Namdini Gold Project Feasibility Study
NI 43-101 Technical Report

Ghana, West Africa



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1.0 SUMMARY

In this Summary Section some Figures and Tables are repeated from the body of this Report. This Section is consistent with the press release to the ASX and TSX on 28 October 2019 by Cardinal (2019d).

1.1 Introduction

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

The main focus of activity is the Namdini Gold Project where a Measured and Indicated Mineral Resource of 182 Mt grading 1.1 g/t Au for 6.5 Moz Au and an Inferred Mineral Resource of 12 Mt grading 1.2 g/t Au for 0.5 Moz Au, reported at a 0.5 g/t Au cut-off grade, have been established as announced to the ASX and TSX on 3 April 2019 (Cardinal, 2019a).

Contained within this, the Ore Reserve estimate forming the basis of the Namdini Gold Project Feasibility Study (the "**2019 Namdini FS**") is 7.4 Mt at 1.31 g/t Au for 0.4 Moz Au of Proved Ore Reserve and 131 Mt at 1.12 g/t Au for 4.7 Moz of Probable Ore Reserve, as previously announced to the ASX and TSX on 3 April 2019 and 18 April 2019 (Cardinal, 2019b).

This NI 43-101 Technical Report presents the result of the Feasibility Study ("**FS**") carried out by Cardinal on the Namdini Gold Project. This FS updates and supersedes the Preliminary Feasibility Study ("**Pre-feasibility Study**" or "**PFS**") released to the TSX and ASX on 18 September 2018.

Figure 1 shows the location of the Namdini Gold Project and the Company's other projects in Ghana.

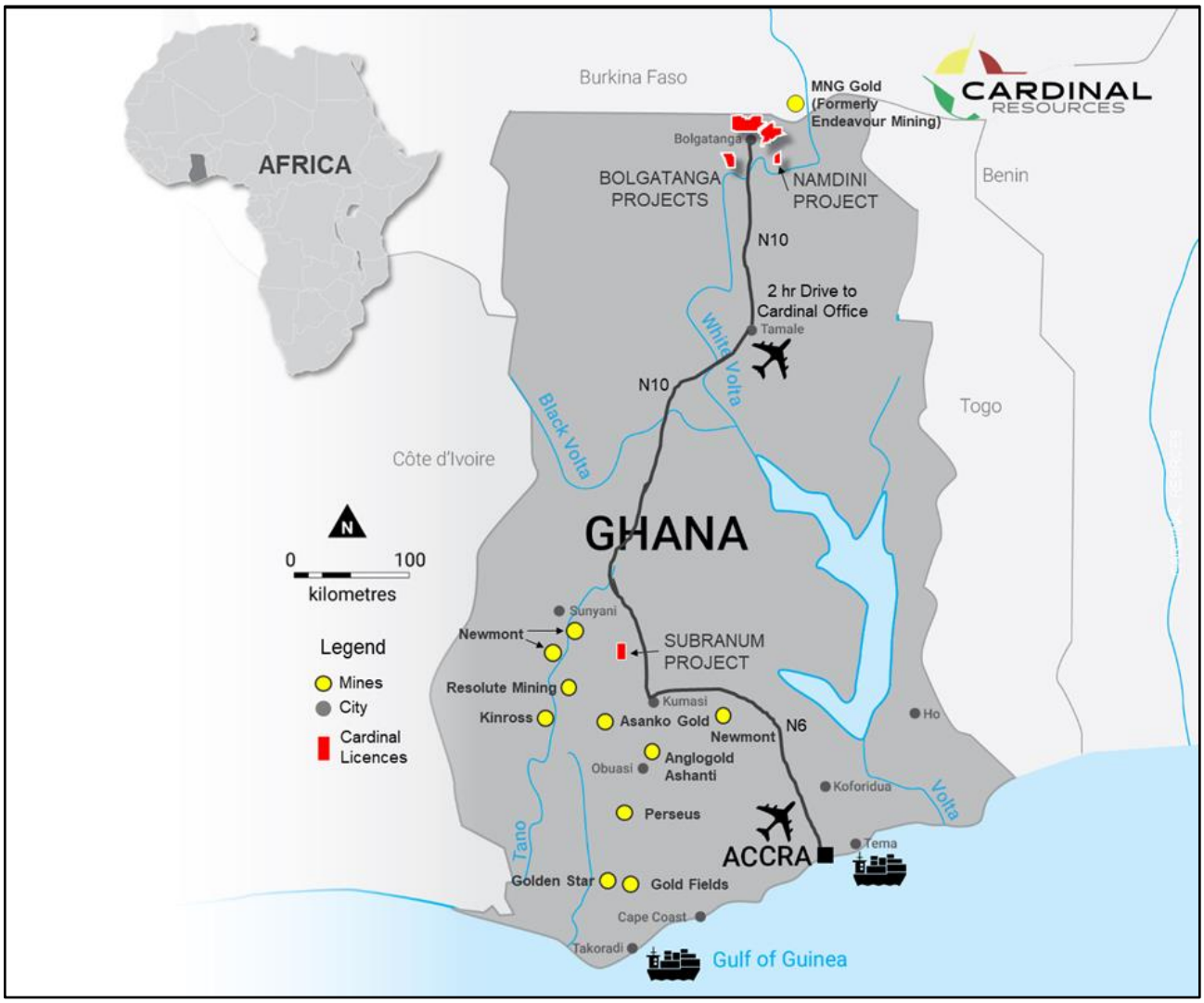


Figure 1: Map of Cardinal's projects in Ghana (source: Cardinal 2019)

1.2 Feasibility Study material assumptions

The capital cost estimates were completed to an accuracy of +15% to -5% for the planned process plant run of mine ("**ROM**") throughput of 9.5 Mtpa and based on open pit mining of the 3 April 2019 Measured and Indicated Mineral Resource as announced to the ASX and TSX on 3 April 2019 (Cardinal, 2019b).

Metallurgical testwork carried out to date indicates that gold can be satisfactorily recovered from Namdini ore using conventional process routes comprising crushing, grinding, gravity, flotation, regrind and carbon-in-leach (CIL) treatment of the flotation concentrate. The testwork was considered sufficient to determine that the Namdini Mineral Resource represents a deposit with a reasonable prospect of eventual economic extraction ("**RPEEE**"), for which the Modifying Factors and economic parameters have been applied to declare an Ore Reserve. Estimates of capital costs for the process plant and associated infrastructure were prepared by Lycopodium Minerals Pty Ltd (Lycopodium).

The proposed process plant incorporates primary crushing, grinding, gravity, flotation, concentrate regrind and CIL gold extraction. Planned production throughput is 9.5 Mtpa of ROM ore.

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 optimization was selected and used to generate engineering pit designs that included a Starter Pit and Phases for the Life of Mine ("**LOM**") production schedule.

Golder estimated the Ore Reserve in accordance with the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves or 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 the Canadian Institute of Mining, Metallurgy and Petroleum 'CIM Definition Standards for Mineral Resources and Mineral Reserves' (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 Code Table 1 is not required, but it was provided as Appendix 1 in Cardinal (2019d), the ASX/TSX press release dated 28 October 2019.

The Ore Reserve estimate is based on the revised Mineral Resource estimate announced on 3 April 2019 (Cardinal, 2019a). The Ore Reserve was derived from the Mineral Resource after consideration of the level of confidence in the Mineral Resource and application of relevant modifying factors including mining, processing, infrastructure, environmental, legal, social and commercial factors under the guidelines of ASX Listing Rule LR 5.9.1. The Ore Reserve announced to the ASX and TSX on 3 April 2019 and 18 April 2019 (Cardinal, 2019b) represents the economically mineable part of the Measured and Indicated Mineral Resources.

The Proved and Probable Ore Reserve estimate is based on Measured and Indicated Mineral Resources. No Inferred Mineral Resource was included in the Ore Reserve. The Ore Reserve represents the economically mineable part of the Measured plus Indicated Mineral Resources. 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 used in the gold industry. The financial modelling completed as part of the FS demonstrates 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 FS incorporates a number of factors and assumptions as outlined in the various sections within this NI 43-101 Technical Report.

Golder provided an estimate of mining costs, including drill and blast, haulage, rehabilitation, and administration costs. Lycopodium provided process plant design and cost estimates.

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 at varying real discount rates. The financial analysis was used to determine the potential economic return of the Project over the LOM.

The preliminary Project development schedule is shown in Table 1. This is focused on further de-risking of the project and is subject to finalising funding, plus favourable timelines for permitting, contracting, and procurement.

Table 1: Project schedule

Milestone	Target timeline
Commence Front End Engineering and Design ("**FEED**")	Q4 2019
Advance Relocation Action Plan to completion	H1 2020
Target production commencement	H2 2022

The gold price assumptions used for this FS and the financial analysis are described in Table 2.

Table 2: Gold price assumptions

Selection case	Gold price
Trial open pit optimizations were run in Whittle 4X at a US$1,300/oz gold price to define the base of potentially economic material. Five cut-back pits (Phases) were then selected and full mine designs applied.	US$1,300/oz
Mine scheduling was used to maximise 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 shell of approximately $1,235/oz using estimated LOM input prices and costs. Pit shells were converted into engineering designs.	US$1,235/oz
The financial model input gold price was based on a long-term average of US$1,350/oz.	US$1,350/oz

The FS has been evaluated at a gold price of US$1,350 per ounce. The average US$ gold price per ounce for the period from October 2017 to October 2019 is US$1,387 per ounce. During this period the US dollar gold price has been in steady uptrend from a low of US$1,177 in August 2018 to a high of US$1,550 per ounce traded during September 2019.

1.3 Study Team

Cardinal commenced its FS in January 2019 to further advance the Namdini Gold Project. The resources utilized included an Owner's Team, continued engagement with previous consultants and the appointment of new consultants to assist with phased development of the Namdini Gold Project.

Study Manager Lycopodium managed the full integration of all project disciplines and coordinated the efforts of Cardinal's other study partners.

Lycopodium is a highly respected mining services company with over 25 years' global experience in the minerals industry, designing and building large-scale mines, processing plants and associated infrastructure, particularly in Africa and West Africa. The independent specialist consultants and their roles are shown in Table 3.

Table 3: The Namdini FS independent specialist consultants – the Study Team

Company	Role
Lycopodium Minerals Pty Ltd ("**Lycopodium**")	Feasibility Study manager. Process plant and associated infrastructure. Capital cost and process operating cost estimates.
Golder Associates Pty Ltd ("**Golder**")	Mine design, planning, optimization and scheduling. Mine operational costs. Compilation of the JORC, NI 43-101 and Feasibility Study reports.
Golder Associates Africa (Pty) Ltd	Hydrology and hydrogeological engineering studies.
Golder Associates Ghana Limited	Geotechnical engineering studies.
Knight Piésold Consulting Pty Ltd ("**Knight Piésold**")	Tailings Storage Facility and selected infrastructure design.
ALS Laboratory (Perth) ("**ALS**")	Metallurgical testwork to support the process design criteria.
Orway Mineral Consultants Pty Ltd ("**OMC**")	Comminution data analysis, crushing and grinding option studies.
Independent Metallurgical Operations Pty Ltd ("**IMO**")	Metallurgical testwork management, analysis and process flowsheet development.
Maelgwyn Mineral Services Africa Pty Ltd ("**MMSA**")	Aachen™ process metallurgical optimization.
MPR Geological Consultants Pty Ltd ("**MPR**")	Mineral Resource modelling of the Namdini Deposit.
Orefind Pty Ltd ("**Orefind**")	Geology and deposit structural genesis.
NEMAS Consult Pty Ltd ("**NEMAS**")	Environmental Impact Assessment Study.
Geosystems Consulting Limited ("**Geosystems**")	TSF Environmental Impact Assessment Study.
MKM Social Limited ("**MKM**")	Socio-Economic Study and Resettlement Action Plan.
Sebbag Group International Pty Ltd ("**SGI**")	Mine design review.
Whittle Consulting Pty Ltd ("**WC**")	Enterprise optimization.
BDO Advisory Pty Ltd ("**BDO**")	Financial model integrity and reviewer.

1.4 Mining Licence

During the quarter ended 31 December 2017, Large-Scale Mining Licence LVB14619/09 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 (Figure 2), which covers 19.54 km^2 in the Datoko 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.

1.5 Accessibility

Namdini is located 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 via 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 4 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.

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 approximately 6 km southeast of the operating Shaanxi underground gold mine, which is supplied by grid power.

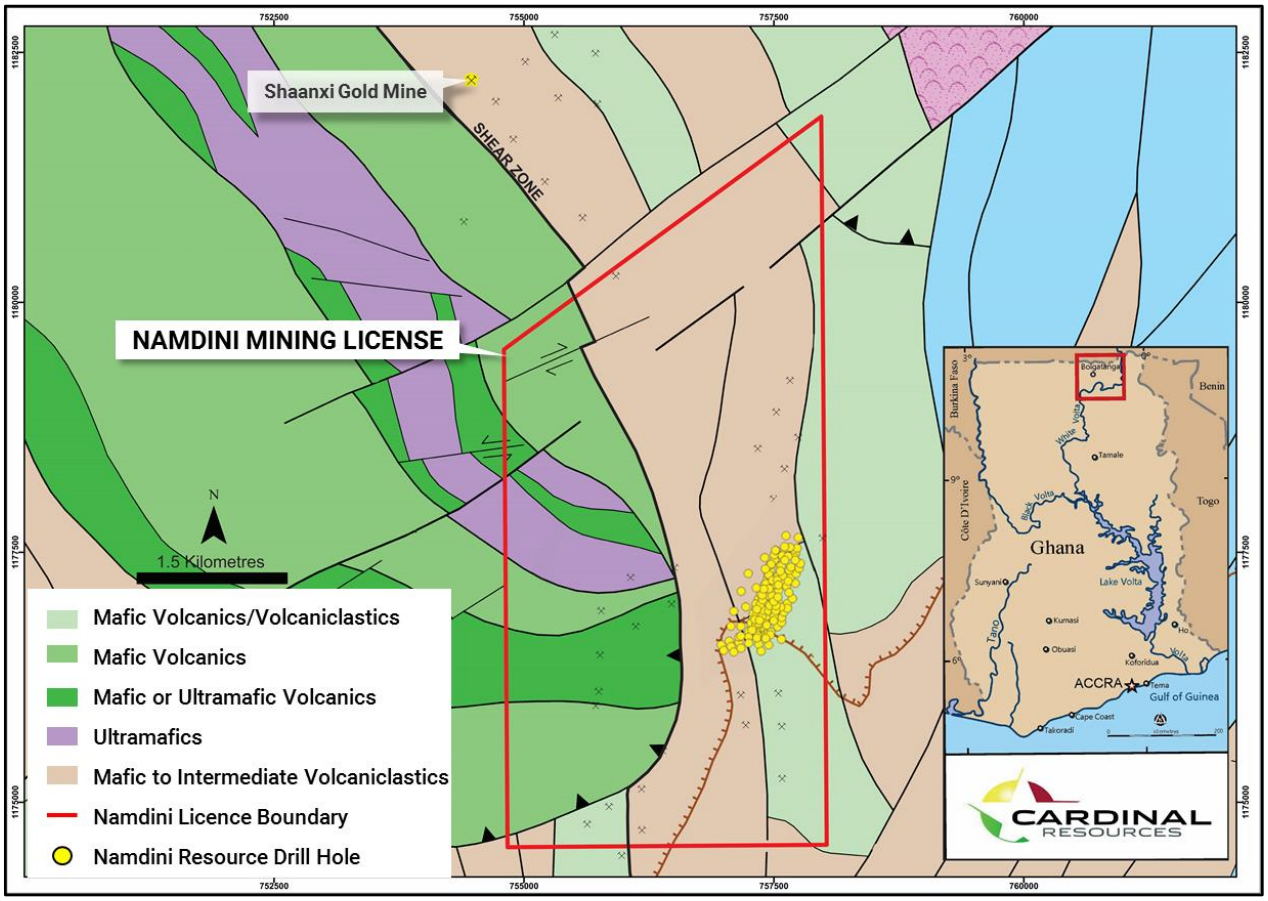


Figure 2: Namdini Gold Project area geology and Mining Licence boundary (source: Cardinal 2019)

1.6 History

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 reported a population of 66,685. There are two small settlements in the vicinity of the Namdini Gold Project that 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 Licence area is 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.

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. An application by Savannah for a Large-Scale Mining Licence over an area of approximately 19.54 km^2 covering the Namdini Gold Project was subsequently granted by the Minister of Lands and Natural Resources of Ghana.

During the December 2017 quarter, the Large-Scale Mining Licence was assigned to Cardinal Namdini by the Minister of Lands and Natural Resources.

1.7 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. The Namdini gold deposit has so far been delineated over a strike length of 1,150 m, up to 300 m wide and 700 m deep.

A geological map showing the gold deposits of north-eastern Ghana is provided as Figure 3.

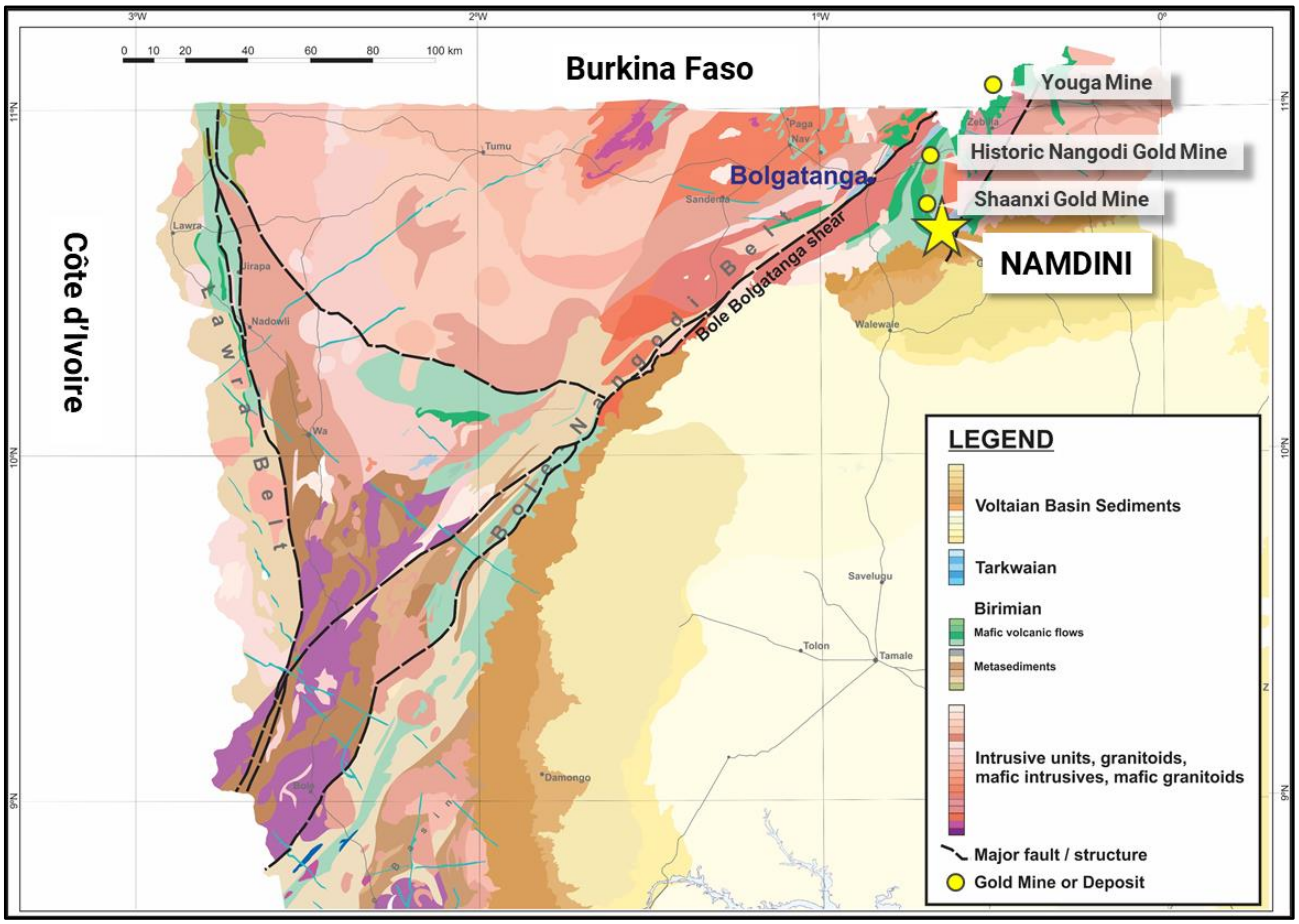


Figure 3: Map of north-eastern Ghana showing geology and gold deposits (source: Cardinal 2019)

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

Orefind concluded that the rock types include a steeply west-dipping Birimian sequence of interbedded, foliated, metasedimentary and metavolcanic units which have been intruded by a medium-grained granitoid and a 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 host 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 granitoid (tonalite) is extensively altered and has been overprinted by silica-sericite-carbonate assemblages.

In all rock types, the gold mineralization is accompanied by visible disseminated sulphides, mainly 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 display yellow-brown sericite-carbonate selvages. Rare visible gold occurs in strongly altered granite and is associated with sub-millimetre wide silica-sericite shears.

A summary of the logged lithologies at Namdini is provided in Table 4.

Table 4: Summary of Namdini geological logging codes and descriptions

Code	Description
LAT	Laterite, ferruginous duricrust developed *in situ*
SPR	Saprock (<20% alteration of weatherable minerals)
SAP	Saprolite (>20% alteration of weatherable minerals)
GRA	Granodiorite or tonalite, altered felsic rocks (sericite, muscovite, carbonate and K-feldspar)
DIO	Intermediate rocks, altered (shearing, silicification, chlorite and sericite) and unaltered diorite, quartz diorite speckled with quartz and feldspar
MVO	Mafic rocks, volcaniclastics, altered (sericite, chlorite + silicification)
MSE	Metasedimentary rocks, dominated by quartz-carbonate veining + hematite and chlorite
LTF	Pyroclastic rocks and tuffs
DAC	Dacite, felsic extrusive rock, intermediate in composition between andesite and rhyolite. Often found associated with lava flows, volcanic equivalent of granodiorite.
BX	Breccia
QTZ	Quartz

1.8 Deposit types

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.

1.9 Exploration

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 regional aeromagnetic surveying, regional and prospect-scale geological mapping and rock chip sampling.

Cardinal commenced reverse circulation ("**RC**") percussion and diamond core 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 was established near the Namdini Gold Project site in Bolgatanga.

A Small-Scale Mining Licence was approved in 2014 and 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 after which the scale and extent of the deposit was established.

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 ongoing exploration target development and assessment activities.

1.10 Drilling and sampling

The Namdini Mineral Resource estimate is based on RC and diamond resource drilling information available for Namdini on the 4 February 2019, totalling 326 holes for 87,140 m of drilling. RC and diamond drilling provide around one quarter and three quarters of the estimation dataset respectively.

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

The RC percussion drilling was either 130 mm (nominally 5¼ inch) or 125 mm in diameter, and drilled to 200 m or less in depth. RC samples were collected over 1 m down-hole intervals using a cyclone, with sub-sampling using a three tier-riffle splitter. Rare un-mineralized intervals were composited over 4 m intervals for analysis.

For drilling prior to approximately April 2016, diamond core was halved for sub-sampling with a diamond saw. From approximately April 2016 to June 2017 core was quartered for assaying. For drilling after June 2017 diamond core was halved for sub-sampling. Sample intervals range from 0.2 to 1.8 m long, with the majority of samples assayed over 1 m intervals.

Collar locations of most resource holes were accurately surveyed by differential GPS techniques. The drilling at Namdini is considered to have been surveyed with sufficient accuracy for Mineral Resource estimation. Most diamond holes and deeper RC holes were down-hole surveyed at generally 30 m intervals. Hole paths have been located with sufficient accuracy for the current estimate.

Geological logging was undertaken by Cardinal's technical staff and captured on field Toughbook laptops using Maxwell Geoservices (Perth) proprietary Logchief™ software. Structural interpretation and modelling were completed by specialist consultants Orefind, resulting in a three-dimensional model of key lithologies, structures and weathering zones.

1.11 Sample preparation and assaying

1.11.1 QAQC

Quality assurance and quality control ("**QAQC**") of the data used for Mineral Resource estimation is described in this Section.

All sample preparation and gold assaying of primary samples from Namdini resource drilling were undertaken by independent commercial laboratories. Analyses by Cardinal were limited to around 87% of the density samples. No employee, officer, director, or associate of Cardinal carried out any sample preparation work on gold-assay samples from the Namdini Gold Project exploration programme.

As part of Cardinal's QAQC programme, a suite of internationally accredited and certified reference materials ('standards') and locally sourced blanks were included in the sample submission sequence. The standards covered gold grade ranges expected at Namdini. Interlaboratory umpire analyses were also conducted. The sampling, sample preparation and analysis processes were found to be appropriate and acceptable for Mineral Resource estimation.

Most primary samples were submitted to SGS Ouagadougou or SGS Tarkwa for analysis by fire-assay with assays from these laboratories contributing around one third and two thirds of the estimation dataset respectively. Samples from several peripheral resource holes analysed by Intertek Tarkwa provide around 0.5% of the estimation dataset.

SGS Tarkwa and Ouagadougou, and Intertek Tarkwa have been 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.

The assay laboratories employed consistent sample preparation and analytical procedures. After sorting and weighing samples they are oven dried and crushed to 75% passing 2 mm. A 1.5 kg riffle split subsample was pulverized to nominally 85% at 75 microns (µm). Samples were fire assayed for gold using a 50 g charge with an AAS finish, with a detection limit of 0.01 g/t. Assays of greater than 100 g/t were re-analysed with a gravimetric finish.

Certified reference materials (blanks and standards) were submitted into the sample stream at a frequency rate of 5% of the samples. Duplicate samples of RC chips were taken at a rate of 5%.

Average assay results of certified reference standards reasonably reflect expected values supporting the general accuracy of assaying. Several sets of inter-laboratory repeat analyses provide additional support for the primary analyses.

Assays of coarse blanks included in assay batches from before May 2015 and after September 2017 show no indication of significant contamination or sample misallocation. Sample batches submitted between May 2015 and September 2017 included fine blanks of pulverized material. These fine blanks showed no significant sample misallocation.

Diamond core and RC samples were transported from the drill site by a Cardinal vehicle to the secure core yard facility at the Bolgatanga Field Exploration Office.

All samples submitted 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.

1.11.2 Bulk density

Resource data acquisition included routine water immersion measurements for bulk density determination of representative diamond core samples.

Density measurements available for Namdini comprise 11,047 immersion measurements performed by either Cardinal (9,652) or SGS Tarkwa or Ouagadougou (1,395) on diamond core. 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.

Bulk densities were assigned to the Mineral Resource estimate by rock type and oxidation zone on the basis of the average of the available measurements. Assigned bulk densities vary from 2.00 t/bcm for strongly weathered Metavolcanics to 2.82 t/bcm for Fresh Diorite and Metasediments.

1.12 Data

Independent verification checks were undertaken to confirm the validity of the database compiled for the Mineral Resource estimate. These checks included internal consistency between and within database tables, comparison of assay values between nearby holes, and comparison of database entries with original source files supplied by Cardinal for collar survey, down-hole surveys and gold assays. The consistency checks showed no significant issues.

1.13 Metallurgy and mineral processing testwork

1.13.1 Introduction

Metallurgical testwork continued to focus on the same flowsheet as presented in Cardinal's PFS (Cardinal, 2018c; Golder, 2018b).

Gold is recovered using crushing, SABC milling, gravity, flotation, concentrate regrind, high-shear oxidative pre-treatment, and CIL cyanidation of the flotation concentrate. The milling circuit comprises a SAG mill, ball mill and pebble crushing ("**SABC**").

The metallurgical testwork on Fresh material was carried out by ALS Laboratory in Perth, Australia and at the Maelgwyn Mineral Services Africa ("**MMSA**") Laboratory in Johannesburg, South Africa.

1.13.2 Results

Key findings from the testwork were:

Mineralogy:

- Native gold is the predominant gold bearing mineral with very low silver content (<2 ppm Ag).
- Pyrite is the dominant sulphide mineral where the majority of the gold is associated within the three main lithologies referred to in this Report as Transition and Fresh material. The main Transition and Fresh ore lithology types are Metavolcanics (MVO), Granite (GRA) and Diorite (DIO). The small portion of Metasediments (MSE) in the ore behaves similarly to MVO metallurgically, therefore its results are deemed to conform to those of MVO.
- A small portion of gold resides in Oxide material at surface accounting for approximately 3% of the total ounces in the Ore Reserve. No detailed mineralogy was necessary on the Oxide material since previous metallurgical testwork established its consistent metallurgical behaviour and high gold recovery.
- The pyrite particle size of 80% passing ("**P_{80}**") ranges 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 stereomicroscope of the unmounted gravity concentrates. These gold grains are approximately 200 µm in diameter and rounded in appearance. Gravity gold recovery is supported based on the testwork.
- Mineralogy and laboratory testwork results do not indicate the presence of preg-robbing carbonaceous matter. Organic carbon varied from 0.03% to 0.09%; leach kinetic response did not indicate preg-robbing behaviour which was verified by a preg-robbing index ("**PRI**") test ranging from 0.03 to 0.06 PRI.

Comminution:

- Comminution design was based on the competencies and behaviour of the Transition and Fresh material only. The Oxide material from previous studies did not affect comminution design outcomes.
- Average Bond Ball Mill Work index ("**BBWi**") values for MVO, GRA and DIO ranged from 12.6 to 18.6 kWh/t with Diorite (DIO) reporting the highest work index. Gyratory crusher design has been based on the 85th percentile values, initially 19.8 kWh/t (Starter Pit) reducing to 19.0 kWh/t (LOM).
- JK Drop Weight calibration was performed on PQ drill core samples, supported by SAG mill comminution ("**SMC**") variability testing applied to HQ samples. Analysis of the distribution of separate lithology results supports Comminution circuit design based initially on an A*b value of 31.3 reducing to 28.8 (LOM).
- The BBWi data indicated a range between 14.7 and 19.7 with an average of 16.9 kWh/t. Ball mill circuit design has been based on 85th percentile values, initially 17.3 kWh/t increasing to 18.9 kWh/t (LOM).
- Bond Abrasion Index ranged between 0.03 and 0.29 with Granite (GRA) reporting the highest value. Circuit design is based on a value of 0.285 with an average (blended) value of 0.139 used for the operating cost estimates.

- Pilot plant testing of the Outotec high intensity grinding mill ("**HIGmill**") to establish grind energy signature plots, proved that a regrind (P_{90}) size of 9 µm was the most economical for the purpose of design.

Flotation:

- Oxide material will bypass the flotation process as it does not contain floatable sulphide material. No flotation testwork on Oxide material was performed.
- Fast flotation kinetics of Transition and Fresh ore was observed with most of the gold recovered in sulphides in less than 8 minutes.
- Gold recovery averaged 95% to concentrate for the majority of composites.
- Gravity recovery improved total gold recovery and mostly eliminated variability in the flotation tails grade.
- Gravity and flotation separation resulted in an average flotation tail grade of 0.08 g/t Au over a total of 72 tests and across all lithologies.
- Flotation mass recovery averaged 7.3% w/w; concentrate grade based on the Starter Pit composite averaged approximately 30 g/t without gravity recovery and approximately 15 g/t Au with prior gravity recovery.

Leach:

- No leach testwork was deemed necessary on the Oxide material during the FS. Previous Oxide testwork results by Gekko dated 28 August 2017 were carried forward into the FS. Leach extractions averaging 90% at a grind size P_{80} of 75 µm were consistently observed.
- Concentrate leach feed grades based on Starter Pit and separate Life of Mine lithology composites ranged between 6 g/t and 37 g/t, reporting an average at 15.6 g/t Au.
- Leach residues based on standard direct cyanidation leach tests without high shear oxidative pre-treatment averaged 0.21 g/t Au based on an average flotation concentrate mass recovery of 7.6% and a concentrate regrind P_{90} size of 9 µm.
- Maelgwyn Aachen™ shear reactor technology was tested at the MMSA laboratory facility in South Africa with encouraging results, supporting improved leach extraction relative to standard direct cyanidation. At a regrind P_{90} size of 9 µm, total extraction increased by 4.0% relative to the Baseline direct cyanidation via 10-Pass Leachox treatment. The testwork results have been incorporated into the circuit design based on 10-Pass treatment; extraction improvement has also been incorporated into recovery values based on a concentrate regrind P_{90} size of 9 µm.
- Prior Leachox treatment also acted to oxidize cyanide-consuming species, indicating potential for reduced leach cyanide consumption and improved operating cost.

1.13.3 Expected metallurgical recovery

Early testwork established that Namdini Oxide ore is free milling with an average recovery of 90% at a grind size P_{80} of 75 µm. Previous testwork has shown that a non-flotation conventional gold process can be applied to all Oxide material (less than 3% of the Ore Reserve in terms of tonnes and contained gold) which will be stockpiled and processed at the end of the mine life. The metallurgical work carried out to date indicates that gold can be satisfactorily recovered from Namdini Transition and Fresh ore feed using conventional flotation, regrind and Carbon In Leach (CIL) cyanidation techniques, and from Oxide ore feed using conventional CIL cyanidation techniques.

All gravity gold, flotation, regrind and leach results were analysed to produce three regression recovery curves, one for each of the key Namdini lithologies. These curves were then applied to the varying head grades of the mine schedule to derive overall recovery.

The metallurgical testwork has also provided variability information characterising the comminution, gravity, flotation and leach response of the main three Namdini lithologies.

Gold mineralization is mainly associated with pyrite in all three main rock types: Metavolcanic, Granite and Diorite. Bulk samples were collected for bench-scale and pilot-scale metallurgical testwork for the key areas of the process plant flowsheet. Sample selection and therefore metallurgical testwork has included high-grade, low-grade, ore body variability and sulphur variability for all ore lithologies.

Metallurgical testwork has proved that the ore is amenable to conventional flotation techniques due to the gold being associated with sulphides. Metallurgical testwork has also confirmed that the ore is amenable to conventional milling and flotation, followed by regrinding, high-shear oxidation and cyanide leaching of the flotation concentrate in a carbon-in-leach process.

The selected processing route for Namdini ore is crush, primary grind, sulphide flotation followed by regrind, high-shear oxidative treatment, and CIL cyanidation of the flotation concentrate.

Aachen™ is a relatively simple, proven process already being used at gold producing mines globally and specifically in Africa. These operations have consistently demonstrated an uplift in gold recovery with Aachen™. The premise of the process is that high shearing of slurry particles and increased oxygen mass transfer to the slurry results in improved kinetics by scouring the particle surfaces and accelerating the oxidation of the sulphide species. This allows for less residence time and less reagent requirements within the leaching circuit**.** In addition to a potential increase in gold recovery (and therefore a potential uplift to annual gold production rates), there are typically power and reagent (Opex) savings and power requirements (Capex) savings that can be realised.

Cardinal has tested 4,447 kg of material from 47 drill holes (2,048 one metre intervals), comprising 7 pilot scale composites across the entire deposit. A further detailed testwork programme was then completed comprising 2,310 kg from 24 drill holes (805 m) as part of an optimization program.

Aachen testwork on Namdini flotation concentrate indicated the following improvements (Cardinal, 2019c):

- Increased gold recovery (+4.0% over Baseline, without Leachox treatment)
- Coarser regrind size compared with the PFS flowsheet for certain lithologies
- Reduced power consumption (i.e. lower operating power requirements and Opex)
- Improved mass transfer of oxygen, leach kinetics and reactivity of reagents
- Reduced reagent consumption leading to reduced operating costs.

