtake restriction

The river abstraction tower embankment has been sized with the crest at RL146.4 m (the estimated 1 in 100 year flood level plus freeboard allowance). This will require additional work in the detailed design phase to confirm the final tower height to ensure it can withstand extreme flooding events, and a detailed survey of the river bank and bathymetry in order to produce an accurate hydraulic model to accurately estimate the 1 in 100 year ARI flood level and low flow levels in the river base.

The river abstraction system will comprise submersible pumps situated within a vertical steel (slotted) tower, located within a trench excavated into the northern abutment of the White Volta River. The base of the trench at the inlet will coincide with the lowest allowable river flow level for abstraction. The abstraction tower will comprise a 25 MPa concrete base and 2.6 m diameter (10 mm thickness) slotted steel pipe sections. The tower will be surrounded by free-draining coarse rockfill (Zone G) with an inlet pipe to convey low level flow into the tower. A pump bipod stand (designed by others) will be installed over the abstraction tower to facilitate pump removal. Abstraction system pumps, pipelines, pump controls, electrical supply and associated infrastructure will be designed by others.

Water from the river abstraction tower will be pumped to a water storage facility located directly to the north of the process plant. This facility will comprise a turkey's nest pond lined with 1.5 mm smooth HDPE geomembrane liner. The facility is sized to provide a capacity of 240,000 m^3 (nominated by the process designer) in order to provide 30 days of process water demand, assuming a restricted offtake period from the White Volta River of 30 days. It should be noted that the process water demand increases with throughput (as summarised in Table 118) and that whilst the water storage facility is indicated to provide sufficient capacity for the 4.5 Mtpa option (based on the assumption relating to 30 day supply period) it may not be large enough for the higher throughputs unless additional make-up water is proposed to be sourced elsewhere. The offtake restriction period, maximum river abstraction and throughput option will need to be confirmed to finalise the design of the water storage facility.

Table 118: Process water demand

Throughput (Mtpa)	Process Water Demand (t/h)	Process Water Demand – 30 days (m^3)
4.5	134	96,480
7.0	408	293,760
9.5	554	398,880

18.17.11 Infrastructure design (roads and airstrip)

18.17.11.1 Roads

A new site access road is to be constructed between the main N10 route and the project site. The access corridor will originate at a junction with N10 approximately 16 km south-south-east of Bolgatanga and terminate at the process plant approximately 23 km to the west. The corridor will be shared with the mains power supply to the project over the whole alignment and with the raw water supply from the White Volta River over the last 9 km. The site access road will connect with a network of internal roads which link the various site facilities.

18.17.11.2 Site access road

Design parameters for the site access road are provided in Table 119.

Table 119: Design parameters – site access road

Parameter	Value
Design Vehicle	Semi-trailer Truck
Road Cross Section	Formation Width – 11.3 m Lane Width – 3.65 m Shoulder Width – 2.0 m Crossfall (road and shoulder) – 3% Max. Superelevation – 5%
Minimum Horizontal Curve Radius	450 m
Minimum Vertical Curve Length	70 m
Maximum Vertical Grade	6%
Design Vehicle Speed	80 km/h
Minimum Culvert Diameter	2000 mm
Culvert Design Criteria	1 in 10 year ARI
Pavement	150 mm sub-base 150 mm basecourse

The total length of the access road is 24,188 m. The road is designed to comply with the Ghana Highway Authority guidelines and a classification of “Primary Road” was adopted for the road design.

18.17.11.3 Internal access roads and TDRT

A network of internal access roads will link the river abstraction tower, airstrip, process plant, accommodation camp, and TSF with the primary site access road and tailings and decant return trench (“TDRT”).

Design parameters for the internal access roads are provided in Table 120.

Table 120: Design parameters – internal access roads

Parameter	Value
Design Vehicle	Semi-trailer Truck
Road Cross Section	Formation Width – 6.0 m Lane Width – 3.0 m Crossfall (road and shoulder) – 3%
Minimum Horizontal Curve Radius	50 m
Minimum Vertical Curve Length	50 m
Maximum Vertical Grade	10%
Design Vehicle Speed	30 km/h
Minimum Culvert Diameter	1200 mm
Culvert Design Criteria	1 in 10 year ARI
Pavement	150 mm basecourse

18.17.11.4 Airstrip

An airstrip is proposed to be constructed to provide air freight capability both to and from the site. The runway was designed in accordance with the following standards:

- The runway shall be constructed to service a Beechcraft Kingair 200 aircraft or similar operating into and out of the airstrip 3-4 times per week
- Aerodrome design life shall be 20 years, and
- The aerodrome is to be utilitarian and shall contain basic firefighting facilities as well as re-fuelling facilities.

Specifications for the Kingair 200 include up to 6 passenger numbers, a maximum take-off weight of 5,670 kg, tyre pressure of 735 kPa, aeroplane reference field length of 592 m, aircraft length of 13.3 m, wingspan of 16.6 m and an outer main gear wheel span of 5.6 m. The recommended runway length for this aircraft is 1,200 m.

Based on the design aircraft the aerodrome facility is classified as Code 1B in the CASA standards. These standards require that Code 1B aerodromes meet the following requirements:

Table 121: Design standards – airstrip

Parameter	Value
Runway width	18 m
Runway shoulders	Not required
Runway strip width	90 m
Runway End Safety Area (RESA) length	90 m
Stopway length	30 m
Maximum longitudinal slope	2%
Maximum transverse slope	2%

Parameter	Value
Runway turning area	3 m (minimum)
Taxiway width	10.5 m
Taxiway shoulders	Not required
Taxiway strip width	20 m
Maximum taxiway longitudinal slope	3%
Maximum taxiway transverse slope	2%

In addition to the above a sufficient apron area for the parking of 2 aircraft and a re-fuelling facility shall be allowed for.

Based on wind rose data collected at a weather station in Bolgatanga approximately 23 km north-west of the Project site the airstrip runway will most likely be aligned in a north-northeast to south-southwest direction. The proposed airstrip site is approximately 2.5 km to the west of the accommodation camp facility and directly to the south of the mine access road, and has been selected to suit the existing topography and operational constraints.

18.17.12 Surface water management

The majority of the proposed site infrastructure is situated within three catchments:

- The pit shell, northern half of the waste dump, a quarter of the CIL TSF footprint, process plant, ROM pad, and the accommodation camp are situated downstream of an approximately 18,589 Ha catchment. However, no major diversion channels are required as the proposed site infrastructure is situated to the east of the catchment watercourse.
- The southern half of the waste dump, the Flotation TSF, and the western half of the CIL TSF footprint are incorporated in a separate catchment to the south and west of the above catchment.
- The catchment to the east of this incorporates approximately one quarter of the CIL TSF footprint.

A site-wide surface water management design was developed with the following purposes:

- To capture run-off from the downstream embankment faces of the Flotation TSF.
- To capture run-off from the downstream embankment faces of the CIL TSF.
- To divert and capture run-off and sediment from the western side of the waste dump.
- To capture run-off and sediment from the northern and eastern sides of the waste dump and western side of the pit perimeter.
- To divert run-off from the north-eastern side of the CIL TSF and around the pit perimeter.

To this end the surface water management design incorporates the following components:

- Construction of the accommodation camp-TSF access road which would act as sediment control on the western side of the waste dump, western and southern sides of the Flotation TSF, and the southern and eastern sides of the CIL TSF.
- Construction of a diversion bund (Diversion Bund 01 (DB01)) to the west-north-west of the proposed waste dump.
- Construction of a diversion bund (Diversion Bund 02 (DB02)) downstream of the north-eastern face of the CIL TSF and the eastern perimeter of the open pit.

- Construction of two sediment control dams (Sediment Control Dam 01 (SCS01) and Sediment Control Dam 02 (SCS02)), one to the north of the proposed waste dump and the other to the north-east of the open pit.
- Minor ponding could occur along the north-east perimeter of the waste dump due to its intersection with the natural topography. Typically, this could comprise development of an approximately one metre deep pond before being diverted to SCS01 downstream. In turn this may impact the Mine Services Area depending upon the earthworks design profile. Forced ponding could be carried out in this area if required, which would reduce the run-off and sediment flow to SCS01 and hence provide another form of sediment control.

The preliminary design configurations of the diversion bunds and sediment control structures are summarised in Table 122 and Table 123.

Table 122: Proposed diversion bunds

Structure Reference	Minimum Crest Level (RL m)
DB01	171.0
DB02	194.0

Table 123: Proposed sediment control dams

Structure Reference	Minimum Crest Level (RL m)
SCS01	170.5
SCS02	192.5

18.17.13 Monitoring

18.17.13.1 General

A monitoring programme for the TSFs will be developed prior to commissioning to monitor for any potential problems which may arise during operations. The monitoring will include:

- Monitoring bores and surface water sampling stations downstream of the TSFs
- Standpipe piezometers within each embankment to monitor the phreatic surface
- Survey pins on embankment crests to monitor embankment movement.

If the monitoring programme indicates that potential problems are developing, an increase in monitoring frequency will be implemented and a response plan developed.

18.17.13.2 Seepage monitoring

The TSF designs incorporate a number of measures to reduce the amount of seepage that will occur from the TSFs and to mitigate the extent of any effects on the downstream environment.

A series of groundwater monitoring stations will be installed downstream of the TSF embankments to facilitate early detection of changes in groundwater level and/or quality both during the operating life and following decommissioning.

Each monitoring bore station consists of one shallow bore extending to a depth of approximately 10 m, and one deep bore terminating approximately 5 m below the groundwater table. The shallow bore is intended to detect any seepage from the TSF flowing within the surface sediments, whilst the deep bore is designed to monitor groundwater level and chemistry. Each bore will be cased and screened over an interval set in the field during installation and sealed back to surface with low permeability grout. It is recommended that the boreholes are constructed before commissioning of the TSF in order to accumulate baseline data specific to the TSF location.

18.17.13.3 Stability monitoring

Pore water pressures should be monitored within the TSF embankments to ensure that stability is not compromised. To this end it is proposed that standpipe piezometers are installed at a number of locations along the embankment crest of each facility. The base of the piezometers will be located within the embankment to ensure that the phreatic surface within the embankment fill is measured, as opposed to natural groundwater level.

The piezometers will be monitored at regular intervals and any rises in water level analysed to ensure that the phreatic surface does not reduce the overall stability of the embankments below acceptable levels.

During each embankment raise, the existing piezometers will either be extended if viable or, if not, sealed with a cement/bentonite grout mix. New piezometers will be established on the embankment crest at the end of raise construction.

18.17.13.4 Survey pins

Survey pins will be installed along the embankment crests and downstream face to monitor any movement of the embankment. As-installed details of each pin (date of installation, settlement pin ID, Northing, Easting, and RL) will be recorded at time of installation. Any displacement considered excessive or on-going may indicate embankment stability problems and will need to be assessed by a qualified geotechnical engineer.

18.17.13.4.1 Monitoring and maintenance programme

Monitoring Programme

As part of operation of the TSFs, extensive monitoring of all aspects of the operation should be undertaken. This monitoring falls into three basic categories:

- Short-term operation monitoring – this includes items such as offtake location, whether pipe joints are leaking, etc., which are part of ensuring that the TSF is operating smoothly
- Compliance monitoring – this includes items such as checking settlement pins for movement and monitoring bores for changes in groundwater chemistry etc., which are used to ensure that the project is meeting all of its commitments in regard to a safe, secure operation, and
- Long-term performance monitoring – this includes such items as tailings level surveys and tailings and water flow measurements (using flow meters installed at designated locations), etc., which are used to monitor the long-term performance of the facility and refine future embankment lift levels.

In addition, the TSFs will undergo annual audits by a suitably qualified geotechnical engineer to ensure that the facilities are operating in a safe and efficient manner.

A full monitoring programme will be included in the TSF operating manuals (to be issued prior to commissioning of the TSFs).

Maintenance Programme

Inspection and maintenance of the TSFs is largely aimed at mitigating potential problems by dealing with them before they can develop into major problems.

A full maintenance programme, including the maintenance requirements for each area, will be included in the TSF operating manuals. Modifications to the maintenance programme as a result of emergency situations or annual reviews should be made as required.

18.17.14 Rehabilitation

18.17.14.1 Tailings Storage Facility

At the end of the TSF operations, the downstream faces of the embankments will have a slope of 3H:1V, with 5 m wide benches located at 10 m height intervals, for an overall slope profile of 3.5H:1V. The downstream profile will be inherently stable under both normal and seismic loading conditions, will provide a stable drainage system, and will allow for re-vegetation.

The embankment downstream faces can only be re-vegetated once the final downstream profile is achieved. Prior to rehabilitation of the embankment faces, temporary vegetation and geotextile silt fences can be used to limit erosion and sediment run-off from the embankment slopes.

The TSF closure spillways will be excavated during rehabilitation of the tailings surface following decommissioning and once the remaining supernatant water is proven to be suitable for release. The closure spillways will be constructed in such a manner as to allow rainfall runoff from the finished surface of the rehabilitated TSFs to flow into the surrounding natural drainage system.

Rehabilitation of the tailings surface will commence upon termination of deposition into the TSFs. The final rehabilitated profile will be confirmed during operation based on on-going operational tailings geochemistry test results. The following profile has been assumed for the tailings surface:

- Capillary break rockfill layer (300 mm)
- Low permeability fill layer (300 mm), and
- Topsoil growth medium layer (100 mm).

The finished surface will be shallow ripped and seeded with shrubs and grasses.

The underdrainage system will need to continue to operate for some time after completion of capping and re-vegetation to drain excess water from the tailings deposit. The quantity of water recovered from the underdrainage system will reduce with time and experience with similar facilities suggests that water recovery may continue for a period of up to 2 years following decommissioning. During this time, water from the underdrainage will be pumped back into the TSF for removal by the decant system. After the flow ceases, the underdrainage pumps will be removed and the underdrainage tower backfilled and sealed as part of the rehabilitation process. The supernatant pond left on the surface of the TSF may need to be returned to the Plant Site for treatment until the supernatant water in the TSF can be shown to be suitable for discharge. After the water has been proven to be benign, the runoff can be allowed to discharge via the closure spillway.

18.17.14.2 River abstraction tower

Following de-commissioning of the process plant, pumping from the abstraction tower will cease and the abstraction tower and associated infrastructure will be removed. The rehabilitation process will comprise the following:

- Removal of the raw water pipeline, electrical equipment and generator, pumps, valves and ancillary equipment
- Removal of the steel tower and intake pipe
- The tower backfill will be spread uniformly over the lower portion of the inlet channel, and
- All areas compacted by equipment (including the access road) will be ripped, uniformly graded and revegetated, concrete pads covered and revegetated, and inlet channel slopes revegetated in accordance with the project rehabilitation plan.

18.17.14.3 Sediment control structures

If required, the sediment control structures will be ‘closed’ by breaching the embankments to allow stored water to discharge and to provide an engineered drainage path for all runoff.

18.17.15 Rehabilitation trials

In addition to these objectives, other aspects of the rehabilitation programme will be re-vegetation, erosion control and stormwater management. Establishing a surface cover of verdant vegetation will reduce the potential for adverse environmental impacts such as dust generation and rainfall erosion, as well as improving aesthetics. To this end, rehabilitation trials should be undertaken during operation to determine the most efficient method to effectively cap and rehabilitate the surface of the TSFs. The results of these trials will be used to design the most suitable cover to be placed over the tailings surface.

18.17.16 Quantities and cost estimate

Quantity and cost estimates for the Flotation and CIL TSFs, site access roads, airstrip, river abstraction tower, water storage facility, and surface water management facilities have been developed by KP and are summarised in Table 124.

Table 124: Summary of Cost Estimates

Item	Infrastructure	Option 1 – 4.5 Mtpa	Option 2 – 7 Mtpa	Option 3 – 9.5 Mtpa
		US$	US$	US$
1	Flotation TSF	89,048,481	86,144,377	98,360,615
2	CIL TSF	51,114,849	50,182,103	49,774,443
3	Access Roads	6,707,280	6,707,280	6,707,280
4	Airstrip	470,482	470,482	470,482
5	River Abstraction Tower	86,924	86,924	86,924
6	Water Storage Facility	1,047,588	1,047,588	1,047,588
7	Surface Water Management	684,084	684,084	684,084
8	Total	149,159,688	145,322,838	157,131,415

19.0 MARKET STUDIES AND CONTRACTS

No formal marketing studies have been completed.

Gold is a readily traded commodity and a specific market study was not required.

Gold doré bars will be transported from the project site to an accredited gold refiner for smelting and refining into an LME grade gold bar on a regular basis, and the refined product credited to the company's revenue account.

Advice regarding the forward-looking gold price was provided by Cardinal and the Project assumes US$1,250/oz for the purposes of the financial model at the date of this PFS.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

This Section is based on an independent consultant's report to Cardinal by Nemas Consult Ltd, of Accra Ghana, (NEMAS, 2018). NEMAS was contracted by Cardinal to undertake the Environmental Impact Assessment study for the Project.

20.1 Introduction

NEMAS has undertaken a site reconnaissance visit and completed the scoping stage of the process in accordance with the Ghanaian Environmental Protection Agency procedures for the EIA. NEMAS' scoping study included preliminary field surveys, literature reviews, and examination of appropriate legal and regularity frameworks.

20.1.1 Background

Cardinal acquired a 20 km^2 concession, located within the Talensi District in the Upper East Region to establish a large-scale Gold Mining Company.

The Ghanaian Environmental Protection Agency ("EPA"), EPA Act 490, and Regulation 1 of LI 1652 Regulation 1 & 2 states:

(1) *No person shall commence any of the undertakings specified in Schedule 1 to these Regulations or any undertaking to which a matter in the Schedule relates, unless prior to the commencement, the undertaking has been registered by the Agency and an environmental permit has been issued by the Agency in respect of the undertaking.*

(2) *No person shall commence activities in respect of any undertaking which in the opinion of the Agency has or is likely to have adverse effect on the environment or public health unless, prior to the commencement, the undertaking has been registered by the Agency in respect of the undertaking"*

20.1.2 Registration

In compliance with the above regulations, the Namdini Gold Project was registered with the Ghana EPA for environmental permitting, under the title Savannah Gold Mining Project. The EPA in response to the registration application by the Proponent in a letter dated 23 November 2016 indicated that the project which falls under schedule 2 makes mandatory a full-scale Environmental Impact Assessment ("EIA") study and submission of Environmental Impact Statement ("EIS") to the EPA.

20.1.3 Scoping Report

As part of the permit acquisition process, the Environmental Assessment Regulation 1999 of LI1652, Regulation 11 states "A scoping report shall set out the scope or extent of the environmental impact assessment to be carried out by the applicant, and shall include a draft terms of reference, which shall indicate the essential issues to be addressed in the environmental impact statement".

The scoping study was submitted to commence the process of Environmental Impact Statement (EIS) in accordance with Regulations 15(1b) and (1c) of the Environmental Assessment Regulations, 1999 (LI 1652) and Ghana's Environmental Impact Assessment (EIA) Procedures, the Environmental Protection Agency (EPA).

Accordingly, and in compliance with directives by the EPA as contained in EPA letter to the proponent, a scoping report was prepared and submitted to the Agency on 22nd June 2017, which also set out the Terms of Reference ("ToR") for the EIA and EIS (the "ESIA") study. The scoping report highlighted the following issues among others: Project Description, Environmental and Social baseline conditions (mostly from secondary sources) and key environmental and social issues of impact and some preliminary proposed mitigation measures. The scoping report also captured the various national and internals laws, policies and guidelines that shall be triggered in course of the study. Additionally, the concerns of some key stakeholders consulted were captured in the report and other key stakeholders needed to be consulted were also identified. The key stakeholders in Table 125 were identified.

Table 125: Identified key stakeholders

No.	Organization	Relevance to the Study
1	Cardinal Resources and Savannah Mining Ltd	Proponent/Project owners
2	EPA, Head office (Accra)/Regional office (Bolga)	Environmental Permitting
3	Minerals Commission, Head Office	Mining lease
4	Water Resources Commission	Water abstraction permit/approval
5	Ghana Highway Authority, Regional Office	Haul/link road approval
6	Office of the Administrator of stool lands	Granting of surface land right
7	Volta River Authority	Supply of electricity/Custodian of the White Volta
8	Northern Electricity Distribution Company (NEDCo)	Supply of electricity for operations
9	Talensi District Assembly	Local government authority/building & structural permit
10	Ghana Museum and Monument Board	Archaeological and cultural clearance
11	Ghana Police Service	Provision of security
12	Ghana National Fire Service Authority	Granting of Fire permit/certificate
13	Selected health facilities	Provision of health needs of workers
14	Traditional authorities	Custodian of area of operation
15	Leadership of artisanal miners	Immediate project affected persons
16	Opinion leaders of host communities	Representative of Project affected persons/beneficiaries
17	Identifiable civil society organizations	Representation of project affected persons/beneficiaries
18	Main religious bodies	Spiritual/traditional representative of host communities

The following potential issues and impacts from project implementation were identified at the PEA stage and will be evaluated during the impact analysis stage of the EIA.

Physical Environment

- Air pollution from blasting, loading, transport, dumping of ore, and increased vehicular movement along haul roads, operation of mine equipment, light vehicles, and diesel-powered electrical generators
- Noise/Air blast and Vibration from blasting, crushing and milling operations, movement of haul trucks and operations of electrical generators and maintenance activities at the mechanical workshops
- Impact on soil resources from topsoil removal, erosion and soil contamination from chemical/fuel spills
- Impact on surface water resources from water pollution concerns due to chemical/fuels spills into water, transport of sediment and sewage into water, changes in surface water hydrology and flows due to the construction of mine infrastructure, and raw water abstraction for both domestic and production usage

- Impact on groundwater resources:
  - Ground water pollution concerns from spillage of fuel/oils and chemicals, seepage of chemicals from tailings dam, and leachate of metals from exposed waste rock
  - Changes in the local ground water drawdown/quantity due to pit dewatering concerns.
- Disposal of hazardous and non-hazardous wastes concerns
- Land degradation from creation of mine pits, waste dumps, tailings dam.

Biological Environment

- Impact on terrestrial flora and fauna:
  - Removal/loss of vegetation
  - Damage to or destruction of faunal habitat
  - Displacement and destruction of wildlife
- Impact on aquatic/water ecology
  - Effect of water pollution on aquatic life
  - Effect of increased pollution on water resources and usage
- Impact on birds
  - Poisoning of birds due to access to or contact with contaminated tailings pond or overflows.

Social Environment

The adverse impacts likely to arise during the implementation of the project include:

- Land take and effects on land availability for other usage
- Disruption in land use:
  - Loss of land for cattle grazing
  - Destruction of cash crops (sheanut)
  - Loss of natural resources and gathering opportunities
  - Displacement of small-scale miners (galamsey).
- Influx of new workers/job seekers and new residents, many of whom may have different cultural expectations and values
- Increased potential of social problems such as prostitution, teen pregnancy, drugs, drunkenness, and crime
- Resource use/consumption concerns
- Electricity and fuel consumption concerns
- Potential for inflation of housing costs in the local communities
- Visual intrusion
- Noise, vibration and dust nuisance

- Public/community health and safety concerns
  - Increased potential for spread of infectious diseases, including HIV/AIDS
  - Increased potential for spread of malaria in the local communities
  - Increased potential for accidents on the public/community roads
  - Worker health and safety concerns.

20.1.4 Response of EPA on the Scoping Report

On receiving the Scoping Report the EPA posted a Scoping Report Notification on page 24 of the August 18, 2017 edition of the Ghanaian Times (a government owned daily newspaper with a wide national circulation) requesting persons who have an interest, concern or special knowledge relating to the potential environmental effect of the proposed undertaking to contact or submit such concerns, etc., before the Environmental Impact Statement notification, to the Executive Director at its National Office in Accra and/or the Regional Director at its Regional office in Bolgatanga or the Managing Director of the proponent's company in Bolgatanga. The EPA also provided copies of the Scoping Report to the Talensi District Assembly in Tongo and to its Regional Office in Bolgatanga.

NEMAS are in the process of a detailed Environmental Impact Study which will be submitted to the Ghanaian EPA for approval.

20.2 Primary baseline data generation from field investigations

In order to establish baseline features of the site and immediate areas within the Project Area of Influence as well as to act as a reference and for future monitoring during the implementation stage of the project, various field studies are underway as part of the ESIA studies. All field studies except for public engagement were completed. Draft reports on Ecology, Soils, Environmental Media Quality, Archaeological and Cultural profiling were completed. A report on the aquatic ecosystem is in progress.

A summary of the completed draft reports is provided in the following sections:

20.2.1 Ecological studies

20.2.1.1 General overview

The structure of the vegetation of the project site is rather simple and uniform. In addition to the simple vegetation structure, the project site also has rather uniform species composition; as a result of cultivation and regular annual grass burning.

A total of 39 species belonging to 35 genera in 13 families were identified in the study area. The Family Graminae dominated the flora with 16 species, followed by the Caesalpiniaceae and Mimosaceae with 4 species each. The rest of the families had 3 or less species representation in the flora. The Life Form Composition shows the tree and herb life forms to be the dominant life forms, constituting 53.8% and 41% of the flora respectively. The trees and shrubs that occur on the project site are fire-tolerant and are mostly left standing during cultivation. Some of the dominant trees and shrubs encountered in the study area are *Parkia biglobosa* (Dawadawa), *Vitellaria paradoxa* (Shea tree), *Anogeissus leiocarpos*, *Afzelia africana*, *Diospyros mespiliformes*, *Sarcocephallus latifolius* and *Chromolaena odorata*. The ground flora was predominantly tall grasses including *Andropogon gayanus*, *A. tectorum*, *Hyparrhenis rufa*, *Pennisetum violaceum* and *Panicum maximum*.

The vegetation of the site showed a high level of disturbance. Table 126 shows that weeds occur in the same proportion as the Savanna species i.e. about 46% each. This means that the vegetation of the site is secondary to tertiary in ecological status. This could be attributed to cultivation and wild fires.

Table 126: Composition of species by ecological guild

Ecological Guild	Count	%
NF/SP	18	46.2
NPLD	1	2.6
NE	2	5.1
NFW/P	18	46.2
Total	**39**	**100.1**

Notes: NF/SP is non-forest/savannah planted (planted by people)
NPLD is non-pioneer light demander (primary species that demand light to grow)
NE is not evaluated (ecological guild has not been determined)
NFW/P is non-forest weed/pioneer (normal weed)

20.2.1.2 Plant species of conservation and economic importance

The flora is deficient in species of conservation concern both nationally and globally. *Afzelia africana* is listed as a Red Star Species in FROGGIE while *Vitellaria paradoxa* is listed as Vulnerable in the IUCN Red List. Most of the species occur widely in the savanna and are of no immediate conservation concern.

Some of the tree species encountered are of economic and medicinal importance: the Shea butter tree (*Vitellaria paradoxa*), African oak (*Afzelia africana)*, African copaiba balsam (*Daniellia oliveri)*, *Adansonia digitate* and African locust bean (*Parkia biglobosa*) are of economic and medicinal value.

20.2.1.3 Fauna

Several species of mammals and birds occur in the project area. The proximity of the project to the eastern wildlife corridor means that occasionally large mammals stray into the area. Some of the commonly seen wildlife are several primates, common among them are the Patas monkey (*Erythrocebus patas*), Green monkey (*Cercopithecus aethiops*) and Senegal galago (*Galago senegalensis)*. The African elephant (L*oxodonta africana*), Warthogs (*Phacochoerus aethiopicus*) and the West African Bushbuck (*Tragelaphus scriptus*) are occasionally seen in the project area. The Togo hare (*Lepus zechi)*, Giant rat (*Cricetomys gambianus*) and the Cane rat (*Thrynomys swinderianus*) are common in the area. The avifauna includes species such as the Grey Heron (*Ardea cinerea*), Double-Spurred francolin (*Francolinus bicalcaratus*), and Hammerkop (*Scopus umbretta*).

20.2.2 Archaeology and cultural profile

The survey aimed at identifying and documenting cultural and other heritage assets within the concession and its catchment area to evaluate any potential impact from the proposed mining activities.

20.2.2.1 General overview

A total of fifteen heritage resources, two old archaeological settlement sites and thirteen shrines were identified and documented at Biung, while three shrines and fourteen archaeological settlement sites were identified and documented at Dotoko. With certainty that one of the archaeological sites was in the concession area, two test pits of one meter-square, were excavated to ascertain the nature of cultural materials associated with site. Surface collection was also done at this site of associated material remains. Resources identified were documented, with names, GPS coordinates, and brief descriptions of the resources complemented with photographs where appropriate.

20.2.2.2 Summary of artefacts obtained

A total of 155 artefacts were collected from both the excavated units and the surface. They comprised one 103 local ceramic shards, 30 glass fragments, 2 foreign ceramic shards, 12 metal objects, 3 bones (Figure 134), 4 querns, and 1 plastic bead.

Some glass objects collected from the site

Local ceramic shards collected from the site (Nanbang, Yin and Tobik Dabogu)

An ancestral tomb

The pot marking Naa Gang's tomb

Figure 134: Artefacts excavated on site

Table 127 provides a summary presentation of finds.

Table 127: Summary of found artefacts

	Local Ceramic	Glass	European Ceramic	Metal	Bones	Querns	Bead
Surface Collection (Excavated site)	25	23	2	6	0	4	0
Test Pit 1							
Level 1	16	0	0	0	0	0	0
Level 2	13	0	0	0	0	0	0
Test Pit 2							
Level 1	77	4	0	1	3	0	1
Level 2	25	0	0	1	0	0	0
Surface Collection (Naa Gang Duun)	26	3	0	4	0	0	0
TOTAL	**103**	**30**	**2**	**12**	**3**	**4**	**1**
Percentage Distribution	67.74%	19.35%	1.29%	7.74%	1.94%	2.58%	0.65%

20.2.2.3 Finding implications

The archaeological survey conducted at Biung and Datoko yielded evidence of heritage resources or assets, principally in the form of archaeological sites and shrines. The communities consider these resources and shrines important as they are said to provide various mitigating solutions to their challenges and needs.

Similarly, the belief in life after death has ensured the continued importance of abandoned settlements as libations and other rituals are performed on such ground in veneration of the ancestors and for their intervention in the daily activities of the communities.

It was found that some of the shrines in Biung are located either within the concession or areas adjacent to the concession and could be impacted. These include Kpinkpalyen, Kpagsab Babuk, Sirkugri, Zietod, Voyen Wari, Zoozee, and Kpari Gayi. Two of the shrines in Biung, Zietod and Kpari Gayi had been destroyed and had to be reconstituted, while the former has had part of the sacred grove that protects it cleared despite prohibition by the community members from cutting down trees in the shrine area. It is also certain that one of the old settlements; Nanbang, Yin and Tobik Dabogu is within the concession area and will thus be excavated.

Two communities, the Accra and Tarkwa Communities, which emerged only some two decades ago as a result of small-scale gold mining activities (galamsey) may be affected and require relocation. The EIA study will capture possible issues of resettlement and will recommend the type of Resettlement Action Plan (“RAP”) needed to ensure an acceptable resettlement package.

Maps of the heritage assets are being prepared and will be imposed on the concession map to help fully identify the cultural resources that are within the concession. Mining activities need prior consultation with the local communities to engender cooperation and mutual understanding regarding such resources.

20.2.3 Soil studies

The site is covered largely by Birimian rocks of Precambrian age, which consist of green stones, andesites, amphibolites and schists. The original vegetation, like the climate is classified as Sudanese. This consists of short deciduous trees often widely spaced and a ground flora composed of different species of grasses of varying heights. The soils of the area were developed under the influence of a number of factors; notably among them climatic conditions especially rainfall and temperature, parent materials derived from Upper Birimian, Granite and Voltaian rocks, relief/topography, drainage conditions and time.

Four pits, one of which was outside the project area were dug and formally described (Figure 135), after which samples were taken for laboratory analysis. The sites were selected to conform to proposed infrastructure.

The physical properties of soils at the sites, notably texture, structure, bulk density, porosity, water content, aggregate stability and color, were very good and clearly indicate the availability of oxygen in the soils, the mobility of water into or through soils and the ease of root penetration.