Orway Mineral Consultants Pty Ltd ("**OMC**") used the comminution testwork results for comminution circuit selection and mill sizing. A primary crushing and then open circuit SAG mill with recycle pebble crushing followed by closed circuit ball mill/hydro-cyclones ("**SABC**") was selected by OMC using available comminution parameters.

The process design based on the metallurgical testwork is as follows:

- A primary grind size of 80% passing 106 µm was selected for the primary grinding circuit design.
- Incorporation of a gravity concentration circuit given the presence of gravity recoverable gold.

- Flotation testwork indicated fast sulphide flotation kinetics; the circuit comprises six 300 m^3 rougher flotation cells based on kinetic response and recovery.
- The flotation concentrate is reground and subjected to a high-shear reactor and pre-aeration before CIL cyanidation.
- Gold recovery will be via a conventional CIL cyanidation with elution circuit, electrowinning and gold smelting to recover the gold from the loaded carbon to produce doré.
- Industry typical design parameters were assumed where testwork is pending.
- Detailed metallurgical testwork is continuing for the Namdini Gold Project under the direction of Cardinal to support the design as part of the continuous improvement program.
- Gold is recovered using crushing, SABC milling, gravity, flotation, concentrate regrind, high-shear oxidative pre-treatment, and CIL cyanidation of the flotation concentrate. The milling circuit comprises a SAG mill, ball mill and recycle crushing (SABC).

1.13.4 Metallurgical testing summary

Key aspects supporting selection of the process flowsheet include:

- The Oxide results reported in the PFS were retained. The previous Oxide metallurgical testwork was performed on a whole-of-ore-leach recovery testwork regimen, which yielded an average 90% recovery at a grind size P_{80} of 75 µm.
- Average recovery is dependent on head grade and the ratio of the different lithologies.
- Recovery was calculated by adding the gold recovered by gravity, flotation and leaching gold and allowing for gravity gold loss at scale and CIL carbon and solution loss. This is effectively gravity recovery plus flotation recovery multiplied by leach recovery.
- The Transition material at Namdini is approximately 3% of the ore feed. It is only partially weathered (oxidized) and behaves in metallurgical test work similar to the Fresh material, with similar recoveries.
- Metallurgical testwork carried out to date indicates that the Namdini Gold Project can deliver a standard gold recovery process plant design with no unproven technology required.
- The metallurgical process includes 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 that will be exported for refining.

1.13.5 Conclusions – metallurgy

The Transition material at Namdini is only partially weathered (oxidized) and behaves in metallurgical test work similar to the Fresh material, with similar recoveries. The Transition material is to be processed with the Fresh material as part of the mill feed and thus these are combined in all the mine and mill schedules.

The metallurgical work carried out to date indicates that gold can be recovered from Namdini Transition and Fresh ore feed using conventional flotation, regrind and Carbon In Leach (CIL) cyanidation techniques, and from Oxide ore feed using conventional CIL cyanidation techniques. The work is considered sufficient to define a technically and economically viable gold mining project.

1.14 Mineral Resource estimation

Independent mining industry consultant, MPR Geological Consultants Pty Ltd ("**MPR**") was commissioned by Cardinal to estimate the Mineral Resources of the Namdini deposit. To satisfy the Reasonable Prospects For Eventual Economic Extraction ("**RPEEE**") guidelines under the JORC code and CIM standards, the Mineral Resources are reported within an optimized pit shell produced by Golder. The optimization parameters reflect a large scale conventional open pit operation at a gold price of US$1,950/oz.

The Mineral Resource estimate was reported in accordance with the JORC Code (2012) to the ASX and TSX on 3 April 2019 (Cardinal, 2019a) as shown in the Appendix provided with that release (JORC Code Table 1). The Mineral Resource estimate, summarised in Table 5 and Table 6, on a 100% Project basis shows the Mineral Resources by category and material type (oxidation) at 0.5 g/t Au cut-off. The classification categories of Measured, Indicated and Inferred Mineral Resources under the JORC Code (2012) are equivalent to the CIM categories of the same name (CIM, 2014).

Table 5: Measured and Indicated Mineral Resources at 0.5 g/t Au cut-off, 3 April 2019 (Cardinal, 2019a)

Mineral Resource category	Type	Tonnes (Mt)	Gold grade (g/t Au)	Contained gold (Moz)
Measured	Oxide	1.1	1.23	0.04
Measured	Fresh	6.4	1.33	0.27
Measured Resource	**Total**	7.5	1.31	0.32
Indicated	Oxide	3.3	1.08	0.11
Indicated	Fresh	171	1.11	6.10
Indicated Resource	**Total**	174	1.11	6.21
Measured and Indicated	**Oxide**	4.40	1.12	0.16
Measured and Indicated	**Fresh**	177	1.12	6.38
Measured and Indicated	**Total**	182	1.12	6.53

Table 6: Namdini Inferred Mineral Resource estimate at 0.5 g/t Au cut-off, 3 April 2019 (Cardinal, 2019a)

Mineral Resource category	Type	Tonnes (Mt)	Gold grade (g/t Au)	Contained gold (Moz)
Inferred	Oxide	0.04	1.0	0.001
Inferred	Fresh	12	1.2	0.46
Inferred Resource	**Total**	12	1.2	0.46

Notes: Mineral Resource estimates are reported inclusive of those Mineral Resources converted to Ore Reserves as of 3 April 2019 (refer to the announcement released to the ASX and TSX on 3 April 2019).
The Mineral Resources and Ore Reserves are reported in accordance with JORC Code (2012) guidelines and Canadian Institute of Mining, Metallurgy and Petroleum Standards (CIM, 2014).
Mineral Resource estimates follow the ASX Listing Rule LR 5.8.1.
Numbers may not add due to rounding to appropriate significant figures.

MPR (2019) used the same methods described in the PFS report (Golder, 2018b) to estimate 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 MIK modelling used GS3M™ resource modelling software developed by FSSI Consultants (Australia) Pty Ltd.

The MIK modelling was based on 2 m down-hole composited gold assay grades from RC and diamond resource drilling. It incorporated a mineralized domain interpreted on the basis of the 2 m down-hole composited gold grades. 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°.

The modelling incorporated panels of dimensions 12.5 mE by 25 mN by 5 m elevation reflecting drill hole spacing in the areas of tightest resource drilling. The resource estimates include a variance adjustment to give estimates of recoverable resources at various gold cut-off grades. The variance adjustments were applied using the Direct Lognormal Correction method. The variance adjustment factors are considered by MPR and Cardinal to reflect comparatively large scale, open pit mining consistent with Cardinal's perception of potential mining scenarios. The variance adjustment factors were estimated from the variogram model for gold grades assuming mining selectivity of 5 m by 10 m by 2.5 m (across strike, along strike, vertical) consistent with high quality grade control sampling on an 8 by 12 by 1.25 m pattern. The variance adjustments 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 by Golder (2019a).

The three progressively more relaxed search criteria used for MIK estimation are presented in Table 7. The search ellipsoids were aligned with the general mineralization orientation.

Table 7: Estimation 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

The Namdini Mineral Resource has been classified as Measured, Indicated and Inferred 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. The classification used the search passes and wire-frames outlining more closely drilled portions of the mineralization. The classification approach assigns mineralization tested by generally 50 by 50 m drilling to the Indicated category, with estimates for zones with more closely spaced drilling classified as Measured. Estimates for panels not informed consistently by at least 50 by 50 m drilling are assigned to the Inferred category.

1.15 Ore Reserve estimation

The mine design and Ore Reserve estimate are based on the revised Mineral Resource model announced to the ASX and TSX on 3 April 2019 (Cardinal, 2019a).

The Namdini Ore Reserve reported in this FS is a subset of the Measured and Indicated Mineral Resource, which forms the basis of the technically and economically viable Namdini Gold Project.

The Ore Reserve estimation process converted approximately 80% of the Measured and Indicated Mineral Resources to Proved and Probable Ore Reserves.

The Ore Reserve was estimated for the Namdini Gold Project as part of this FS by Golder, which is summarised in Table 8. The total Proved and Probable Ore Reserve is estimated at 138.6 Mt at 1.13 g/t Au with a contained gold content of 5.1 Moz. Of this total, 92% of the contained gold is within the Probable Ore Reserve category.

The Ore Reserve for the Project is reported in accordance with the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves (JORC, 2012) and Canadian Institute of Mining, Metallurgy and Petroleum "CIM Definition Standards for Mineral Resources and Mineral Reserves" (CIM, 2014). The Mineral Resource was converted after applying Modifying Factors. All the Proved Ore Reserve is based on the Measured Mineral Resource and all the Probable Ore Reserve estimate is based on the Indicated Mineral Resource. Table 8 presents a summary of the Ore Reserves on a 100% Project basis at a US$1,300/oz gold price.

All Measured Resources within the final pit design were converted to Proved Ore Reserves with the Indicated Mineral Resources within the pit shell converted to Probable Ore Reserves.

Table 8: LOM Ore Reserve estimate, 3 April 2019 (Cardinal, 2019b)

Ore Reserve Category	Type	Tonnes (Mt)	Gold Grade (g/t)	Contained Gold (Moz)
Proved	Oxide	1.0	1.21	0.1
Proved	Fresh	6.4	1.33	0.3
Proved Reserve	**Total**	**7.4**	**1.31**	**0.4**
Probable	Oxide	3.0	1.08	0.1
Probable	Fresh	128.2	1.13	4.6
Probable Ore Reserve	**Total**	**131.2**	**1.12**	**4.7**
Proved and Probable	Oxide	4.1	1.11	0.2
Proved and Probable	Fresh	134.5	1.13	4.9
Proved and Probable	**Total**	**138.6**	**1.13**	**5.1**

Note: Costs and totals are rounded

The Ore Reserve conforms with and uses JORC Code (2012) and CIM (2014) recommendations

The Ore Reserve was evaluated using a long-term average gold price of US$1,300/oz with the US$1,235/oz optimization pit shell chosen for the Ore Reserve pit design to maximise the resource base

The Ore Reserve was evaluated and reported using an average cut-off grade of 0.5 g/t Au

Dilution was incorporated through the use of an MIK recoverable resource estimation modelling technique, which was demonstrated to incorporate an expected level of equivalent ore loss and dilution for the scale of mining envisaged. This dilution was incorporated at the resource block model stage prior to delivery to Golder

All relevant modifying factors under the ASX Listing Rule LR 5.9.1 guidelines were applied to the Ore Reserve estimate, as described in the JORC Code Table 1 provided as Appendix 1 in the ASX announcements dated 3 April 2019 and 18 April 2019

All figures are rounded to reflect appropriate levels of confidence

Apparent differences may occur due to rounding.

1.16 Mining

1.16.1 Approach

Cardinal commissioned Golder to carry out mine optimization and design studies and an economic evaluation that led to declaration of the Ore Reserve based on technical work reported by Golder (2019b).

Mining of the Namdini Gold deposit has been planned 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 limited amount of free-dig material with the majority of material assumed to require drilling and blasting.

Mining will comprise a conventional hydraulic shovel operation typically using 400 t class excavators in backhoe configuration for mining ore and face-shovel configuration for mining waste. Rigid body Cat 785 (134 t) class dump trucks will be used for hauling ore and waste on designed access roads. An auxiliary mining fleet of dozers, graders, water carts and utility vehicles will support the mining operation.

Mining will be carried out using staged cut-backs with five identified Phases incorporated within the LOM Final Pit. The mining schedule defines movement of ore and waste on 10 m mining benches, by year, for each of the Phases. It is proposed to mine three flitches in the ore (allowing for blast heave), within 10 m benches.

The Mineral Resource model is the basis for the mining model used for the 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 variance adjustment allows for a selective mining unit ("**SMU**") of 5 m by 10 m by 2.5 m, which has been applied to Namdini's large-tonnage, disseminated deposit.

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 mining contractors and compared with similar gold projects. Gold grades were supplied with the model as estimated proportional grades using the MIK recoverable resource estimation technique.

Oxide ore will be stockpiled temporarily and batch-fed into the process plant when suitable volumes are available, at the end of the mine life. Waste rock will be stockpiled separately on the western side of the pit.

1.16.2 Mining factors

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

For selecting the optimum Whittle pit for mine design Golder estimated a mining base surface cost of US$2.86 /t of rock mined 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 general and administration ("**G&A**") cost were estimated at US$14.30 /t milled plus an additional US$1.50 /t allowance for stockpile reclaim. All costs are based on dry tonnes.

Once the optimum Whittle pit was selected and mine design completed, a detailed mining movement schedule was supplied to five prospective mining contract companies to assist with the provision of a detailed mining cost estimate. Further discussions and negotiations will continue with suitable mining contractors prior to any award of the mining contract.

Metallurgical test work as reported in the Pre-feasibility Study, 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 in each parcel and the gold grade above the cut-off grade were included within the mining model to allow export of the parcelled (ore + waste) blocks for open pit optimization.

A minimum mining width of 80 m was assumed. Mining dilution and recovery are addressed in the modelling method (MIK with variance adjustment) and the utilisation of flitch mining. No Inferred Mineral Resources have been included within the LOM planning. Mining infrastructure requirements were planned 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 m by 15 m with samples taken at 1.5 m intervals down-hole (i.e. 1.25 m vertically). All grade control sampling will be fire assayed at the mine site.

No consideration has been made for underground extensions of the operation in this FS.

1.16.3 Geotechnical parameters

For the mine design, Golder Ghana (2018) carried out a study of existing geotechnical information, reviewed information on Mineral Resource estimates, conducted a detailed pit geotechnical drilling campaign, and gathered detailed rotary core logging data from selected drill locations within the 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 and data and reports made available by Cardinal.

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

For the final wall, bench stacks will consist of four double-benches (i.e. 80 m high) in Fresh material. These will be separated by either 25 m wide catch-berms or ramps. Some flexibility in height is used where a ramp is close to a catch berm.

1.16.4 Pit optimizations

Pit optimizations were completed using the Lerchs-Grossman (LG) algorithm in Whittle 4X™ software to calculate the optimal pit at the specified input parameters. A wire-frame 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 pit of approximately 1.2 Moz was chosen as the Starter Pit (Phases 1 and 2) to maximise discounted cash flow and minimise capital payback time.

1.16.5 Mine scheduling

Mine scheduling was used to maximise value by deferring 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**") targeted shell at around 0.95 RF of US$1,235/oz to maximise cash flow, 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.

The mine scheduling was completed using the Minemax™ Scheduler software in quarterly ("**Qtr**") periods. The mining sequence has been designed to incorporate five mining Phases. The initial mining Phase targets a relatively small initial pit of higher-grade ore with minimal waste stripping requirements (Starter Pit). The overall mining schedule is targeted towards the early generation of cashflow to assist in repayment of the Project capital and to maximise Project discounted value. The Minemax schedule was then used to back-flag the mining model to the quarterly periods mined. Subsequent to the flagging of the period mining sequence in the block model, the model was then used in the Alastri™ Tactical Scheduler ("**ATS**") to identify bottlenecks and to allocate resources for detailed down-stream cost modelling.

The process plant will accommodate an annual feed rate of 9.5 Mtpa as advised by Cardinal over the 15-year life (13 years of direct feed and 2 years of reclaimed stockpiles) including ramp-up. The ramp-up has been assumed to cover a three-month period within the first year. It is planned to treat 90% of annual throughput during that first year of plant operation. An initial pre-strip period has been 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.

The schedule was constrained to a generally maximum vertical sink rate (or Vertical Advance Rate, "**VAR**") of 80 m per year, with the mining Phases set to honour a minimum bench advance lead of 3 benches during any period.

The schedule targets the ore above the marginal cut-off grade, but uses the maximum recoverable value as a determination of process plant feed. Lower grade material is stockpiled as long as the mill feed requirements for deliverable ore can be met.

The scheduler was set up to maximise net present value (NPV) whilst maintaining a reasonable smoothing of tonnage movement per year. The peak tonnage requirement over the period 2026 to 2033 is maintained at 36 Mtpa to allow mainly medium- and high-grade ore to be directly fed to the process plant, with the low-grade ore being stockpiled for treatment after primary mining has completed.

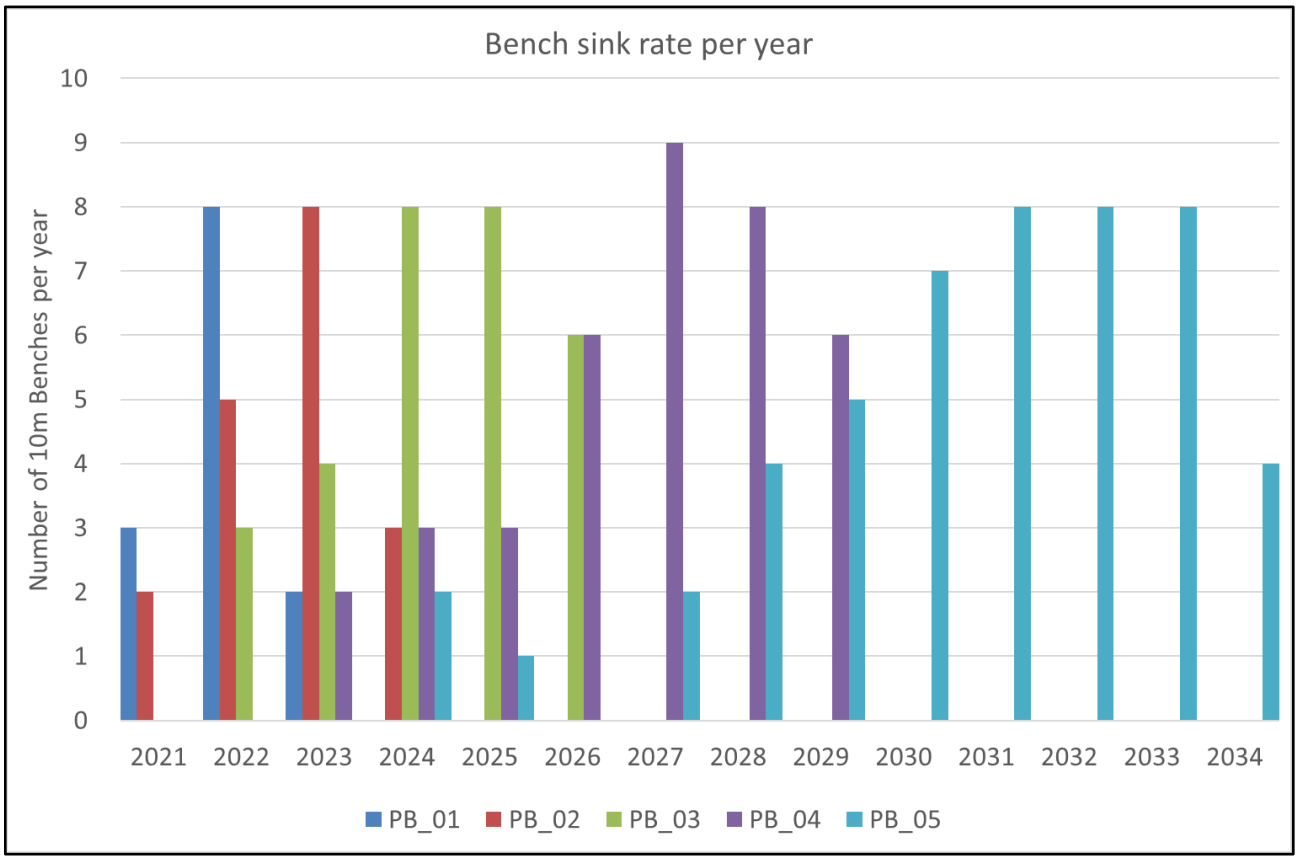


Figure 4: Mining vertical advance rate (source: Cardinal 2019)

1.16.6 Mine design criteria

The mine design criteria were developed to provide a plant feed rate of 9.5 Mtpa.

The mining sequence at Namdini consists of a relatively short period of higher-grade low strip ratio mining in the Phase 1 and Phase 2 pits. The Oxide ore is removed from most of the mining Phases during the early part of the schedule as this material is stockpiled for processing at the end of the LOM following process plant modifications. The maximum mining movement has allowed for a strip ratio of up to 3:1 so that the initial optimizations are not 'mining-limited'. The final Life of Mine strip ratio is 1.9:1.

The pit design considered the geotechnical requirements for berms, face angle batter and catch-berms for the lithology within the block model to establish the engineered pit design in which the Ore Reserve is contained.

The pit was designed with five Phases, the first one (the Starter Pit) having two sub-phases with a single common ramp, allowing early access to the higher-grade ore near the surface. The second Phase is largely an expansion of the initial Phase targeting the ore to a greater depth. The Phase designs were created for optimal ore delivery from the first two Phases due to their low strip ratio and waste rock movement. The subsequent Phases contain a greater proportion of waste rock. A minimum mining width of 80 m was established between the Phases.

The Namdini deposit has been described as a largely diffuse orebody with limited pockets of higher-grade zones. An analysis of the grade-tonnage curve for the LOM Pit (Phase 5), indicates that there is a relatively low proportion of higher-grade material above 1.5 g/t Au. This type of grade-tonnage curve with a very flat curve for grades higher than 1.5 g/t Au shows that the ore attainable from selective mining would be minimal compared to the total available ore within the pit. The grade-tonnage profile for the Ore Reserve within the Final Pit design shows that some 86% of the Ore Reserve sits within a gold grade range of 0.5 to 1.5 g/t Au.

Of the 138 Mt of ore within the Final Pit design, only 20 Mt of the total is above a gold grade of 1.5 g/t. The nature of the ore lends itself to bulk mining, with minimal edge dilution and ore loss expected at the edges of the demarcated ore zones during mining.

The design work targeted the maximum discounted value pit shell at a US$1,235/oz gold price. The pit optimization using the Whittle 4X 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 in order to minimise waste movement and improve the cashflow for the Project. The analysis allowed the selection of five Phases with the initial two Phases targeting a relatively higher-grade area of ore near surface. Access was allowed to the four Phases by a ramp from the northern edge of the pit as the volume of waste rock in the first four Phases is considered modest. The final fifth Phase 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 Phase has a higher strip ratio than the preceding four Phases. Having the primary access on the western side of the pit reduces waste rock haulage costs and thus improves the overall value.

Design of the pit Phases was largely focused on targeting maximum value change-points within practical mining constraint limits, such as the minimum mining width of 80 m between the Phases.

Table 9 shows the key estimated production results.

Table 9: Key estimated production results

Key Estimated Production Results	Unit	9.5 Mtpa
Gold produced (average for full production years)	koz/y	287
Life of Mine production – gold	koz	4,177
Average mine head grade	g/t Au	1.13
Ore Reserve tonnes mined at 0.5 g/t Au cut-off grade	Mt	138.6
Life of Mine strip ratio	W:O	1.9:1
Mine life	years	15

The Final Pit design is shown in Figure 5.

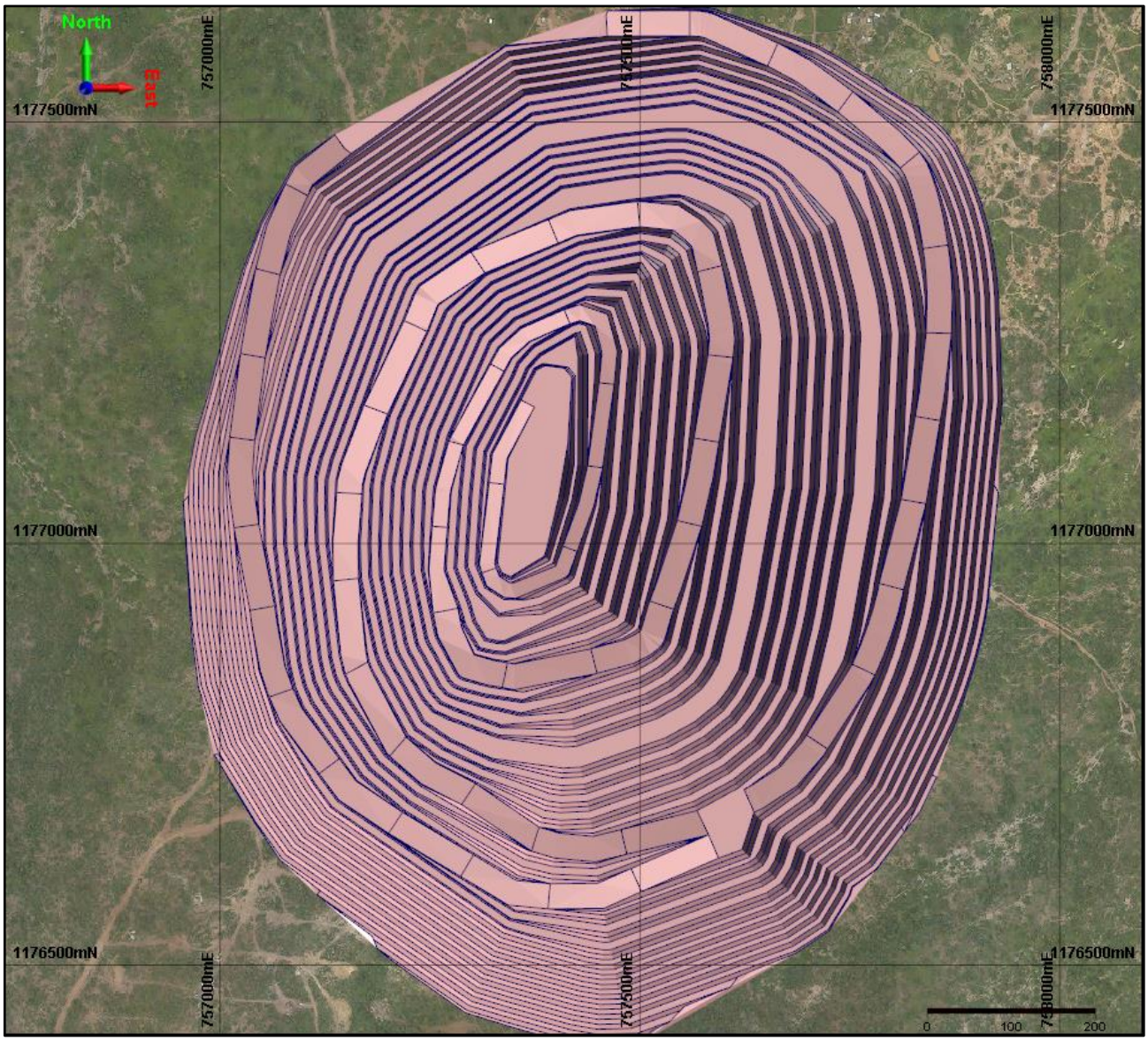


Figure 5: Plan view of Final Pit design

A typical view in cross section of the Namdini Pit and block model is shown in Figure 6.

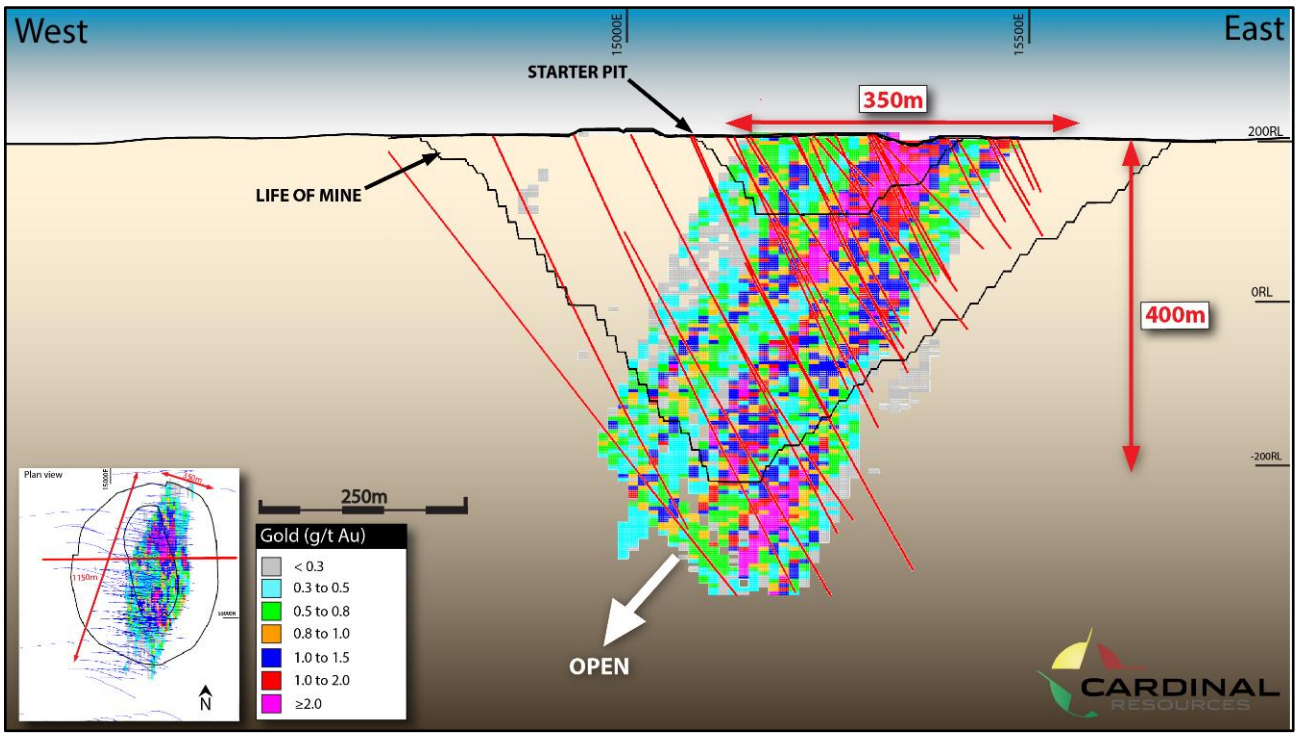


Figure 6: Cross section view of the Final Pit design and block model

A typical view in long section of the Namdini pit and block model is shown in Figure 7.

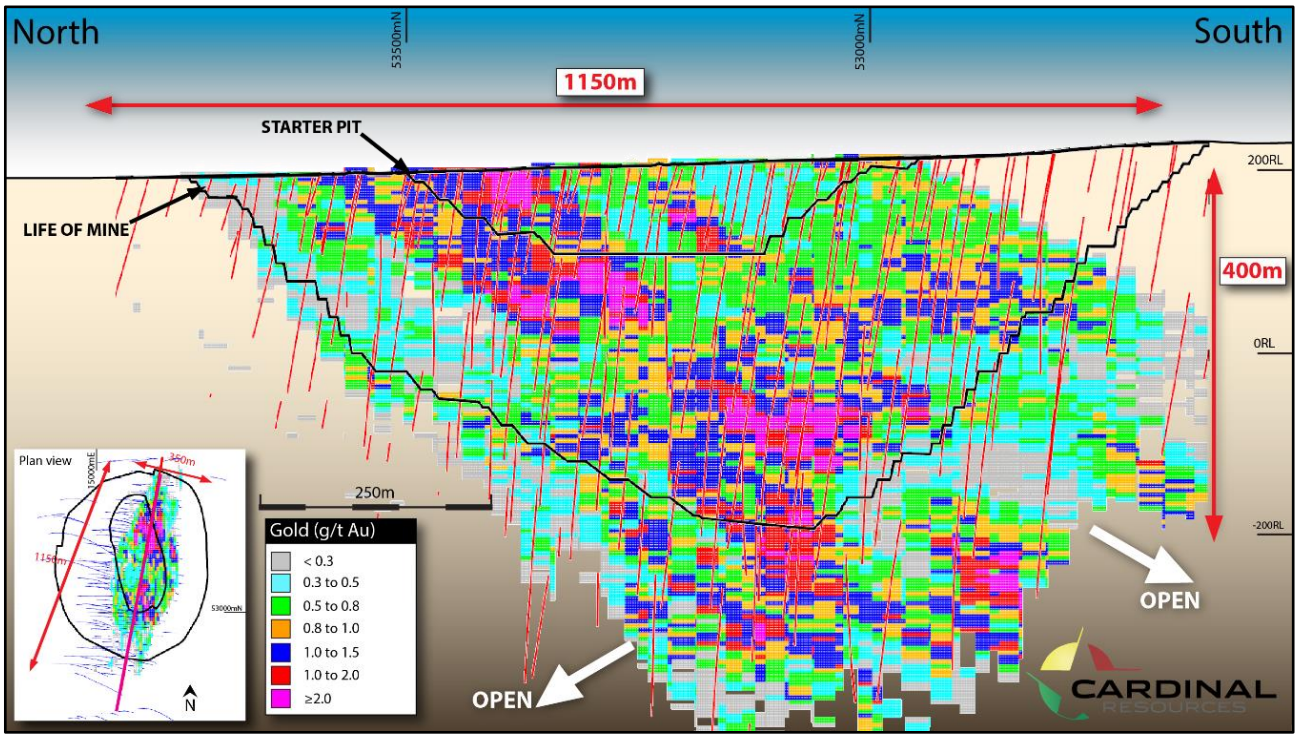


Figure 7: Long section view of the Final Pit design and block model

1.16.7 Mining cost

The FS 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.

Golder approached five leading mining contractor service providers in Ghana and has received budget estimates for the provision and supply of mining operations at the Namdini Project. The budget estimates have been used to provide project cost estimates for the mining operation over the life of the operation. Ghana currently has a very active mining contractor supply base. Specialist skills and core competencies in the mining contractor companies will be key considerations in the final selection criteria.

In addition to the mining contractor budget estimates, mining costs were also developed from first principles and a profit factor applied to estimate contract mining cost. 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 ("**US$**") basis.

1.16.8 Cut-off grade parameters

An estimated marginal cut-off grade was established at 0.5 g/t Au using an assumed long-term gold price of US$1,300/oz. The provided Mineral Resource model was validated and used to develop a mining model, as the basis for a 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 $39.67/g (US$1,234/oz). The input processing cost provided in the ASX and TSX announcement of 18 April 2019 (Cardinal, 2019b) was $14.30/t plus an additional $1.50/t allowed for stockpile reclaim giving a total of $15.80 /t of mill feed (as dry tonnes). Thus, the marginal cut-off grade ("**COG**") was estimated as: *process cost / (net gold price X process recovery)* giving 0.5 g/t Au (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 grade 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 in CIM (2014) and the JORC Code (2012).

The grade-tonnage curve (Figure 8) for the LOM Pit (Phase 5), indicates that there is a relatively low proportion of higher-grade material above 1.5 g/t Au.