The soils were slightly acid to neutral pH in both top and subsoil. The soils were non-saline, had moderate to high organic matter but low to moderate cation exchange capacity. The overall fertility assessment from the chemical analyses indicates an impoverished soil, due partly to continuous cropping and also activities by artisanal gold miners. The situation has led to a serious degradation of the land through severe sheet and gully erosions.

The trend of heavy metal concentration follows a consistent reducing pattern: Mn > Cu > Zn > Hg > Cn > Hg > Cd > Pb. Concentrations of Pb and Cd were in trace amounts in all the soils. Zn, Cu and Mn concentrations were high, noting that there may be possible adverse effects on plants sensitive to high Mn.

Soil profile showing Dorimon series

Soil profile showing Kalini series

Figure 135: Soil profiles in pits

20.2.4 Environmental media assessment

Dust dispersion for Total Suspended Particulates ("TSP") and Particulate Matter less than 10 microns in diameter ("PM_{10}"); gases comprising sulfur dioxide (SO_2) and nitrogen dioxide (NO_2) and noise dispersion were determined from six different sites within three different sampling regimes. The six sites include four communities defined as data generation sites. Two of these communities, the Tarkwa and Wankala are within the concession whilst the other two, Bingo and Accra communities were the closest communities to the concession. The exact positions of sampling were all captured with GPS (Table 128, Figure 136).

Table 128: Sampling sites location for Environmental Media Quality

No.	Sampling Site	Latitudes (N)	Longitudes (W)
1	AQM1 (Bingo)	10°37' 11.8''	000° 39' 54.1''
2	AQM2 (Accra)	10°38' 44.7''	000° 38' 41.4''
3	AQM3	10°39' 49.9''	000° 39' 54.7''
4	AQM4	10°39' 54.0''	000° 38' 35.4''
5	AQM5 (Tarkwa)	10°40' 41.0''	000° 39' 37.9''
6	AQM6 (Wankala)	10°40' 50.1''	000° 37' 59.0''

Figure 136: Environmental media sampling sites

20.2.4.1 Dust concentration

The highest values recorded were 81.9 µg/m^3 from the Tarkwa site and 4.5 µg/m^3 from the Accra site for TSP and PM_{10} respectively. These levels were all lower than the Ghana EPA recommended values of 230 µg/m^3 TSP and 70 µg/m^3 PM_{10}. Dust dispersions were mainly from air laden with dust particles from wind action over bare land, old small-scale (galamsey) mining heaps and crushing machines as well as vehicular/motor/tricycle movement.

20.2.4.2 Gas concentration

The highest mean values of gases recorded were 109.2 µg/m^3 at the Tarkwa site and 36.9 µg/m^3 at the Bingo site for SO_2 and NO_2 respectively. Even though these figures were within the EPA maximum permissible level ("MPL") of 150 µg/m^3, maximum levels of 829.4 µg/m^3 and 108.7 µg/m^3 were recorded. These figures were recorded when there was quite a lot of on-going activities at these sites, including exploration works by subcontracting firms. Intermittent release of fumes from vehicular movement and exploration machines coupled with exhaust fumes from motorbikes and tricycles could account for these readings. Baseline figures recorded when these activities were minimal during the third sampling regime indicates much lower values of 3.9 µg/m^3 and 34.9 µg/m^3 for SO_2 and NO_2 respectively.

20.2.4.3 Noise dispersion

Integrated noise levels from all the sites recorded lower values with respect to the 70 dB(A) maximum permissible levels set by the Ghana EPA. The highest noise level of 50.1 dB(A) was recorded at the Tarkwa site during the second monitoring regime. Baseline noise levels were mainly generated by domestic noise sources within the communities, crushing noise of small-scale (galamsey) miners' machines and vehicular movement

The ambient air quality of the immediate areas of influence within and around the concession indicates a moderately modified environment due to various induced anthropogenic interferences. This modification is due to unregulated activities normally witnessed within unplanned settings with limited controlled structures. The applicability of proper and appropriate control measures during the operation of the Project will sanitize the environment and keep any media pollution load within acceptable thresholds.

20.2.4.4 Water quality

A total of 11 sites were selected for water sampling of 28 physico-chemical parameters, including trace elements of water quality relevance were analysed. This comprised five sites along the White Volta close to the boundaries of the concession and six domestic bores within the communities. The results were compared with WHO drinking water MPLs. Six of the parameters recorded values at some sites that exceeded the MPLs, as follows:

Table 129: Summary of water quality parameters above MPL

Parameters	Exceedance Sites	Water Source
Total dissolved solids	1	Borehole
Turbidity	3	1SW; 2 BH
Sodium	3	All BH
Total hardness	4	All BH
Total iron	6	4SW; 2 BH
Manganese	1	BH

The available water is generally good for drinking and for other domestic uses. However, elevated iron content and hardness of water could create problems for water storage and washing.

20.3 Initial mitigation proposals under consideration

The general rules that will guide the design of mitigation measures are:

- Avoidance where impact is very significant and are considered unacceptable with disastrous consequences unless prompt and adequate measures are taken.
- Reduction where impact is very significant and/or moderately significant but could be mitigated reasonably and practicably through planning, designing and controlling mitigation measures. This implies that mitigation measures will be applied until the limitations of cost effectiveness and practical applications are reached. The limitations are established by best international practice.
- Implementation of good management practices for impacts rated as mildly significant or insignificant, in order to ensure that impacts are managed within good reason.

Classification of mitigation measures

The proposed mitigation measures for the identified impacts will be premised on the three principal methods/guide of mitigations namely Preventive, Control and Compensatory methods.

- Preventive method: Measures will be adopted to include environmental and safety concerns at the design stage to ensure that certain anticipated impacts are completely avoided or reduce to insignificant levels.
- Control method: Sustainable environmental and good mining practices will be devised to ensure impacts are reduced to acceptable levels.

- Compensatory methods: Measures will be adopted for unavoidable but non-disastrous impacts that result in direct losses to communities or individuals.

Mitigation of key impacts

Mitigation measures for key environmental impacts will be proposed in the EIS Report based on empirical studies.

The following measures are envisaged.

Air/noise pollution

- Suppression of dust
- Regular maintenance of equipment
- Adherence to design drawing specification in vegetation clearance
- Speed humps and speed limits
- No idling of vehicles/machines not in use
- Equipment selection with best technology
- Use of good quality fuel/lubricants
- Avoidance of working under aggressive weather condition
- Use of qualified operators and capacity building of machine operators
- Adoption of control/delayed blasting.

Impact on flora and fauna

- Restricted clearance according drawing specifications
- Preservation of patches of original vegetation as biodiversity/game reserves
- Avoidance of bush burning
- Authorization before felling of trees
- Afforestation of degraded lands
- Restriction of gaming/hunting and safe passage for stray animals
- Sensitization of workers on environmental preservation.

Water pollution

- Avoidance of indiscriminate disposal of waste
- Re direction of waste water away from water courses
- Adoption of bund walls around fuel/oil depot
- Ensuring effective collection and storage of spent oil
- Stringent servicing of fuel storage facilities to avoid spillage
- Maintenance of buffer zone around streams
- Use of silt control berms and sediment control structures

- Use of improve lining for TSF construction to prevent seepage
- Re-use of tailings water for ore processing
- Treatment of tailings to meet standards before release into external environment
- Avoiding release of tailings water directly into water bodies
- Ensuring effective containment of liquid waste and proper drainage system for grey water
- Effective disposal of both liquid and solid waste.

Topography/landscape alteration/reduction of erosion tendencies

- Practicing good landscape engineering
- Recontouring, pegging and terracing of areas prone to erosion
- Progressive re-vegetation or re-graveling
- Compacting of loose soil.

Visual effects

- Preservation of natural view by maintaining original vegetation within boundaries of active mining areas and within buffer zones
- Preservation of any natural features of interest
- Designing waste dumps and other disposal facilities to blend with surrounding topography
- Embarking on progressive rehabilitation of disturbed areas.

Solid and liquid waste

- Waste segregation to facilitate reuse and recycling
- Use of engineered landfill sites and incinerators
- Collection of hazardous waste material for disposal by accredited firm
- Treatment of liquid waste before disposal
- Ensuring leakage-free septic effluent system.

Compensation and resettlement

- Preparation of Resettlement Action Plan for compensation and relocation of affected communities
- Implementation of compensation and resettlement in accordance with a RAP Report
- Provision of all facilities such as schools, potable source of drinking water and clinics currently existing, at the new resettlements
- Establishment and implementation of Alternative Livelihood Programmes for community members who by virtue of the establishment of the mine lose their livelihood.

Occupational health and safety

Measures will be instituted to mitigate negative OHS issues.

The following measures are envisaged.

- Use of appropriate PPE

- Adequate visibility/lighting
- Ensuring safety at the pit by appropriate design of berms and pit walls
- Construction of structures to meet local and international building codes
- Effective signage
- Recruitment of qualified staff/operators
- Observance of traffic rules and regulations both on- and off-site
- Observance of break/rest within working hours
- Incentive/rewarding packages to boost morals of workers
- Installation or firefighting equipment
- Hoisting of caution flags and flash lights on operating vehicles and moving machines
- Effective communication system
- Training and refresher courses for workers on OHS compliance
- Well trained and effective security personnel
- Effective security network with the Ghana Police Service
- Open door policy of management for effective evaluation and for grievance channelling options
- Effective dust and noise abatement measures, such as regular dust suppression and routine maintenance of machines
- Periodic medical examination of workers to check their fitness level for work
- Time limitation of workers on exposure to hazardous and dangerous working sites
- Proper system of identification of workers and visitors
- Working to meet local EPA Akoben and other international OHS and Environmental standards, such as ISO14001
- Vaccination of workers against cerebrospinal meningitis bacterial disease ("CSM").

Public health and safety concerns

Measures will be adopted to ensure that the welfare of the public and host communities is not undermined by the operations of the mine.

The following measures are envisaged.

- Constructive and effective relations between community and the mine. A Community Relations Office shall be created in the host community to ensure easy access to management with community grievances
- Prompt attention by management to all community grievances received
- Observance of speed limits
- Escort of haulage trucks and other heavy equipment that move through town
- Restricted of haulage trucks and heavy machinery through town only during day

- Non-involvement of mine workers in local politics and chieftaincy issues
- Periodic community/mine engagement and sensitization, especially on HIV/AIDS awareness and other STDs.

20.4 Conclusion

Exploration activities are undertaken by Cardinal so that any potential emissions and effects associated with exploration activities, which could include habitat modification and associated visual effects, are kept to a minimum.

Auger drilling is used as a primary grassroots exploration tool as this method does not cause significant impact on the hole surroundings. For diamond and RC drilling site preparation and access in the Savannah grassland, it is mostly undergrowth that is cleared, while larger trees are preserved. Drill sites are kept clean of rubbish and free of oil or fuel spills and are then remediated upon completion of drilling.

The environmental studies carried out so far indicate the physical and biological features of the project area have been greatly modified by settlement and various human based activities. These activities have subsequently affected the ambient air and to a lesser extent the water quality within the catchment area.

Major stakeholders were identified and consultation is in progress.

The Project area environment has been already altered due to unregulated human activities, especially from the galamsey miners.

The setting up of the Project under strict environmental regulation, with the adherence of proposed mitigation measures and action plans, will restore environmental integrity of the site.

Cardinal considers that there are unlikely to be any specific environmental issues that would preclude potential eventual economic extraction.

21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimate

21.1.1 Approach

The project capital cost estimate was compiled by Lycopodium and reflects the project scope as described in this PFS report.

The estimate for the 7.0 Mtpa plant throughput option was estimated in sufficient detail to reflect a PFS (+20 to -15%). The 4.5 Mtpa and 9.5 Mtpa options were factored from the 7.0 Mtpa Mid Case and are discussed further below.

The capital cost estimate for the 9.5 Mtpa is summarized in Table 130 below:

Table 130: Preliminary capital cost estimate summary (US$, 1Q18 +30 to-20%)

Main Area	9.5 Mtpa (US$ M)
Construction distributables	42.0
Treatment plant	157.2
Reagents and services	39.8
Infrastructure	76.8
Contractor mining – establishment only	5.1
Subtotal	***320.9***
Management costs	37.1
Owners project costs	41.4
Subtotal	***399.4***
Contingency	14.6
Total	**414.0**

Notes: Rounding may affect totals

The process plant and infrastructure costs were estimated by Lycopodium.

Knight Piésold provided the quantities for the tailings storage facilities (TSF), site access roads and various other earthworks related infrastructure.

All costs are expressed in United States dollars (US$) unless otherwise stated and based on 1Q2018 pricing.

Factored throughput options

The 4.5 Mtpa and 9.5 Mtpa throughput options were factored from the 7.0 Mtpa Mid Case and are therefore classed as scoping or conceptual estimates with an accuracy of +30 to -20%.

The factored estimates were established by assessing the correlation between cost and the process design criteria with factors being determined by discipline for all areas of the estimate to an appropriate PFS level of study.

Contingency for the factored estimates was increased by a further 5%, applicable only to the estimate components that had been factored or adjusted from the Mid Case estimate.

The factored estimates are summarized in Table 131 and are compared directly with the 7.0 Mtpa Mid Case. Additional estimate summaries and detail were provided.

For the purpose of comparison, the main area values include the Mid Case applied contingency percentages. An additional factored contingency adjustment is included at the bottom of the table to achieve the specified estimate accuracy for the factored estimates.

Table 131: Factored estimate summaries (US$ M)

Main Area	9.5 Mtpa (+30/-20%)	7.0 Mtpa (+20/-15%)	4.5 Mtpa (+30/-20%)
Construction distributables	42.0	38.4	33.6
Treatment plant	157.2	128.3	95.8
Reagents and services	39.8	35.5	30.5
Infrastructure	76.8	72.0	66.4
Contractor mining – establishment only	5.1	4.4	3.5
Subtotal	***320.9***	***278.5***	***229.7***
Management costs	37.1	32.1	27.2
Owners project costs	41.4	37.4	33.6
Subtotal	***399.4***	***348.0***	***290.5***
Additional factored estimate contingency	14.6	-	9.7
Total	**414.0**	**348.0**	**300.2**

21.1.2 Estimate basis: Mid Case

Basis and methodology

The capital cost estimate was prepared in accordance with Lycopodium's standard estimating procedures and practices. The basis and methodology are summarized in Table 132 and Table 133.

Table 132: Capital cost estimate basis

Description	Basis
Site	
Geographical Location	Site Plan
Maps and Surveys	Good topographical data available
Geotechnical Data	Not Available. Assumed competent
Process Definition	
Process Selection	Fixed
Design Criteria	Fixed
Plant Capacity	Fixed for Mid Case
P&IDs	Not Required
Mass Balances	Fixed
Equipment List	Fixed
Process Facilities Design	
Equipment Selection	Selection based on duty
General Arrangement Drawings	Preliminary
3D model	Not required
Piping Drawings	Not required
Electrical Drawings	Not required
Specifications/Data Sheets	Not required
Infrastructure Definition	
Existing Services	Nil
Design Basis	Preliminary
Layout	Preliminary

Table 133: Capital cost estimate methodology

Description	Basis
Bulk Earthworks	Volume for bulk earthworks provided by the preliminary project model.
Detailed Earthworks	Allowances for under pad excavation and backfill to prepare site for concrete works.
Concrete Installation	Estimated from the layout and similar projects of comparable scale. Concrete (wet) supply rates and installation rates applied from project specific BQRs.
Structural Steel	Quantities estimated from the layout and similar projects of comparable scale. Supply and install rates applied from project specific BQRs.
Platework & Small Tanks	Details provided in the mechanical equipment list. Large item quantities estimated from reference projects. Smaller items taken from database. Supply and install rates applied from project specific BQRs.
Tankage Field Erect	Details provided in the mechanical equipment list. Supply and install rates applied from project specific BQRs.
Mechanical Equipment	Details provided in the mechanical equipment list. Costs from responses to BQRs from reputable suppliers for all equipment with a value nominally >$5,000. Costs for low value items taken from the Lycopodium database.
Haul Roads	Excluded – Contractor Mining
Mining Fleet	Excluded – Contractor Mining
Power Supply	Excluded as directed by Cardinal.
Conveyors	Concrete & structural estimated from reference projects and layout. Mechanical supply pricing from recent projects and installation rates applied from responses to BQRs.
Plant Piping General	Factored off mechanical costs.
Overland Piping	Size and specification based on engineering selection. Quantity based on site layout. Rates based on recent market inquiries.
Electrical General	Quantities derived from engineering design and site layout. Materials pricing and installation costs from a combination of recent database information and responses to BQRs.
Electrical HV	Quantities derived from engineering design and site layout. Materials pricing and installation costs from a combination of recent database information and responses to BQRs.
Commodity Rates – General	Appropriate rates from responses to project specific and other recent BQRs.
Installation Rates – General	Appropriate rates from responses to project specific BQRs based on preliminary contracting strategy.
Heavy Cranes	Requirements estimated based on largest lifts and likely duration.
Freight General	Combination of rates per freight ton & factors.
Contractor Mobilization/Demobilization	Appropriate rates from responses to project specific BQRs
Fencing	Cost based on measured length and rate.
EPCM	Percentage of EPCM controlled scope.
Vendor Representatives	Allowance based on similar projects.
Owner's Costs	
Site Establishment	Requirements estimated using base rates.
Construction Facilities	Allowance based on projects of a similar size.
Opening Stocks, First Fill Reagents and Consumables	Estimated from consumption rates and costs in operating cost estimate.
Working Capital	Estimated from costs in operating cost estimate and likely commissioning and ramp up schedule.
Spares	Allowance factored from mechanical supply cost.

Description	Basis
Owner’s Project Costs	Provided by Cardinal (retained from the PEA study).
Project Insurances and Permits	Part of Owner’s Project Costs
Sterilization Drilling	Assumed in sunk costs.
Land Compensation	Excluded.
Community Relations	Excluded.
Plant preproduction expenses	Estimated from costs in operating cost estimate and likely commissioning and ramp up schedule.
Training	Estimated from costs in operating cost estimate manning schedule.
Duties and Taxes	Excluded.
Escalation	Excluded.

General estimating methodology

Overall plant layout and equipment sizing was prepared with sufficient detail to permit an assessment of the engineering quantities for the majority of the facilities for earthworks, concrete, steelwork, and mechanical items. The layouts enabled preliminary estimates of quantities to be taken for all areas and for interconnecting items such as pipe racks.

Unit rates for labour and materials were derived from responses to Bulk Quantity Requests (“BQR”s) sent to fabricators and contractors experienced in the scale and type of work in the region.

Budget pricing for equipment was obtained from reputable suppliers with the exception of low value items which were costed from Lycopodium’s database of recent project costs.

For the accommodation camp, offices, workshops and similar items, appropriate budget pricing was obtained from reputable suppliers of similar prefabricated designs.

Knight Piésold provided the design and quantities of the following infrastructure items that were subsequently costed by Lycopodium:

- Tailings Storage Facility
- Water Storage Facility
- Airstrip
- Site Access Roads
- Surface Water Management
- River Abstraction Tower.

Pricing basis

Pricing was identified by the following cost elements, as applicable, for the development of each estimate item.

Plant equipment

This component represents prefabricated, pre-assembled, off-the-shelf types of mechanical or electrical equipment, either fixed or mobile. Pricing is inclusive of all costs necessary to purchase the goods ex-works, generally excluding delivery to site (unless otherwise stated) but including operating and maintenance manuals. Vendor representation and commissioning spares were allowed for separately in the estimate.

Bulk materials

This component covers all other materials, normally purchased in bulk form, for installation on the project. Costs include the purchase price ex-works, any off-site fabrication, transport to site (unless otherwise stated), and over-supply for anticipated wastage.

Installation

This component represents the cost to install the plant equipment and bulk materials on site or to perform site activities. Installation costs are further divided between direct labour, equipment and contractors' distributables.

The labour component reflects the cost of the direct workforce required to construct the project scope. The labour cost is the product of the estimated work hours spent on site multiplied by the cost of labour to the contractor inclusive of overtime premiums, statutory overheads, payroll burden and contractor margin.

The equipment component reflects the cost of the construction equipment and running costs required to construct the project. The equipment cost also includes cranes, vehicles, small tools, consumables, PPE and the applicable contractor's margin.

Contractors' indirect costs encompass the remaining cost of installation and include items such as offsite management, onsite staff and supervision above trade level, crane drivers, mobilization and demobilization, R&Rs, meals and accommodation costs, and the applicable contractors' margin.

Temporary construction facilities

Facilities will be capable of servicing the owners and EPCM teams.

Included in the estimate for construction facilities are:

- construction offices
- computers and computing servers, telephones, printers, etc. and office furniture
- provision of services.

Heavy lift cranage

A heavy lift crane of 250 t capacity was allowed for in the estimate for the duration of the installation of the mills.

Mobilization/demobilization

Costs for mobilization/demobilization of labour and equipment to/from the project site were, where practical, adopted from budget quotation enquiries to contractors or adjusted from current tenders/contracts to reflect the project location.

Earthworks

Quantities for plant site bulk earthworks were taken from the preliminary project model. Rates were derived from recent budget quotation requests for similar projects.

Quantities for the ROM pad (at the process plant) are limited to the engineered fill and drainage and site earthworks required around the ROM retaining wall. The cost of the balance of the ROM pad is included in the mining estimate.

Concrete

Quantities for concrete works were established using:

- material take-offs from layouts prepared for the PFS
- benchmarking against detailed drawings for similar sized projects completed by Lycopodium.

Rates for this estimate were based on responses to BQRs from regional subcontractors with experience on this kind of work and capacity to perform the works.

Rates and quantities were prepared on a composite per cubic metre basis. Mobilization, demobilization and indirect costs were separated to reflect contract methodology.

Steelwork

Quantities for structural steel were established using:

- the layout and equipment elevation drawings/sketches prepared for the PFS
- benchmarking against detailed drawings for similar sized projects completed by Lycopodium.

Rates for this estimate were based on responses to BQRs from fabricators with experience on this kind of work and capacity to perform the works.

Site installation hours were based upon responses to BQRs from regional contractors with experience on this kind of work and capacity to perform the works.

Platework/tankage

Platework and tankage quantities were determined using the sizing provided in the mechanical equipment list prepared for the PFS as the basis. A preliminary design was undertaken for each tank to select appropriate plate thicknesses to develop tank tonnages. Lining materials, where applicable, were quantified separately.

Rates for this estimate were based on responses to BQRs from regional subcontractors with experience on this kind of work and capacity to perform the works.

Mechanical equipment

The mechanical equipment list prepared for the PFS provided the quantities and sizing for the cost estimate.

Budget pricing was obtained from reputable suppliers for the majority of mechanical equipment, based on equipment data sheets prepared for the PFS.

Equipment installation hours were estimated based on responses to BQRs solicited from contractors and installation hours estimated by Lycopodium. For each individual item of equipment due allowances were made for retrieval from the storage location, handling, placing, installing and commissioning the equipment.

Plant pipework

The supply and installation estimate for in-plant piping was determined using factors derived from previously built projects. These factors are a percentage of the mechanical equipment supply and installation costs, and are calculated per individual plant area. The plant piping costs allow for the supply and installation of pipe, fittings, mountings and manual valves.

Overland pipework

The overland piping, i.e. tailings discharge line and decant water return line were estimated from first principle engineering. Quantities were based on material take-offs.

Electrical instrumentation

The supply and installation estimate for electrical and instrumentation was estimated in detail and compiled using electrical equipment lists, loads lists, GA drawings and supplier pricing. The electrical and instrumentation estimate was added into the master estimate at WBS Level 3 subtotals.

Erection and installation

Included in the discipline assessment of erection/installation costs detailed above, allowances were made for major construction cranage and equipment and construction costs such as site establishment, construction personnel meals, accommodation, flights and fuel usage, etc.

Architectural/buildings

Budget pricing for prefabricated and steel frame buildings were sourced from reputable suppliers based on preliminary layout drawings.

Transport

The transport costs included in the estimate are a combination of rates per freight ton and factors on supply costs.

EPCM

The Engineering, Procurement and Construction Management ("EPCM") estimate was based on a percentage of the EPCM controlled scope.

Expenses such as catering and accommodation for the Engineer's site personnel, as well as site telecommunications costs are included in the estimate.

Vendor commissioning

The estimate was based on Lycopodium's experience for similar plant and equipment.

Spares

Spares were taken as an allowance of the capital costs and benchmarked against the spares expenditure on projects of a comparable scale. A minimalist approach was assumed, with spares stocks progressively expanded during operations.

First fill and opening stocks of consumables

Quantities for opening stocks and first fill consumables were assembled from basic principles and using the project design criteria. Unit rates are based on budget quotations solicited from suitable suppliers.

21.1.3 Qualifications

The estimate is subject to the following qualifications:

- All labour rates, materials and equipment supply costs are 1Q18. Contingency was allowed based on the quality of the various estimate inputs, but no allowance for escalation was included.
- Construction contractor rates include mobile equipment, vehicles, fuel, construction power and consumables for the duration of construction. Potable water and raw water supply will be provided by the Client and available at site for the use by contractors.
- Accommodation, meals and mobilization/demobilization/R&R flights of construction contractor personnel are incorporated in the contractor indirect labour rates on the basis of individual contractors making their own accommodation arrangements.
- Project spares included as a percentage allowance of the mechanical supply cost based on similar size projects.
- A commissioning assistance crew is allowed for in the EPCM allowance.
- PLC programming for the process plant was allowed for in the EPCM allowance.

- Owners costs general were retained from the PEA and reused for the PFS as directed by Cardinal.

Contingency

The purpose of contingency is to make specific provision for uncertain elements of cost within the Project scope. Contingencies do not include allowances for scope changes, escalation or exchange rate fluctuations.

Contingency is an integral part of an estimate. It has not been applied at to all line items resulting in an overall project contingency of approximately 15%.

21.1.4 Exclusions

The following is excluded from the overall project capital costs:

- Power supply was excluded from the capital cost estimate as directed by Cardinal.
- Duties/taxes/fees (on the basis that the project will negotiate duty/tax-free status).
- Project sunk costs.
- Project escalation.
- Mining facilities as they will be provided by the mining contractor.

21.1.5 Escalation and foreign exchange

Escalation

Escalation is excluded from the estimate.

Exchange Rates

All items in the capital estimate were included in US$ and no allowances for exchange rate variations were included in the estimate.

21.1.6 Working and sustaining capital

Working capital, other than allowance for six weeks operating costs in the Opex estimate, was excluded from the estimate.

Sustaining capital was excluded from the estimate.

Mining

Mining costs other than establishment costs were excluded from the estimate.

21.2 Operating cost estimate

21.2.1 Summary

The purpose of this operating cost estimate is to provide substantiated costs which can be utilized for a preliminary assessment of the viability of the Namdini Gold Project. The operating costs were compiled by Lycopodium based on costs developed by:

- Contractor supplied estimates and Golder – Mining costs (Section 16.0)
- Lycopodium – Processing and General and Administration costs (Section 17.0).

Operating costs were determined for a single throughput rate of 7.0 Mtpa operating 24 hours a day, 365 days a year, with a primary grind size of P_{80} 106 µm and a flotation concentrate regrind grind size of P_{98} 15 µm. The estimate is considered to have an accuracy of ±25%, is presented in US$ and is based on prices obtained during the first quarter of 2018 (1Q18). Study currency exchange rates were confirmed by Cardinal.

The 4.5 Mtpa and 9.5 Mtpa options were factored from the 7.0 Mtpa Mid Case and are discussed further below.

Operating cost summary

The processing operating cost estimate is summarized in Table 134 with the 9.5 and 4.5 Mtpa options factored off the 7.0 Mtpa Mid Case. The operating costs were compiled from a variety of sources, including the following:

- The LOM design mass recovery to flotation concentrate is 7.5%. This is based on recent testwork showing high gold recovery to concentrate at this mass pull.
- Flotation reagent consumption based on recent Pre-feasibility optimization testwork.
- Leaching reagent consumption based on industry norms in anticipation of final testwork results.
- Calculated reagent usage regimes for cyanide detoxification prior to testwork.
- Modelling by OMC for crushing and grinding energy and consumables, based on the final comminution testwork.
- Typical industry data from equipment vendors.
- Budget pricing or Lycopodium's database of prices for consumables.
- Lycopodium's database of costs for similar sized operations.

Table 134: Operating cost summary (US$/t)

Cost Center	9.5 Mtpa (±40%)	7.0 Mtpa (±25%)	4.5 Mtpa (±40%)
Power	4.8	5.0	5.2
Operating consumables	4.4	4.4	4.6
Maintenance	0.5	0.6	0.8
Laboratory	0.2	0.3	0.4
Process & maintenance labour	0.4	0.6	0.9
Administration labour	0.5	0.6	1.0
Total Processing	***10.8***	***11.5***	***12.9***
General & Administration costs	0.7	0.9	1.4
Total G & A	***0.7***	***0.9***	***1.4***
Total	**11.5**	**12.4**	**14.3**

The operating cost estimate presented in this section is exclusive of the following:

- All head office costs.
- Withholding taxes and other taxes. Import duty on consumable cost is included.
- Any impact of foreign exchange rate fluctuations.
- Any escalation from the date of the estimate.
- Any contingency allowance.
- Any land or crop compensation costs.
- Any rehabilitation or closure costs.
- Any licence fees or royalties.

- Tailings storage costs, including future lifts and rehabilitation.
- Government monitoring/compliance costs.
- Gold refining and transport of gold from site.
- All costs associated with areas beyond the battery limits of Lycopodium's scope of work.

The 4.5 Mtpa and 9.5 Mtpa throughput options were factored from the 7.0 Mtpa Mid Case and are therefore classed as scoping or conceptual estimates with an accuracy of ±40%.

The factored estimates were established by reviewing the basis for estimation for each cost center. Generally:

- grinding power varies with throughput but power for services and reagents is relatively fixed. As grinding power is the dominant consumer overall power cost per tonne is relatively constant
- operating consumables vary with throughput so the cost per tonne is relatively constant
- maintenance was calculated based on cost of equipment which does not vary linearly with throughput
- laboratory, labour and administration are fixed costs so the cost per tonne varies with throughput

21.2.2 Estimate basis – Mid Case (7.0 Mtpa)

Labour costs

The processing labour cost includes all labour costs associated with plant operations and maintenance personnel. The site laboratory is assumed to be operated on a contract basis with the personnel included in the process labour count but the labour costs included in the contract laboratory cost.

The administration labour cost includes all labour costs associated with site-based administration personnel. All head office costs are excluded (included in the Owner's costs). The camp is assumed to be operated on a contract basis with the personnel included in the administration labour count, but the labour costs included in the camp catering cost.

All labour costs for mining personnel are excluded (included in the mining costs).

The labour manning numbers for plant operations and maintenance and site G&A are summarized in Table 135. A full listing of labour positions, number of personnel and labour rates and costs is available separately.

The manning levels and rosters used to determine the labour operating cost estimate were based on similar operations. The estimate of the labour contingent was based on a three-shift operation (two shifts working 12 hours per day, one rotation shift), to provide continuous coverage for the plant operation with allowance for leave and absenteeism coverage. Provision was made for four weeks leave and two weeks sick leave per year per person.

Unit rates for labour were based on information from the Lycopodium database. An overhead cost allowance was made to cover such items as payroll taxes, worker's compensation, death and disability insurance, leave provisions and superannuation contributions. Camp and bus transportation costs for the workforce are excluded from the labour cost as they are included in the G&A cost.

Table 135: Plant and administration personnel requirements

Personnel	Count
Foreign expat	16
Regional	74
Local	221
Total	**311**

Power costs

The power cost estimate was based on grid power at a unit cost of US$0.12/kWh as advised by Cardinal. The power requirements and costs for the mine services area ("MSA") and the accommodation camp are also included.

The power consumption for the SAG and ball mills was calculated from typical ore properties, as determined by OMC. The power consumption for the regrind mill is based on the nominal mass pull of 7.5% to flotation concentrate and an assumed specific grinding energy of 50 kWh/t as advised by the mill vendor. The power consumption for the remainder of the Project was estimated from calculated power required for the process plant equipment and infrastructure. Typical drive efficiency and utilization factors were applied to the installed power to estimate the plant average continuous power draw.