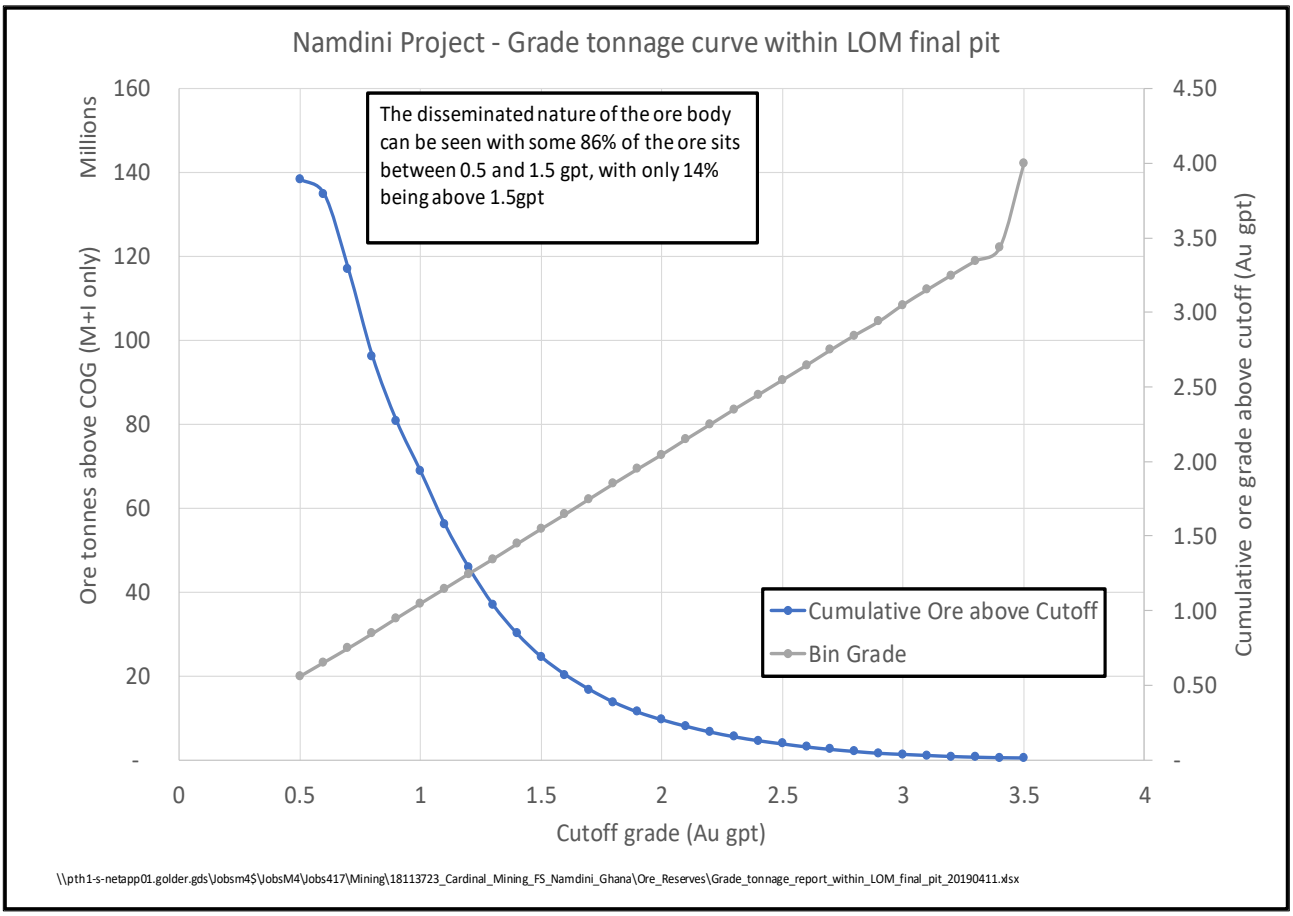


Figure 8: Grade-tonnage curve for Phase 5 Final Pit

1.16.9 Waste rock geochemical characterisation

To define the Acid and Metalliferous Drainage ("**AMD**") characteristics of the Namdini Project based on the preliminary mine planning, Cardinal requested that Golder undertake a geochemical characterisation of the waste rock, low-grade ore and mineralized waste to be produced by mining of the deposit. The main findings (Golder, 2019g) were:

- The bulk of the waste rock materials are Non Acid Forming ("**NAF**"), especially Metasediments (MSE), Dacite (DAC) and Pyroclastics and tuffs (LTF).
- The Granite (GRA), Diorite (DIO) and Mafic Volcaniclastics (MVO) are the key lithologies that are consistently characterised as being Potential Acid Forming ("**PAF**") or of uncertain classification ("**PAF-UC**"), for 20% of the samples of these lithologies analysed to date. These materials have elevated sulphur mainly associated with reactive sulphides, and insufficient Acid Neutralising Capacity ("**ANC**") to buffer the acidity produced by sulphide oxidation, although results suggest that even samples with elevated sulphidic sulphur may have significant lag times before producing acid leachate.
- Water extractable element testing suggested that there is likely to be minimal immediate mobilisation of trace elements in NAF and PAF materials represented by these samples, other than arsenic.
- MVO samples showed substantial mobilisation of arsenic, with deionised water leachate concentrations above 0.1 mg/L (World Bank Group International Finance Corporation ("**IFC**") mining guidelines and Ghana EPA Akoben mining effluent guidelines) in 90% of samples tested.

- GRA and DIO samples showed some mobilisation of arsenic, with deionised water leachate concentrations above 0.1 mg/L in over 20% of the samples tested, and concentrations above 0.01 mg/L in 67% and 100% of the samples tested respectively.

Based upon the above findings, considering the availability of beneficial oxide zone saprolite (which is non-acid forming and has characteristically low infiltration rates) and laterite (which has the ability to adsorb dissolved arsenic from leachate) it is planned that:

- The waste rock dump and the low-grade ore stockpiles should both be constructed with a basal layer of oxide zone saprolite.
- Waste rock materials that are higher risk (PAF, PAF-UC and arsenic-bearing) should be contained within a central cell of the waste rock dump.
- Low-risk waste rock materials should be dumped around the outside of the high-risk cell; they could be end-tipped in a doughnut shape, with the high-risk cell at the centre.
- The high-risk cell and the low-grade ore stockpile should be constructed with a layer of laterite to manage arsenic in the seepage (above the basal layer of oxide zone saprolite).
- The Tailings Storage Facility (TSF) embankments, which are to be earthfill, should be constructed with oxide zone saprolite, with low-risk waste rock cladding where required (with suitable geotechnical properties).

1.17 Process plant

The information presented in this Section is derived from Lycopodium (2019a) and Knight Piésold (2019a).

A nominal throughput processing rate of 9.5 Mtpa was selected as part of this FS.

The assessment of the comminution circuit identified that 9.5 Mtpa is the largest throughput that can be achieved with dual pinion mill drives.

The process plant design incorporated the following unit process operations:

- Single stage primary crushing with a gyratory crusher
- Crushed ore feeding a stockpile with ore reclaim by two apron feeders
- A SAG mill and ball mill in closed circuit with cyclones with recycle crushing of pebbles from the SAG mill
- A gravity recovery circuit consisting of centrifugal concentrators and an intensive cyanidation leach system
- Rougher flotation to produce a gold-rich sulphide concentrate
- Thickening of the flotation tails for water recovery prior to disposal in a separate non-cyanide Tailings Storage Facility ("**TSF**")
- Thickening of flotation concentrate followed by regrind utilizing Outotec HIGmill™ technology
- Maelgwyn's Aachen high-shear oxidation process on the regrind product before CIL
- A concentrate CIL facility incorporating one pre-leach aeration tank and seven CIL tanks
- A split AARL elution circuit, electrowinning and smelting to recover gold and silver and produce doré bullion
- CIL tailings treatment incorporating cyanide destruction
- Concentrate CIL tailings disposal in a lined tailings storage facility.

The plant is designed to comply with the International Cyanide Management Code (ICMI, 2016) for the manufacture, transport, and use of cyanide in the production of gold.

Figure 9 shows the simplified flowsheet for the Namdini Gold Project. More detail can be seen on Figure 140.

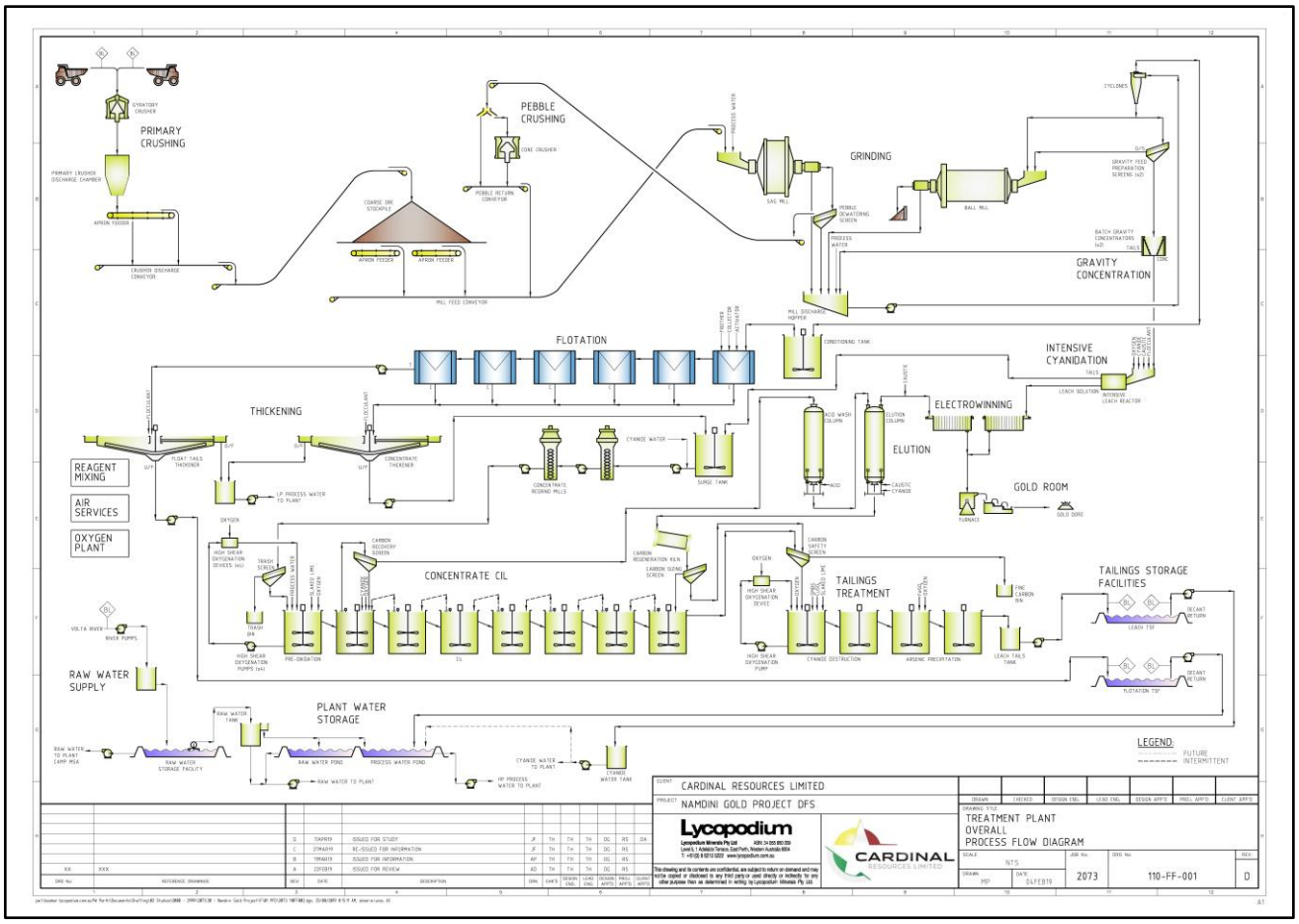


Figure 9: Simplified process flow diagram for the process plant

A summary of the plant design basis and the process plant components follows:

1.17.1 Process plant

The process plant design was based on a nominal capacity of 9.5 Mtpa. The feed will comprise blended ore from the pit consisting predominantly of three rock types being approximately 56% Metavolcanics, 29% Granite and 15% Diorite over the LOM. Oxide material will be fed at the end of the LOM once all Fresh + Transition material is depleted.

1.17.2 ROM Pad

The Run of Mine stockpile (“**ROM Pad**”) will hold the mill feed ore and be used as a buffer between the pit and the plant. The ROM Pad will allow for some blending of feed stock to help produce a consistent feed type and rate to the plant. A fixed hydraulic rock breaker will be used to break any oversized rocks in the crushing area.

1.17.3 Comminution selection

Previous comminution testwork started in 2016. This work focused on the main feed lithologies according to a preliminary mining schedule. This only comprises SAG mill power index (“**SPI**”) and bond ball mill work index (“**BBWi**”) data.

Additional testwork was completed in two campaigns during early and late 2017 and further testwork was completed in June 2018.

As part of optimizing the feed methodology for the Namdini Project, Cardinal geologists identified that the testwork undertaken during early 2017 consisted of material close to the contact zone that would be mined in the latter part of the production schedule.

It was also noted that the November 2017 and 2018 testwork was more representative of the plant start-up blend (effectively operating Year 1 to 5). In terms of identifying the longer-term LOM feed blend, all the testwork data was considered, including the 2016 and early 2017 data.

The comminution circuit design and throughput predictions reflect this methodology.

A future circuit upgrade is possible, transitioning from a 'primary crush only' fed circuit to a 'secondary crush' fed circuit.

The findings and conclusions were summarized in the 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 29.9 to 40, placing these ore samples in the hard category.
- A gyratory crusher was selected for primary crushing supported by a fixed rock breaker.
- The SAG mill power efficiency ("**fSAG**") for this ore is low. The fSAG is poor at 1.36 but could be improved by partial crushing.
- Assessment of the comminution circuit identified that 9.5 Mtpa is the largest throughput that can be achieved with dual pinion mill drives.
- Pebble crushing is included to manage build-up of critical size material in the SAG mill.
- The 85th percentile design abrasion index range of 0.090 for Metavolcanics to 0.290 for Granite shows relatively low ore abrasiveness and therefore translates into low and favourable wear liner and steel media consumption.

1.17.4 Selected comminution circuit

The comminution circuit option selected using the design criteria was a 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 minimises operating cost while still producing the relatively fine product size required.

1.17.5 Crushing and ore storage

The primary crusher was sized on the basis of achieving 75% utilization. The mine design is based on direct feed to the crusher with minimum rehandle. The crusher was 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 150 t trucks 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.

1.17.6 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 minimizing 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 range of mill speed with a nominal design of 75% of critical speed. The ball mill will also be variable speed. The variability in ore competence can be addressed by varying the speed of the mills.

A gravity circuit was included as some of the ore types tested, in the Starter Pit samples in particular, appear to have a gravity recoverable gold component.

1.17.7 Flotation concentrate thickening

Inclusion of a flotation concentrate thickener before the regrind facility will achieve a higher regrind feed slurry solids content. Thickening also recycles water that is not cyanide-bearing for reuse in the grinding and flotation sections of the process plant.

The higher regrind feed slurry solids allows the CIL tailings decant to be used for process water dilution to meet the regrind mill vendor's requirements for limiting the mill feed slurry solids content without compromising the plant water balance.

The thickener was sized to allow for potential froth management.

A surge tank is installed to store concentrate thickener underflow slurry prior to delivery to the regrind circuit. This allows the primary grinding and flotation circuit to remain operational in the event of a short-term maintenance stoppage or breakdown in the regrind area.

1.17.8 Flotation

Testwork on samples of the Transition and Fresh ore has indicated that a high gold recovery can be achieved by flotation, with a design point of 7.5% mass recovery from the feed selected by Cardinal for design.

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 each with individual level control. All flotation concentrate will report to a single concentrate hopper and be pumped to the concentrate thickener before transfer to the regrind circuit.

1.17.9 Concentrate regrind

Leach testwork indicated that regrinding of flotation concentrate would be necessary to improve leach extraction. The capability to regrind to a P_{90} grind size of 9 µm has been incorporated in the design.

A high intensity grinding mill (the HIG mill) was selected for the regrind duty. This equipment minimizes footprint, provides internal classification and has minimal overgrind issues, while providing suitable access for maintenance. The mill was sized based on testwork carried out by the vendor (Outotec) on behalf of Cardinal.

The reground concentrate will be pumped to the trash screen ahead of the Aachen process and CIL circuit.

1.17.10 Pre-leach shear and oxidation

The Aachen process has proved at pilot scale to enhance leach kinetics and gold extraction for all the ore types fed to the process. The process uses a pipe shear device designed to improve oxidation of slurries using oxygen fed into an aerator. The aerator has an engineered slot to introduce micron-sized (200 µm) oxygen bubbles into the device which has different chambers separated by orifice plates creating a highly efficient contact of oxygen to the slurry. The equipment is especially efficient for high-rate oxidation of sulphides which is ideal for Namdini's ore since the gold is particulate and occurs as free gold, in fractures in pyrite or in pyrite grains.

1.17.11 Leach and adsorption circuit

Metallurgical testwork indicated that pre-oxidation improved both the gold leaching kinetics and overall dissolution. As a result, a pre-aeration tank was included in the design. Cardinal nominated a leach residence time of 48 hours for design.

A configuration of one pre-aeration tank and seven CIL tanks was selected. The pre-leach and CIL tanks will be the same size. As well as the pre-aeration tank oxygen addition facility, extra oxygen may be added to all CIL tanks to oxidize any cyanicides and maintain an adequate dissolved oxygen level.

1.17.12 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 4.5 t capacity split AARL elution circuit was selected requiring just under seven strips per week. The AARL circuit also offers the flexibility to run more than one elution per day if needed.

1.17.13 Flotation tails thickening

Tails from the flotation circuit will be dewatered in a high rate thickener to recover cyanide-free process water for re-use in the grinding and flotation circuits.

1.17.14 Cyanide destruction

Concentrate leach generally requires a higher cyanide concentration to achieve efficient extraction. As a result, tailings from the CIL circuit may have high weak acid dissociable cyanide (CN_{WAD}) levels. A cyanide destruction facility incorporating sodium metabisulphite and oxygen was included to reduce the levels of CN_{WAD}, compliant with the International Cyanide Management Code (ICMI, 2016) limits.

1.17.15 Arsenic precipitation

Testwork has indicated that elevated levels of arsenic can occur in leach solutions. The arsenic levels in CIL tailings solution can be reduced by the addition of iron sulphate. This process has been included in the design.

The design of the arsenic precipitation circuit has two agitated tanks, the same size as the cyanide destruction tanks. This provides additional residence time. One tank is designed for use as either a cyanide destruction or arsenic precipitation tank to provide additional cyanide destruction capacity if required.

1.17.16 Tailings disposal

Two separate tailings streams will be produced. CIL 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 a plastic lining. This will allow the two decant water streams to be recycled separately.

1.17.17 Water and air services

Three water systems will be used:

- Process water recovered from the flotation tails thickener and tailings storage facility will be recycled in the grinding and flotation areas.

- Cyanide water recovered from the cyanide tailings storage facility will be used in the CIL circuit.
- Raw water sourced from the White Volta River via a raw water storage dam will be used for reagent mixing, cooling water make-up, pump gland seal water and will feed a water treatment plant for elution water and potable water plant feed.

Three air systems will be used:

- Low pressure air will be used in the flotation cells.
- Oxygen will be used in the CIL circuit and cyanide destruction and arsenic precipitation areas.
- Compressed air will be used for instruments and plant services.

1.18 Infrastructure

The information presented in this Section is derived from Lycopodium (2019b) and Knight Piésold (2019a).

1.18.1 Location and layout

The site is located approximately 50 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 to and from site as required.

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 Pwalugu 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 Pwalugu.

Cardinal has sufficient area available on its leases to cater for all its planned infrastructure requirements.

Lycopodium and Knight Piésold completed FS level analyses covering all related aspects of the infrastructure requirement including power, water, road access and waste management.

The site infrastructure layout has been developed and is shown in Figure 10.

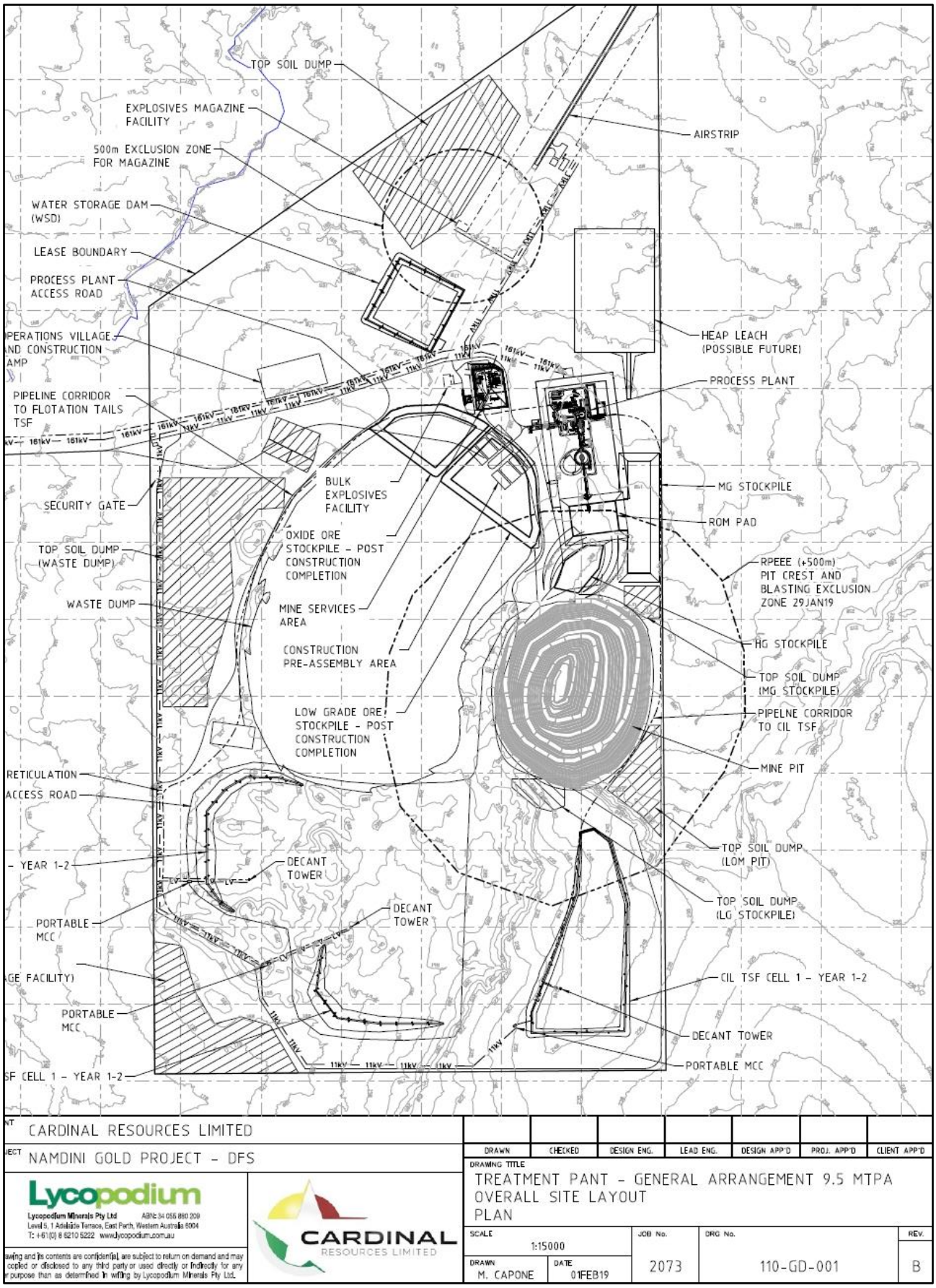


Figure 10: Project site infrastructure layout

The site layout was developed based on the following information:

- Total available ore tonnage for the Life of Mine 138.6 Mt
- Process throughput 9.5 Mtpa.
- Flotation tail/concentrate split 92.5% : 7.5%
- Tailings to flotation TSF 128.2 Mt
- Tailings to CIL TSF 10.4 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.

Infrastructure costing includes the following dedicated elements:

- Unsealed access road
- High voltage power supply line to the process plant
- Water supply pipeline from the White Volta River
- Raw water storage facility
- Airstrip
- Accommodation village
- Mine service area including heavy vehicle workshops
- Fuel depot.

Approximately 260 Mt (95 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 northwest of the open pit.

The process plant and mine services area will be located on relatively flat terrain to the north-northwest of the open pit and directly to the northeast of the waste dump.

For the FS, the TSF designs have been further optimized using locations defined in the PFS and incorporating more recent topography for the site area together with the updated design parameters.

A hydrogeological fieldwork programme was undertaken comprising a hydro-census of surrounding properties to identify groundwater users. Characterisation 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.

Further geotechnical investigations of the flotation TSF, CIL TSF and process plant sites were carried out by Knight Piésold as part of the Feasibility Study to investigate the sub-surface conditions and to provide final geotechnical parameters for design. A geotechnical investigation of the site access road, water storage facility, airstrip and potential borrow sources was carried out as part of the FS.

The power design is based on establishing a grid power supply to the process plant. This will be achieved by constructing a new GRIDCo switchyard near the process plant to step-down the incoming 161 kV supply to 11 kV for plant power distribution. The grid supply is seen to be reliable, based on a HV supply monitoring study previously carried out by Cardinal. Construction power and emergency power for process plant essential drives and facilities during grid power supply outages is provided for.

A water management model was developed to understand the 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 quantify the volume of water required from external sources. Results indicate the CIL plant will only require 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 will require some additional water make up, some of which may be sourced from an abstraction tower on the northern bank of the White Volta River approximately 8.5 km to the west 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.

Waste will be treated or stored appropriately. Grey water and effluent from all water fixtures will drain to gravity sewerage systems at the camp and plant site. Effluent will be treated in a sewage treatment plant located adjacent to the camp. Solid waste will be sorted and reused or recycled as far as the limited access to recycling facilities allows. Waste lubricating oils will be returned to the supplier for recycling.

Construction and permanent accommodation camps have been allowed for. 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 centres 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.

Requirements for site communications, IT services, mobile phone, radio and satellite TV connections have been estimated. Diesel fuel and explosives storage and handling have been considered.

Site security is based on concentric lines of fencing and control. 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 and exit will be via a single monitored security post and will be strictly controlled. Access to the gold room within the plant will be restricted and strictly controlled.

Site facilities, buildings and the airstrip have all been allowed for in the design and costings. An administration office, site warehouse and clinic combined with the emergency response facility will be located together.

An airstrip will be constructed to service the site with freight shipments both inwards and outwards, and gold shipments outwards. The airstrip will be located approximately 2.5 km to the west-southwest of the accommodation camp and directly to the south of the site access road.

1.18.2 Tailings Storage Facilities

1.18.2.1 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 in terms of percent water release, 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 pH values. Therefore, the Diorite and Metavolcanics flotation tailings are classified as acid consuming ("**AC**") and the Granite rougher tailings as non-acid forming ("**NAF**") following the AMIRA (2002) classification.
- The CIL tailings sample recorded a positive NAPP and a low net acid generation pH, resulting in a classification of potentially acid forming ("**PAF**").
- On the basis of the multi-element results, both the flotation and CIL TSFs are 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 ongoing 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 a compacted soil liner with overlying HDPE liner and underdrainage system.

1.18.2.2 Design of the 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 474 Ha and is designed to accommodate 139 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 (Knight Piésold, 2019b).

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 under-drainage system comprising a network of branch and collector drains in each cell. The system will drain 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 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.

1.18.2.3 Design of the 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 earth fill 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 16 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 (Knight Piésold, 2019b) to a final elevation of 266.0 m RL.

The TSF basin area will be cleared, grubbed and topsoil stripped, and a 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 smooth HDPE geomembrane liner.

The TSF design incorporates an underdrainage system comprising a network of branch and collector drains. The underdrainage system will drain 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 mounted in a decant tower located along the western embankment of the facility.

In order to mitigate seepage losses through the basin area, minimise 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
- Leachate collection and recovery 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 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 that cannot be stored within the facility will discharge via an engineered spillway.

1.19 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$1,350/oz for the purposes of the financial model at the date of this FS.

1.20 Environment

1.20.1 Studies

Cardinal engaged NEMAS Consult Ltd ("**NEMAS**") as an independent consultant to undertake the Environmental Impact Assessment ("**EIA**") study for the Project (NEMAS, 2017). This culminated in preparation and submission of an Environmental Impact Statement ("**EIS**") report (NEMAS, 2018) to the Ghanaian Environmental Protection Agency ("**EPA**") on 26 October 2018 to satisfy the Environmental Protection Agency Act 1994 (EPA Act 490) and the Environmental Assessment Regulations 1999 (LI 1652). The EIS Report is still in Draft form pending approval from the EPA.

Cardinal has also registered the Tailings Storage Facility (TSF) portion of the Project with the EPA as a standalone project. This enables a TSF-specific EIA to be conducted to ultimately obtain an environmental permit for the construction, operation and closure of the TSF. Therefore, Cardinal engaged Geosystems Consulting Limited ("**Geosystems**") to prepare a Draft TSF Scoping Report (Geosystems, 2019), which precedes the EIA specifically for the TSF. The Draft TSF Scoping Report sets out the scope and extent of the EIAs to be carried out and includes the draft terms of reference to be addressed in the EIS during the EIA process.

This NI 43-101 Technical Report provides a high-level summary of the environmental and social baseline conditions with a focus on key findings from the baseline studies conducted to date for the Project. Numerous desktop and field studies and assessments were undertaken to support the EIA Study and EIS, including examination of stakeholder engagement, flora, fauna, aquatic biology, hydrology, soils, environmental quality, archaeology and culture.

There are four main sub-basins within the Volta catchment, consisting of the Black Volta River, White Volta River, Oti-Pendjari River and the Lower Volta River. 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 is located approximately 7 km north of the White Volta River (Figure 11).

A small ephemeral stream, the Zoan Buliga, is the only visible source of surface water that traverses the Project area and passes through the northern boundary before joining the White Volta. The Zoan Buliga stream flows from a few days to weeks during the months of heavy rainfall, typically between May and October.

Stagnant ponds of water are a result of rainfall collecting in pits left by illegal mining.

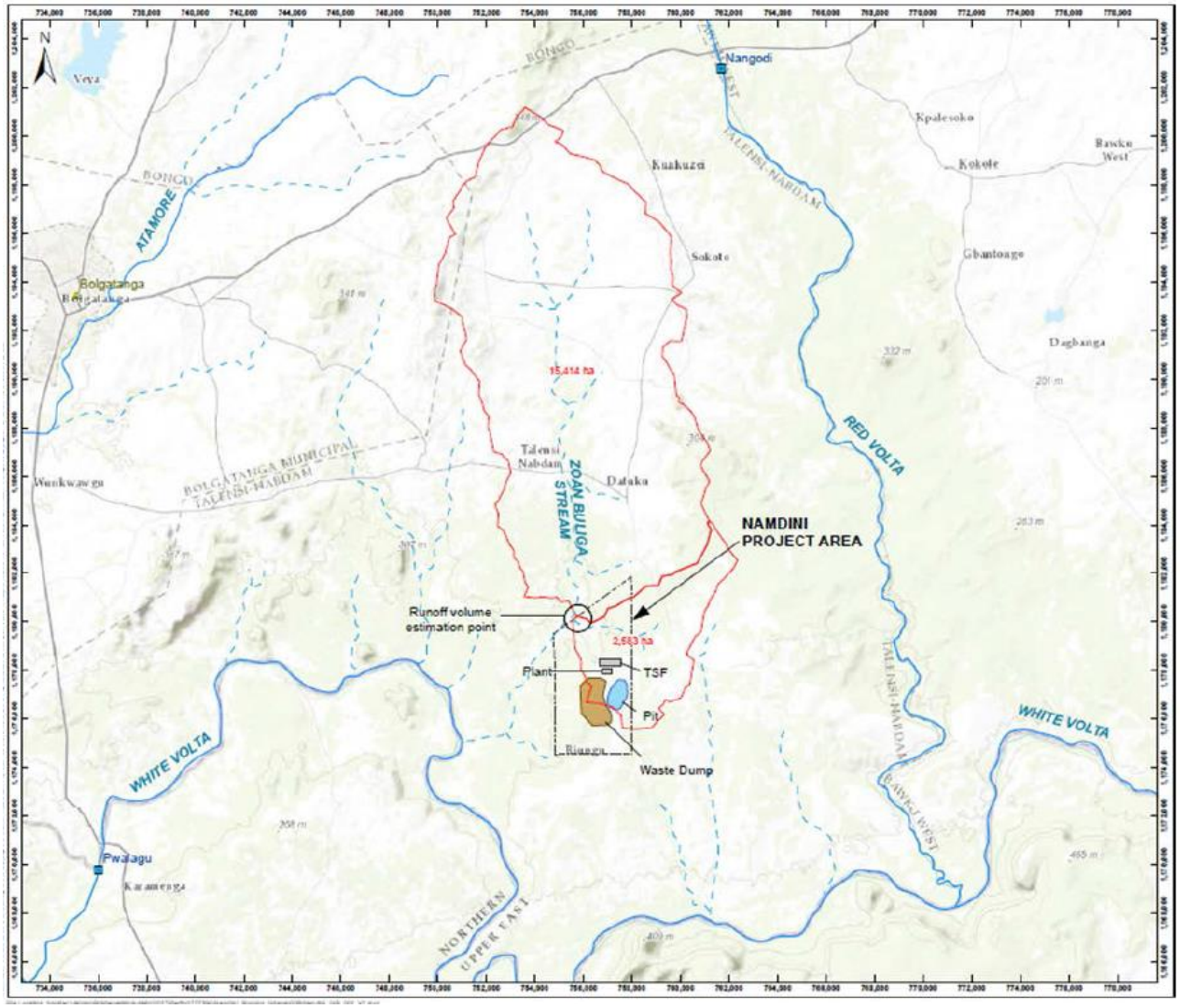


Figure 11: Location of the Project area in relation to nearby rivers

The broader hydrogeological aspects of the region and the impacts on community use of groundwater are discussed in Section 20.3.3.

The hydrology is discussed in Section 20.3.4 and the results of water quality sampling are discussed in Section 20.3.5.

1.20.2 Social setting

A social baseline study was conducted as part of the EIA Study to outline the socio-economic and cultural settings of the host communities for a reference against which to measure impacts (positive and negative) of the Project on the host communities.

The host communities affected by Cardinal's 19.5 km^2 Project area are the Biung (Bingo) and Datoko which also captures Wankala and Zangore.

The dominant tribe is the Talensi which forms about 90% of the local population with some migrants from Southern Ghana and neighbouring countries. The area is under the Nabdam Traditional Council and is male dominated, both in terms of demography and cultural practices. The people are mostly traditionalist and/or Christians with Muslims forming the minority.

Fifteen heritage resources, two archaeological settlement sites and 13 shrines were identified and documented at Biung. Three shrines and 14 archaeological settlement sites were identified and documented at Datoko.

Most residents either have no formal education or are without any employable skills. The local economy is based on subsistence farming. Illegal artisanal gold mining has dominated economic activity particularly within the Project area over the past 20 years. Concerns regarding loss of access to land and therefore loss of this economic activity due to Project development is a key concern for these communities.

As part of the EIA Study and to support the EIS, Cardinal identified approximately 29 key stakeholders for the Project, including Government regulatory agencies, Government institutions, traditional and political authorities, Project communities and leaders, and non-Government authorities.

Extensive social consultations and stakeholder engagements have been held with key stakeholders.

All key stakeholders consulted expressed their support for the Project. Key concerns raised related to security, Social Corporate Responsibility, payment of royalties, and protection of juveniles from engaging in artisanal mining activities. Stakeholders advised Cardinal to operate an ‘open door’ policy based on mutual respect.

1.20.3 EIS

As noted above, Cardinal has recently completed an EIS report for Namdini and has filed the EIS with the EPA. In accordance with EPA Regulations 15(1b) and (1c) of the Environmental Assessment Regulations, 1999 (LI 1652) and Ghana’s EIA Procedures, the EPA issued a public notification on the proposed Namdini Gold Mining Project.

The FS environmental studies included active engagement with local and state regulatory bodies and local stakeholders. Cardinal has an excellent relationship with neighbours and the granted leases cover all the required mining and processing areas. There are no title claims pending.

1.20.4 Tailings Storage Facilities

Reclamation and subsequent closure of the proposed TSFs will be completed in accordance with the provisions of LI 2182 and with the conditions of the environmental permit and EPA guidelines so that water leaving the Project site meets regulatory requirements.