21.2.3 Consumables costs

Costs for processing operating consumables, including reagents, liners, fuels and process supplies were estimated and are summarized by plant area in Table 136. The consumables cost for mining is included in the mining operating cost.

Table 136: Processing operating consumables cost summary

	Annual Cost (US$ M)	Unit Cost (US$/t)
Crushing	0.07	0.01
Milling	14.98	2.14
Flotation	2.76	0.39
Regrind	1.39	0.20
Leaching	7.25	1.04
Thickening	0.62	0.09
Refining	0.31	0.04
CN Detox	2.89	0.41
Water	0.08	0.01
Fuel	0.64	0.09
Other	0.02	0.00
Total	**30.99**	**4.43**

The consumption of reagents and other consumables was calculated from laboratory testwork and comminution circuit modelling or was estimated based on experience with other operations. No additional allowance for process upset conditions and wastage of reagents was allowed for. Details of consumption rates are available separately.

Reagent costs were sourced from budget quotations and in-house data relating to similar projects in the region. Transport and freight to site and import duties and taxes were added.

Cyanide destruction cost was based on the Air/SO_2 method, with the treatment of concentrate CIL tailings containing 500 ppm CNWAD down to below 50 ppm CNWAD after cyanide destruction.

A diesel price, delivered to site, of US$0.90 per litre was advised by Cardinal. This price is based on recent quotations from local suppliers. Diesel will be used in plant mobile equipment, for carbon elution and regeneration and for the gold room furnace. The diesel consumption for plant mobile equipment is based on industry standard vehicle consumption rates and estimated equipment utilization. The diesel usage for carbon treatment and the gold room was calculated from first principles.

Allowances were made for mill lubricants, water treatment reagents and operator supplies.

21.2.4 Maintenance materials costs

The plant maintenance cost allowance was factored from equipment supply capital costs using factors from the Lycopodium database and is summarized in Table 137 with further detail available separately.

Table 137: Maintenance cost summary

Area	Plant Maintenance Cost		Fixed Annual Plant Maintenance Cost
	US$/y	US$/t	US$/y
Plant maintenance	3.2	0.46	2.7
Mobile equipment	0.3	0.04	0.2
Maintenance general	0.2	0.03	0.2
Contract labour	0.5	0.07	0.2
Total cost	**4.2**	**0.60**	**3.3**

The plant maintenance cost allowance covers mechanical spares and wear parts, but excludes crushing and grinding wear components, grinding media and general consumables, which are captured in the consumables cost. It excludes site maintenance labour, which is included in the labour cost. Contract labour was allowed for crusher and mill liner changes and plant shutdowns.

Allowances for plant mobile equipment, plant building maintenance and general maintenance expenses were made. The mobile equipment allowance is based on unit costs for maintenance of the light vehicles, portable generators and other mobile equipment for the process plant. General maintenance expenses include specialist maintenance software, maintenance manuals and control system licence fees.

21.2.5 Laboratory costs

The laboratory cost is based on assaying and analyses of periodic plant samples and 100 mining grade control samples per day, as summarized in Table 138.

Laboratory costs were based on in-house data for contract laboratories of a similar size.

Table 138: Laboratory cost summary

Item	Samples/Month	US$ M/Year
Fixed fee		1.93
Variable fee (based on the following samples)		0.08
Mine exploration & grade control	3,000	
Plant solids (assay, moisture, sizing)	342	
Plant solutions (assay)	402	
Plant carbon (assay)	120	
Bullion (Au)	22	
Environmental (CNWAD, Total CN)	24	
Total		**2.00**

21.2.6 General and Administration costs

The general and administration costs were based on information from the Lycopodium database. The G&A costs are summarized in Table 139 with further detail provided separately. The administration labour cost was calculated as described in Section 21.2.2 above and is included below.

Table 139: Annual General & Administration costs summary

	7.0 Mtpa
Site Office	0.81
Insurance	1.45
Financial	0.18
Government Charges	0.0
HR Administration	0.50
Contracts	1.99
Community Relations	0.17
Other	0.90
Total G&A	**6.0**
Administration Labour	4.53
Total G&A including labour	**10.54**
Total G&A including labour ($/t)	**1.51**

The basis of costs for the accommodation contract based on camp requirements for the processing and administration personnel (but excluding mining personnel) is shown in Table 140.

Table 140: Camp and catering costs

Classification	Count of Personnel	Rate (US$/person/day)	Time on Site (weeks/year)	Annual Cost (US$)
Personnel in camp	87	25	32	488,538
Personnel not in camp	224	12	32	603,766
Total	**311**			**1,092,305**
Total Annual cost incl. 10% contingency for visitors				**1,201,535**

Notes: *Processing & administration personnel only, including laboratory staff

21.3 Operations preproduction and working capital costs

The costs incurred by the operations during the latter stages of construction and commissioning are included in the capital cost estimate but are derived in this estimate. Labour costs reflect the need to recruit key operating personnel in time for them to set up and establish operating procedures and undergo training as required. The pre-production cost estimate is summarized below with details available separately. Pre-production costs associated with mining are excluded.

- Administration Labour and Expenses US$2.15 million
- Process Plant Labour and Expenses US$0.45 million

21.3.1 First fill reagents and opening stocks

Costs were allowed to purchase the consumables and reagents required to fill the reagent tanks, charge the mills with media and provide the initial stocks of materials to sustain the operations for the first month until regular ordering of supplies can be established. Quantities allowed were based on either consumption over a minimum period or minimum shipping quantities, considering package size. Costs are summarized as follows with details available separately:

- First fill US$1.60 million
- Opening stocks US$5.97 million
- Total US$7.57 million

21.3.2 Vendor representatives and training

These costs allow for specialist vendor representatives to oversee commissioning of their equipment. The training allowance covers the cost of providing training for operations maintenance staff, but not their salaries, as these are covered in the pre-production labour costs. Costs are summarized as follows with details available separately:

- Vendor representatives US$0.58 million
- Training US$0.30 million

21.3.3 Working capital

Six weeks of processing and administration operating costs was allowed to cover the cost of operating the plant before the first revenue is received from bullion sales. The value of this is US$9.39 million.

22.0 ECONOMIC ANALYSIS

22.1 Forward-looking information

The results of the economic analysis represent forward-looking information that is subject to a number of known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented within this report. Forward-looking statements in this report include, but are not limited to, statements with respect to future gold prices, estimation of Ore Reserves and Mineral Resources, the realisation of Ore Reserve estimates, unexpected variations in quantity of mineralized material, grade or recovery rates, geotechnical and hydrogeological factors, unexpected variations in geotechnical and hydrogeological assumptions used in mine designs including seismic events and water management during the construction, operations, closure and post-closure periods, the timing and amount of estimated future production, costs of future production, capital expenditures, future operating costs, costs and timing of the development of new ore zones, success of exploration activities, permitting time lines and potential delays in the issuance of permits, currency exchange rate fluctuations, requirements for additional capital, failure of plant, equipment or processes to operate as anticipated, government regulation of mining operations, environmental, permitting and social risks, unrecognized environmental, permitting and social risks, closure costs and closure requirements, unanticipated reclamation expenses, title disputes or claims and limitations on insurance coverage.

22.2 Methodology

The Project has been evaluated using a discounted cash flow ("DCF") analysis. Cash inflows consist of quarterly and annual revenue projections. Cash outflows consist of capital expenditures, operating costs, taxes and royalties. These are subtracted from the inflows to arrive at the annual cash flow projections.

To reflect the time value of money, annual net cash flow ("NCF") projections are discounted back to the Project valuation date using selected discount rates. The discount rate appropriate to a specific project depends on many factors, including the type of commodity and the level of project risks (market risk, technical risk and political risk). The discounted, present values of the cash flows are summed to arrive at the Project's net present value ("NPV").

In addition to the NPV, the internal rate of return ("IRR") and the payback period are also calculated. The IRR is defined as the discount rate that results in an NPV equal to zero. The payback period is calculated as the time required to achieve positive cumulative cash flow for the Project.

Cardinal's Free Cash Flow calculation for the 9.5 Mtpa option is the total revenue generated, minus total costs. The calculation is total recovered ounces (3.975 Moz), multiplied by the net revenue per ounce (i.e. the selected gold price US$1,250/oz minus the total cost per ounce of US$833/oz).

Pre-Tax cash flow is before tax is deducted, while Post-Tax is after tax is deducted.

22.3 Operating costs

Operating Costs per tonne of ore processed (129.6 Mt of ore) are tabulated below (Table 141).

Table 141: Operating costs

	9.5 Mtpa (US$/t)	7.0 Mtpa (US$/t)	4.5 Mtpa (US$/t)
Mining	7.9	8.5	8.6
Processing	10.8	11.5	12.9
G & A	0.7	0.9	1.4
TOTAL	**19.4**	**20.9**	**22.9**

Owners Costs are tabulated below (Table 142).

Table 142: Owners costs

	9.5 Mtpa (US$/t)	7.0 Mtpa (US$/t)	4.5 Mtpa (US$/t)
Grade control	0.5	0.5	0.5
Owners G & A	0.6	0.8	1.0
TOTAL	**1.1**	**1.3**	**1.5**

Sustaining Capital Costs provided by consultants and Cardinal were compiled from a variety of sources and compared against existing and planned operations elsewhere in Ghana.

Sustaining Capital Costs which include rehabilitation and mine closure are tabulated below (Table 143).

Table 143: Sustaining costs

Unit	9.5 Mtpa	7.0 Mtpa	4.5 Mtpa
US$ (M)	170	159	164

22.4 Financial model parameters

22.4.1 Basis of analysis

The financial analysis is based on the Ore Reserve presented in Section 15.0, the mine and process plant and assumptions detailed in Sections 16.0 and 17.0, the projected infrastructure requirements outlined in Section 18.0, the doré and concentrate marketing assumptions in Section 19.0, the permitting, social and environmental regime discussions in Section 20.0, and the capital and operating cost estimates detailed in Section 21.0.

22.4.2 Metal pricing

The gold price assumptions used for the purposes of this PFS and the project gold price for the financial analysis are presented in Table 144.

Table 144: Gold price

Selection Case	Gold Price
Value was maximized by deferring larger strip-ratio cut-backs until later in the mine life. The maximum value pit was selected using a discounted average NPV and a Revenue Factor (“RF”) shell of approximately $1,105/oz using estimated LOM input prices and costs. Pit shells were converted into engineering designs prior to export of the contained resource model for scheduling.	US$1,105/oz
Trial open pit optimizations were run in Whittle 4X at a US$1,300/oz gold price (which was the appropriate gold price at the time of the optimization runs) to define the base of potentially economic material. Four Phases (cut back pits) were then selected for full mine design.	US$1,300/oz
The Financial Model Input gold price for all options was US$1,250/oz.	US$1,250/oz

22.4.3 Transport costs – doré

An airstrip will be developed near the process plant area. Doré will be delivered to a security contractor at the process plant. Fixed wing aircraft transport will be used to fly doré from site to an international commercial airport in Ghana. The doré will then be delivered by air to the nearest commercial refinery. For the purposes of the study, it is assumed that the refinery will handle the security and logistics chain from the point of receipt at the process plant through delivery to the refinery.

Doré transport and insurance costs are assumed to average US$0.65/oz of gold produced.

22.4.4 Working capital

Working capital cash outflow and inflows are included in the model. The calculations are based on the assumptions that accounts payable will be paid within 30 days and accounts receivable will be received within 60 days.

22.4.5 Royalties

The Ghanaian government, as the owner of non-renewable natural resources, is entitled to receive mineral royalties from mining companies that have exploitation agreements. The rate is 5% of the net sales. Royalties are assumed to be paid annually in December of each year.

22.4.6 Taxes

Mining companies domiciled in Ghana are subject to an annual corporate income tax at a rate of 35%. The Ghanaian income tax year is 1 January to 31 December with assessments being undertaken by the Ghana Revenue Authority. The income tax basis is determined by the total taxable income less allowable deductions according to the tax law. All deductions and rates are based on currently-enacted legislation, and are subject to change in the future.

The Minister of Lands and Natural Resources, upon advice from the Minerals Commission may enter into a Development Agreement under a Mining Lease with a company where the proposed investment by the company exceeds US$500 million. A Development Agreement may contain provisions relating to the mineral right or operations to be conducted under the mining lease, the circumstance or manner in which the Minister of Lands and Natural Resources will exercise discretion conferred by the Minerals and Mining Act on tax stabilisation as indicated above. Environmental issues and obligations of the holder exist, to safeguard the environment in accordance with any enactment and dealing with the settlement of disputes. A Development Agreement is subject to ratification by Parliament.

Cardinal is currently in negotiation with the Ghanaian government to establish a Development Agreement to potentially introduce tax concessions for the Namdini Project.

22.5 Government carried interest

Under the Mining Code of Ghana, the government is entitled to a 10% free carried interest in the rights and obligations of the mineral operations where the mineral right is for mining or the exploitation of minerals for which the government is not required to make any financial contribution. The government is not precluded from any other or further participation in mineral operation subject to the agreement of the holder. There is no special listing requirement for the project company.

Based on guidance provided by Cardinal, the mechanism planned to develop and operate the mine and incorporate the government’s interest has been modelled as follows:

- An operating company will be formed, with Cardinal holding 90% of the shares and the government of Ghana 10% of the shares.
- Cardinal’s sunk costs and funds provided to develop the mine will be booked as a loan to the operating company, to be repaid with interest out of available cash flow.
- The loan to the operating company will be repaid before any distributions are made to the two shareholders of the operating company.

- Following repayment of the loan, the operating company free cashflow will be distributed to the two shareholders in the form of dividends, with 10% of the dividends going to the government of Ghana and 90% to Cardinal.
- Dividends and interest received by Cardinal will be subject to Ghanaian withholding taxes which are 8% and are not recoverable.

22.5.1 Financing

The model does not include any costs associated with financing.

22.5.2 Inflation

There is no adjustment for inflation in the financial mode and all cash flows are based on 2018 US dollars.

22.6 Funding

Cardinal will use a staged funding approach for the on-going development of the Namdini project. Cardinal has budgeted for the Feasibility Study out of their existing cash balance, which includes the recent senior secured credit facility of US$25 M with Sprott (Cardinal, 2018b). The Board believes that there are strong reasonable grounds to assume that future funding will be available to fund Cardinal's pre-production capital for the development of Namdini as envisaged in this announcement.

- Cardinal is confident that it will continue to increase Mineral Resources at the project to extend the mine life beyond what is currently assumed in the PFS.
- As of September 2018, the gold price was trading at an average of approximately US$1,200/oz which compares favourably to the project's financial assumption of US$1,250/oz. The recent improvement in market conditions and an encouraging outlook for the gold market enhances the Company's view of the ability to finance the Namdini project.
- The strong production and economic outcomes delivered in the Namdini PFS are considered by the Cardinal Board to be sufficiently robust to provide confidence in the Company's ability to fund its pre-production capital through conventional debt and equity financing.
- Cardinal is in early discussions with a number of financial advisors and substantial mining investment funds with a view to fund Namdini in stages to production. These financiers have extensive track records of funding similar companies through PFS and FS, construction financing and into commercial production.
- Cardinal's Board has a financial track record and experience in developing projects:
  - Non-Executive Charmain Kevin Tomlinson has over 30 years of experience in Mining and Finance within Toronto, Australian and London Stock markets. Mr Tomlinson has extensive experience in development and financing of mining projects internationally.
  - Non-Executive Director Jacques McMullen has had a distinguished 35-year career in the mining industry of which the last 17 years were with Barrick Gold Corporation where he held the positions of Senior VP Special Projects and Technical Services. In his role as Senior VP of Barrick, Mr McMullen was instrumental in the development of many mines including gold strike, Veladero, Lagunas Norte, Cowal and Bulyanhulu. His experience includes all aspects of development including feasibility, construction, commissioning, ramp-up and operational optimization.

22.7 Financial results

This Section summarizes the financial results for the preferred 9.5 Mtpa option and then compares this to the results for the other two options studied in this PFS, 7.0 and 4.5 Mtpa.

The Ore Reserve on which all these estimates are based is:

Ore Reserve September 2018 (0.5 g/t Au cut-off grade)
Probable Ore Reserve 129.6 Mt at 1.14 g/t Au for 4.76 Moz within Life of Mine Pit

This was derived from the following Mineral Resource:

Mineral Resource March 2018 (0.5 g/t Au cut-off grade)
Indicated Mineral Resource 180 Mt at 1.1 g/t Au for 6.5 Moz at 0.5 g/t cut off

This is in accordance with the guidelines of Reasonable Prospects for Eventual Economic Extraction ("RPEEE") per the Canadian Institute of Mining, Metallurgy and Petroleum "CIM Definition Standards for Mineral Resources and Mineral Reserves" (CIM, 2014) and the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC, 2012).

For the preferred 9.5 Mtpa option, the Post-Tax NPV at a 5% discount rate over the estimated mine life is US$586 million. The Post-Tax IRR is 38.0%. The Post-Tax payback of the initial capital investment is estimated to occur 1.8 years after the start of production.

The LOM all-in sustaining cost ("AISC") per ounce is US$769. Key economic and operating statistics for the 9.5 Mtpa option are provided in Table 145.

Table 145: Financial summary of 9.5 Mtpa option (gold price of US$1,250/oz)

Key Economic Results		Unit	9.5 Mtpa
Development Capital Cost		US$ M	414
All in Sustaining Costs (AISC)[1]	Starter Pit	US$/oz	599
	Life of Mine		769
Total Project Payback		Years	1.8
Pre-Tax NPV US$ (at 5% discount)[2]		US$ M	927
Post-Tax NPV US$ (at 5% discount)[2]		US$ M	586
Pre-Tax IRR		%	49
Post-Tax IRR		%	38

Notes: [1] Cash Costs + Royalties + Levies + Life of Mine Sustaining Capital Costs (World Gold Council Standard)
[2] Royalties calculated at flat rate of 5%, corporate tax rate of 35% (both subject to negotiation).

The key estimated results achieved by the Starter Pit (Phase 1) are presented in Table 146. The Starter Pit includes the first 2.5 years of operation (24 Mt at 1.31 g/t for 1.06 Moz at 0.5 g/t cut off).

Table 146: Key estimated production results for the 9.5 Mtpa option for the Starter Pit (Phase 1)

Key Estimated Production Results	Unit	9.5 Mtpa
Gold Price	US$/oz	1,250
Gold Produced (Average for full production years)	koz/year	361
All in Sustaining Costs (AISC)[1]	US$/oz	599
Gold Head Grade (Starter Pit)	g/t Au	1.31
Ore Reserve Mined (0.5 g/t Au cut-off grade)	Mt	24
Gold Recovery	%	86
Waste Mined	Mt	12
Strip Ratio (Starter Pit)	W:O	0.5:1
Starter Pit Life (Inc. Ramp up)	Years	2.5
Total Project Payback	Years	1.8

Notes: [1] Cash Costs + Royalties + Levies + Life of Mine Sustaining Capital Costs (World Gold Council Standard)

Cash flows for the LOM Pit are provided in Table 147.

Table 147: The 9.5 Mtpa option production and cash flow on annualized basis

	Unit	LOM Total	Year															
			-2	-1	0	1	2	3	4	5	6	7	8	9	10	11	12	13
PRODUCTION																		
Gold Produced	**koz**	**3,975**	-	-	420	343	279	268	262	293	291	298	299	302	264	331	248	76
Head Grade	**g/t**		-	-	1.62	1.22	1.05	1.05	1.05	1.12	1.13	1.16	1.14	1.17	1.04	1.29	0.97	0.80
Recovery	**%**		-	-	85.9%	85.9%	84.1%	83.6%	82.6%	83.0%	82.7%	82.9%	82.8%	82.9%	82.3%	83.4%	82.3%	80.7%
REVENUE																		
Gold Price	**US$/oz**		-	-	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250
Revenue	**US$ M**	**4,957**	-	-	523,944	427,969	347,953	334,920	327,032	364,896	362,901	371,792	372,929	377,174	328,762	412,575	308,975	95,410
OPERATING COSTS																		
Mining Cost	**US$ M**	**1,024,694**	-	25,463	59,348	66,277	71,207	60,204	62,830	101,136	103,282	108,061	113,501	91,834	75,554	57,080	23,385	5,528
Process & GA Cost	**US$ M**	**1,481,002**	-	-	107,470	112,337	112,113	108,701	109,415	112,025	111,266	111,014	112,749	111,290	110,296	109,809	110,213	42,303
Owners Cost	**US$ M**	**146,350**	-	-	10,594	11,085	11,078	10,765	10,805	11,104	11,021	10,949	11,173	11,018	10,874	10,823	10,891	4,170
Royalties	**US$ M**	**247,862**	-	-	26,197	21,398	17,398	16,746	16,352	18,245	18,145	18,590	18,646	18,859	16,438	20,629	15,449	4,771
CAPITAL COSTS																		
Development	**US$ M**	**414,018**	82,804	248,411	82,804	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining	**US$ M**	**146,017**	-	-	3,120	8,320	2,427	8,320	11,992	9,707	9,707	11,094	14,766	13,867	13,867	19,414	19,414	3,120
Reclamation	**US$ M**	**23,789**	-	-	-	-	-	-	-	-	-	-	-	-		7,930	7,930	7,930
CASH FLOW																		
Net Cash Flow, Pre-Tax	**US$ M**	**1,469**	-83	-261	245	218	125	134	119	115	109	114	106	126	99	190	118	0
Net Cash Flow, Post-Tax	**US$ M**	**957**	-83	-261	185	149	108	117	106	82	64	69	62	86	65	142	75	(5)

The key estimated results achieved over the life of the mine LOM Pit (Phase 4) are presented in Table 148. This includes the Starter Pit results.

Table 148: Key estimated production results for the 9.5 Mtpa option for the LOM Pit (Phase 4)

Key Estimated Production Results	Unit	9.5 Mtpa
Gold Price	US$/oz	1,250
Gold Produced (Average for full production years)	(koz/year)	294
Gold Produced (Life of Mine)	(koz)	3,975
Gold Head Grade (Life of Mine)	g/t Au	1.14
Gold Recovery (Life of Mine)	%	84
Ore Reserve Mined (0.5 g/t Au cut-off grade)	Tonnes (Mt)	129.6
Waste Mined	Tonnes (Mt)	181
Strip Ratio (Life of Mine)	W:O	1.4:1
Mine Life (Inc ramp-up and mine closure)	years	14
Development Capital Cost (Including owner's cost and 15% contingencies)	US$ M	414
Total Project Payback	Years	1.8

22.8 Sensitivity analysis

An analysis of the NPV after taxes was performed for the 9.5 Mtpa option to examine the sensitivity to gold price at varying Post-Tax discount rates. The results of the sensitivity analysis are shown in Table 149 for the Post-Tax scenario.

In the Pre-Tax and Post-Tax evaluations, the Project is most sensitive to changes in gold price, process recovery and head grade, less sensitive to changes in operating costs, and least sensitive to capital cost. As expected, the gold grade in the sensitivity graphs (Figure 137 to Figure 140) mirror the impact of changes in the gold price.

Based upon Life of Mine production and cost parameters, the Post-Tax NPV sensitivities are shown in Table 149 for the 9.5 Mtpa option.

Table 149: 9.5 Mtpa option Net Present Value and gold price sensitivities (Post-Tax) for the LOM Pit

Post-Tax Real Discount Rate (%)	Gold Price (US$/oz)				
	US$1,150	US$1,200	US$1,250	US$1,300	US$1,350
0	682	805	**928**	1,051	1,174
5	415	501	**586**	672	758
10	251	314	**376**	439	501

Notes: All NPVs are Post-Tax values shown in US$ M

The following four bar charts illustrate the 9.5 Mtpa option Pre-Tax and Post-Tax economic sensitivities at a gold price of US$1,250/oz.

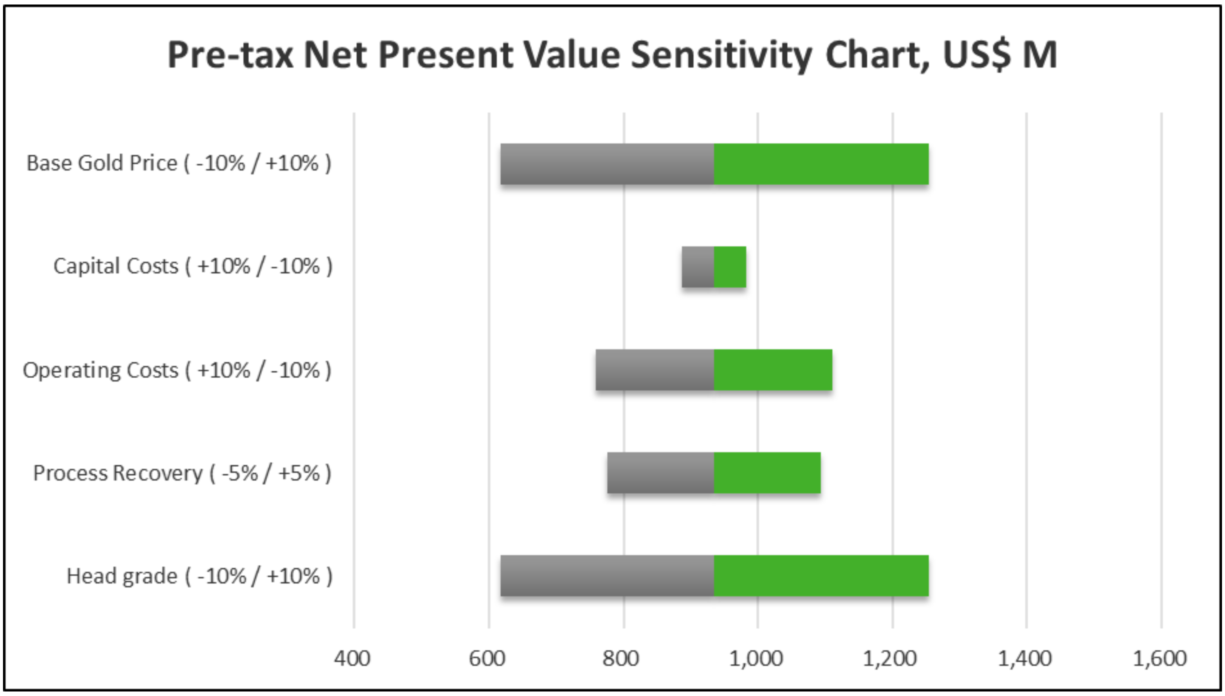


Figure 137: 9.5 Mtpa option – Pre-Tax NPV sensitivity at 5% discount (US$ M)

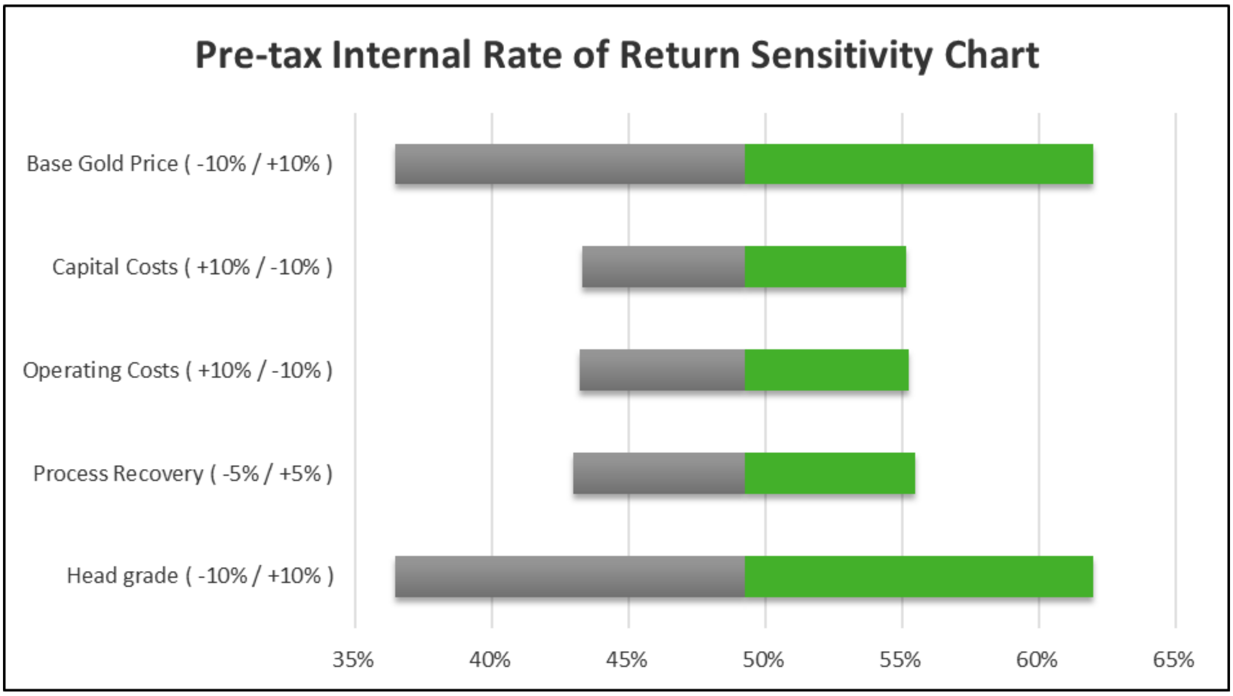


Figure 138: 9.5 Mtpa option – Pre-Tax Internal Rate of Return (%)

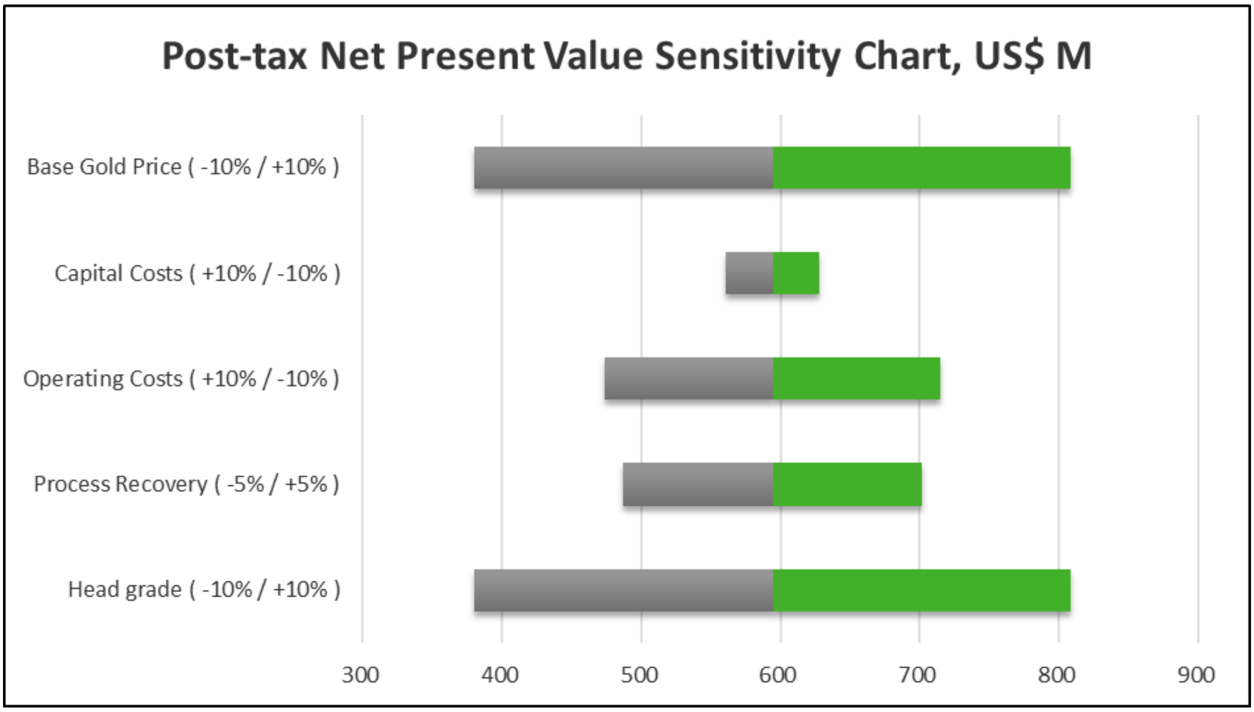


Figure 139 9.5 Mtpa Option – Post-Tax NPV Sensitivity at 5% discount (US$ M)