The proposed TSFs will be rehabilitated to promote long term stability, minimise erosion, and provide a discharge water quality that meets the baseline quality of the receiving environment or the EPA guidelines for discharge into waterbodies. Rehabilitation of the TSF would begin when process plant operation ceases and tailings are no longer generated, with initial measures taken to stabilise the tailings surface. This is a key issue that will be appropriately addressed to ensure compliance with the legislation and result in long term environmental sustainability.

Decommissioning and rehabilitation of the TSF will be linked to the general rehabilitation of the mine site and the permitting conditions for the mining lease.

1.20.5 Potential environmental and social impacts, and mitigation

The Project shall interact both positively and negatively with the biophysical and socioeconomic resources and/or receptors within and beyond the Project area. The key potential positive impacts and opportunities, and key potential adverse impacts from Project construction, operation, and decommissioning/closure identified in the draft Project EIS (NEMAS, 2018), and draft Scoping Report for the TSF (Geosystems, 2019) are summarised in Section 20.4.

1.20.6 Resettlement

All issues related to relocation and/or resettlement shall come under a Resettlement Action Plan ("**RAP**") to be produced as a standalone report for the Project. The handling of relocation and/or settlement will consider the following principles:

- Fair and adequate compensation
- Compensation based on negotiation
- Payment before destruction
- Maintenance and/or betterment of facilities to pre-project condition
- Regards to traditional and spiritual needs of the people
- Grievance redress mechanism.

Cardinal will liaise with relevant stakeholders as provided in the RAP to establish the Relocation and Compensation Negotiation Committee ("**RCNC**") within the Project approval phase prior to commencement of construction. The RCNC will approve or agree on compensation rates for affected properties/assets, livelihood restoration programmes for project affected persons and deprivation of use of land/surface rights issues as provided in the *Minerals and Mining Act 2006* and the Minerals and Mining (Compensation and Resettlement) Regulations, 2012 (LI 2175).

Potential compensation amounts cannot be predicted at this stage.

1.20.7 Environmental management and monitoring

A provisional Environmental Management Plan ("**EMP**") has been developed for the Project which captures Cardinal's corporate policies on environment, health and safety with clear policy objectives and targets as well as establishing a comprehensive Environmental Management Structure.

Cardinal is committed to financing the environmental aspects of the Project from its annual operating budget for the Mine and will make provisions for financial allocations during budget preparations for the implementation of all environmental management actions, including monitoring. The detailed cost estimates will be calculated prior to the construction and operational phases.

A detailed EMP will be prepared and submitted to the Ghana EPA within 18 months of commencement of operations under Regulation 24 of LI 1652. Cardinal will update the EMP every three years throughout the life of the Project.

Environmental monitoring programmes will be implemented to confirm the extent and magnitude of predicted impacts and to ensure effectiveness of the proposed mitigation measures as well as to confirm any residual impacts, which are expected with the development of such Projects. The proposed monitoring programme will cover meteorology, ambient air quality and noise levels, blast vibration, water resources and the socio-economic environment.

1.20.8 Closure, decommissioning and reclamation

The Project's Mining Lease was issued for 15 years, renewable for a further 30 years. The projected mine life is 15 years (including ramp-up and mine closure) based on a 9.5 Mtpa processing plant throughput.

Within two years of commencement of operation Cardinal is required to prepare and submit a detailed Reclamation and Decommissioning Plan for the Project to guide the reclamation security agreement to be signed by Cardinal and the Ghanaian EPA. The Plan shall address land stabilisation and erosion, sediment control among other issues associated with the Project. The Plan will serve as the basis for concurrent reclamation during the Project's life. A final Reclamation and Decommissioning Plan is required to be submitted to the EPA two years before closure of the Project.

Cardinal's aim of reclamation is to achieve a socially acceptable balance between mining and the biophysical and social environment, and to ensure that internationally accepted standards of health, safety and environmental protection are observed during and after active mine life.

The short-term reclamation approach will be to stabilise all areas heavily impacted during mining operation as well as its immediate areas not disturbed from any land degradation, either by erosion, sediment transport or deposition.

The long-term reclamation approach will be to ensure complete site stabilisation and re-establish a productive floristic heterogeneous community capable of supporting and sustaining a healthy faunal population. The post-mining decommissioning is also to ensure public safety within the mine sites and its immediate areas of influence and establish and regenerate the sites to acceptable land use that conforms to the pre-mining land use or land use that enhances land utilisation.

The post-mining land use objectives will be determined through stakeholder engagement during the operational phase, and the final land use will be compatible with community expectations and needs.

A comprehensive monitoring programme is required to be prepared and shall include physical observation and assessment of reclaimed sites. Closure and post-closure monitoring shall aim to ensure that the mine area after closure does not suffer from residual mine effects and to satisfy statutory authorities on both environmental and public safety of areas decommissioned.

Cardinal proposes to finance the closure, decommissioning and reclamation of the Project from the annual operations budget which is included as part of sustaining capital expenditure in Cardinal's financial model.

1.21 Cost estimates

The information presented in this Section is derived from Lycopodium (2019c), Golder (2019b) and Cardinal.

1.21.1 Capital costs

The capital cost ("**Capex**") estimate for the Namdini Gold Project has been compiled by Lycopodium with input from Knight Piésold on water infrastructure and tailings storage.

The FS capital cost estimate was completed to an accuracy of +15% to -5% for both Plant and Infrastructure Capital Costs, based on open pit mining of the Ore Reserve.

The proposed plant comprises crushing, SABC milling, gravity, flotation, concentrate regrind, high-shear oxidative pre-treatment, and CIL cyanidation of the flotation concentrate.

Cardinal compiled the Mining and Owner's costs. The mining establishment cost was based on information provided by in-country mining contractors. The mining equipment is identified in Section 1.16.1, 16.12 and detailed in 16.13. The capital costs include Owner's costs and contingency as calculated by Lycopodium.

The capital cost estimate is summarised in Table 10 for which the detail is available separately to Cardinal.

Table 10: Capital estimate summary (US$, 1Q19, +15/-5%)

Main area	US$'000
Construction distributables	41,699
Treatment plant costs	149,727
Reagents and Plant services	28,562
Infrastructure	60,138
Mining Services	6,253
Management costs	36,313
Owner's Project costs	25,207
Subtotal – Process Facility & Infrastructure	**347,911**
Contingency	42,196
Taxes & duties	*
Escalation	Excl.
TOTAL PLANT DEVELOPMENT	**390,107**

Note: * Taxes of $37M are excluded and are subject to negotiation and/or deferment under a Development Agreement.

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 including 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 requests 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 infrastructure items that were subsequently costed by Lycopodium.

The Capex estimate includes:

- Direct costs of the Project development including infrastructure
- 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
- HV Powerline to the process facility and switchyard
- Taxes and duties (subject to Development Agreement).

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

1.21.2 Operating costs

The operating costs ("**Opex**") have been developed by:

- Lycopodium – Processing and General and Administration costs
- Golder – Mining costs
- Cardinal – Owner's costs.

Operating costs have been determined for a throughput rate of 9.5 Mtpa operating 24 hours per day, 365 days per year at a primary grind size of P_{80} 130 µm and a flotation concentrate regrind size of P_{90} 9 µm.

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

- OMC modelling for crushing and grinding energy and consumables, based on comminution testwork
- The LOM expected mass recovery of gold to flotation concentrate is 7.5% based on testwork managed by Cardinal
- Flotation reagent consumption based on testwork managed by Cardinal
- CIL reagent consumption based on based on testwork managed by Cardinal
- Calculated reagent usage for cyanide destruction and arsenic precipitation prior to testwork
- Typical industry data from equipment vendors
- Budget pricing from Vendors or Lycopodium's database of unit prices for consumables
- Lycopodium's database of costs for similar sized operations
- Mining costs were solicited from five 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 increases in the depth of mining in line with standard mining cost principles.

1.21.3 Processing and G&A costs

Estimated process operating costs per tonne of ore processed (138.6 Mt of ore) at a throughput of 9.5 Mtpa are shown in Table 11.

Table 11: Processing operating cost estimate summary (±15%,1Q19)

Opex – Process costs	US$ M/y	US$/t	% of cost
Power[1]	49.5	5.21	45%
Operating consumables	48.0	5.06	44%
Maintenance	8.0	0.84	7%
Laboratory	1.9	0.20	2%
Process & maintenance labour	2.7	0.28	2%
Total Processing	**110.1**	**11.59**	**100%**
Administration labour	7.7	0.82	50%
General & administration costs	7.2	0.76	47%
Camp power[1]	0.4	0.05	3%
Total G&A	**15.4**	**1.62**	**100%**
TOTAL – Process	**125.4**	**13.21**	

Notes: [1]The average power tariff for the LOM is US$0.12/kWh. Cardinal's financial model uses a third party power offer at a variable tariff as detailed in Table 168 and Table 169

Costs and totals are rounded.

1.21.4 Contract mining costs

Contract mining costs per tonne of ore processed (138.6 Mt of ore) at a throughput of 9.5 Mtpa are shown in Table 12.

Table 12: Contract mining costs

Item	Cost/t milled (US$)
Variable mining cost – Waste	5.08
Variable mining cost – Ore	3.53
Management fees	1.68
Services	0.30
Total	**10.59**

The variable mining cost for ore and waste includes drill and blast (i.e. production drill and blast, presplits, blast movement monitors and drains holes), load and haul, road and other ancillary works. The variable mining cost for ore also includes rehandle of ore from stockpiles to the ROM and crusher.

Management fees include contractor fixed monthly management fees, supervision, explosives management, training, dewatering, health and safety, freight, communication, vehicles, transport and other general costs.

Services includes the mine dispatch system, contractor mobilisation and demobilisation, survey, dewatering, technical services running costs and other costs (e.g. consultants).

1.21.5 Owner's costs

Owner's costs per tonne of ore processed (138.6 Mt of ore) at a throughput of 9.5 Mtpa are shown in Table 13.

Table 13: Owner's costs

Opex – Owner's costs	US$/t
Grade control	0.76
10% ROM rehandle	0.07
Owner's G & A	0.60
TOTAL	**1.43**

1.21.6 Sustaining capital costs

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. These costs include rehabilitation and mine closure.

Sustaining capital costs per tonne of ore processed (138.6 Mt of ore) at a throughput of 9.5 Mtpa are shown in Table 14.

Table 14: Sustaining costs

Sustaining capital cost	US$ '000	US$/t
Flotation TSF sustaining capital	102,696	0.74
CIL TSF sustaining capital factor	20,910	0.15
Vehicles & equipment sustaining capital	6,678	0.05
TSF sustaining costs – mining waste to TSF	27,513	0.20
Mine closure cost	23,789	0.17
TOTAL	**181,586**	**1.31**

1.22 Economic analysis

1.22.1 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, the estimation of the 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.

1.22.2 Methodology

The economic assessment was carried out Pre-Tax and has assumed a total allowance for royalties of 5% of the gross gold price.

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 5% discount rate applied is appropriate to the specific project and 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 annual 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.

The Free Cash Flow calculation is the total revenue generated, minus total costs. The calculation therefore is total recovered ounces (4.196 Moz), multiplied by the gold price selected (US$1,350/oz), minus the total cost per ounce (US$/oz). The Pre-Tax cash flow is before tax is deducted, while the Post-Tax cash flow is after tax is deducted.

1.22.3 Financial summary

1.22.3.1 Key economic results

The key economic results based on a gold price of US$1,350/oz are provided in Table 15.

Table 15: Key economic results

Key economic results		Unit	9.5 Mtpa
Capital cost (*including 11% contingency*)		US$ M	390
All-in Sustaining Cost (AISC)[1]	*Starter Pit*	US$/oz	585
	Life of Mine		895
Total Project payback		Months	21
Pre-Tax NPV US$ (@ 5% discount)[2]		US$ M	914
Post-Tax NPV US$ (@ 5% discount)[2]		US$ M	590
Pre-Tax IRR		%	43
Post-Tax IRR		%	33

Notes: [1] Cash Costs + Royalties + Levies + Life of Mine Sustaining Capital Costs (World Gold Council Standard)
Royalties calculated at a rate of 5.5% and a corporate tax rate of 32.5%; both subject to negotiation and expected to be finalised over coming months.

1.22.3.2 Production summary – Starter Pit and Life of Mine

A summary of the expected production is provided in Table 16 for the Starter Pit and in Table 17 for the Life of Mine (including the Starter Pit).

Table 16: Starter Pit proposed production summary

Key Estimated Production Results	Unit	Feasibility Study
Gold price	US$/oz	1,350
Gold produced *(average for full production years)*	koz/y	361
Gold head grade *(Starter Pit, Ore Reserve)*	g/t Au	1.41
Gold recovery *(Starter Pit)*	%	85
Strip ratio *(Starter Pit)*	W:O	0.9 : 1
Ore tonnes mined *(0.5 g/t cut-off grade)*	Mt	47
Waste tonnes mined	Mt	43
Starter Pit life *(including ramp-up)*	Months	27
Total Project payback	Months	21

Note: [1] Cash Costs + Royalties + Levies + Life of Mine Sustaining Capital Costs (World Gold Council Standard)

Table 17: LOM proposed production summary, including the Starter Pit

Key Estimated Production Results	Unit	Feasibility Study
Gold price	US$/oz	1,350
Gold produced *(average for LOM full production years)*	koz/y	287
Gold produced *(LOM)*	Moz	4.2
Gold head grade *(LOM, Ore Reserve)*	g/t Au	1.13
Gold recovery *(LOM)*	%	83
Strip ratio *(LOM)*	W:O	1.9 : 1
Ore tonnes mined *(0.5 g/t cut-off grade)*	Mt	138.6
Waste tonnes mined	Mt	263
Mine life *(Including ramp-up and mine closure)*	years	15
Capital cost *(including 11% contingency)*	US$ M	390
Total Project payback	Months	21

1.22.4 Economic sensitivity studies

Based upon Life of Mine production and cost parameters, the Post-Tax NPV sensitivities are shown for the 9.5 Mtpa throughput in Table 18.

Table 18: Post-Tax NPV (in US$ M) and gold price sensitivity for the 9.5 Mtpa throughput

Post Tax Real Discount Rate (%)	Gold Price (US$/oz)				
	US$1,150	US$1,250	US$1,350	US$1,450	US$1,550
0	441	720	**980**	1,257	1,527
5	228	411	**590**	769	950
10	105	233	**355**	486	614

Note: All NPVs are Post-Tax values in US$ M

Bar charts in Figure 12 to Figure 15 show the expected Pre-Tax and Post-Tax economic sensitivities for the 9.5 Mtpa throughput at a gold price of US$1,350/oz.

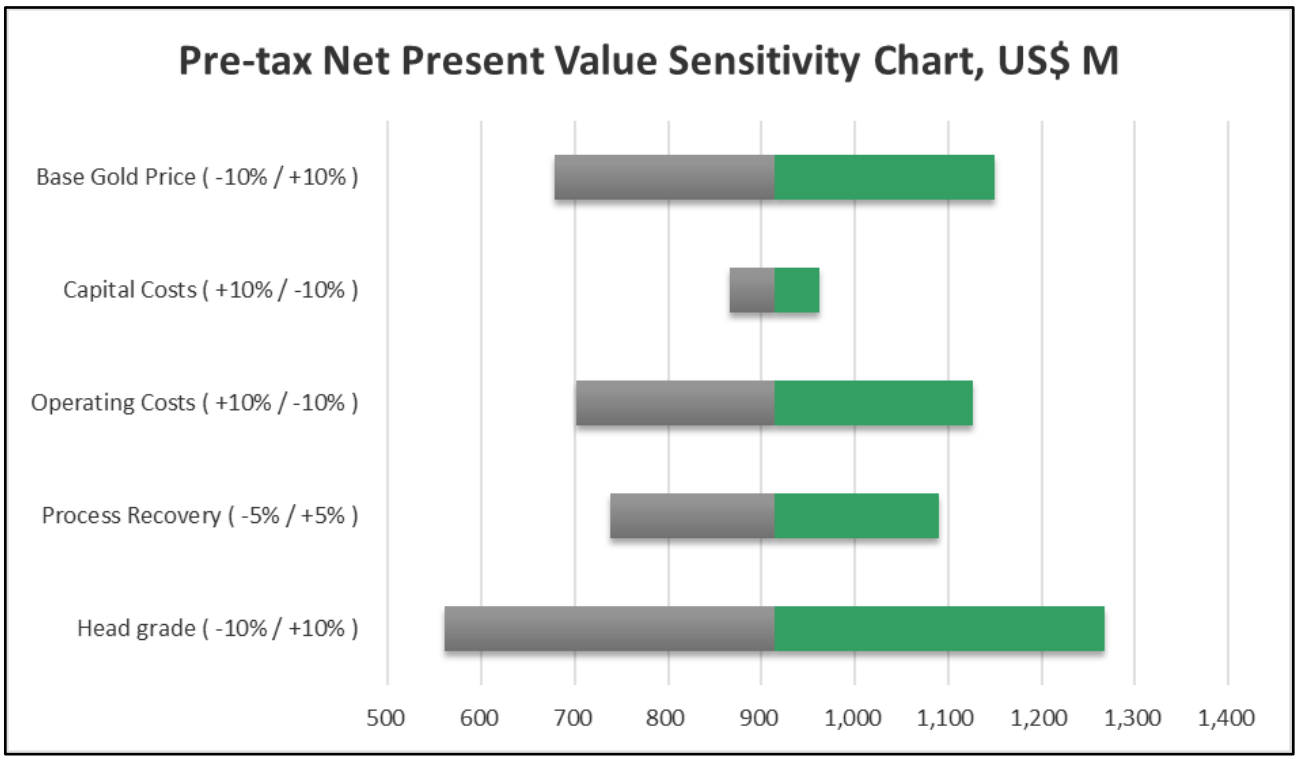


Figure 12: Pre-Tax NPV sensitivity at 5% discount (US$ M) for the 9.5 Mtpa throughput

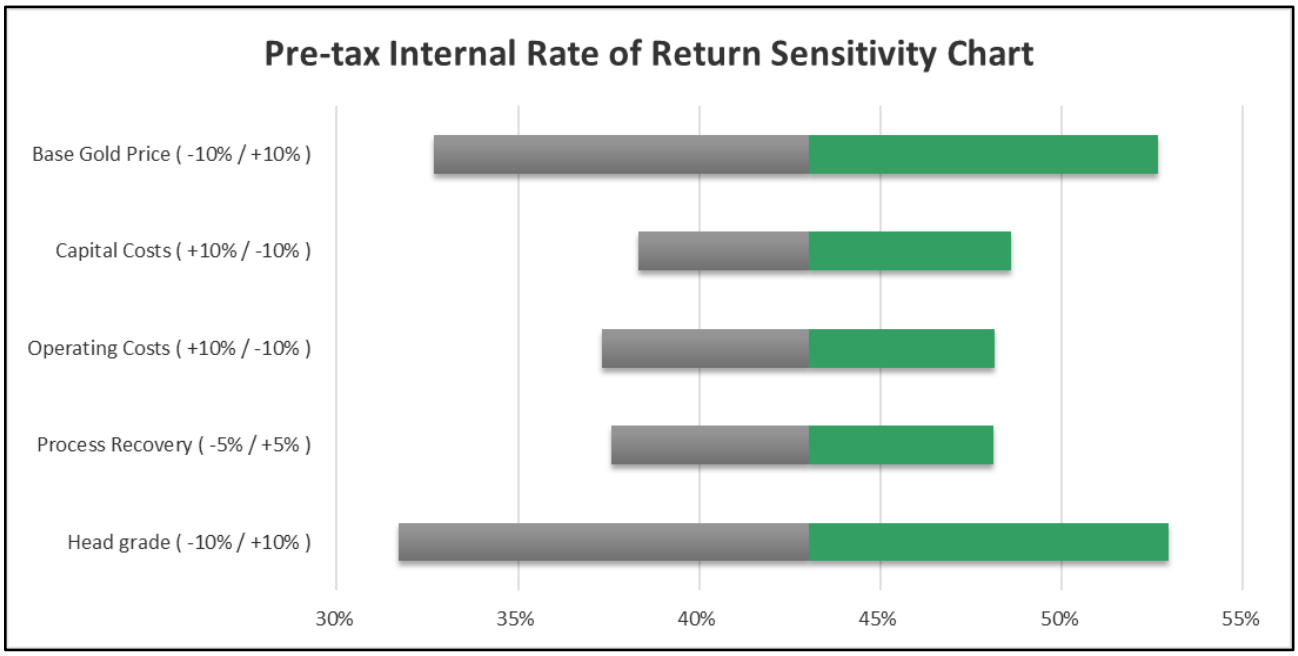


Figure 13: Pre-Tax IRR (%) sensitivity for the 9.5 Mtpa throughput

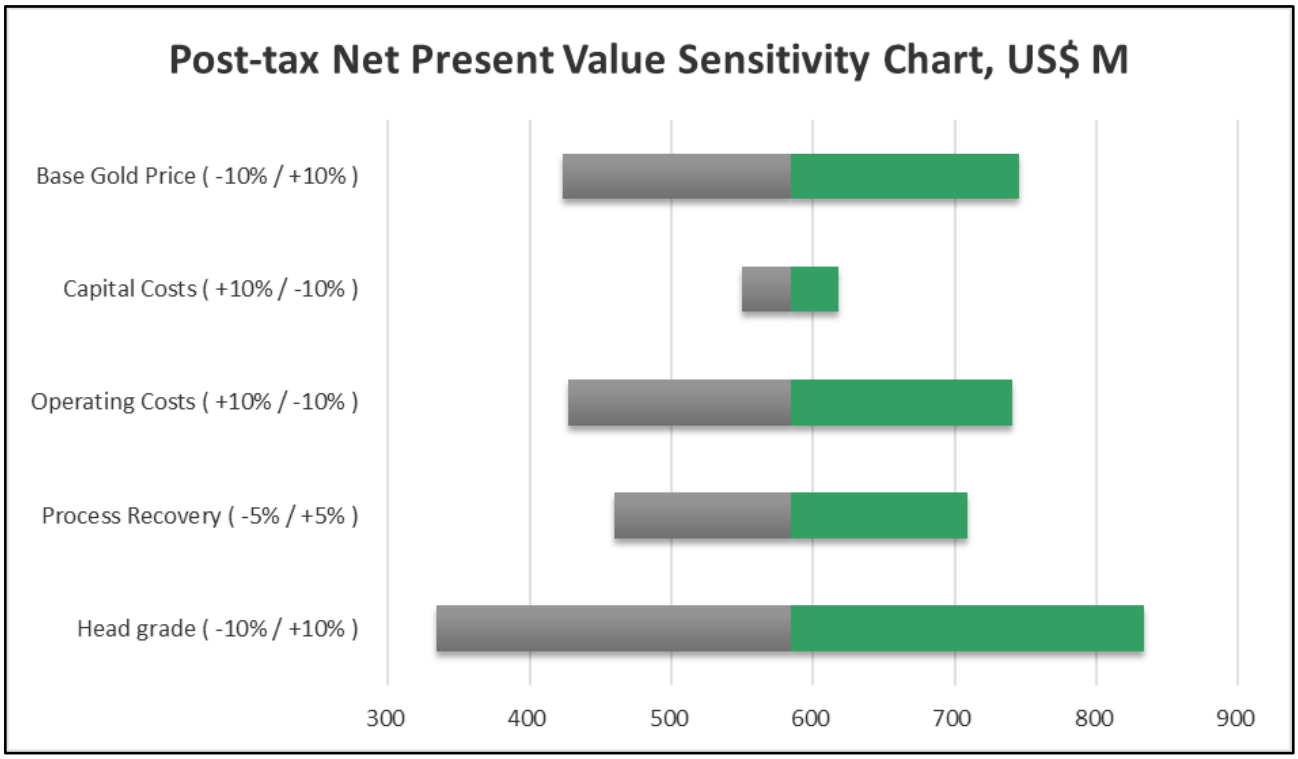


Figure 14: Post-Tax NPV sensitivity at 5% discount (US$ M) for the 9.5 Mtpa throughput

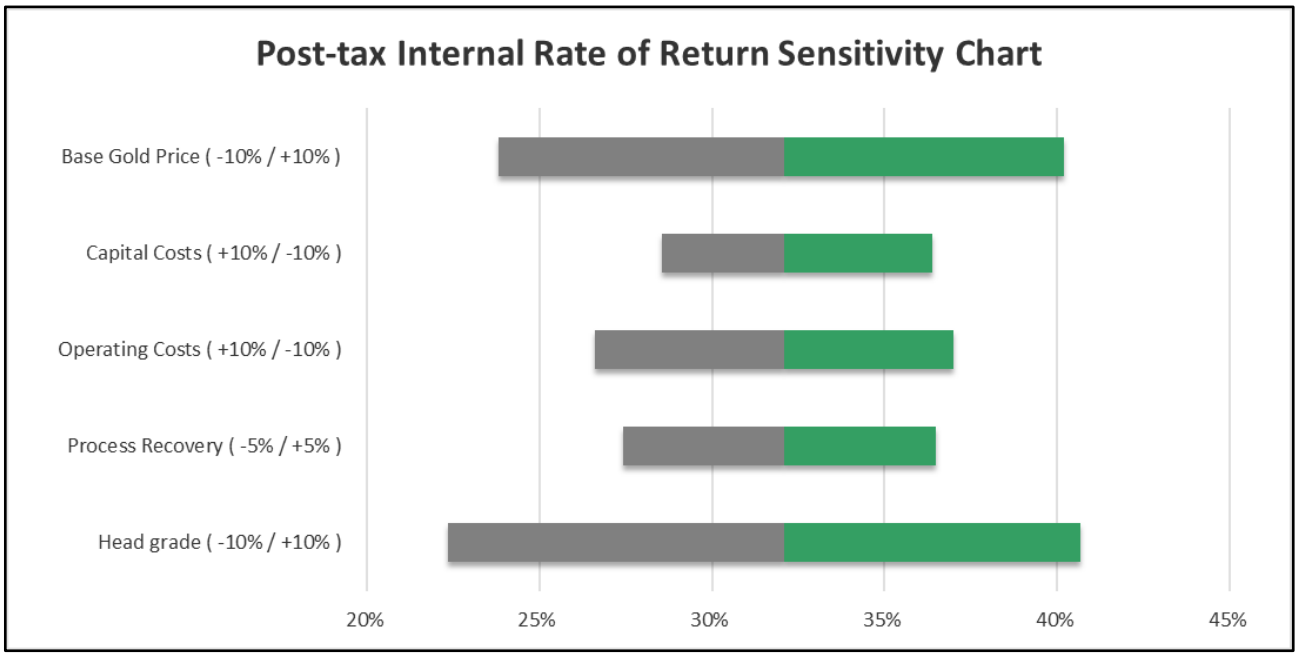


Figure 15: Post-Tax IRR (%) sensitivity for the 9.5 Mtpa throughput

Figure 16 shows the annual gold production against the accumulative post tax cash flow for the 9.5 Mtpa throughput at a gold price of US$1,350/oz. Figure 17 shows the annual gold production against the All-in Sustaining Cost (AISC) in US$/oz for the 9.5 Mtpa throughput.

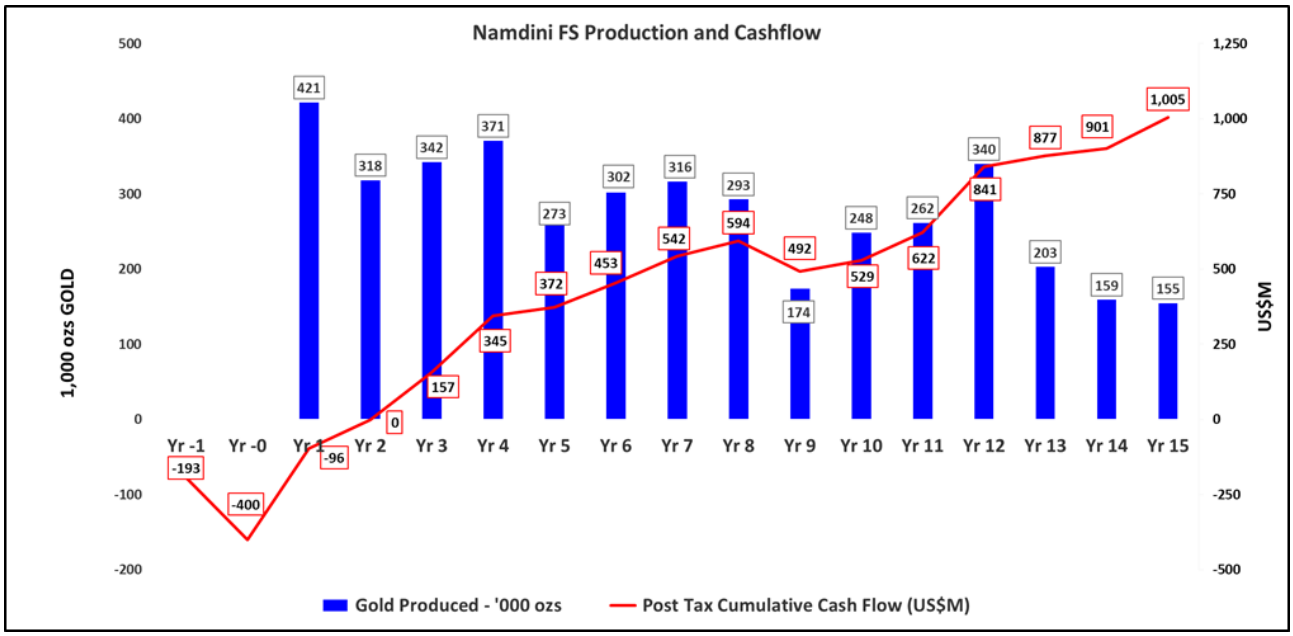


Figure 16: Namdini FS ounce production and net cash flow (100%)

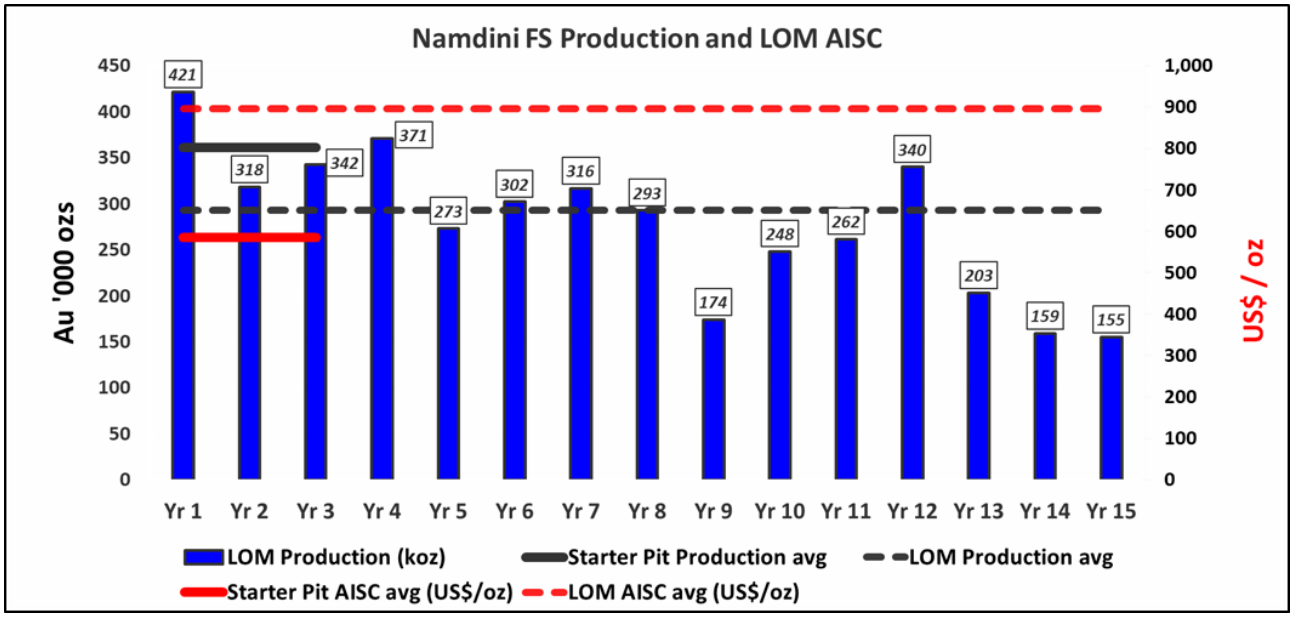


Figure 17: Namdini FS ounce production & All-in Sustaining Cost (100%)

1.22.5 Funding

Cardinal's Board has approved this Feasibility Study and recommends progressing the Project to construction pending successful completion of financing activities.

- The gold price is currently trading at approximately US$1,500/oz, which compares favourably to the Project's financial assumption of US$1,350/oz.
- The Company has been actively working with its debt advisor Cutfield Freeman to secure Project finance on competitive terms and to ensure that Project finance can be secured in a timely manner following completion of the Feasibility Study. A number of leading financial institutions are involved in the process which is well advanced and is continuing. Cardinal looks forward to providing more details in the future.
- In addition to traditional financing solutions, the Company will also evaluate alternatives to bring the project into production.
- The strong production and economic outcomes delivered in the Namdini FS are considered by Cardinal's Board to be sufficiently robust to provide confidence in the Company's ability to secure project funding.
- A final financing decision will be made at a time assessed as the most appropriate and beneficial to the Company.
- In the meantime, the Company remains in a strong financial position with cash reserves of approximately A$27 million.

1.23 Adjacent properties

The Namdini Gold Project site is located approximately 6 km southeast of the operating Shaanxi underground mine, which uses grid power.

1.24 Other relevant information

In the opinion of the Authors, all relevant and material information is provided in this NI 43-101 Technical Report, based on the Feasibility Study documentation.

1.25 Next stages

Based on the positive FS outcome, the Cardinal Board has approved initiatives that will further de-risk the Project. These include immediate advancement of the Namdini Gold Project to the FEED and Early Works Construction stages, plus advancement of the Relocation Action Plan subject to final project finance.

The preliminary Project development schedule is shown in Table 19 and a Project Gantt Chart is provided as Figure 18. This is subject to available funding based on the positive outcome of this FS and to favourable timelines for permitting, contracting and procurement.

Table 19: Proposed Project development schedule (subject to financing*)

Milestone	Target timeline
Commence Front End Engineering and Design (FEED)	Q4 2019
Advance Relocation Action Plan to completion	H1 2020
Target production commencement	H2 2022

Note: The Company's proposed project development schedule assumes that the Company is fully funded to proceed throughout and beyond the FEED process, into early works and full construction. The actual development schedule will depend upon the manner and timing of the Company's financing plans. Dates are therefore indicative only.

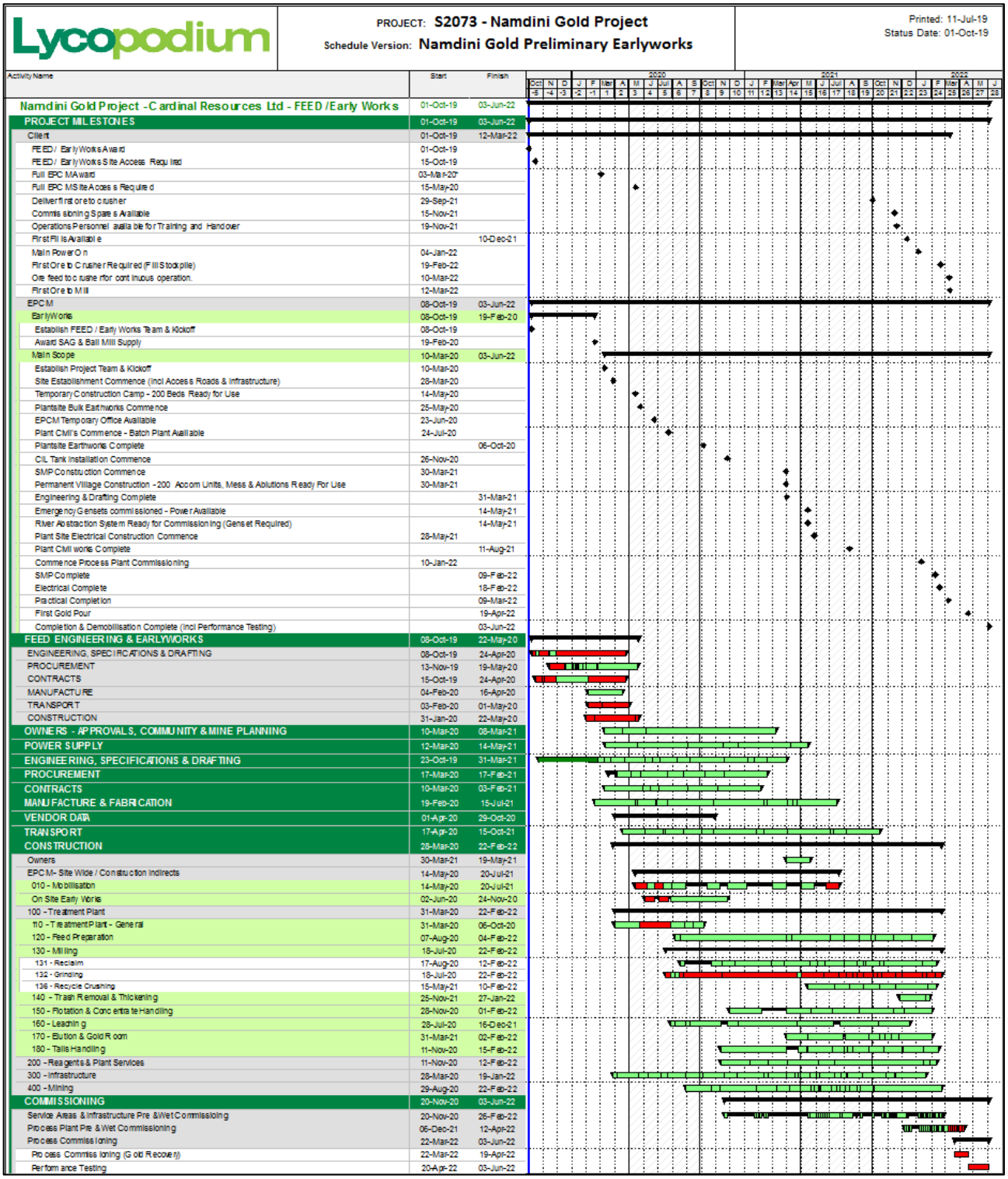


Figure 18: Project schedule Gantt timeline (source: Lycopodium, 2019d)

2.0 INTRODUCTION

2.1 Scope of work

This Technical Report, documenting the 2019 Feasibility Study of the Namdini Gold Project (the FS), was prepared for Cardinal by Golder and Lycopodium, with reliance on other expert consultants (the Authors).

Cardinal is a gold exploration and development company based in Western Australia 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.

The Preliminary Feasibility Study (PFS) was released to the ASX and TSX on 25 October 2018 (Cardinal, 2018d) and is referred to in this Report as the PFS Report or Golder (2018b). The PFS 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 could be planned.

This FS was commissioned to better evaluate the costs for mining and processing to demonstrate the economic viability of Cardinal proceeding with development of the proposed gold mine.

This Technical Report was produced as part of the FS and published parts are compliant for Public Reporting under Canadian National Instrument ("**NI**") 43-101 in Canada (NI 43-101, 2014) and under the JORC Code (2012) in Australia. It was prepared in accordance with the requirements of:

- disclosure and reporting requirements of the Toronto Stock Exchange 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).

This Technical Report is subject to the material assumptions and market conditions outlined in Section 1.2 and the Important Notice at the start of this Report. As such it provides **forward-looking information** and there is no certainty that the conclusions of the FS 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 The Namdini Gold Project

The Namdini Gold Project (the "**Project**" or "**Namdini**") is located in the Upper East Region of Ghana approximately 50 km southeast of the regional centre of Bolgatanga, close to the southern border of Burkina Faso.

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 134 t trucks. The ore will be fed to a large sulphide flotation circuit with concentrate fed to a conventional CIL process plant to enhance project economics. 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 the Ghanaian Government, the owners, shareholders, employees, and local communities.

2.3 Statement of independence

Lycopodium is an independent engineering company contracted by Cardinal to carry out this FS. Neither that company, 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. The 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 Lycopodium, nor the Authors, is or is intended to be a director, officer or other direct employee of Cardinal.

In the preparation of this Independent Technical Report Lycopodium has used information provided by Cardinal and other experts, as described in Table 20 in Section 3.0. The Authors have verified this information, making due inquiry of all material issues that are required in order to comply with NI 43-101 and JORC Code requirements.

There is an ongoing consultancy agreement between Lycopodium and Cardinal regarding conducting further work for Cardinal as the Project progresses.

The positive result of this FS 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, with ongoing drilling and preparatory works, while meeting all required permits and approvals, to add significant value to the Upper East Region of Ghana.

Information on the sources used to prepare this Report is provided in Section 3.3.

The dates of the site inspections by the QPs and others is provided in Section 3.4.

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, Lycopodium 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 Project, and changes to regulations affecting them.

Although the Authors believe the expectations reflected in the forward-looking statements to be reasonable, they do not guarantee future results, levels of activity, performance or achievements.

2.5 Use of the term ‘ore’ in this FS

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 FS.

The term ore is used in the mining and processing sections of this FS 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 Associates Pty Ltd 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 optimization was selected and used to generate engineering pit designs that included a Starter Pit and Phases for the Life of Mine (“**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 Code Table 1 is not required, but it is provided as Appendix 1 in press releases by Cardinal to the ASX, e.g. Cardinal (2019c).

3.0 RELIANCE ON OTHER EXPERTS

3.1 Introduction

This NI 43-101 Technical Report (this "**Report**") was prepared by Lycopodium for Cardinal for Public Reporting. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available 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 Report.

For the purpose of this Report the Authors have relied on ownership information and other local knowledge provided by Cardinal. The Authors have no independent information regarding property title or mineral rights for the Namdini Gold Project and express 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 the FS comprised: 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 development of the Project as an operating gold mine and processing plant.

Cardinal commenced its FS in January 2019 to further advance the Namdini Gold Project. This consisted of an Owner's Team and continuation with previous and new consultants to assist with the phased development of the Namdini Gold Project. The independent specialist consultants and their roles are shown in Table 20.

Table 20: The Namdini FS independent specialist consultants – the Study Team

Company	Role
Lycopodium Minerals Pty Ltd ("**Lycopodium**")	Feasibility Study manager. Process plant and associated infrastructure. Capital cost and process operating cost estimates.
Golder Associates Pty Ltd ("**Golder**")	Mine design, planning, optimization and scheduling. Mine operational costs. Compilation of the JORC, NI 43-101 and Feasibility Study reports.
Golder Associates Africa (Pty) Ltd	Hydrology and hydrogeological engineering studies.
Golder Associates Ghana Limited	Geotechnical engineering studies.
Knight Piésold Consulting Pty Ltd ("**Knight Piésold**")	Tailings Storage Facility and selected infrastructure design.
ALS Laboratory (Perth) ("**ALS**")	Metallurgical testwork to support the process design criteria.
Orway Mineral Consultants Pty Ltd ("**OMC**")	Comminution data analysis, crushing and grinding option studies.
Independent Metallurgical Operations Pty Ltd ("**IMO**")	Metallurgical testwork management, analysis and process flowsheet development.
Maelgwyn Mineral Services Africa Pty Ltd ("**MMSA**")	Aachen^{TM} process metallurgical optimization.
MPR Geological Consultants Pty Ltd ("**MPR**")	Mineral Resource modelling of the Namdini Deposit.
Orefind Pty Ltd ("**Orefind**")	Geology and deposit structural genesis.
NEMAS Consult Pty Ltd ("**NEMAS**")	Environmental Impact Assessment Study.
Geosystems Consulting Limited ("**Geosystems**")	TSF Environmental Impact Assessment Study.
MKM Social Limited ("**MKM**")	Socio-Economic Study and Resettlement Action Plan.

Company	Role
Sebbag Group International Pty Ltd ("**SGI**")	Mine design review.
Whittle Consulting Pty Ltd ("**WC**")	Enterprise optimization.
BDO Advisory Pty Ltd ("**BDO**")	Financial model integrity and reviewer.

3.3 Sources of information

Sources used in the compilation of this NI 43-101 Technical Report include:

- Namdini Gold Project Mineral Resource Estimate Study Report. Prepared for Cardinal Resources Limited by ERGM Consulting Pty Ltd (Gossage, 2017) at 17 February 2016, for which the estimate was released publicly to the ASX in Cardinal (2016)
- Technical Report on the Namdini Gold Project, Ghana, West Africa, by Roscoe Postle Associates Inc (RPA, 2017) describing the Mineral Resource estimated at 5 April 2017 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; 2019)
- The previous Preliminary Economic Assessment on the Namdini Gold Project dated 5 February 2018 released publicly to the TSX under NI 43-101 (Golder, 2018a)
- The previous Preliminary Feasibility Study on the Namdini Gold Project dated 25 October 2018 released publicly to the TSX under NI 43-101 (Golder, 2018b)
- Studies by Lycopodium on the process plant design and costings (Sections 16.15 and 21.0)
- Studies by Lycopodium and Knight Piésold on the infrastructure, including tailings storage, and costings (Sections 18.0 and 21.0)
- Report on the Environmental Impact Statement 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
- Geotechnical (rock mechanics) studies supporting the preliminary Namdini Pit design (Golder, 2019d; 2019e; Golder Ghana, 2018) by Reginald Hammah (see Section 16.4).

The Authors have reviewed the independent technical reports and are reasonably assured, having made due inquiry, 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 in Perth visited the Namdini Gold Project site on 11 to 14 January 2017.
- Dr Frank Anim of Nemas Consult Ltd in Accra visited the site on 1 to 4 May 2017 and again in April 2019.
- Dr Charles F.A. Akayuli of Geosystems Consulting Limited visited site on 16 April 2019, 20 June 2019 and 12 July 2019.

- 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 in Perth visited the Namdini Gold Project site from 11 to 15 December 2017.
- Simon Smith, Civil Engineer of Knight Piésold Pty Ltd in Perth visited the Namdini Gold Project from 11 to 15 December 2017 and again from 18 to 25 February 2019.
- 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).
- Mark Addo of MKM Social Limited visited site on 6 to 7 March 2019, 12 July 2019 and 1 to 20 August for the socio-economic study and the Resettlement Action Plan.
- Givarn Singh (Water Resources Engineer) of Golder Associates Ghana Limited in Accra visited site from 3 to 12 September 2018 assisted by Edna Addai.
- Michael Cicanese of Lycopodium Minerals Pty Ltd completed a site review from 18 to 19 February 2019.

The QP for Sections 17, 18, 21 and 24 of this Report, David Gordon has not personally inspected the Namdini Gold Project site. He has relied on the information about site setting and infrastructure provided by his colleagues Michael Cicanese of Lycopodium and Simon Smith of Knight Piésold who have made site inspections and have been involved in developing the relevant parts of this Report.

The QP for Section 13 of this Report, Daryl Evans, has not personally inspected the Namdini Gold Project site because that was not required for him to do the metallurgical testwork studies. He was involved in the selection of the samples to be tested and they were collected under his instruction by Cardinal geologists. He has relied on the information about site setting, drilling, sampling, sample preparation and QAQC provided by Nicolas Johnson as documented in this Report.

3.5 Qualified Persons and experts relied upon

The Qualified Persons ("**QP**s") identified as responsible for this FS Technical Report have written specific sections of the Technical Report and have also relied on other experts as defined in NI 43-101 (Item 3a). These other experts and the Sections for which they take responsibility are identified in Table 21.

Table 21: Qualified Persons for this FS

Company	Report section	FS 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
IMO	Section 13	Metallurgy	Daryl Evans	QP
Lycopodium	Sections 17, 18, 21 and 24	Plant Design, Infrastructure, Capital and Operating costs	David Gordon	QP
Golder	Section 14.13 to 14.14, 15, 16 and all other Sections (i.e. Sections 1 to 6, 14.14, 14.15, 19, 20, 22, 23, 25, 26 and 27)	The costings and basis for the Ore Reserve estimate (the Modifying Factors) and compilation of all other information	Glenn Turnbull	QP

Other personnel involved in preparing this FS included:

- Cardinal: Richard Bray, Bruce Lilford
- Golder: Ross Bertinshaw, Allan Blair, Rosana Crumpler, Henry Dillon, Richard Goldsmith, Dr Sia Khosrowshahi, Alan Puhalovich, David Reid, Dr Bill Shaw, Russell Staines, Dr Marilena Stimpfl, Andrew Weeks, Karl Wellwood
- Golder Africa: Trevor Coleman, Johan Jordaan, Jennifer Pretorius, Dr David Love
- 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 FS are provided at the end of this Report.

3.6 Conventions used in this Report

Where appropriate:

- **Significant conclusions material to the outcome of this study are highlighted in bold font** and collated in Section 25.0
- ***Significant recommendations for further work material to the outcome of this study are highlighted in bold italic fond***
- Generally where first used, acronyms and abbreviations are formally defined in bold font within double quotes and brackets.

4.0 PROPERTY DESCRIPTION AND LOCATION

The information in this Section is derived from the PFS (Golder, 2018b) which is a comprehensive report provided on Sedar (https://www.sedar.com).

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 km south of the Burkina Faso border (Figure 19). 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 constitutes 7.8% of the regional population. It is made up of 96 towns and villages with a settlement pattern which is predominantly rural.

The District context of the Lease boundary is shown in Figure 20.

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 mean sea level ("**AMSL**") with average elevation at approximately 190 m. Physiography is primarily savannah grassland characterised 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.

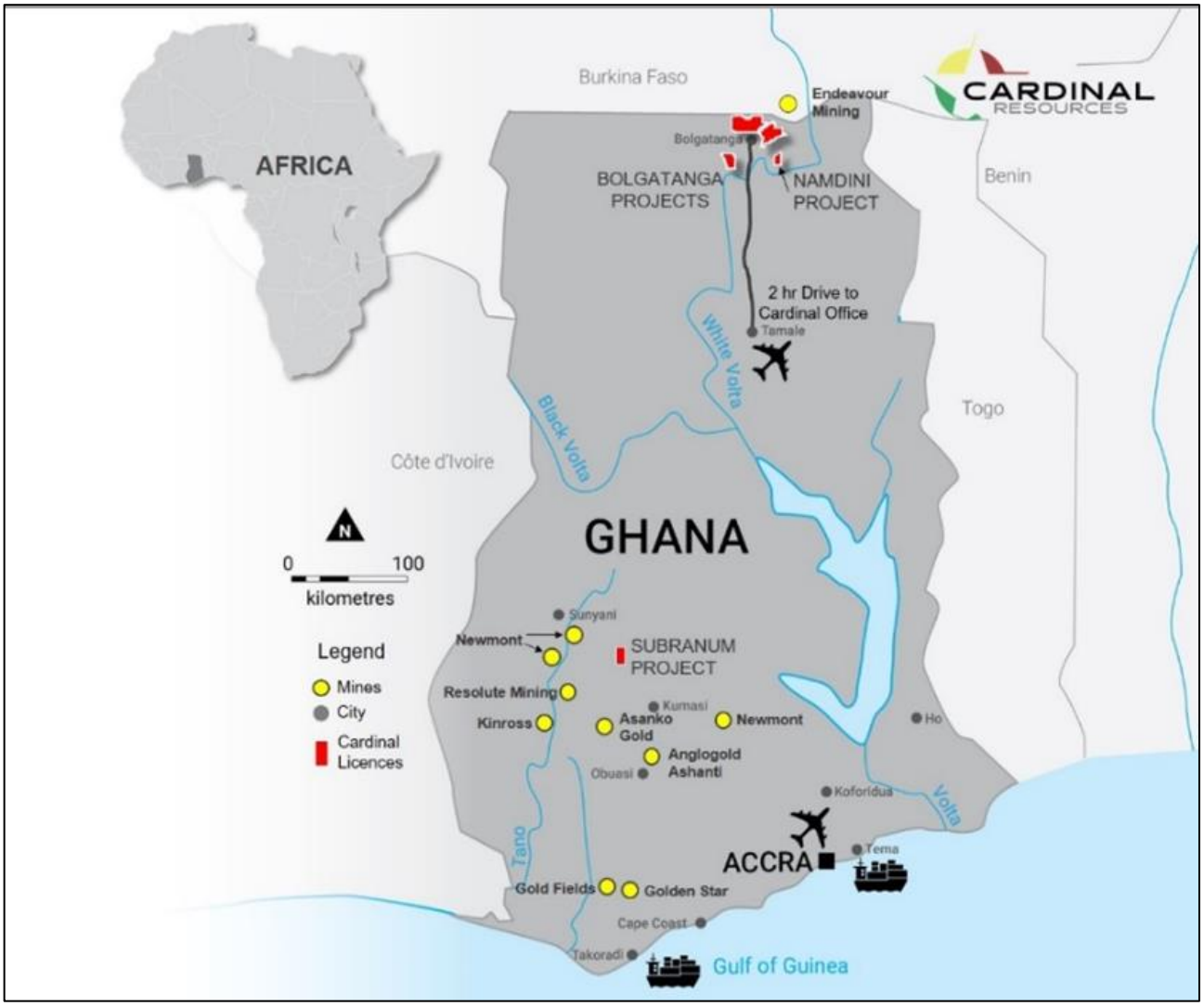


Figure 19: Location of Namdini Gold Project in northern Ghana (source: Cardinal 2019)

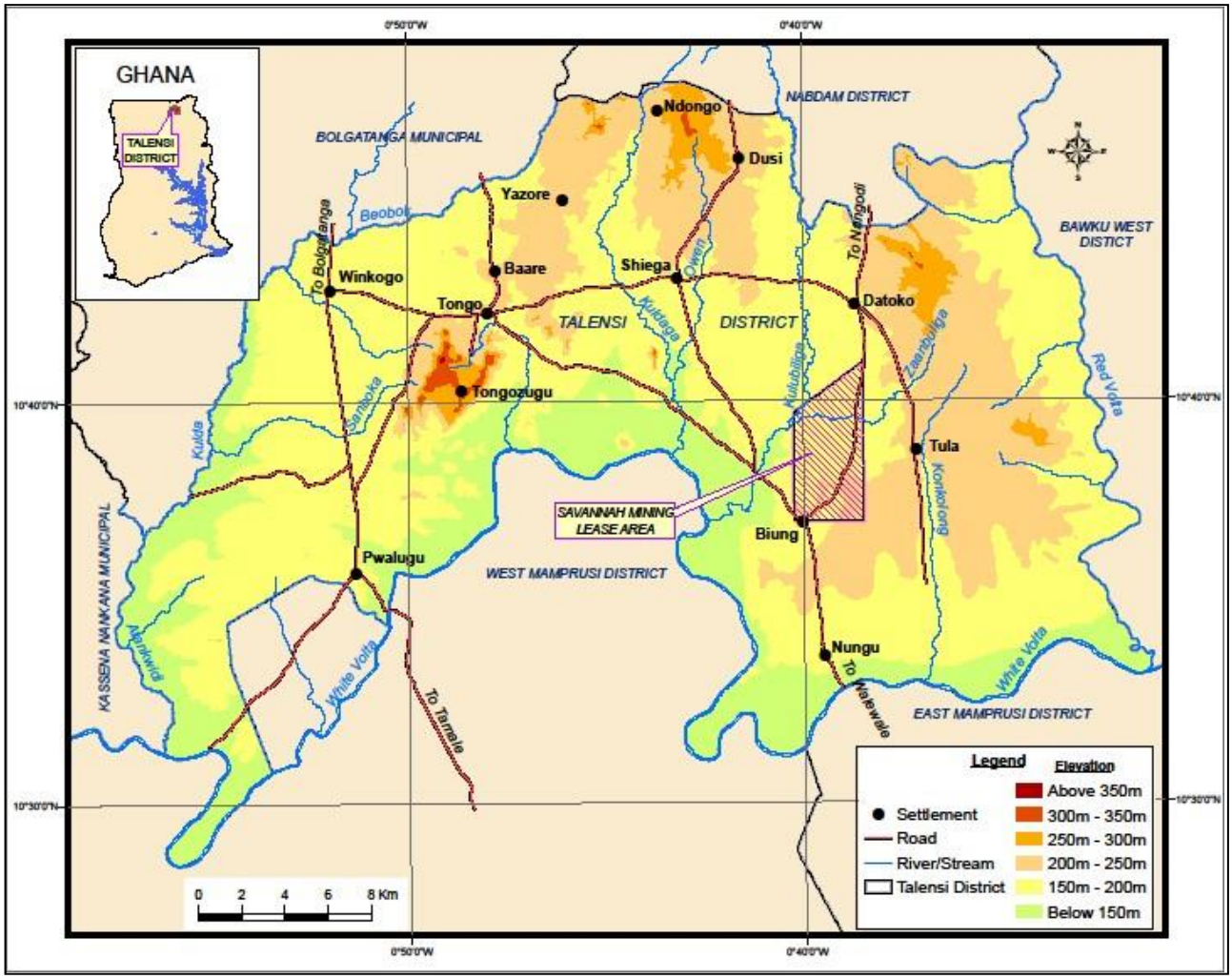


Figure 20: District map of Talensi showing the Namdini Mining Lease (source: NEMAS, 2017)

4.2 Tenement description

Savannah Mining Ghana Limited 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. An application by Savannah for a Large-Scale Mining Licence over an area of approximately 19.54 km^2 covering the Namdini Project was subsequently granted by the Minister of Lands and Natural Resources of Ghana.

During the December 2017 quarter, the 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. This Mining Licence is for an initial period of 15 years and is renewable for up to a further 30 years.

Table 22 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. The stamped survey document was viewed by Golder during the PFS.

Table 22: 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

Figure 21 shows the lease superimposed on the regional physiography image.

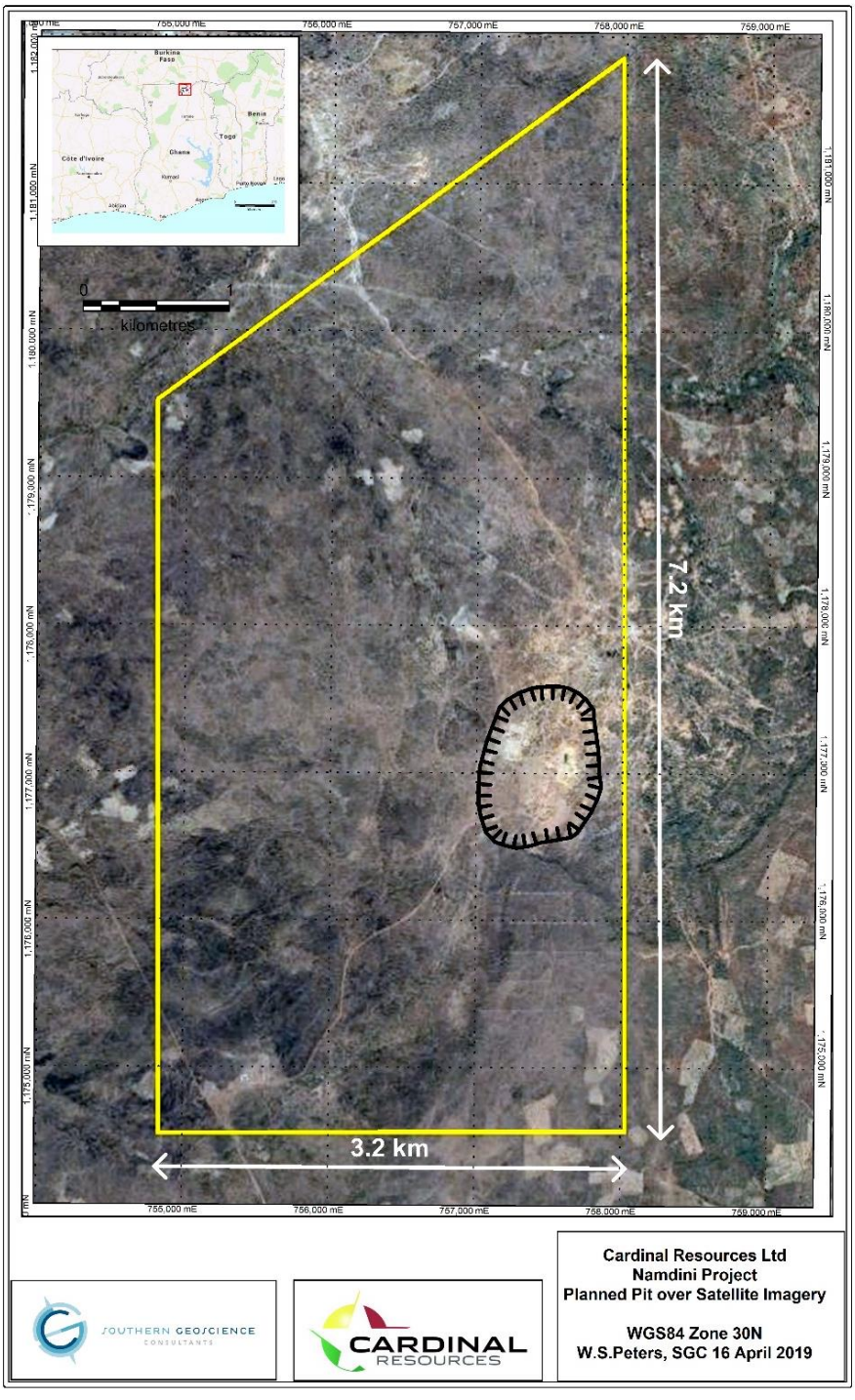


Figure 21: Namdini Gold Project lease boundary (source: Cardinal 2019)

4.3 Ownership

The following description of mineral tenure and property ownership for the Namdini deposit is derived from information in (Golder, 2018b) 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).

Transfer of the Large-Scale Mining Licence covering the Namdini Mining Lease was assigned to Cardinal Namdini as described above.

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.

The FS environmental studies included active engagement with local and state regulatory bodies and local stakeholders. Cardinal has an excellent relationship with neighbours and the granted leases cover all the required mining and processing areas. There are no title claims pending.

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 ongoing work programs on the Namdini Gold Project.

4.4 Royalties, payments or encumbrances

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 Mining 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 Mining Licence.

An assumption of 5% was made to account for all royalties in the Mineral Resource estimation (Section 14.10), the Mining cost estimates (Section 16.6) and the Economic Analysis (Section 22.3).

4.5 Environmental studies

Cardinal Namdini has recently completed an Environmental Impact Statement **(**EIS) 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 EPA issued a public notification on the proposed Namdini Gold Mining Project.

Further information regarding the EIS 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 Project. The Authors of this Report and the other signatory QPs are not aware of any specific environmental liabilities on the Project. They consider that Cardinal has all required permits to conduct the proposed work on the Project. They are not aware of any other significant factors and risks that may affect access, title, or the right or ability to perform ongoing work programs on the Project.

5.0 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE, AND PHYSIOGRAPHY

The information in this Section is derived from the PFS (Golder, 2018b) which is a comprehensive report provided on Sedar (https://www.sedar.com).

5.1 Project layout

The site layout is shown in Figure 22. 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 23. 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.

All other relevant aspects of the site layout are detailed more appropriately elsewhere in this Report including sites for the open pit mine and waste rock dumps (Section 16.0), the Process Plant (Section 17.0), and the Tailings Storage Facilities, roads, power, waste sites, and all other infrastructure (Section 18.0).

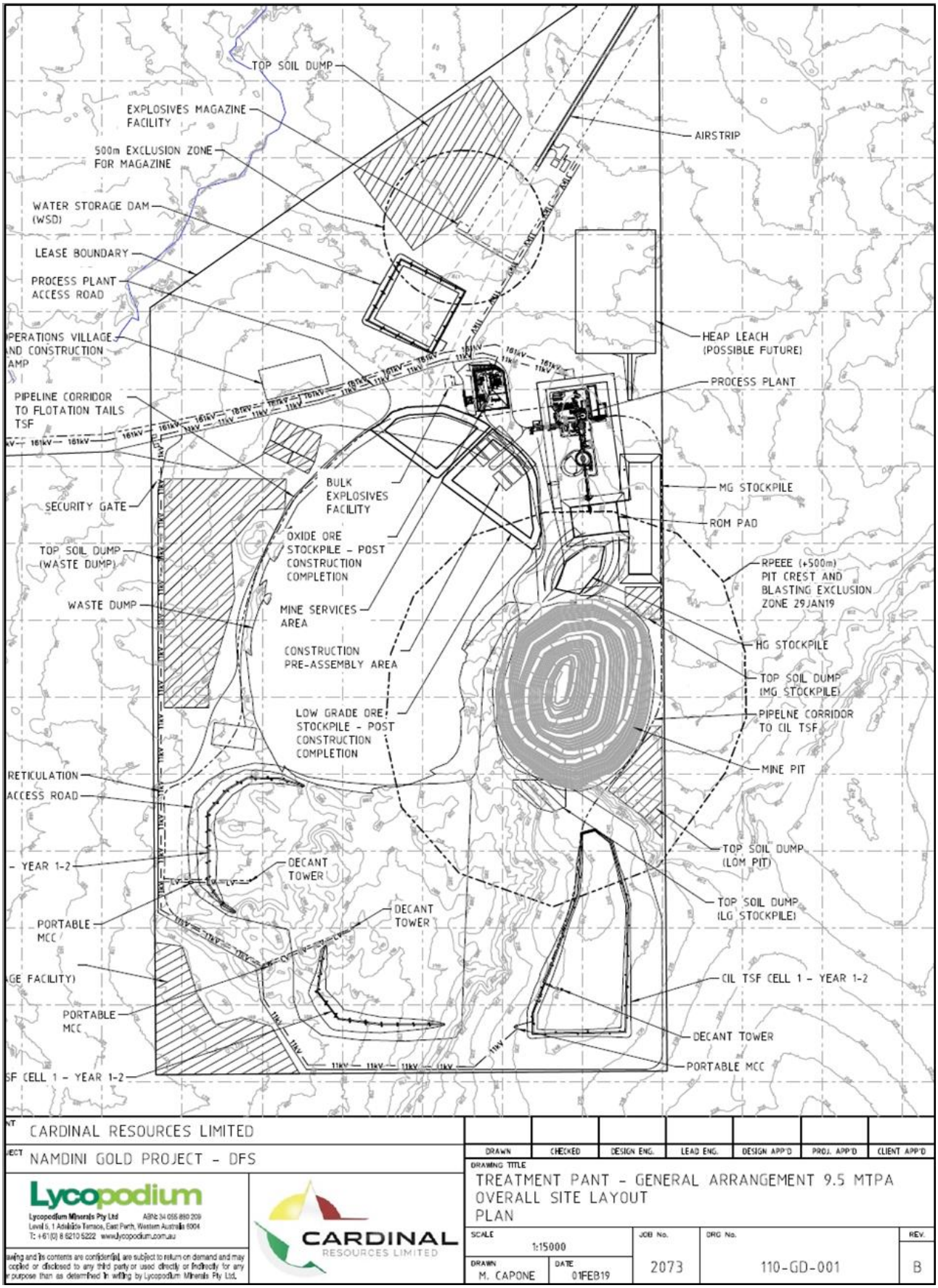


Figure 22: Namdini Gold Project infrastructure layout (source: Cardinal 2019)

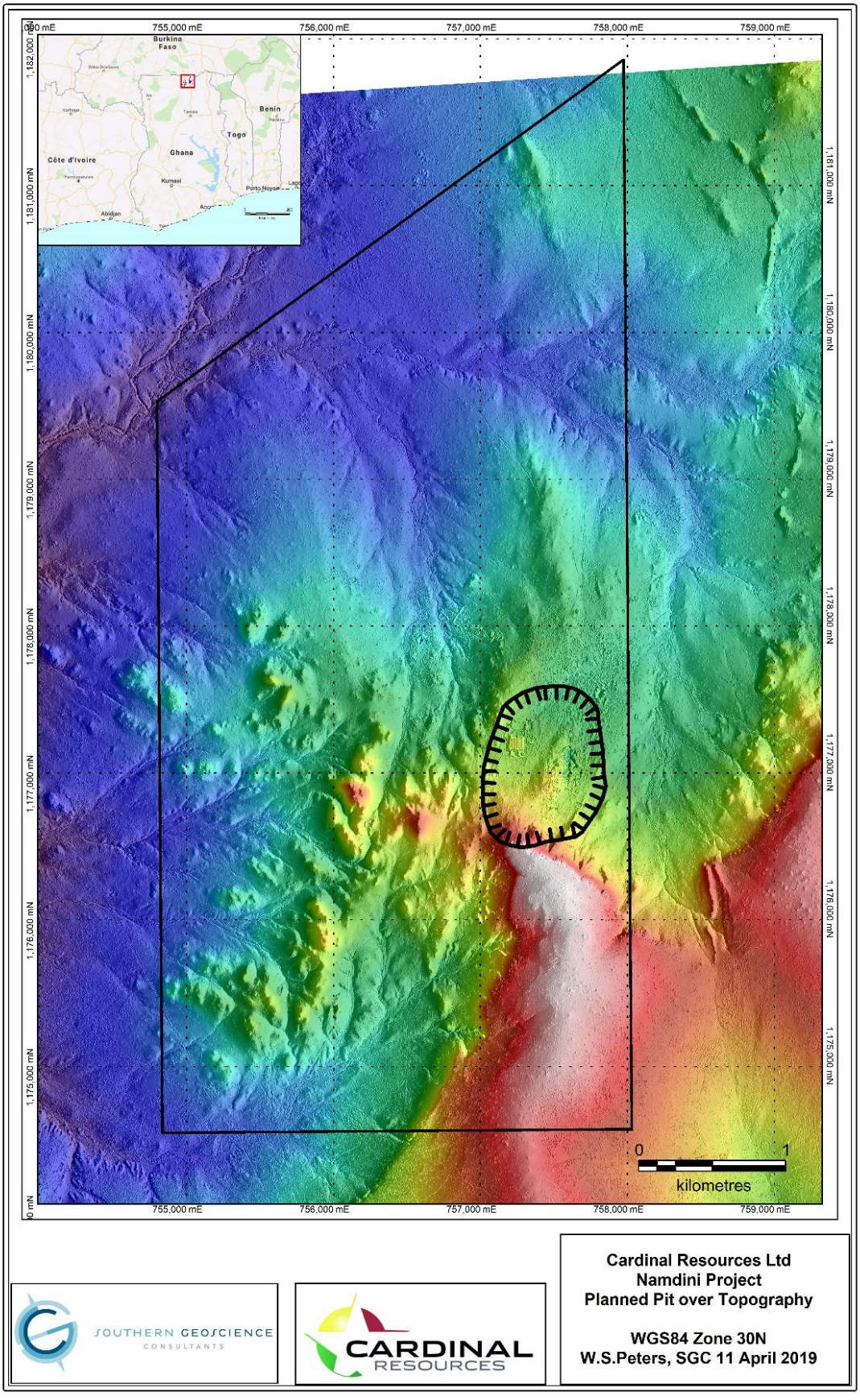


Figure 23: Project LOM Pit outline and site topography (source: Golder 2018)

5.2 Access, infrastructure and population

The following descriptions of accessibility, local resources and infrastructure are derived from (Golder, 2018b) 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.