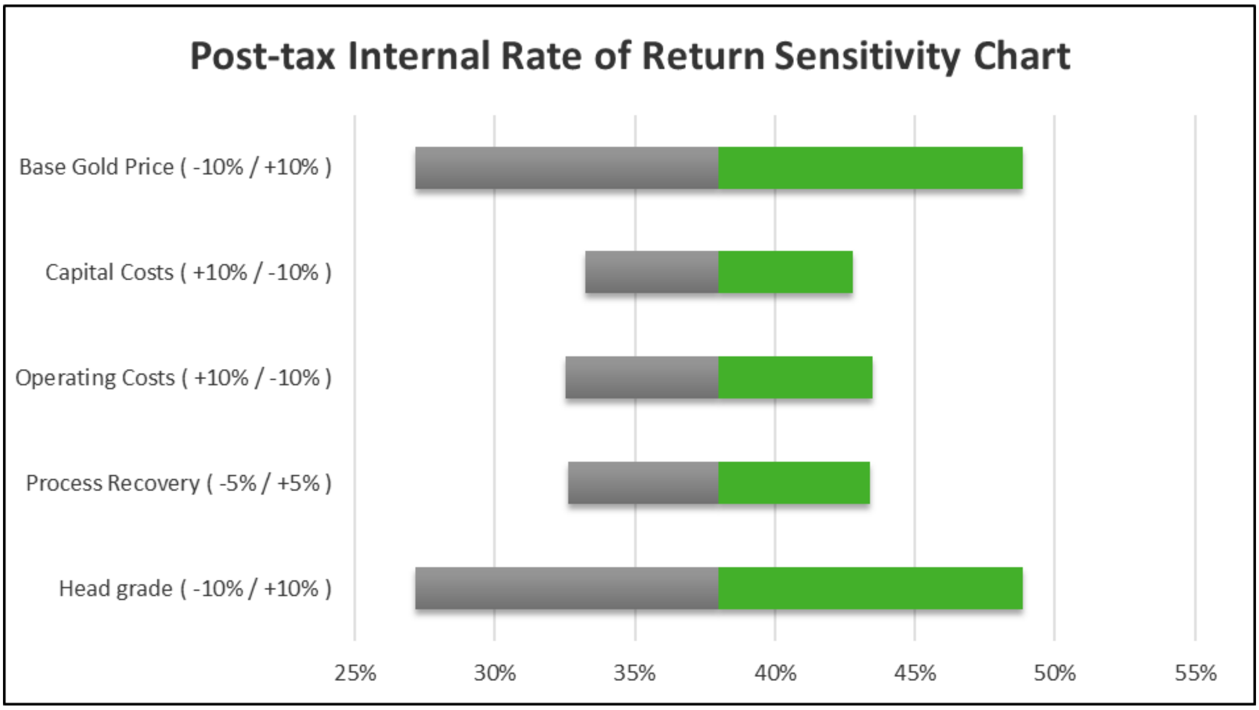


Figure 140 9.5 Mtpa option Post-Tax Internal Rate of Return (%)

22.9 Economic comparison for 9.5, 7.0 and 4.5 Mtpa options

The same basis of financial analysis used for the 9.5 Mtpa option was applied to the 7.0 and 4.5 Mtpa options presented in this Section.

Key economic statistics for the comparison of the 9.5 Mtpa, 7.0 Mtpa and 4.5 Mtpa option are included in Table 150.

Table 150: Key economic results for 9.5, 7.0 and 4.5 Mtpa options

Key Economic Results		Unit	9.5 Mtpa	7.0 Mtpa	4.5 Mtpa
Development Capital Cost		US$ M	414	348	300
All in Sustaining Costs (AISC)[1]	Starter Pit	US$/oz	599	652	708
	Life of Mine		769	823	895
Total Project Payback		Years	1.8	2.5	2.8
Pre-Tax NPV US$ (at 5% discount)[2]		US$ M	927	759	514
Post-Tax NPV US$ (at 5% discount)[2]		US$ M	586	478	317
Pre-Tax IRR		%	49	42	31
Post-Tax IRR		%	38	32	24
Pre-Tax Free Cash Flow		US$ M	1,469	1,324	1,044
Post-Tax Free Cash Flow		US$ M	945	849	667

Notes: [1] Cash Costs + Royalties + Levies + Life of Mine Sustaining Capital Costs (World Gold Council Standard)

[2] Royalties calculated at flat rate of 5%, corporate tax rate of 35% (both subject to negotiation).

The Starter Pit key estimated production comparison results are presented in Table 151. The Starter Pit includes the first 2.5 years of operation (24 Mt at 1.31 g/t for 1.06 Moz at 0.5 g/t cut off).

Table 151: Starter Pit production summary comparison for the three throughput options

Key Estimated Production Results	Unit	9.5 Mtpa	7.0 Mtpa	4.5 Mtpa
Gold Price	US$/oz	1,250	1,250	1,250
Gold Produced (Average for full production years)	koz/year	361	257	171
All in Sustaining Costs (AISC)[1]	US$/oz	599	652	708
Gold Head Grade (Starter Pit)	g/t Au	1.31	1.31	1.31
Ore Reserve Mined (0.5 g/t Au cut-off grade)	Mt	24.0	24.0	24.0
Gold Recovery (Starter Pit)	%	86	86	86
Waste Mined (Starter Pit)	Mt	12.0	12.0	12.0
Strip Ratio (Starter Pit)	W:O	0.5:1	0.5:1	0.5:1
Starter Pit Life (Inc. Ramp up)	Years	2.5	3	5
Total Project Payback	Years	1.8	2.5	2.8

Notes: [1] Cash Costs + Royalties + Levies + Life of Mine Sustaining Capital Costs (World Gold Council Standard)

The LOM key estimated production comparison results for the three throughput options are presented in Table 152.

Table 152: LOM Pit production summary comparison for the three throughput options

Key estimated production results	Unit	9.5 Mtpa LOM	7.0 Mtpa LOM	4.5 Mtpa LOM
Gold Price	US$/oz	1,250	1,250	1,250
Gold Produced (Average for full production years)	koz/year	294	216	140
Gold Processed (Life of Mine)	koz	3,975	3,975	3,975
Gold Head Grade (Life of Mine)	g/t Au	1.14	1.14	1.14

Key estimated production results	Unit	9.5 Mtpa LOM	7.0 Mtpa LOM	4.5 Mtpa LOM
Gold Recovery (Life of Mine)	%	84	84	84
Ore Reserve Mined (0.5 g/t Au cut-off grade)	Mt	129.6	129.6	129.6
Waste Mined	Mt	181	181	181
Strip Ratio (Life of Mine)	W:O	1.4:1	1.4:1	1.4:1
Mine Life (Including ramp-up and mine closure)	Years	14	19	29
Development Capital Cost (Including owner's cost and 15% contingencies)	US$ M	414	348	300
Total Project Payback	Years	1.8	2.5	2.8

23.0 ADJACENT PROPERTIES

The Namdini Gold Project site is located approximately 6 km southeast of the operating Shaanxi underground gold mine.

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Study of higher throughput options

During the PFS, Cardinal evaluated higher process throughput options to assess whether the 6.5 Moz Mineral Resource was able to support higher throughputs. Golder Associates and Intermine Engineering Consultants ("Intermine") were separately tasked to perform an order of magnitude study on the larger throughput options using different Whittle optimizations and mining Phases (pushbacks or stages). Cardinal provided order of magnitude capital and operating costs for the larger throughput options to Golder and Intermine.

24.1.1 Inputs

Optimization parameters applied included:

- Options were completed for mill throughputs of 4.5, 7.0, 9.5, 12.0 and 14.0 Mtpa.
- The 12.0 and 14.0 Mtpa scenarios were divided into a Diesel and High Voltage option. This was done to evaluate potential HV power supply limitations.
- Mining costs for ore and waste based on 10 metre incremental unit rates provided by local contractors for four pit stages designed by Golder were used.
- Overall slope angles of 45° were used.
- Process operating costs and capital costs for the study were factored for the larger options based on the 4.5, 7.0 and 9.5 Mtpa throughput costs that were available at the time.
- A discount factor (DCF) of 10% was used for economic comparison using the industry standard Whittle pit optimization software.

24.1.2 Methodology

The methodology of the Whittle optimization was as follows:

- Whittle shells were selected based on the weighted average of the best and worst discounted cashflow. The weighting was 60% to the best and 40% to the worst.
- Whittle NPV Pushback software was then utilized to evaluate the final Whittle shells to determine the number of stages that would return the best NPV for each of the mill throughput scenarios.
- A minimum mining width of 60 m was used.
- All material was processed if cashflow positive. No high grading was applied.
- The final pit shell was allowed to expand if necessary.
- Vertical advance rate was limited to 80 m per year.
- No limit on total movement was applied.
- Stockpiles were used to store Oxide material mined in the initial years, until Fresh ore supply to the mill diminishes. Processing of Oxide would then commence, to use the available mill capacity and sustain optimum metallurgical response.

24.1.3 Results

The results from the evaluation are shown by NPV and IRR comparison graphs (Figure 141).

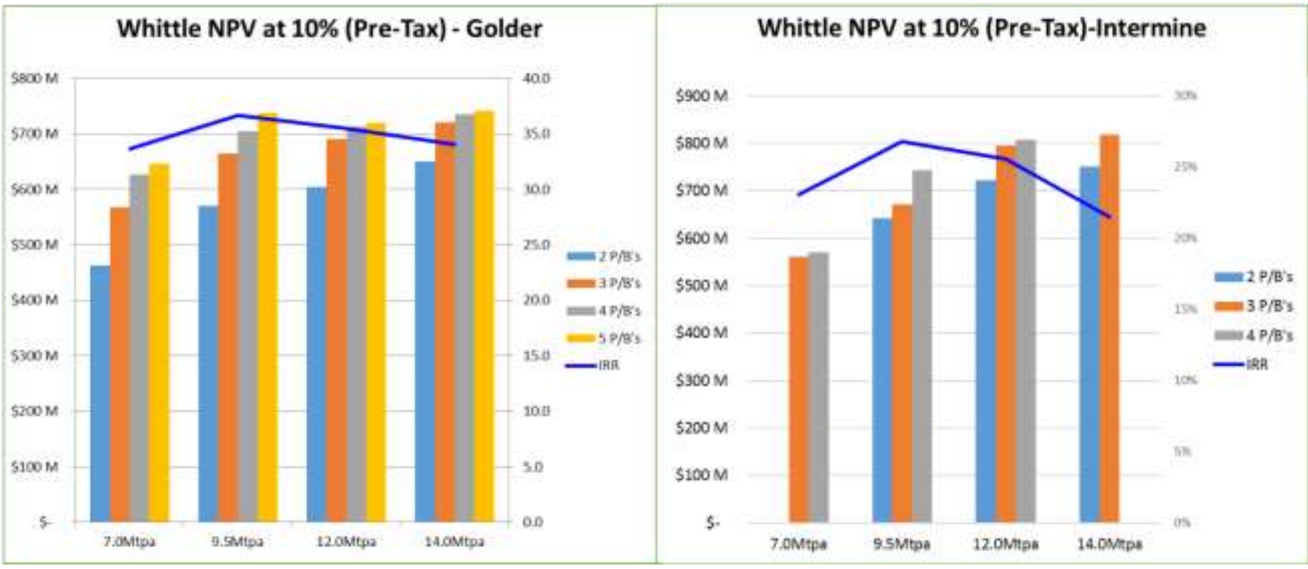


Figure 141: Whittle Optimization NPV and IRR comparisons

24.1.4 Conclusions

From the work completed the following was concluded:

- The best economic scenario was the 9.5 Mtpa option using 4 pit stages of development. Following this was a 7.0 Mtpa option with 4 stages.
- The 12.0 and 14.0 Mtpa throughput options showed practical mining extraction limitations at depth due to the number of working faces required and mining fleet numbers.
- Power supply for the 12.0 and 14.0 Mtpa larger options would require separate studies to assess whether the available grid could supply increased demand. Water supply would also need detailed assessment for these options.
- There are some limitations as to the ability of Whittle to produce realistic nested stage shells and associated mining and production schedules in some cases. These include:
  - A global vertical advance rate limit of 80 m has been applied on the basis of eight 10 m benches in a year. This has affected the potential in some of the scenarios to achieve the required mill throughput in the first year. For example, in the case of using either 3 or 4 stages in the 9.5 Mtpa option, in Year 1 only 4.5 Mt was processed using 3 stages compared to 9.2 Mt for 4 stages. This is because by using 3 stages, the starting top bench of shell 15 is 240 m RL and it can only be mined down to the 190 m RL whereas for 4 stages using shell 14 the starting bench is 230 m RL and goes down to the 180 m RL, hence accessing another 3.1 Mt of fresh mill feed.
  - Positioning of ramps in the practical design process can result in changes to the physicals when compared to those in the staged schedules.
  - The economic evaluation of mill throughput and the number of stages could be combined with variable high grading scenarios. There is potential for higher grade material to be fed to the mill in the earlier years.

25.0 CONCLUSIONS

The positive results of this PFS of the Namdini Gold Project have confirmed that progression of the Project to the Feasibility Study (FS) stage is warranted.

This PFS is preliminary in nature and there is no certainty that the conclusions of this PFS will be realized.

Inferred Resources are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as Ore Reserves. Inferred Mineral Resources have not been used in determining the potential viability of the Namdini Gold Project.

The orebody presents limited opportunity for mining selectivity of higher-grade zones outside the targeted Starting Pit area.

25.1 Mineral Resource estimation

Further updates to the Mineral Resource are planned to support the FS. The updated geological model will form the basis of further pit optimization studies and subsequent pit/stage designs.

An update to the Mineral Resource model for the FS will:

- Identify any additional resources that can be targeted as extensions to the ore body
- Convert Inferred Resources to Indicated by increasing confidence in the identified mineralization
- Define Measured Resources in the first two years of mining
- Update the resource model with additional data for S and As.

25.2 Mine planning and Ore Reserve

The preliminary mining schedule was based on the requirement to identify a range of preferred possible processing throughputs to maximize value: a Low Case (4.5 Mtpa), Mid Case (7.0 Mtpa) and High Case (9.5 Mtpa) of plant feed from the Namdini pit.

The mining plan has scheduled 130 Mt of ore containing 4.760 Moz Au, and 181 Mt of waste. The Low Case option gives an anticipated mine life of some 28 years, the Mid Case option gives 18 years and the High Case option gives 13 years.

25.3 Process plant design

The Return On Capital Employed, is a simple metric used for comparing risk to the scale of the project and suggests a maximum Project value for a process plant throughput size around 9.5 Mtpa. The economic assessment was carried out pre-tax and has assumed a total allowance for royalties of 5% of the gross gold price.

25.4 Other aspects

Cardinal’s proposed work programs for updating Mineral Resources and further trial grade control drilling are appropriate. Additional resource drilling and sampling requirements should be assessed after evaluation of results from the planned drilling.

26.0 RECOMMENDATIONS

The PFS has highlighted the following specific items of focus for the next stage of Project development (the Feasibility Study). These are essentially trade-off studies and optimization studies to clarify options going forward:

26.1 Mineral Resource estimation

Cardinal's planned future work is intended to support development of the FS. Critical aspects will be to infill RC and diamond drilling with the goal of converting Indicated Mineral Resources to Measured Mineral Resources, and further trial grade control drilling to drill out the first year of production, including deeper drilling to intersect the Fresh material below 20 m.

Additional work planned by Cardinal for the PFS includes the following:

- Further exploration drilling targeting potential resource extensions to the north and south
- Further sterilization drilling of potential sites for the process plant, tails disposal sites and waste disposal sites.