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.

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 6 km 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 studies summarised in Section 20.3.4 (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 24) and rises to the south where the area is overlain by sediments. Elevation varies from 175 to 250 m AMSL, averaging approximately 190 m.

The topography of the district is characterised 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 25.

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 24: Namdini Gold Project site showing terrain facing north (source: Cardinal 2019)



Figure 25: View of Namdini Gold Project site facing south (source: Cardinal 2019)

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 26).

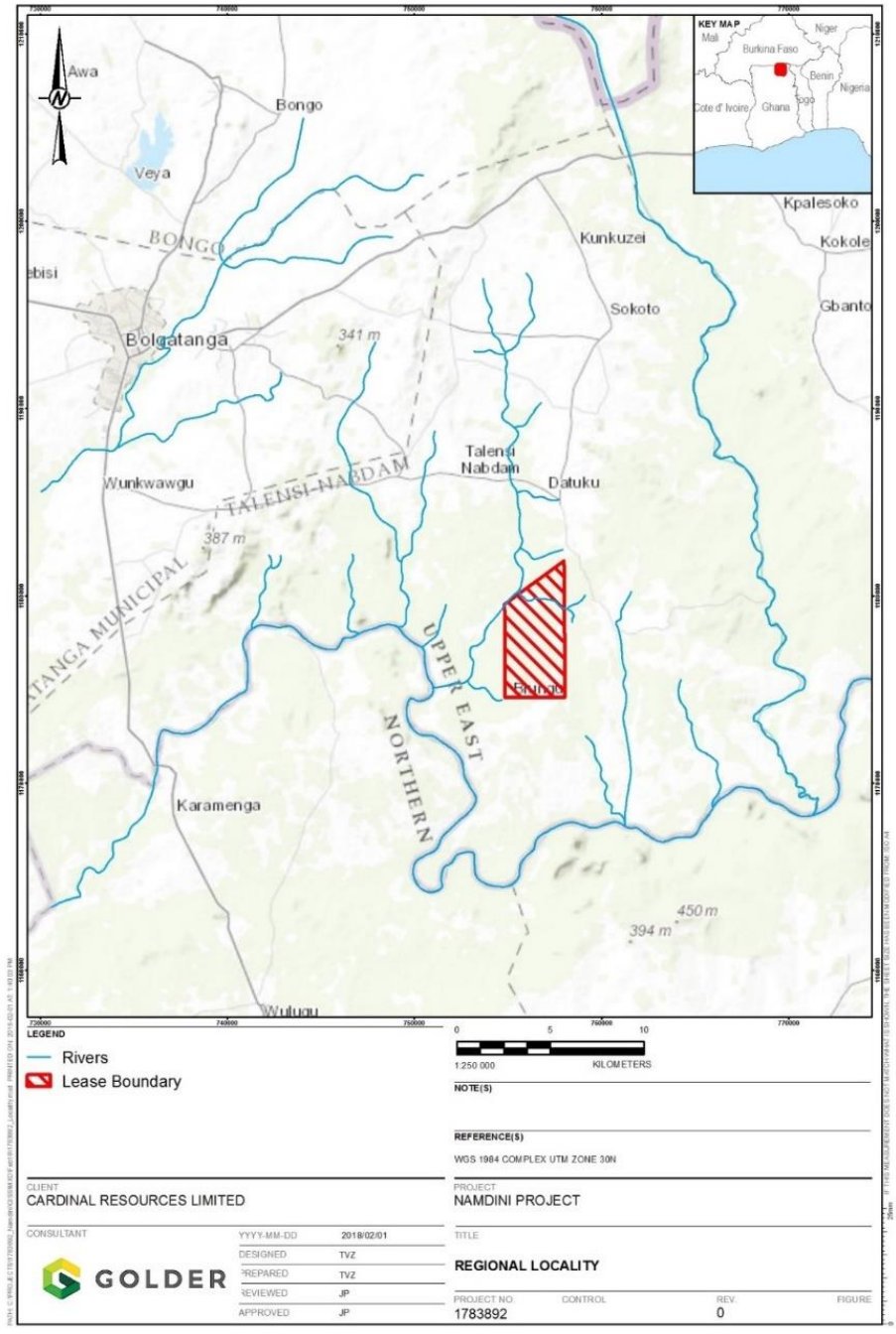


Figure 26: Drainage and rivers around the Namdini Gold Project site (source: Golder 2018)

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 27). A small non-perennial stream, the Zoan Buliga, passes the northern boundary of the site before joining the White Volta.

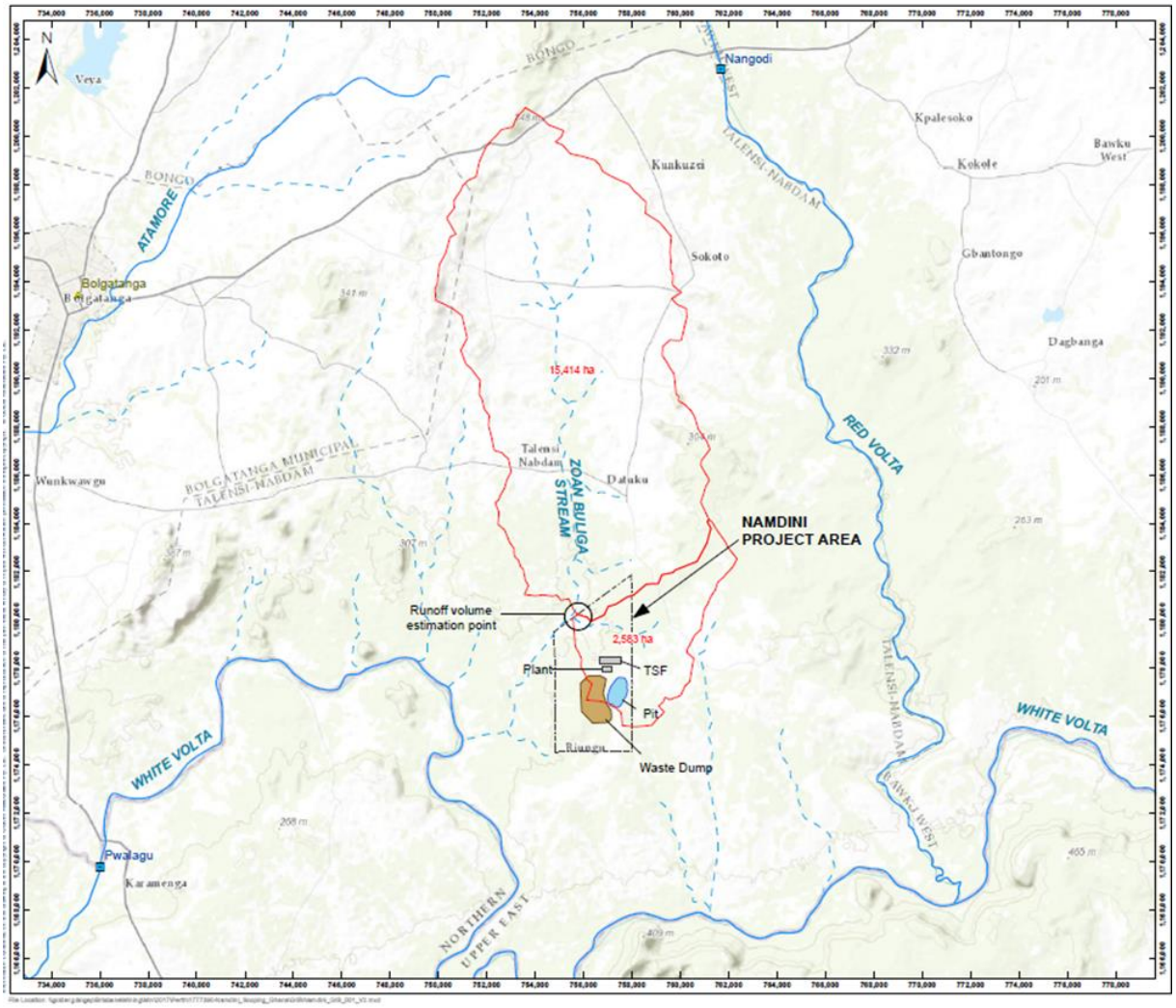


Figure 27: Location of the Namdini Gold Project Site in relation to rivers (source: Golder 2018)

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 characterised by short scattered drought-resistant trees, scattered scrub, and grass. The most common trees are the sheanut, dawadawa, and baobab (Section 20.3.12).

5.5 Climate

5.5.1 Annual range of climate variables

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

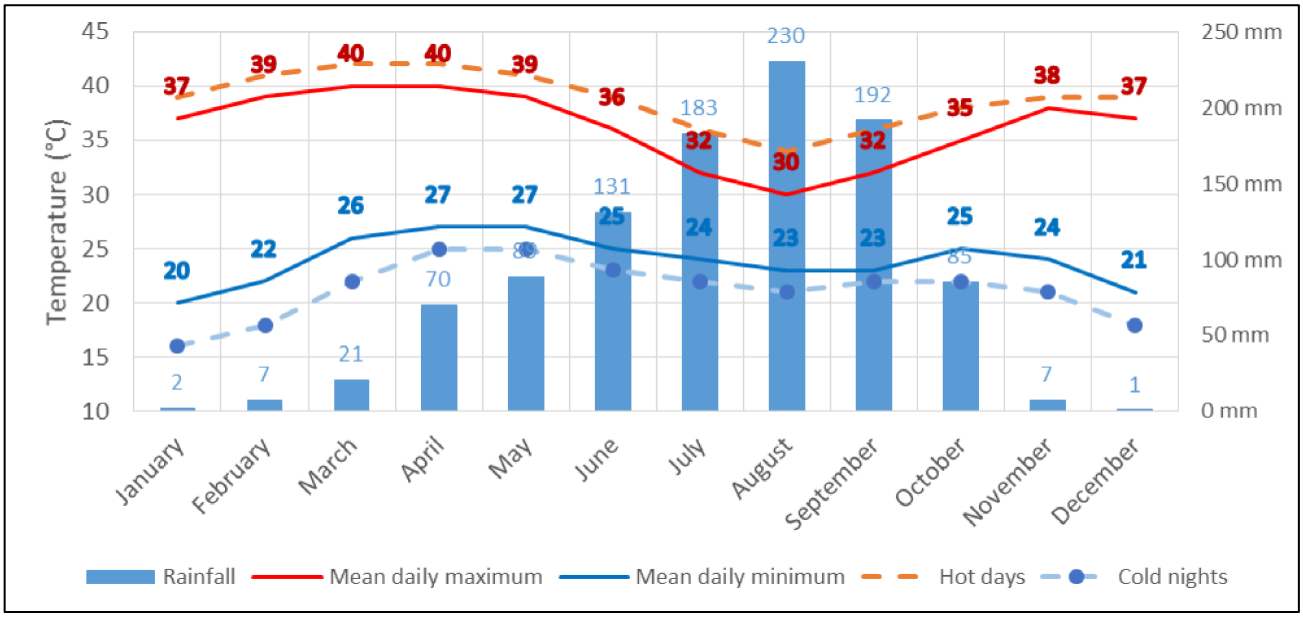
The mean annual temperature in Bolgatanga is 28.3°C. There is a long dry season from November to late February, characterised by cool, dry and dusty Harmattan winds. Temperatures during this period can be as low as 15°C at night and as high as 40°C during the day.

The rainy season is between May and October when the rainfall is erratic spatially and in duration. Mean annual rainfall varies between 800 mm and 1,100 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 28.



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

Figure 28: Average minimum and maximum daily temperatures for Bolgatanga, from Meteoblue (2017)

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 28 (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 23 below.

Table 23: 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 summarised by Lycopodium for the purpose of designing the Project Infrastructure during the PFS (Golder, 2018b) as shown in Table 24.

Table 24: 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 information in this Section is derived from the PFS (Golder, 2018b) which is a comprehensive report provided on Sedar (https://www.sedar.com).

6.1 Historic production

Prior to the agreements described in Section 4.2 and 4.3 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. Golder observed artisanal mining operations during the May and December 2017 site visits. The artisanal mining is small-scale and targets visible gold in quartz veins near surface (Figure 29). The quantity of gold extracted by the artisanal miners is expected to be negligible with minimal effect on the Ore Reserve.



Figure 29: 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 northeastern Ghana is taken from Golder (2018b), after RPA (2017). Cardinal considers 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 the 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**") which optioned the property in 1936 and acquired a large prospecting licence area that 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, which 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 groundwork. 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

The tenement status was discussed in Section 4.2.

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 centre 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.

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 2016. This was reported in accordance with the JORC Code (2012) to the Australian Stock Exchange (ASX) as summarised in Table 25.

Table 25: 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 downhole 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 2 December 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 6 April 2017 (RPA, 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 Au, 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 5 April 2017. A summary of the results is provided in Table 26.

Table 26: 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/oz.
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 NI 43-101 Technical Report as Golder (2018a).

Table 27: Mineral Resource estimate by MPR (2017) at 0.5 g/t cut-off, used for the PEA (Golder, 2018a)

September 2017 Mineral Resource Estimates			
Resource Category	Tonnage (Mt)	Grade (g/t Au)	Contained Metal (Moz Au)
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 (Cardinal, 2018a; 2018b; 2018c) is now superseded, based on MPR (2019). The current estimate is fully discussed in Section 14.0. There was a change in tonnes, grade and contained metal based on additional drilling, an alternative estimation method and revised parameters for the constraining pit shell compared to the MPR (2018) estimate (see Section 14.10, Table 120).

Table 28: Mineral Resource estimate by MPR (2018) at 0.5 g/t cut-off, used for the PFS (Golder, 2018b)

March 2018 Mineral Resource Estimates			
Resource Category	**Tonnage (Mt)**	**Grade (g/t Au)**	**Contained Metal (Moz Au)**
Indicated	180	1.13	6.5
Inferred	13	1.2	0.5

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 (2019). The 2019 estimate is fully discussed in Section 14.0.

6.5 Preliminary Economic Assessment results

In February 2018 Cardinal announced results of a Preliminary Economic Assessment ("**PEA**") for Namdini (Cardinal, 2018b; Golder, 2018b). The PEA was based on the September 2017 Mineral Resource model reported at 0.5 g/t cut-off within a pit design (Table 29).

Table 29: February 2018 PEA summary

From Golder (2018a)			
Gold price	US$1,300/oz		
Resource at 0.5 Au g/t cut-off grade	See Table 27		
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 PFS stage was warranted.

6.6 Preliminary Feasibility Study results

In October 2018 Cardinal presented the PFS Report for Namdini (Cardinal, 2018c; Golder, 2018b). The PFS was based on the March 2018 Mineral Resource (Table 28) reported at 0.5 g/t cut-off within a pit design (Table 30).

Table 30: February 2018 PFS summary

From Golder (2018b)			
Gold price	US$1,300/oz		
Resource at 0.5 Au g/t cut-off grade	See Table 28		
Strip ratio	1.4:1 (waste:ore)		
Metallurgical recovery	84% LOM		
	9.5 Mtpa	**7.0 Mtpa**	**4.5 Mtpa**
Life of gold mine production	3.98 Moz	3.98 Moz	3.98 Moz
Mine life	14 Years	19 Years	29 Years
Development capital cost	US$414 M	US$348 M	US$300 M
Life of mine sustaining capital cost	US$170 M	US$159 M	US$164 M
All in Sustaining Costs (AISC)	US$769/oz	US$823/oz	US$895/oz

The declared Probable Ore Reserve announced in the PFS (Golder, 2018b) was 129.6 Mt grading 1.14 g/t Au with a contained gold content of 4,760 koz and is reproduced in (Table 31). This is now superseded.

Table 31: Ore Reserve estimate declared in the September 2018 PFS (superseded)

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/oz.
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.

Mining was assumed to be medium-scale using conventional open pit mining equipment, including drill and blast as well as conventional load and haul operations. Mining was planned using five staged cut-backs to reach the LOM Final Pit. The mining schedule incorporated movement of ore and waste on 10 m mining benches, by year for each of the four mining stages. The initial stage (Starter Pit) provided early access to the higher-grade ore near the surface. The second stage was 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 80 m was established between the stages.

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 considered 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. Design sectors were defined around the Namdini Pit based on geotechnical and hydrogeological considerations from site investigations.

The positive result of the PFS resulted in progression of the Namdini Gold Project to this FS stage.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

The information in this Section is derived from the PFS (Golder, 2018b) which is a comprehensive report provided on Sedar (https://www.sedar.com).

7.1 Geological background

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 30).

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), characterised 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 characterised 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; 2019):

The Namdini Gold Project covers the southern extension of the Nangodi Greenstone Belt (Figure 30 and Figure 31).

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.

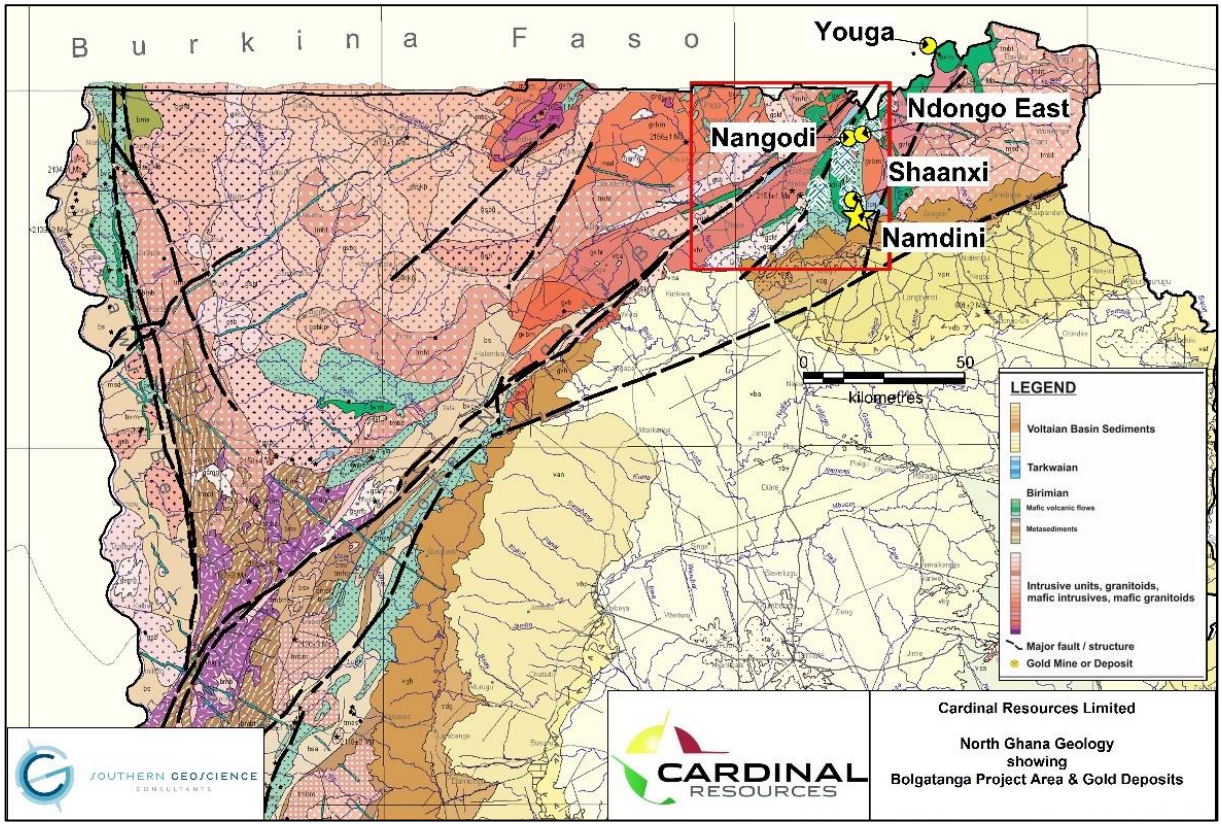


Figure 30: Regional geology of northern Ghana (source: Cardinal 2019)

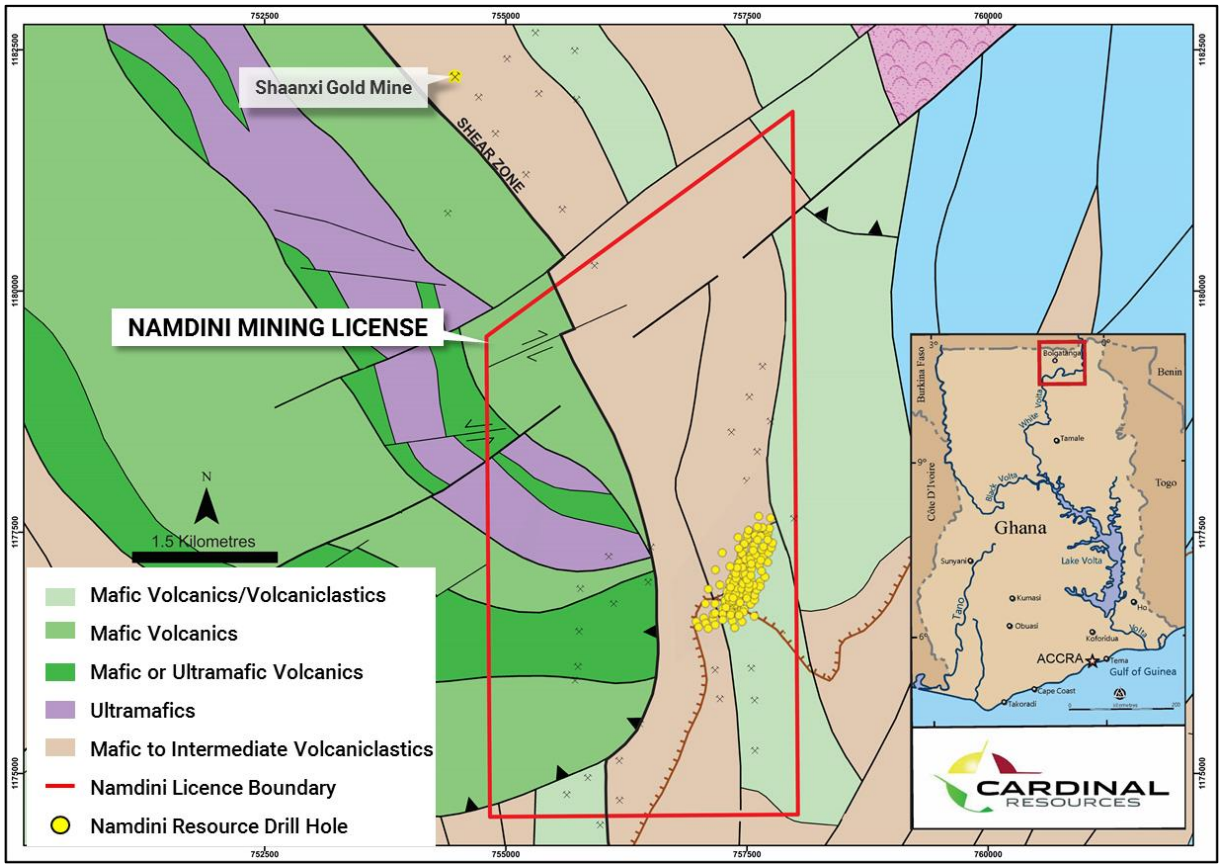


Figure 31: Local geology of the Namdini Gold Project and the licence boundary (source: Cardinal 2017)

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 (Davis & Cowan, 2016).

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, sulphide-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 sericite) or unaltered diorite, as well as quartz diorite speckled with quartz and feldspar.

Figure 32 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.

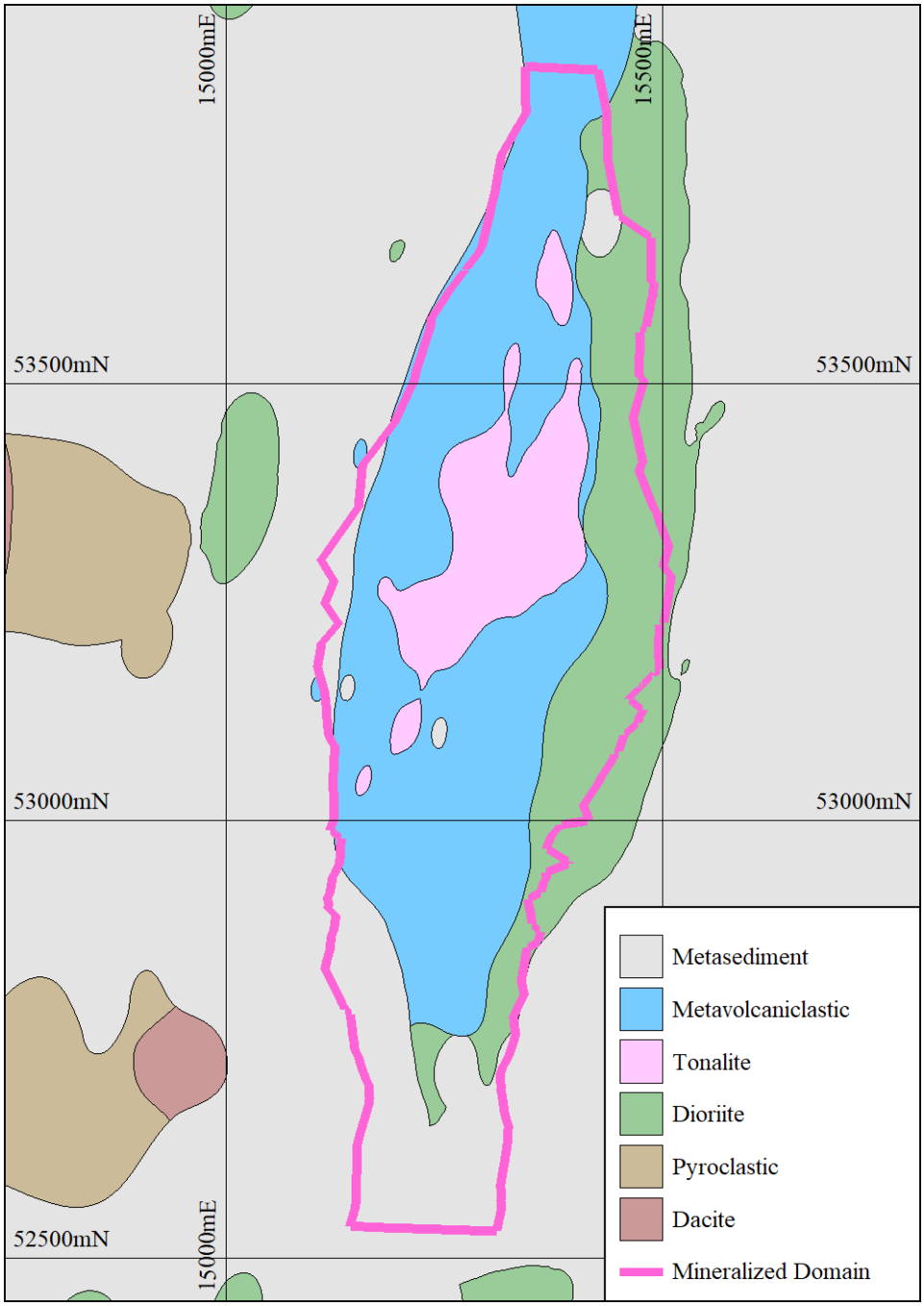


Figure 32: Namdini geological setting (source: MPR, 2018)

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:** total oxidation of all primary minerals with little or no primary rock texture. This zone ranges from 1.0 to 7.5 m in thickness.
- **Moderately oxidized:** some primary rock texture, total oxidation of feldspar to clay, and total oxidation of sulphides. This zone ranges from 0.5 to 13.0 m in thickness.
- **Transition:** strong primary rock textures with partial oxidation of feldspars and sulphides. 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 summarised 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 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.

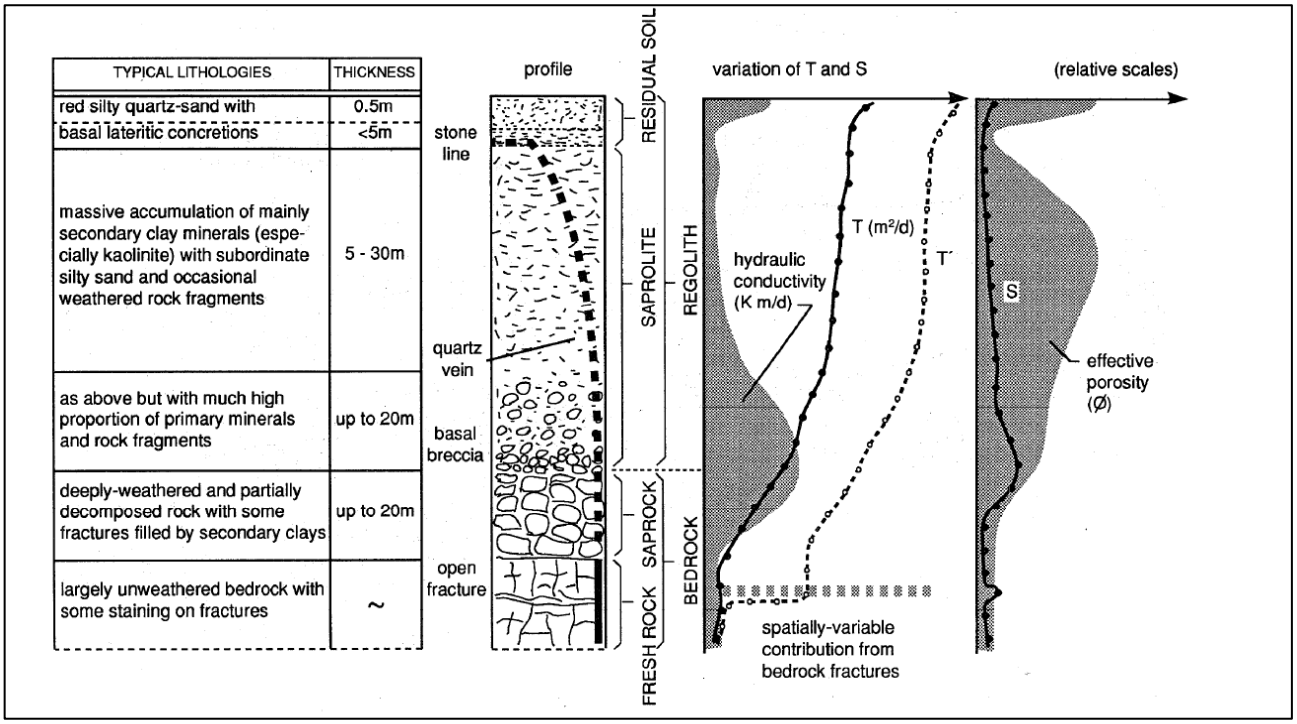


Figure 33: Conceptual model of weathered layer in Africa (source: CIDA 2011)

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 33 shows a conceptual model of the weathered layer for basement rock areas (CIDA, 2011). 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 characterisation. Fe-dolomite and ankerite units were noted 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 sulphides 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 34).

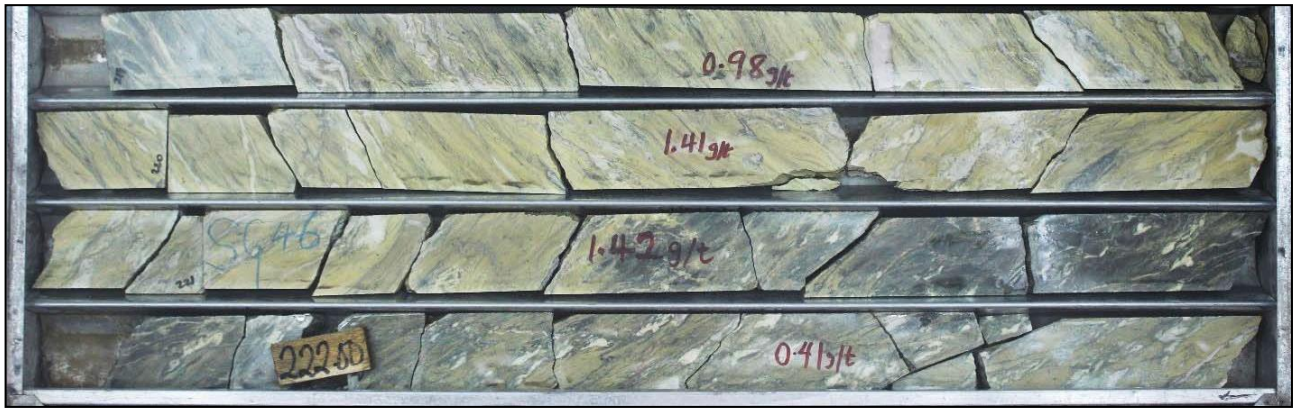


Figure 34: Alteration in drill hole NMDD007 at 227.33 m to 231.95 m, from RPA (2017)

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.

Petrography and mineralogy on thin and polished sections has been carried out by Townend Mineralogy Laboratory ("**TML**") of Perth. Gold is observed associated with sulphides, in particular pyrite, where it commonly occurs as inclusions and on the crystal margins. Gold is 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 sulphides and less commonly silicate minerals.

7.2.6 Cardinal geological logging

Logging, interpretation and geological modelling were undertaken by Cardinal's 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 32 presents a summary of the revised lithological codes and descriptions used by Cardinal geologists for geological logging purposes.

Table 32: Summary of Cardinal geological logging codes and descriptions

Code	Description
LAT	Laterite, ferruginous duricrust developed *in situ*
SAP	Saprock (<20% alteration of weatherable minerals)
SPR	Saprolite (>20% alteration of weatherable minerals)
GRA	Granodiorite or tonalite, altered felsic rocks (sericite, muscovite, carbonate and K-feldspar)
DIO	Intermediate rocks, altered (shearing, silicification, chlorite and sericite) and unaltered diorite, quartz diorite speckled with quartz and feldspar
MVO	Mafic rocks, volcaniclastics, altered (sericite, chlorite + silicification)
MSE	Metasedimentary rocks, dominated by quartz-carbonate veining + hematite and chlorite
LTF	Pyroclastic rocks and tuffs
DAC	Dacite, felsic extrusive rock, intermediate in composition between andesite and rhyolite. Often found associated with lava flows, volcanic equivalent of granodiorite.
BX	Breccia
QTZ	Quartz

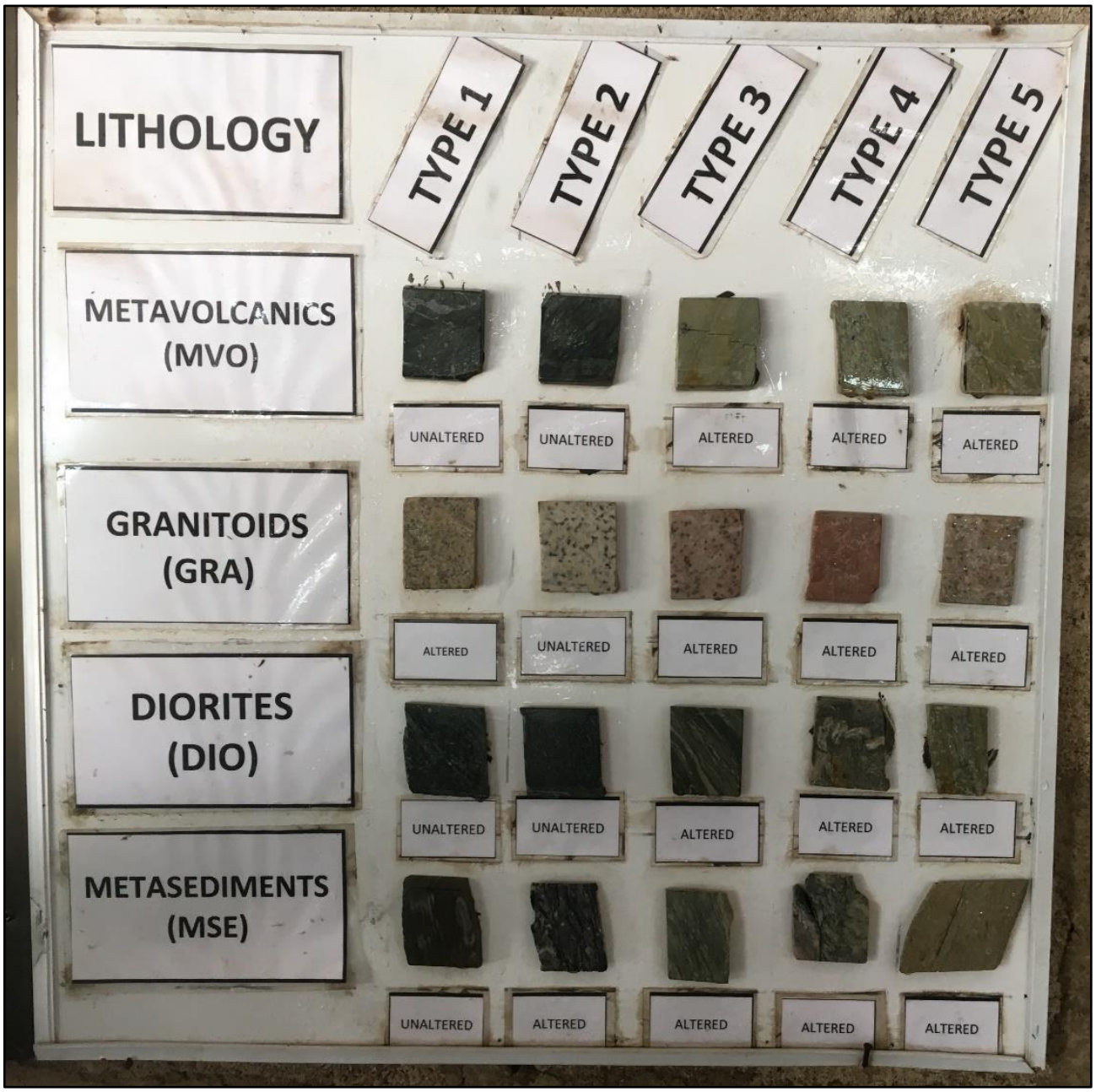


Figure 35: Logging guide of lithology and alteration types (source: Cardinal, 2019)

8.0 DEPOSIT TYPES

The information in this Section is derived from the PFS (Golder, 2018b) which is a comprehensive report provided on Sedar (https://www.sedar.com).

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 sulphide 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 summarised 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 sulphides in the host rocks surrounding the veins. The vein- and sulphide-hosted gold is strongly associated with deformational fabrics formed by the Eburnean extensional and compressional events respectively, suggesting that disseminated sulphide 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 km 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 sulphides in small, steeply dipping stockworks and as sulphide 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 FS.

9.0 EXPLORATION

The information in this Section is derived from the PFS (Golder, 2018b) which is a comprehensive report provided on Sedar (https://www.sedar.com).

9.1 Summary of relevant work

A detailed discussion of early exploration in the region is provided in Section 6.2.

Cardinal's regional exploration activities outside the Namdini deposit area are of little relevance to the current Project and are not detailed in this NI 43-101 Technical 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.

A Small-Scale Mining Licence was approved in 2014 and 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.

All subsequent exploration data that is relevant to the Namdini Gold Project is retained in the Mineral Resource database. Information about the sampling methods, sample and assay quality, representativity, potential for bias and other relevant aspects is all provided in Sections 10.0 to 12.0 of this Report.

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 33), with no elevation change. The transformation rotates the obliquely (WGS) trending drill traverses to east-west (local) grid.

Table 33: 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 36 and Figure 37).

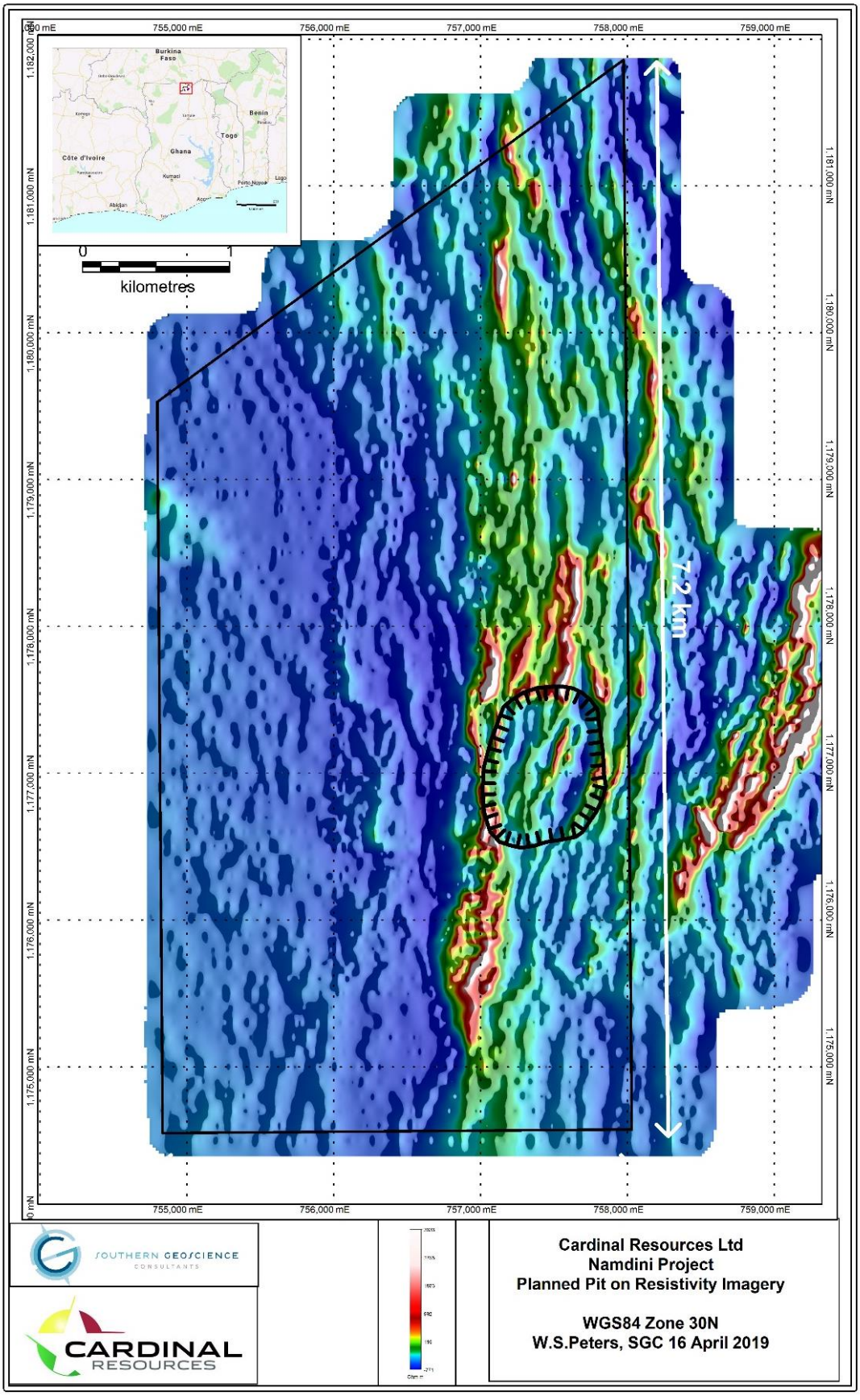


Figure 36: Planned Pit and tenement on resistivity map (source: Cardinal 2019)

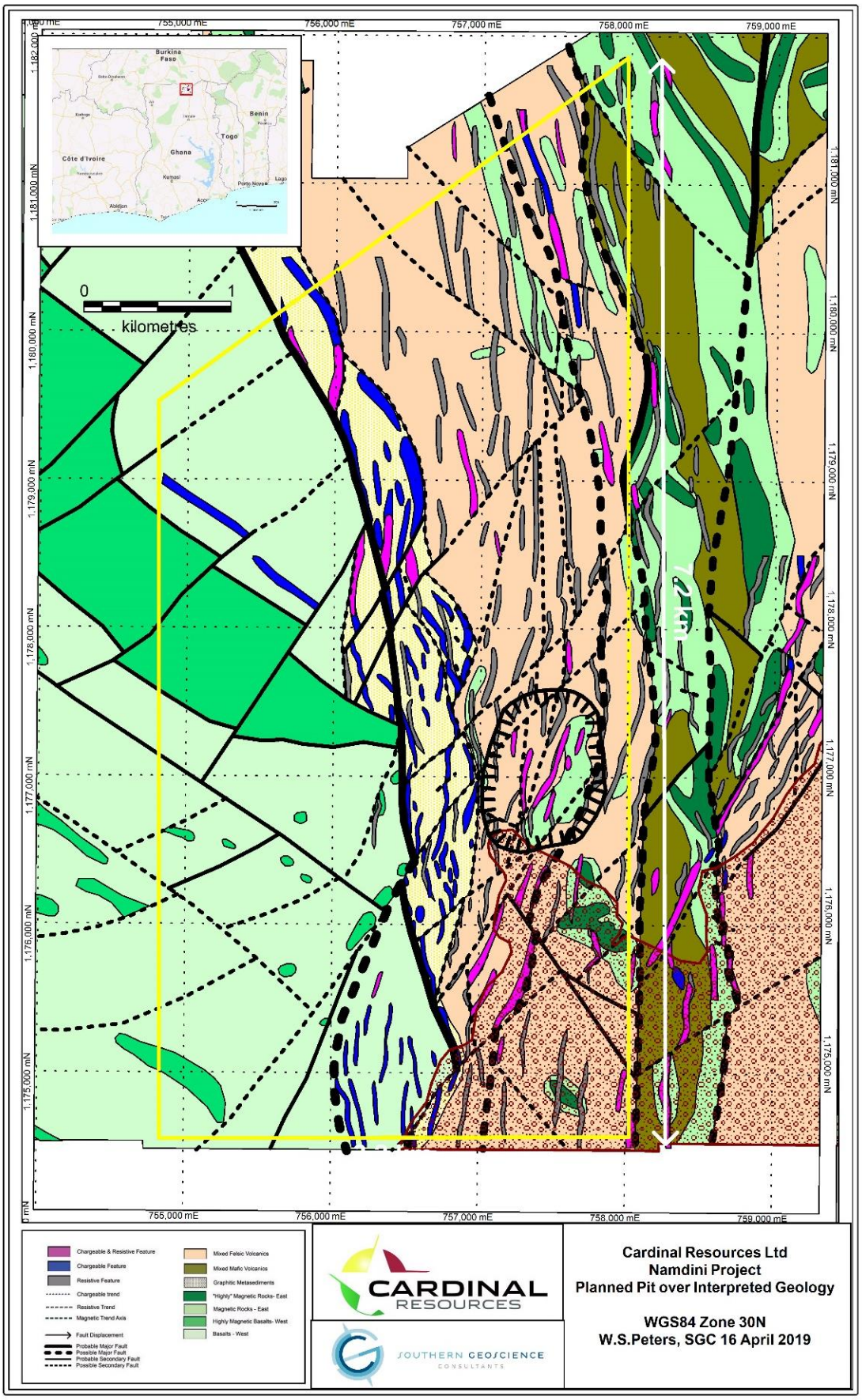


Figure 37: Interpreted geological structural features and lithology (source: Cardinal 2019)

10.0 DRILLING

The information in this Section is derived from MPR (2019) and is presented in a similar format to that in the PFS (Golder, 2018b) which is a comprehensive report provided on Sedar (https://www.sedar.com).

10.1 Summary

The Mineral Resource estimates described in this Report are based on RC and diamond information available for Namdini on 4 February 2019, totalling 326 holes for 87,140 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 of RC samples 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 4 February 2019 as summarised in Table 34 and shown in Figure 38. RC drilling in this table and figure include pre-collared portions of 39 diamond holes which average 99 m deep. In the right-hand plot in Figure 38, collars for additional drill holes available since the March 2018 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 34 and Figure 38, and exclude non-resource drilling, comprising:

- 14 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,
- 196 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 the 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 March 2018 resource estimates, the dataset available for the current estimates includes an additional 4,271 metres of drilling within the area of the resource model (in Section 14.14.1 the full available drilling data set for the Project is examined). This drilling, which represents an increase of around 5% comprises:

- Six RC holes and eight diamond holes drilled during 2017 and 2018 in the central and northern portions of the deposit, and
- One diamond tail drilled in 2018 extending an RC hole initially drilled in 2016 from 150 to 416.39 metres depth (NMRD115). This hole is included in the summary of 2018 drilling in Table 34.

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 35 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:

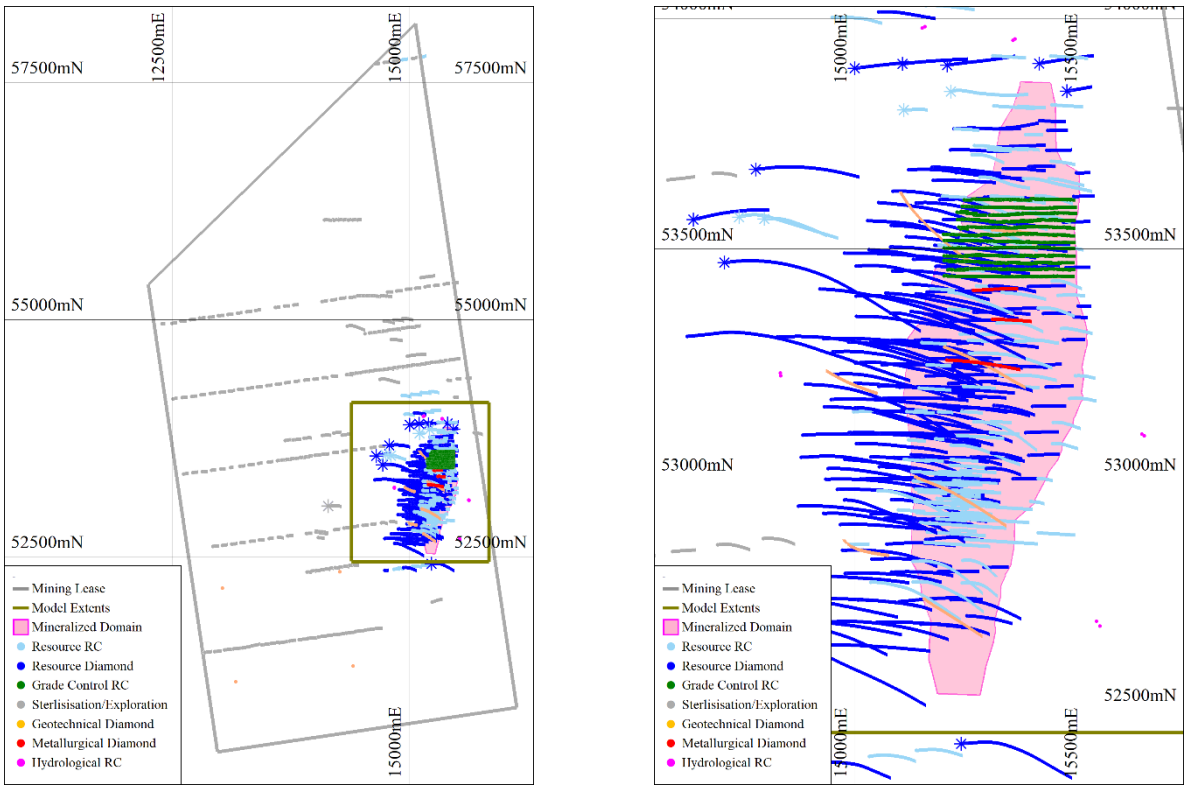
- Diamond drilling provides around three quarters of the dataset with RC sampling contributing the remainder.
- Half and quarter core samples represent around two thirds and one thirds of the diamond core dataset, respectively.
- Most mineralized domain RC composites are from holes drilled by the Cardinal (53%) and AMS rigs (29%) with the Minerex and Toomahit rigs contributing comparatively minor amounts.

Table 34: Namdini RC and diamond drilling

Group	Year	Number of holes			Drill metres		
		RC	Diamond	Total	RC	Diamond	Total
Resource	2014	45	-	45	4,815.00	-	4,815.00
	2015	42	9	51	4,939.30	2,128.53	7,067.83
	2016	19	71	90	3,943.70	16,399.74	20,343.44
	2017	41	86	127	8,651.70	42,183.67	50,835.37
	2018	4	9	13	1,166.00	2,912.52	4,078.52
	Subtotal	**151**	**175**	**326**	**23,515.70**	**63,624.46**	**87,140.16**
Grade Control		317	-	317	13,277.00	-	13,277.00
Metallurgical/Geotech/Hydro		10	14	24	538.20	4,174.84	4713.04
Sterilisation/Exploration		194	5	199	22,678.80	1,284.82	23,963.62
Subtotal		**521**	**19**	**540**	**36,494.00**	**5459.66**	**41,953.66**
Total		**672**	**194**	**866**	**60,009.70**	**69,084.12**	**129,093.82**

Table 35: Mineralized domain composites by sampling type

Drilling Type	Group		Number of Composites	Proportion of Drill Type	Proportion of Total
RC	Drill Rig	Cardinal	4,294	53%	15%
		AMS	2,345	29%	8%
		Minerex	967	12%	3%
		Toomahit	430	5%	2%
		Subtotal	***8,036***	***100%***	***28%***
Diamond	Sample Type	Half core	13,193	64%	46%
		Quarter core	7,383	36%	26%
		Subtotal	***20,576***	***100%***	***72%***
Total			**28,612**		**100%**



Mining Lease **Resource Area**

Figure 38: Namdini drilling, mineralized domain, model extents and tenement (source: MPR, 2019)

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 6 m of PVC casing.
- Samples were collected over 1 m down-hole intervals using a cyclone, with sub-sampling by a three-tier riffle splitter. Some non-mineralized intervals were composited over 4 m intervals for analysis.
- The riffle splitter was routinely cleaned with a rubber mallet and compressed air.
- Hole clearance and stabilisation on every rod change and “blow-backs” for each metre.

- Recovered sample material was routinely weighed for each sample 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. From approximately April 2016 to June 2017 core was quartered for assaying. For drilling after June 2017 diamond core was halved for sub-sampling. Sample intervals range from 0.2 to 1.8 m in length, with the majority of samples assayed over 1 m intervals.

The photographs in Figure 39 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 39: 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.

Table 36 and Figure 40 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 m depth, and subsequent surveys at generally 30 m intervals to hole end. Drilling intervals with such comprehensive down-hole surveying contribute 77% of mineralized domain composites. The remaining mineralized composites are from intervals with variable down-hole survey coverage including the following:

- No down-hole surveys are available for holes providing around half of the mineralized domain RC estimation dataset, which represents around 15% of the combined RC and diamond composites.
- Rare RC holes, which provide 0.8% 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 m provide 8% of the combined mineralized composites. For diamond holes without poorly surveyed pre-collars, the locations of diamond tails are not accurately defined.

The proportion of mineralized domain composites with incomplete or no down-hole surveying reduces with depth. To depths of around 50 m, less than half of these data have comprehensive down-hole surveys. Below 100 m depth, around 90% have full down-hole survey information.

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

Table 36: Mineralized domain composites by downhole survey type

Downhole Survey Availability	Number of Composites			Proportion of Composites		
	RC	DDH	Total	RC	DDH	Total
Un-surveyed	4,249	-	4,249	53%	-	15%
Dip only	68	-	68	0.8%	-	0.2%
Sparsely surveyed > = 60 m spacing	673	1505	2,178	8%	7%	8%
Comprehensive generally < = 30 m	3,046	19,071	22,117	38%	93%	77%
Total	**8,036**	**20,576**	**28,612**	**100%**	**100%**	**100%**

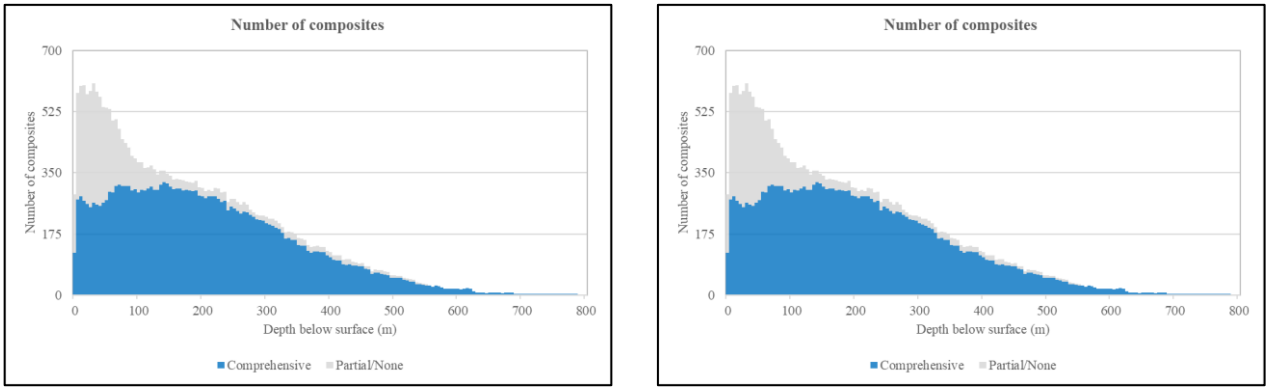


Figure 40: Downhole surveying for mineralized domain composites (source: MPR, 2019)

10.5 Diamond core recovery

Core recovery measurements are available for around 98% 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 in length and are dominated by intervals of around 3 m in length. These data were composited to 3 m intervals to provide a consistent basis for analysis. Table 37 summarises core recoveries for the 3 m 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 shown by experience of high quality diamond drilling.

Table 37: Diamond core recovery by domain

Oxidation Domain	Background		Mineralized		Total	
	Number	Ave. Recovery	Number	Ave. Recovery	Number	Ave. Recovery
Oxide	312	93.6%	242	95.0%	554	94.2%
Transition	476	97.4%	230	96.5%	706	97.1%
Fresh	6,571	99.9%	13,072	99.9%	19,643	99.9%
Total	**7,359**	**99.4%**	**13,544**	**99.8%**	**20,903**	**99.7%**

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 38 summarises sample condition logging for assayed RC samples from resource drilling 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 38: RC sample condition logging

Drill Rig	Number of composites				Proportion of composites			
	Dry	Moist	Wet	Total	Dry	Moist	Wet	Total
Cardinal	9,945	-	1	9,946	99.99%	-	0.01%	100.00%
AMS	7,590	29	4	7,623	99.57%	0.38%	0.05%	100.00%
Minerex	3,374	7	2	3,383	99.73%	0.21%	0.06%	100.00%
Toomahit	1,599	-	-	1,599	100.00%	-	-	100.00%
Total	**22,508**	**36**	**7**	**22,551**	**99.81%**	**0.16%**	**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 the October 2017 Technical Report (MPR, 2017), reflecting the limited additional RC drilling 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 potentially significantly biased assay grades.

Due to progressive wear, average bit diameters are likely to be less than then nominal values of 5.0 or 5.5 inches shown in supplied data files. For assessment of sample recovery, specified diameters were reduced by 1/8th of an inch. This value is based on experience and is of uncertain accuracy.

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 39 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 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 41 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 6 m increments representing the first sample of each six metre long 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 41, 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 representativity of RC samples. Additional investigations of this trend may be warranted.

Table 39: Average RC sample recovery by domain

	Oxide		Transition		Fresh		Total	
	No.	Avg.	No.	Avg.	No.	Avg.	No.	Avg.
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%**

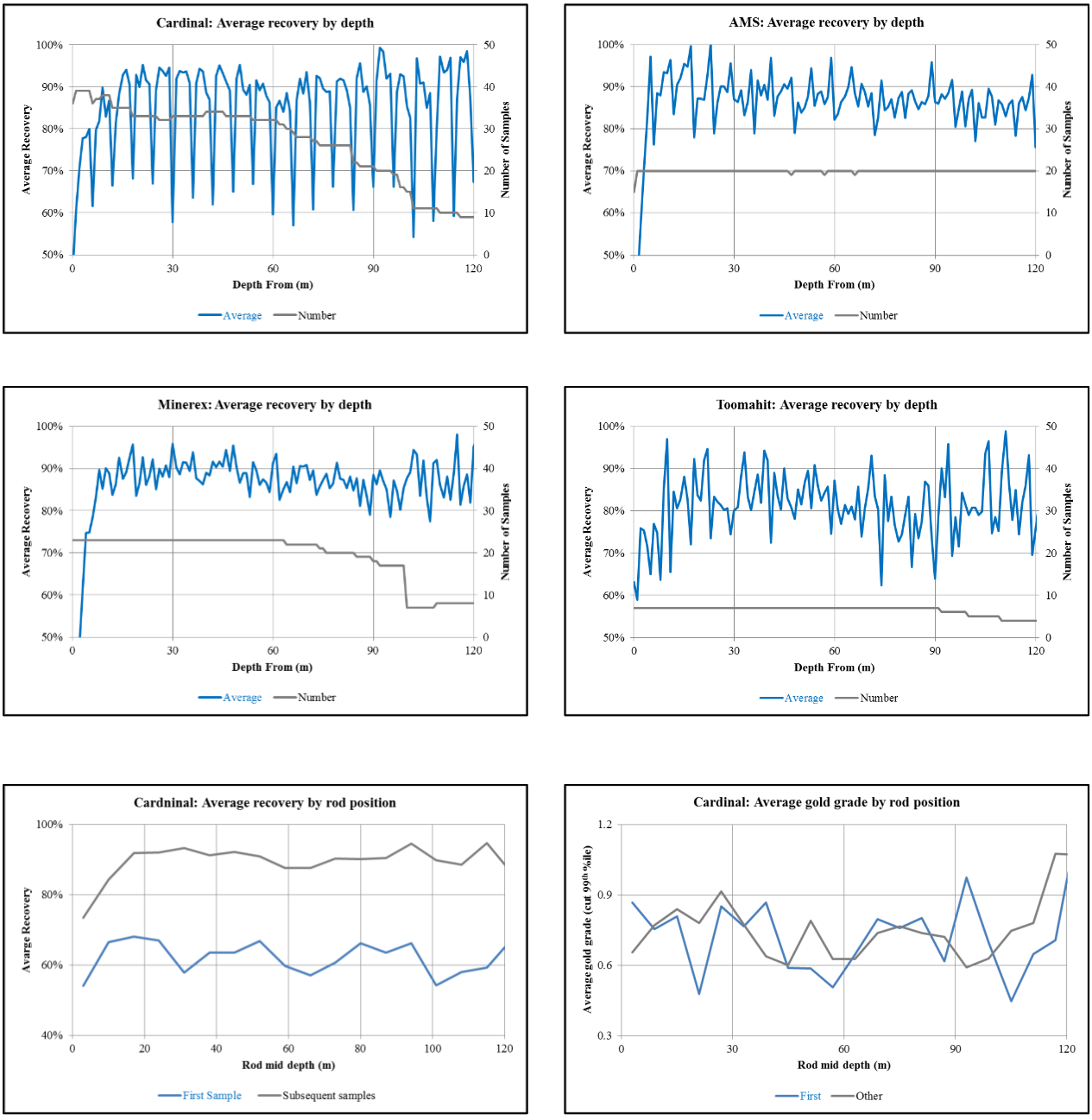


Figure 41: RC sample recovery by drilling depth (source: MPR, 2019)

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. This approach reflects the consistency of sampling and assaying and maximises the size of the review dataset. The compiled field duplicates represent a frequency of around one duplicate per 20 primary samples.

Table 40 and Figure 42 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 of greater than approximately 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 40: 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	502		1,367		154		298	
Mean	**0.74**	**0.69**	**0.48**	**0.47**	**0.57**	**0.62**	**0.15**	**0.17**
Mean dif.		**-7%**		**-2%**		**8%**		**13%**
Coef. Var.	2.36	2.23	5.28	4.25	2.94	2.91	5.16	5.49
Minimum	0.00	0.01	0.01	0.01	0.01	0.01	0.01	0.01
1st Quartile	0.04	0.03	0.01	0.01	0.01	0.01	0.01	0.01
Median	0.20	0.19	0.01	0.01	0.12	0.11	0.01	0.01
3rd Quartile	0.63	0.63	0.12	0.14	0.42	0.43	0.03	0.03
Maximum	15.8	19.2	75.1	42.5	15.0	14.7	10.3	12.9
Correl.	**0.89**		**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		321		75		29	
Mean	**0.99**	**0.97**	**0.96**	**0.96**	**0.76**	**0.78**	**0.70**	**0.73**
Mean dif.		**-2%**		**1%**		**3%**		**4%**
Coef. Var.	1.28	1.21	1.11	1.12	1.39	1.37	1.19	1.07
Minimum	0.10	0.10	0.10	0.10	0.10	0.11	0.11	0.10
1st Quartile	0.28	0.27	0.27	0.28	0.22	0.25	0.16	0.17
Median	0.50	0.51	0.55	0.59	0.40	0.41	0.26	0.29
3rd Quartile	1.12	1.25	1.21	1.17	0.74	0.77	0.91	1.17
Maximum	8.62	7.94	5.76	5.80	6.00	6.65	3.20	2.72
Correl.	**0.92**		**0.91**		**0.91**		**0.92**	

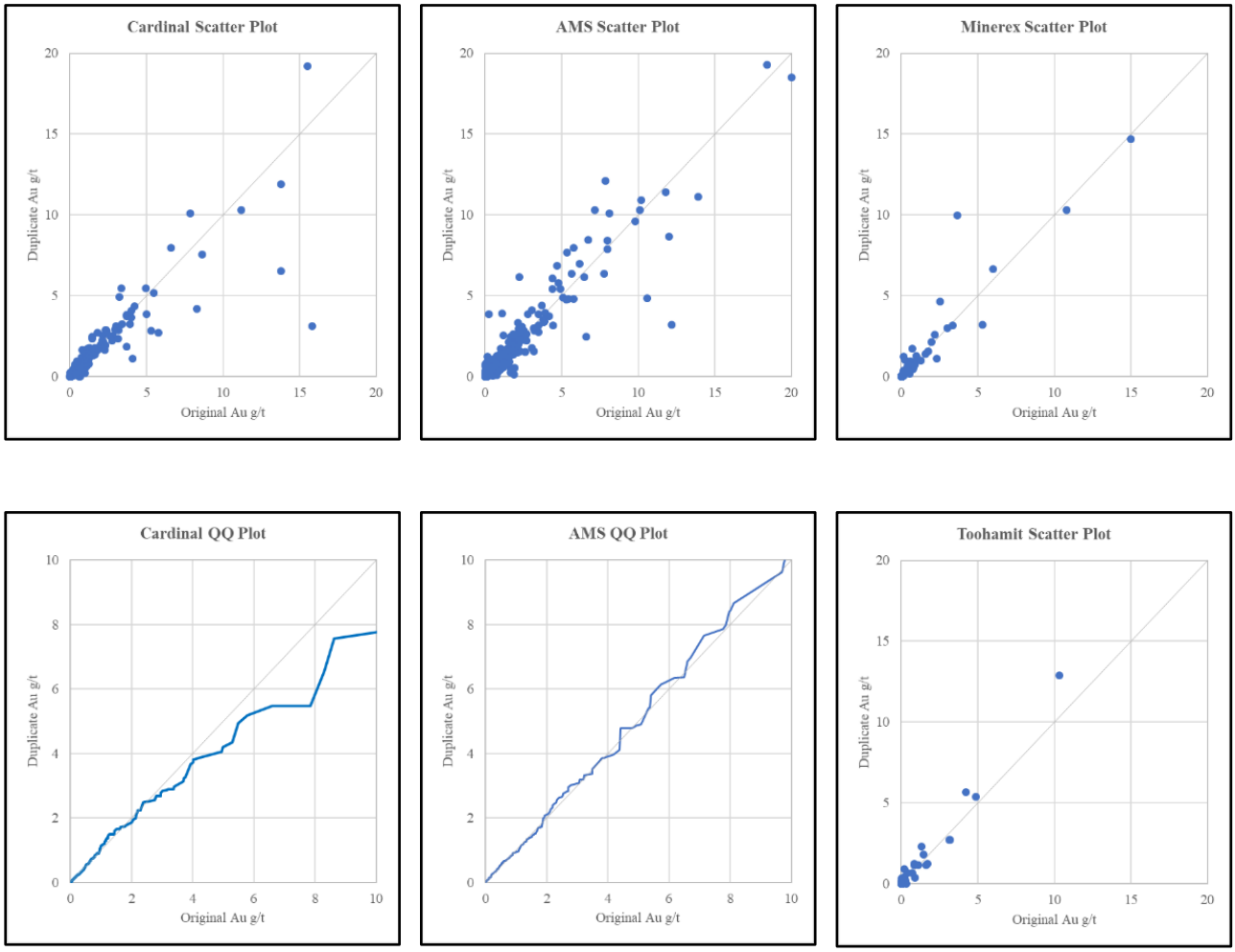


Figure 42: RC field duplicates (source: MPR, 2019)

11.0 SAMPLE PREPARATION, ANALYSES AND SECURITY

The information in this Section is derived from MPR (2019) and is presented in a similar format to that in the PFS (Golder, 2018b) which is a comprehensive report provided on Sedar (https://www.sedar.com).

11.1 Summary

Sample preparation and gold analysis of primary samples from Namdini resource drilling were undertaken by independent commercial laboratories. Most primary samples were submitted to SGS Ouagadougou or SGS Tarkwa for analysis by fire-assay with assays from these laboratories contributing around one third and two thirds of the estimation dataset respectively. Samples from several peripheral resource holes analysed by Intertek Tarkwa provide around 0.5% of the estimation dataset.

Key aspects of the information available to demonstrate reliability of these assays includes:

- Samples were transported from drill sites to secure storage by Cardinal employees before being delivered to the assay laboratory by laboratory personnel.
- All composites in the estimation dataset assayed by Intertek Tarkwa are from the background domain. These data have little impact on estimated resources, and reliability of Intertek's assaying does not significantly affect confidence in estimated resources.
- Assays of coarse blanks included in assay batches from before May 2015 and after September 2017 show no indication of significant contamination or sample misallocation.
- Sample batches submitted between May 2015 and September 2017 included fine blanks of pulverized material. These fine blanks showed no significant sample misallocation.
- Average assay results of certified reference standards submitted at an average frequency of around one standard per 41 primary samples reasonably reflect expected values supporting the general accuracy of assaying.
- Inter-laboratory Intertek repeats broadly support the general accuracy of primary SGS assaying. However, the reliability of Intertek assays has not been established with sufficient confidence for these data to represent a robust check of primary assaying.
- In 2016 a consultant geologist employed by Cardinal collected and submitted 49 independent duplicate core samples to SGS Tarkwa. For 15 intervals with original SGS Tarkwa assays, the duplicate results closely match original assays. For 34 intervals with original SGS Ouagadougou assays, the duplicate results average around 35% higher than the original assays. Reasons for this trend are unclear.
- In 2017, 165 independent quarter core duplicates collected by another consultant geologist were analysed by ALS Ireland giving very similar average gold grades to original assays and providing additional confidence in the general reliability of SGS Tarkwa assaying.
- Density measurements total 11,047 immersion measurements by Cardinal and SGS. For oxidation types and rock units with reasonable numbers of Cardinal and SGS measurements, the two datasets have very similar average values providing some confidence in the reliability of Cardinal's density measurements.