Specific recommendations regarding the planned drilling and sampling includes:

- Investigate anomalous inter-laboratory repeats.
- Further investigate primary assay accuracy, such as inter-laboratory repeats, with comprehensive QAQC monitoring.
- Investigate the detailed reliability of Cardinal's density measurements by a program of repeat measurements of representative intervals for all mineralization styles.
- Comprehensively down-hole survey future infill resource drilling by an appropriately accurate method, such as gyroscopic downhole surveying.
- Survey the collars of infill resource drill holes by an appropriately accurate method.

The collars of some holes should be surveyed by an appropriately accurate method if possible Any resource hole collars that cannot be located should have their GPS elevations adjusted to the LIDAR triangulation.

Routine submission of appropriate coarse blank material consistent with the protocols adopted for recent drilling should be continued for future drilling.

Sensitivity of the mineable part of the Mineral Resource to local variations in the S and As should be considered at the Feasibility Study stage, by more formal modelling of this data and reporting of the results. A more accurate model with S and As grades will also be useful for understanding how these elements may present in the waste dumps and how to best manage this to prevent acid formation.

Golder recommended that the impact of using separate lithology domains to control the grade estimation should be considered for future estimations.

26.2 Mine planning and Ore Reserve

- The PFS mine design is conservative. The FS pit design will be based on better rock mechanics, geotechnical and hydrogeological analysis allowing more aggressive slope design.
- The mine schedule can be refined and optimized for ore blending strategies to enhance the comminution characteristics of the ROM ore and reduce costs.

- Detailed assessment of the drilling and blasting characteristics of the waste and ore material will be required as part of the mining cost estimate for FS inputs.
- Detailed hydrogeological and geotechnical analysis will be required for the area encompassed by the final pit design.

The requirement to maintain the pits in a de-watered condition during mining is critical for both safety and operational success.

26.3 Process plant design

- As the High Case throughput option is the most favourable, a single process plant design should be developed targeting a throughput of 9.5 Mtpa.
- The proposed sites for the process plant, TSFs and waste dumps will be tested to confirm their suitability for permanent infrastructure by programmes of geotechnical and sterilization drilling.

26.4 Other aspects

Mitigation measures for key environmental impacts will be proposed in the EIS Report based on empirical studies.

Measures will be instituted to mitigate negative OHS issues.

Measures will be adopted to ensure that the welfare of the public and host communities is not undermined by the operations of the mine.

Most of the studies detailed in this PFS Report will be further progressed in the Feasibility Study. The budget for this program is US$8.4M (Table 153).

Table 153: Feasibility Study budget for Namdini Gold Project

Item	Cost (US$ k)
Feasibility Study Value Engineering	**101**
Feasibility Study	**1,408**
Detailed design and long lead equipment Procurement	**5,732**
Namdini drilling	**1,121**
Namdini geophysics	**13**
Total	**8,375**

The preliminary Project schedule is shown in Table 154 and is subject to available funding, positive outcomes for the PFS and FS and favorable timelines for permitting.

Table 154: Preliminary Project schedule

Milestone	Target timeline
Completion of PFS	Q3 2018
Completion of FS	Q3 2019
Final investment decision	Q4 2019
Target production commencement	H2 2021

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ABBREVIATIONS AND ACRONYMS

AACE	Association for the Advancement of Cost Engineering
AARL	gold elution circuit (developed by AngloAmerican Research Laboratory)
AC	acid consuming
AEP	annual exceedance probability (for rainfall events)
AIF	Annual Information Form (company report to the TSX)
AMSL	above mean sea level
ARI	average recurrence interval (for rainfall events)
ASX	ASX Group Ltd (the Australian Securities Exchange)
BBWI	Bond ball mill work index
BFA	bench face angle (for open pit design)
BGL	below ground level
BOQ	bill of quantities
BQR	Bulk Quantity Request (for engineering quotation)
C.Eng	Chartered Mining Engineer
Capex	Capital expenditure
Cardinal	Cardinal Resources Limited
CCL	compacted clay liner
CIL	carbon-in-leach
CMS	Cardinal Mining Services Limited
CN	curve numbers (for hydrological stochastic simulation modelling)
CNM	Cardinal Namdini Mining Limited (“Cardinal Namdini” or “CNM”), a wholly owned subsidiary of Cardinal
CNWAD	weak acid dissociable cyanide
COG	cut-off grade (in g/t Au)
COx	carbon oxides
CSM	cerebrospinal meningitis bacterial disease
CSV	comma separated value computer format
CV	coefficient of variation
DAVg	discounted average NPV (used in Whittle 4X)
DGPS	differential GPS
EIA	Environmental (and Social) Impact Assessment
EIS	Environmental Impact Statement (a comprehensive public report)
EPA	Ghana Environmental Protection Agency
EPCM	Engineering Procurement Construction & Management
Eur.Ing.FEANI	Registered European Mining Engineer
FEL	front-end loader
FS	Feasibility Study
G&A	General and Administration (cost category)
Ga	Billion years (giga annum)
GCST	Gold Coast Selection Trust
Golder Africa	Golder Associates Africa (Pty) Ltd
Golder Ghana	Golder Associates Ghana Limited
Golder	Golder Associates Pty Ltd

GPCP	Global Precipitation Climatology Project
GRG	gravity recoverable gold
HDPE	High-density Polyethylene
HIG	High Intensity Grinding (mill type)
ICMC	International Cyanide Management Code
IMO	Independent Metallurgical Operations Pty Ltd
Intermine	Intermine Engineering Consultants
IRA	inter-ramp angle (for open pit design)
IRR	Internal Rate of Return
LCS	local control station
LOM	Life of Mine
LV	low voltage
Lycopodium	Lycopodium Minerals Pty Ltd
M+I	Measured plus Indicated Mineral Resource categories
MAP	Mean Annual Precipitation
MC	Master Composite metallurgical testwork sample
MCC	motor control center
MIK	Multiple Indicator Kriging
MOX	moderately oxidized weathered rock
MPL	maximum permissible level (e.g. for water quality)
MPR	MPR Geological Consultants Pty Ltd
MSA	mine services area
NAF	non-acid forming
NAG	net acid generating
Namdini	Namdini Gold Project (or the Project)
NAPP	net acid producing potential
NEDCo	Northern Electricity Distribution Company
NEMAS	Nemas Consult Ltd
NI 43-101	National Instrument 43-101
NOx	nitrogen oxides
NPV	Net Present Value
OIT	operator interface terminals
OK	Ordinary Kriging
OMC	optimum moisture content (for hydrology sampling)
OMC	Orway Mineral Consultants (for comminution and milling)
Opex	Operating expenditure
Orefind	Orefind Pty Ltd
OSA	overall slope angle (for open pit design)
P_{80}	product size of 80% passing
PAF	potentially acid forming
PAI	project area of influence
PAR	population at risk
PB	pushback, stage or Phase of development of the open pit mine design
PCD	Pollution Control Dam

PEA	Preliminary Economic Assessment (first stage of study under NI 43-101)
PFS	Pre-feasibility Study (second stage of study under NI 43-101)
PLC	programmable logic controller (process plant control system)
PM_{10}	Particulate Matter less than 10 microns
PPE	personal protective equipment
QP(s)	Qualified Person(s) under NI 43-101
R&R	rostered breaks off-site
RAP	Resettlement Action Plan
RC	reverse circulation (drilling)
RF	revenue factor (for Whittle optimization) RF_{1300} is based on a gold price of US$1,300/oz.
ROCE	return on capital employed
ROM (Pad)	Run Of Mine mill feed (stockpile)
RPA	Roscoe Postle Associates Inc.
RPEEE	reasonable prospect for eventual economic extraction (for Mineral Resource reporting)
SABC	SAG mill followed by closed circuit ball mill and recycle pebble crushing
SAG	semi-autogenous grinding (mill)
SANAS	South African National Accreditation System
SANS	South African National Standards
Savannah	Savannah Mining Ghana Limited
SCADA	supervisory control and data acquisition (process plant control system)
SGL	Suntech Geomet Laboratories in Johannesburg
Shaanxi	Shaanxi Mining Company Limited
SMBS	sodium metabisulfite $Na_2S_2O_5$
SMC	SAG mill comminution (test)
SML	small-scale mining licenses
SOX	strongly oxidized weathered rock
SOx	sulfur oxides
SPI	SAG mill power index
STDs	sexually transmitted diseases
TDRT	tailings and decant return trench
TML	Townend Mineralogy Laboratory, Perth
ToR	terms of reference
TRANS	Transition zone of partially weathered rock below MOX zone
TSF	Tailings Storage Facility
TSP	Total Suspended Particulates (for dust sampling)
TSX	Toronto Stock Exchange
UCS	uniaxial compressive strength

UNITS

Many of these units are combined in measurement, e.g. kWh is thousand-Watt hours

µm	micron (millionth of mm)
bcm	banked cubic metre (volume of *in situ* material)
g	gram
g/t Au	gold grade in grams per tonne
Ga	Billion years (*giga annum*)
h	hour
Ha	hectare
k	thousand (as in koz or kWh)
km	kilometre
km^2	square kilometre
ktpa	thousands of tonnes a year
l or L	litre
l/sec	litres a second (flow rate)
LG	Lerchs-Grossmann (algorithm used in pit optimization)
m	metre
M	million
m^3	cubic metre
m^3/day	cubic metres a day (flow rate)
mAHD	metres (above) average height datum
mg	milligram (thousandth of g)
mm	millimetre
MPa	megapascal (compressive strength)
Mtpa	million tonnes a year
Nm^3	normal cubic metre (unit of gas flow e.g. for oxygen consumption)
pH	measure of acidity
t	tonne
t/m^3	tonnes per cubic metre (for *in situ* dry bulk density)
ton	imperial ton
tph	tonnes per hour
US$	United States dollars
V	volt
v/v	volume for volume
W	watt
w/w	weight for weight (e.g. for slurry density)
y or a	year (a *for annum*)

28.0 QUALIFIED PERSON CERTIFICATES

The Qualified Persons named in this PFS Report are:

Glenn Turnbull (Golder)

Nicolas Johnson (MPR)

Daryl Evans (IMO).

Certificates for these Qualified Persons are provided in the following pages.

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report prepared for Cardinal Resources. ("Cardinal") entitled: "National Instrument 43-101 Namdini Gold Project Preliminary Feasibility Study" signed on 23 October 2018 (the "Technical Report") and effective 25 October 2018.

I, Glenn Turnbull, of Perth, Australia, do hereby certify that:

1) I am employed as a Principal Mining Engineer with Golder Associates Pty Ltd, at Level 3, 1 Havelock Street, West Perth, WA, 6005, Australia.

2) I graduated with a Mining Engineering degree with Honours from Trent Polytechnic, UK in 1984. I have worked as a Mining Engineer for a total of 34 years since my graduation from college.

3) I am a fellow in good standing of the Institute of Materials, Minerals and Mining (29840) and Member of the Australian Institute of Mining and Metallurgy (313380). I have been a registered Chartered Mining Engineer (C.Eng) and a Registered European Mining Engineer (Eur.Ing. FEANI) since 1986. My relevant experience includes over 40 years in open-pit and underground mining, including management positions and over five years consulting for Golder.

4) I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that, by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be an independent qualified person for the purpose of NI 43-101.

5) I am responsible for the preparation of all sections of the technical report except Sections 7 to 12, 13, 14.1 to 14.13, and 17, in the report "National Instrument 43-101 Namdini Gold Project Preliminary Feasibility Study" and dated 25 October 2018.

6) I visited the Project site from 11 to 15 December 2017.

7) I have had prior involvement with the property that is the subject of the Technical Report. The nature of my involvement has been in providing advice on mining methods, pit optimisation and pit selection and in preparation of the Preliminary Economic Assessment dated 5 February 2018.

8) As of the date of this Certificate, to my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.

9) I am independent of the Issuer as defined by Section 1.5 of the Instrument. I have read National Instrument 43-101 and the sections for which I am responsible in this Technical Report have been prepared in compliance with National Instrument 43-101 and Form 43-101F1.

10) I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 23 October 2018 at Perth, Australia.Glenn Turnbull

Glenn Turnbull

MPR Geological Consultants Pty Ltd

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report prepared for Cardinal Resources. (“Cardinal”) entitled: “National Instrument 43-101 Namdini Gold Project Preliminary Feasibility Study” signed on 23rd October 2018 (the “Technical Report”) and effective 25th October, 2018.

I, Nicolas James Johnson, MAIG, do hereby certify that:

1. I am a Consulting Geologist, with the firm of MPR Geological Consultants Pty Ltd, 19/123A Colin Street, West Perth, WA 6005, Australia.
2. I graduated with an Honours degree in Geology (1988) from the Latrobe University, Melbourne, Australia . I have worked as a Geologist for a total of 30 years since my graduation from university and have worked continuously in the Mining Industry for over 30 years.
3. I am a practicing Geologist and a Member of the Australian Institute of Geoscientists.
4. I have read the definition of “qualified person” set out in the National Instrument 43-101 - Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfil the requirements to be an independent qualified person for the purposes of NI 43-101.
5. I am responsible for Sections 7.0, 8.0, 9.0, 10.0, 11.0, 12.0 and 14.1 to 14.13 only in the report “National Instrument 43-101 Namdini Gold Project Preliminary Feasibility Study” and dated 25 October 2018.
6. I visited the Namdini Gold Project, Ghana site between the 11 January and 14 January, 2017.
7. I have had prior involvement with the property that is the subject of the Technical Report. The nature of my involvement has been in providing advice on geology, drilling, sampling, sample preparation, data verification and resource estimation in preparation of the Preliminary Economic Assessment dated 5 February 2018.
8. As of the date of this Certificate, to my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the technical report not misleading. I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
9. I am independent of the issuer as independence is described in Section 1.5 of NI 43-101. I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with that instrument and form.
10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 23rd October, 2018

Nicolas James Johnson, MAIG

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CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report prepared for Cardinal Resources. ("Cardinal") entitled: "National Instrument 43-101 Namdini Gold Project Preliminary Feasibility Study" signed on 23 October 2018 (the "Technical Report") and effective 25 October 2018.

I, Daryl Evans, FAusIMM, do hereby certify that:

1. I am a Consulting Metallurgist with Independent Metallurgical Operations Pty Ltd and am employed at: 88 Thomas St, West Perth Western Australia 6005.
2. I graduated with a BSc degree from Murdoch University, WA in 1985 and have worked continuously in the Mining Industry as a metallurgist for over 30 years. My operations experience includes senior roles as Concentrator Manager within process plants employing gravity pre-concentration, sulphide conventional and column flotation, free-milling and refractory sulphide treatment as well as conventional oxide CIL/CIP leaching and elecrowinning. n addition to direct operational experience, I have also undertaken a range of consulting roles across a wide range of Gold oxide and sulphide projects, including projects located in Africa, Australia and the Pacific Rim.
3. I am a Fellow of the Australasian Institute Mining and Metallurgy; member number 224540.
4. I have read the definition of "qualified person" set out in the National Instrument 43-101 - Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that, by reason of my education, affiliation with a professional association, and past relevant work experience, I fulfil the requirements to be an independent qualified person for the purposes of NI 43-101.
5. I was responsible for preparation of Section 13.0 and Section 17.0 of the technical report titled 'National Instrument 43-101 Namdini Gold Project Pre-feasibility Study', dated 25 October 2018 (the "Technical Report") relating to Namdini Gold deposit, Ghana.
6. I have not visited the Namdini Gold Project, Ghana site.
7. I have not had prior involvement with the property that is the subject of the Technical Report.
8. As of the date of this Certificate, to my knowledge, information and belief, the sections of this Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the technical report not misleading.
9. I am independent of the issuer as independence is described in Section 1.5 of NI 43-101. I have read National Instrument 43-101 and Form 43-101F1 and the Technical Report has been prepared in compliance with that instrument and form.
10. I consent to the filing of the Technical Report with any stock exchange and other regulatory authority and any publication by them for regulatory purposes, including electronic publication in the public company files on their websites accessible by the public, of the Technical Report.

Dated this 23 October 2018

Daryl Evans FAusIMM (224540)

Signature Page

Golder Associates Pty Ltd

Glenn Turnbull
Qualified Person
FIMMM, MAusIMM, Eur. Ing, C.Eng

Effective Date: 25 October 2018

A.B.N. 64 006 107 857

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