The quality control measures adopted for assaying of the Namdini resource drilling 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.

11.2 Sample security

Sample preparation and gold assaying of resource samples were 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 submitted 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 43).

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



Core trays laid out for logging



Core tray secure storage

Figure 43: Core tray logging and storage (source: Cardinal 2019)

11.3 Sample preparation and analysis for gold

Field sub-sampling procedures are described in Section 10.3. All sample preparation and gold assaying of samples from the Namdini resource drilling was undertaken by independent commercial laboratories. Analyses by Cardinal were limited to around 87% of the density samples.

Some samples from sterilization and grade control drilling, were analysed by ALS in Kumasi, Ghana. These samples are not included in estimated Mineral Resources and are not discussed in this Report. Selected samples from the resource drilling were assayed for additional attributes including sulphur and arsenic. These data were not included in the current estimates.

Most gold analyses of resource drill hole samples were undertaken at SGS laboratories in Ouagadougou in Burkina Faso or Tarkwa in Ghana. A small proportion of background domain samples were analysed by Intertek in Tarkwa, Ghana.

Table 41 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 mineralized domain estimation dataset with SGS Ouagadougou contributing around one third. The subdivision of SGS Ouagadougou assays by phase reflects variability in information available to demonstrate assay reliability.

Samples assayed by Intertek Tarkwa provide around 0.5% of the estimation dataset. All composites from these samples, which have a maximum gold grade of 0.38 g/t, lie within the background domain and have negligible impact on estimated Mineral Resources.

SGS Tarkwa and Ouagadougou, and Intertek Tarkwa have been 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 and crushed to 75% passing 2 mm.
- A 1.5 kg riffle split subsample was pulverized to nominally 85% at 75 µm. Remaining coarse reject was retained.
- Samples were fire assayed for gold using a 30 g or 50 g charge with an atomic absorption finish, with a detection limit of 0.01 g/t Au. 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.

Intertek Tarkwa employed comparable sample preparation and analytical methods to Ouagadougou and SGS Tarkwa.

Table 41: Estimation dataset by assay laboratory

Assay Laboratory		Number of Composites			Proportion of Composites		
		Background	Mineralized	Total	Background	Mineralized	Total
SGS Ouaga.	April '14-May'16	1,311	6,981	8,292	10%	24%	20%
	May 16'-Dec'17	1,438	2,983	4,421	11%	10%	11%
	Subtotal	**2,749**	**9,964**	**12,713**	**21%**	**35%**	**31%**
SGS Tarkwa	May 16'-Dec'17	8,354	17,017	25,371	65%	59%	61%
	2018	1,544	1,627	3,171	12%	6%	8%
	Subtotal	**9,898**	**18,644**	**28,542**	**77%**	**65%**	**69%**
Intertek Tarkwa 2018		215	-	215	2%	-	0.5%
Total		**12,862**	**28,608**	**41,470**	**100%**	**100%**	**100%**

11.4 Monitoring of assay reliability

11.4.1 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 41 primary samples.

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

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 blank samples 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 42 summarises the composite estimation dataset subdivided by the assay laboratory and the type of blanks inserted in each assay batch. Although some Intertek Tarkwa batches included fine blanks, none of these batches included samples from resource drilling. Table 42 indicates that assay batches with coarse blanks represent around 60% of the estimation dataset. For the remaining 40% 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 a significant proportion of the Namdini resource analyses is unusual. 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).

Table 43 summarises blank assay results by type and assay laboratory and Figure 44 shows blank assay results by type and assay date. These demonstrate that the blank assays show generally low gold grades with no indication of contamination or sample misallocation. Additional features shown by Table 43 and Figure 44 include the following:

- Relative to SGS Ouagadougou, SGS Tarkwa reports notably higher proportions of coarse and fine blank assays as above detection limit. Reasons for this trend are uncertain. However, relative to typical Namdini mineralization, the magnitude of the comparatively elevated gold grades reported by SGS Tarkwa are low and not suggestive of consistent significant contamination.
- For all three laboratories, most of the samples with comparatively elevated gold grades for coarse blanks are from mid to late 2017. Although uncertain, the consistency between laboratories suggests that rather than laboratory contamination, this trend may reflect variability in source material, or increased but still rare sample misallocation or field contamination for a comparatively short period.

Table 42: Estimation dataset by blank type

Assay Laboratory	Number of Composites				Proportion of Composites			
	Coarse	Fine	None	Total	Coarse	Fine	None	Total
SGS Ouagadougou	7,630	4,783	302	12,715	60%	38%	2%	100%
SGS Tarkwa	17,118	11,004	423	28,545	60%	39%	1%	100%
Intertek Tarkwa	165	-	50	215	77%	-	23%	100%
Total	**24,913**	**15,787**	**775**	**41,475**	**60%**	**38%**	**2%**	**100%**

Table 43: Blank assay results

Blank Type	Assay Laboratory	Assays (Au g/t)			Above Detection	
		Assays	Average	Maximum	Number	Prop.
Coarse	SGS Ouagadougou	447	0.005	0.06	6	1%
	SGS Tarkwa	984	0.007	0.17	178	18%
	Intertek Tarkwa	124	0.010	0.16	20	16%
	Subtotal	**1,555**	**0.007**	**0.17**	**73**	**5%**
Fine	SGS Ouagadougou	242	0.005	0.01	-	-
	SGS Tarkwa	578	0.006	0.04	67	12%
	Intertek Tarkwa	25	0.005	0.01	-	-
	Subtotal	**845**	**0.006**	**0.04**	**73**	**9%**
Combined	SGS Ouagadougou	689	0.005	0.06	6	1%
	SGS Tarkwa	1,562	0.007	0.17	245	16%
	Intertek Tarkwa	149	0.009	0.16	20	13%
	Subtotal	**2,400**	**0.006**	**0.17**	**73**	**3%**

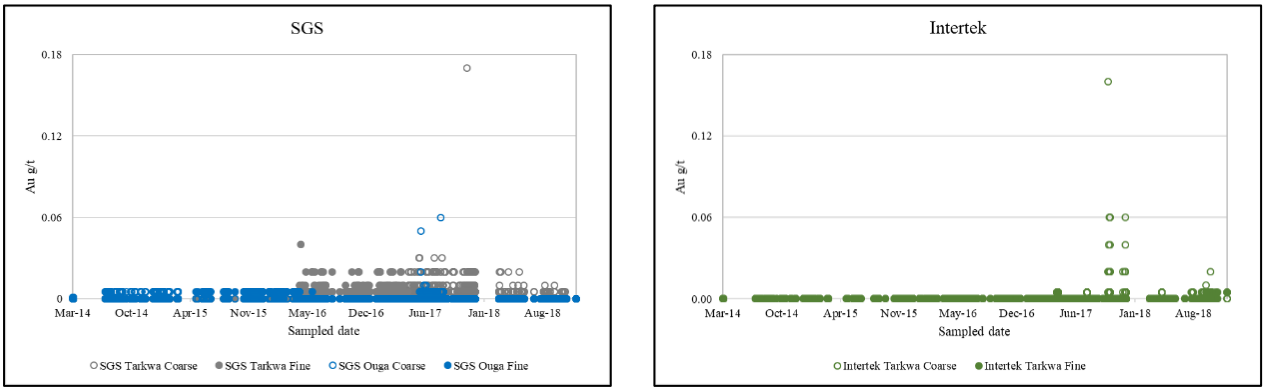


Figure 44: Blank assay results by date

11.4.2 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 one standard per 41 primary samples, have expected gold grades of 0.27 to 6.70 g/t.

Table 44 and Figure 45 summarise reference standard assays by laboratory. This table and figure demonstrate that although there is some variability for individual samples, for all three 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 2% 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 marginally 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 considered significant.

Table 44: Summary of certified reference standards assays

Period	Expected	Number	Average grade Au g/t		Avg vs.
	Value Range	Assays	Expected	Assay	Expected
SGS Ouagadougou	<1.5 g/t	454	0.50	0.50	-1%
	>1.5 g/t	342	3.67	3.69	1%
	Subtotal	**796**	**1.86**	**1.87**	**0%**
SGS Tarkwa	<1.5 g/t	781	0.50	0.50	0%
	>1.5 g/t	841	4.23	4.33	2%
	Subtotal	**1,622**	**2.44**	**2.49**	**2%**
Intertek Tarkwa	<1.5 g/t	106	0.56	0.57	1%
	>1.5 g/t	138	3.98	4.14	4%
	Subtotal	**244**	**2.50**	**2.59**	**4%**

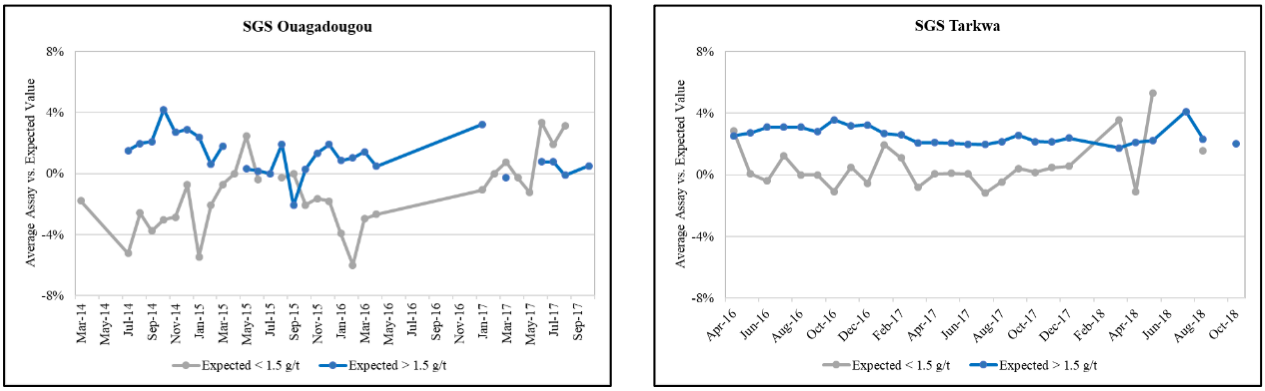


Figure 45: Certified reference standards results by date

11.4.3 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 with the exception of assaying since December 2017 which provides around 8% 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 on the basis of 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 summarised in Table 45 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 is 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 46 as blue dots. The QQ plots in Figure 46 and summary statistics in Table 46 exclude these anomalous pairs. For many of these pairs the magnitude of the difference is suggestive of sample misallocation.

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 is 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 is broadly comparable to trends shown by Intertek standards assays.

Each set of inter-laboratory repeats broadly supports 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 is uncertain.

Table 45: Reference standards results for Intertek repeat batches

Standard	Expected	Assay results (Au g/t)				Avg. vs.
	Au g/t	Number	Minimum	Average	Maximum	expected
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-Nov'17) 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 46: Intertek inter-laboratory repeats

Au g/t	Intertek vs. SGS Tarkwa			
	Full set		0.1 to 50 g/t	
	SGS.	Inter.	SGS.	Inter.
Number	1,002		967	
Mean	**2.89**	**2.96**	**2.54**	**2.58**
Mean difference		**2%**		**2%**
Coef. Var.	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	

Au g/t	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%**
Coef. Var.	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	

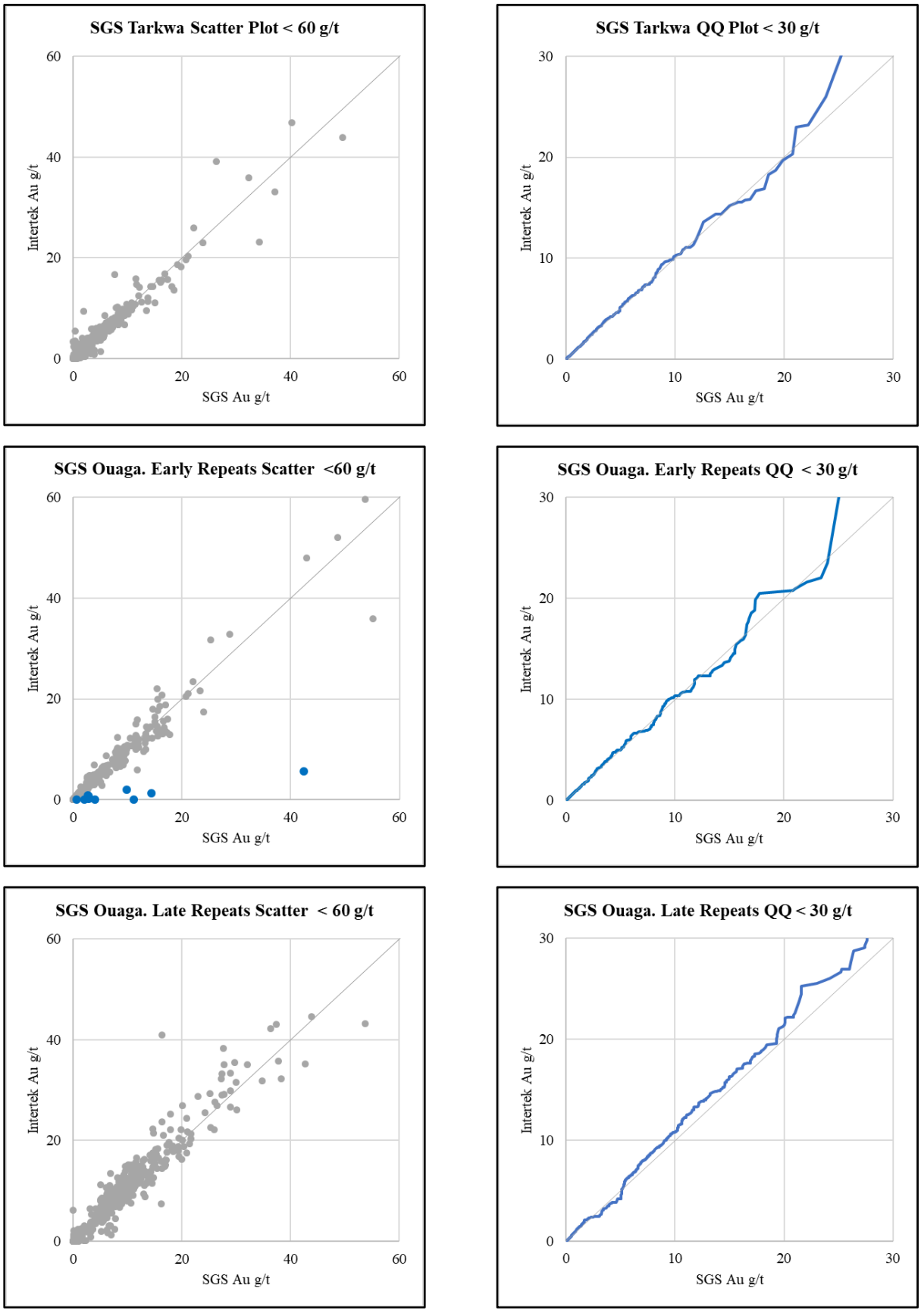


Figure 46: Intertek Inter-laboratory repeats

11.4.4 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 47 and Figure 47, the Intertek core duplicate assays show similar average grades to those reported by SGS Tarkwa, providing additional support for SGS Tarkwa analyses.

Table 47: Intertek Core Duplicates

Au g/t	Full Set		0.1 to 9.0 g/t	
	SGS Tarkwa	Intertek	SGS Tarkwa	Intertek
Number	433		233	
Mean	**0.76**	**0.77**	**1.26**	**1.26**
Mean difference	**2%**		**0%**	
Coef. Var.	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	

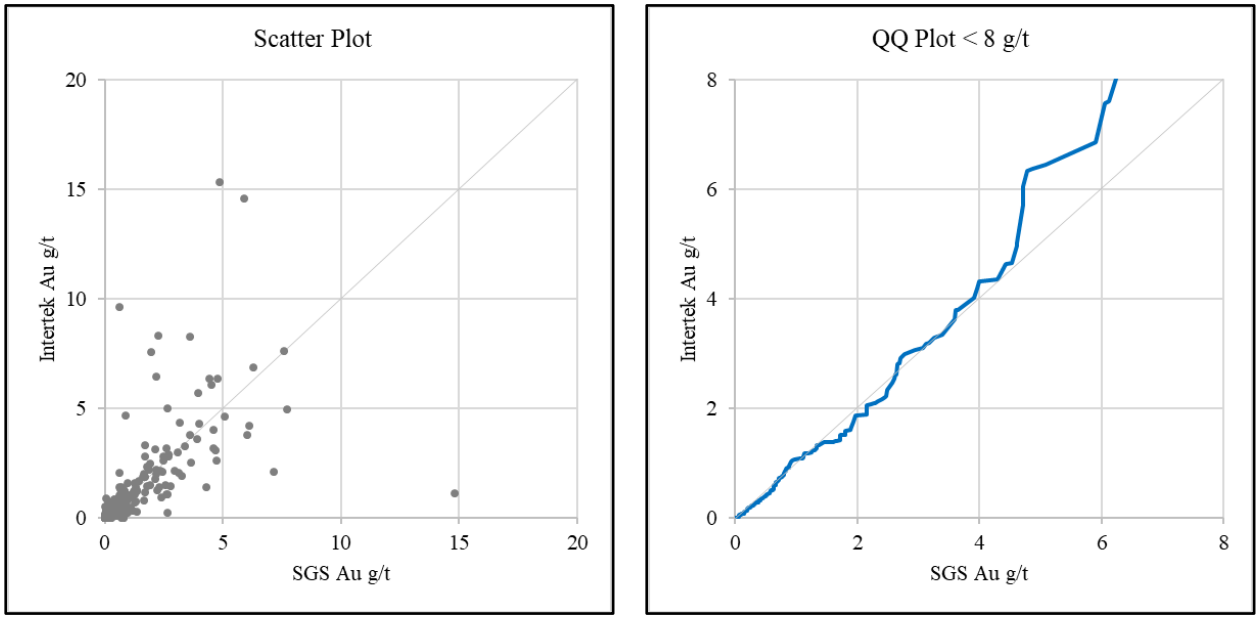


Figure 47: Intertek core duplicates

11.4.5 Independent core duplicate sampling

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

- 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.

11.4.5.1 July 2016 duplicates

Sample identifiers assigned to duplicate samples in the supplied data are the 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 48).

For intervals with original SGS Ouagadougou assays, duplicate results average around 35% higher than the original assays. Reasons for this trend are unclear and it may simply represent an artefact of the small dataset.

11.4.5.2 January 2017 duplicates

The duplicate ALS assay results generally reasonably match original SGS Tarkwa assays, with no notable difference in mean grades (Table 48, Figure 48). These data provide additional confidence in the general reliability of SGS Tarkwa assaying.

Table 48: 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%**
Coef. Var.	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	

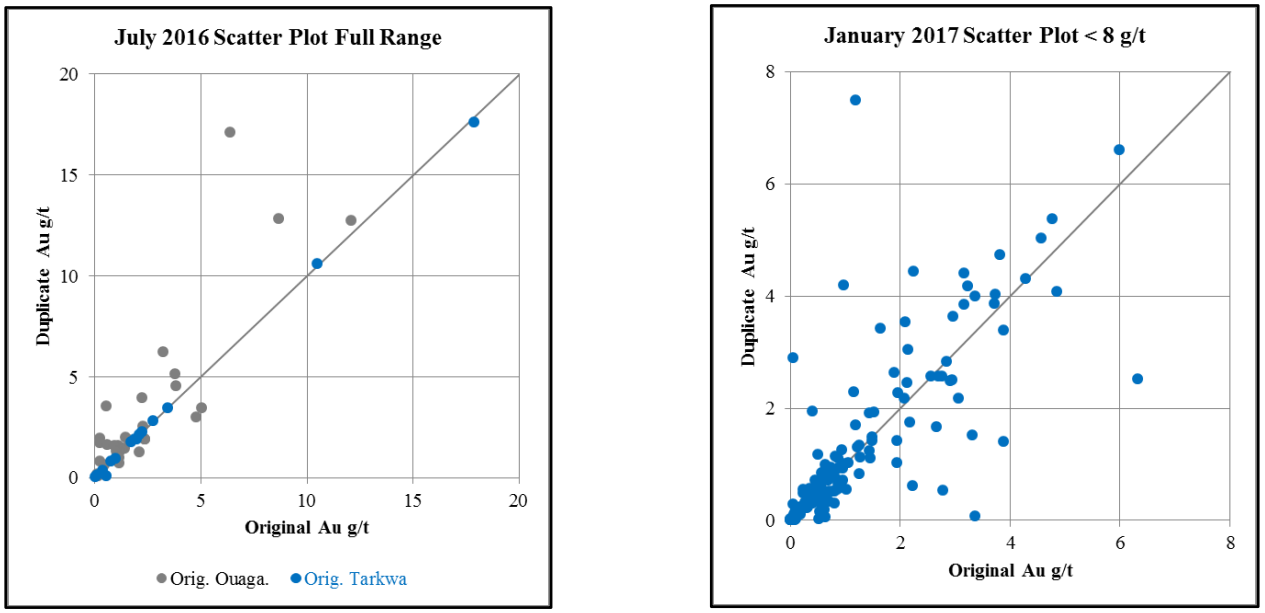


Figure 48: Independent core duplicates

11.5 Bulk density measurements

Bulk density measurements available for Namdini comprise 11,047 immersion measurements performed by either Cardinal (9,652) or SGS Tarkwa or Ouagadougou (1,395) on diamond core. 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.

Table 49 summarises the density measurements coded by the oxidation and rock type wire-frames used for resource model coding. This table demonstrates that for datasets with reasonable numbers of samples for both sets of measurements, Cardinal and SGS measurements have very similar averages providing some confidence in the reliability of Cardinal's measurements.

For the Oxide and Transition material subsets, Cardinal's measurements average slightly higher than SGS's. The maximum average difference of +10% was recorded for the Oxide MVO. These subsets contain too few measurements to establish if the differences are material.

The trend plots in Figure 49 compare density and gold grade for Fresh samples demonstrating that bulk density measurements are not strongly correlated with gold grades. Tonalite samples show no notable association between density and grade. Average densities for Metavolcanic and Diorite show a slight general increase with increasing grade, increasing from around 2.80 t/bcm at low grades to 2.85 t/bcm at gold grades of around 4 g/t, an increase of around 2%. The magnitude of the differences is not material given the average ore grade of 1.13 g/t Au.

Table 49: Density measurements by rock type

Rock Unit	Oxid. Zone	SGS		Cardinal		Combined		Cardinal vs. SGS
		No.	Avg. t/bcm	No.	Avg. t/bcm	No.	Avg. t/bcm	
Metavolcanic	Oxide	7	1.98	48	2.18	55	2.16	10%
	Trans.	26	2.57	47	2.63	73	2.61	2%
	Fresh	620	2.82	2,494	2.80	3,114	2.81	-1%
Tonalite	Oxide	8	2.44	13	2.54	21	2.50	4%
	Trans.	24	2.53	16	2.65	40	2.58	4%
	Fresh	227	2.73	933	2.72	1160	2.72	0%
Diorite	Oxide	-	-	48	2.08	48	2.08	-
	Trans.	-	-	60	2.63	60	2.63	-
	Fresh	329	2.82	3,124	2.80	3,453	2.80	-1%
Metasediment	Oxide	4	2.22	107	2.27	111	2.27	2%
	Trans.	59	2.60	172	2.61	231	2.61	0%
	Fresh	91	2.83	2,355	2.80	2,446	2.80	-1%
Dacite	Oxide	-	-	6	2.47	6	2.47	-
	Trans.	-	-	12	2.62	12	2.62	-
	Fresh	-	-	127	2.76	127	2.76	-
Pyroclastic	Oxide	-	-	8	2.18	8	2.18	-
	Trans.	-	-	28	2.60	28	2.60	-
	Fresh	-	-	54	2.75	54	2.75	-

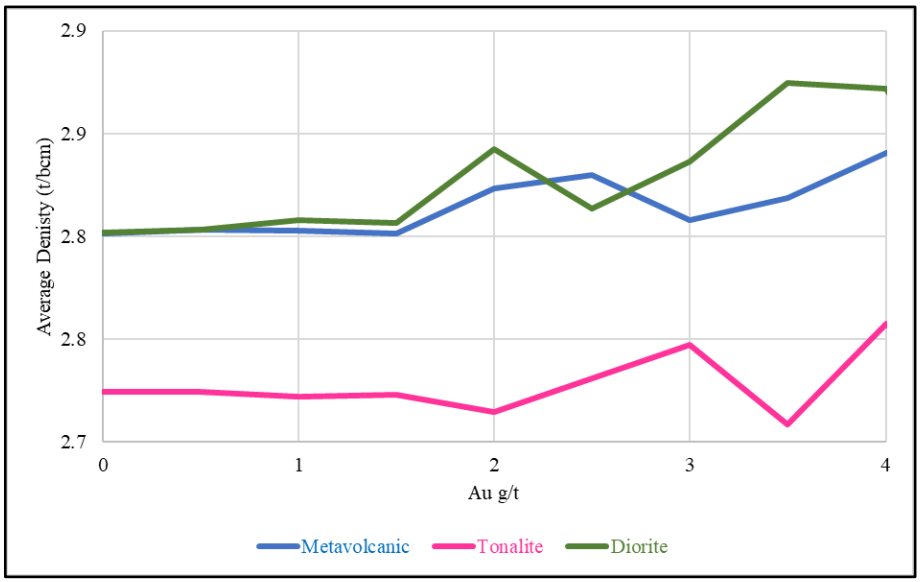


Figure 49: Bulk density versus gold grade for Fresh material

12.0 DATA VERIFICATION

The information in this Section is derived from MPR (2019) and is presented in a similar format to that in the PFS (Golder, 2018b) which is a comprehensive report provided on Sedar (https://www.sedar.com).

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 (2019) 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 99% of primary assays in the compiled database (Table 50). Comparing assay entries in these files with database entries showed no significant inconsistencies.

For 1,660 records representing around 53% of down-hole survey entries for resource holes, entries in original source files were compared to database records, with no significant inconsistencies noted.

For all resource drill holes, database collar UTM grid coordinates were compared to entries in original surveyor's spreadsheets, with no significant inconsistencies noted.

The available information indicates that the drilling database has been generally carefully compiled and validated and forms an appropriately reliable basis for Mineral Resource estimation.

Table 50: Laboratory source file checks

Comment	Count of assays	Proportion
No significant differences noted	86,075	99%
Not checked	1,152	1%
Total	**87,227**	**100%**

12.3 Validity of the data for resource estimation

The estimates described in this Report are based on RC and diamond resource drilling information available for Namdini on 4 February 2019, totalling 326 holes for 87,140 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:

- All resource drill hole collar locations were accurately surveyed by differential GPS techniques defining collar locations 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 77% of the mineralized domain estimation dataset. Hole paths have been located with sufficient accuracy for the current estimates.

- Core recovery measurements, which are available for around 98% of diamond drilling average 99.7% consistent with experience of 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%, average estimated average RC sample recovery is consistent with experience of high quality RC sampling.
- Field duplicates show generally reasonable repeatability consistent with experience of 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 lower than average estimated recoveries for the first sample of each drill rod and lower average gold grades for higher-grade field duplicates.

Primary samples from the resource dataset were generally submitted to independent commercial laboratories SGS Ouagadougou or SGS Tarkwa for analysis for gold by fire-assay. Assays from these laboratories contribute around one third and two thirds of the estimation dataset respectively. Round 0.5% of resource samples were analysed by Intertek Tarkwa. Key aspects of the information available to demonstrate reliability of these assays includes following:

- Diamond core and RC samples were transported from drill sites to secure storage at Cardinals' Exploration Office by Cardinal employees before being delivered to the assay laboratory by laboratory personnel.
- All estimation dataset composites assayed by Intertek Tarkwa are from the background domain. Reliability of Intertek's assaying does not significantly affect confidence in estimated resources.
- Assays of coarse blanks included assay batches from before May 2015 and after September 2017 which contribute 60% of the composite estimation dataset, show no indication of significant contamination or sample misallocation.
- Samples submitted after May 2015 and before September 2017 included fine blanks which did not require preparation by the analytical laboratory. These samples primarily test for sample misallocation and showed no significant misallocation.
- Average assay results of certified reference standards submitted at an average frequency of around 1 standard per 41 primary samples reasonably reflect expected values supporting the general accuracy of assaying.
- Inter-laboratory Intertek repeats broadly support the general accuracy of primary SGS assaying. However, reliability of the repeat assays has not been established with sufficient confidence for these data to represent a robust check of primary assaying.
- In 2016 a consultant geologist employed by Cardinal collected and submitted 49 independent duplicate samples to SGS Tarkwa. For 15 intervals with original SGS Tarkwa assays, the duplicate results closely match original assays by SGS Tarkwa. For 34 intervals with original SGS Ouagadougou assays, the duplicate results average around 35% higher than the original assays. Reasons for this trend are unclear.
- In 2018 a consultant geologist collected and submitted 165 quarter core duplicates to ALS Ireland giving very similar average grades to original SGS Tarkwa assays providing additional confidence in general reliability of SGS Tarkwa assays.

- Available density information totals 11,047 immersion measurements by Cardinal and SGS. For oxidation/rock units with reasonable numbers of Cardinal and SGS measurements, SGS measurements give similar average values to Cardinal providing some confidence in the reliability of Cardinal's density measurements.

The quality control measures adopted for assaying of the Namdini resource drilling 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.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

The information in this Section is derived from IMO (2019).

13.1 Introduction

The feasibility testwork reported in this Section follows on from the PFS testwork reported in Golder (2018b).

A key focus of this current phase has been the assessment of the MMSA Aachen™ shear reactor technology targeting increased recovery and reduction in cyanide consumption over the Base Case reported at the close of the PFS. The FS test scope included Aachen shear reactor pilot testing at the MMSA laboratory in South Africa as well as a range of Process Design Criteria ("**PDC**") supporting testwork performed by ALS in Perth. Note that all results and data for "Baseline" in this Report are results that exclude the application of the Aachen technology.

The following metallurgical flowsheet development testing is reported:

- Comprehensive master and variability composite head assay analysis
- Bulk Knelson gravity gold recovery and intensive leaching of Knelson concentrate performed by MMSA and ALS
- Batch and bulk flotation variability and copper sulphate optimization testing conducted by ALS
- Bulk flotation conducted by MMSA (South Africa) in assessing the application of Aachen shear reactor technology
- Aachen reactor sighter and feasibility piloting by MMSA targeting improved gold recovery and reduced cyanide consumption
- Concentrate leaching by ALS to assess the impact of lead nitrate dosing on the Baseline leach extraction
- Flotation concentrate and tailings flocculant screening, static and dynamic thickening by ALS
- Slurry rheology characterisation by ALS on key process streams: flotation concentrate, concentrate thickener underflow, CIL feed and tailings and flotation tailings thickener underflow
- Bulk concentrate leaching and activated carbon kinetic and equilibrium loading testwork by ALS
- Bulk concentrate leaching and cyanide detoxification testing completed by ALS.

13.2 Sample provenance

13.2.1 Sample provenance (PFS)

All relevant metallurgical testwork completed and reported in the Pre-feasibility Study (PFS) has been utilised in this Feasibility Study (FS) as the testwork results are still relevant and forms the basis in supporting the current Cardinal flowsheet and process design criteria. The process flowsheet has not changed from the PFS to the issuance of this FS.

For the PFS, 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.

13.2.2 Samples for the Starter Pit master composite

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:

- 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.

13.2.3 Flotation samples (S-MC)

The holes, intersection lengths, samples and predicted grades to make the LOM flotation optimization sulphide master composite sample ("**S-MC**") as shown in Table 51, were:

- GRA: five holes, 60 m (30% of the combined sample), 12 samples of 5 m composite length, 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 51: 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

13.2.4 Life of Mine samples (LOM)

The holes sampled to produce the LOM metallurgical sample, subdivided by dominant lithology as shown in Table 52, 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 52: 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
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

13.2.5 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 53.

Table 53: Variability sample – target grade ranges

Metavolcanic, Granite and Diorite	
Gold (g/t Au)	Sulphur (% 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 54 shows the statistics of the samples tested for metallurgical analysis.

Table 54: 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.

13.2.6 Comminution samples

13.2.6.1 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 55. Figure 50 to Figure 52 shows typical core selected.

Table 55: 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 50: Variability comminution (SMC) sample – Metavolcanic HQ NMDD008



Figure 51: Variability comminution (SMC) sample – Granite HQ NMDD005

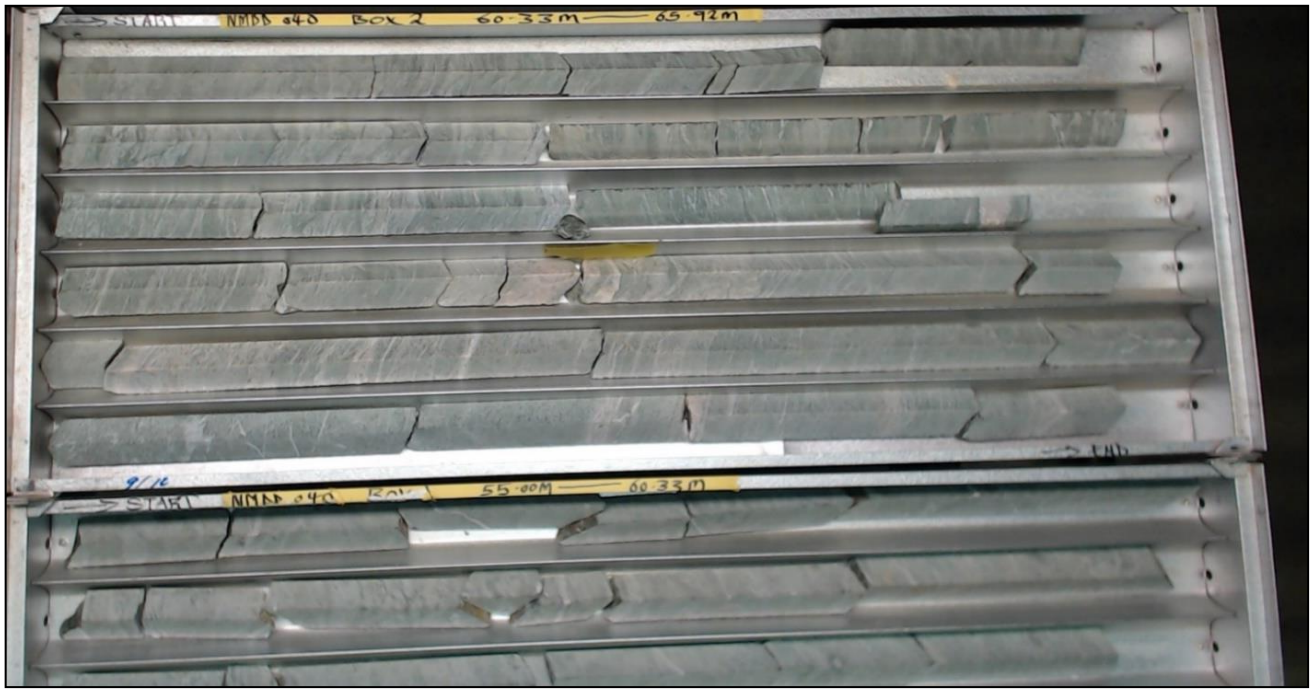


Figure 52: Variability comminution (SMC) sample – Diorite HQ NMDD040

13.2.6.2 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 56 and shown for diorite in Figure 53.

Table 56: 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 53: JKDWT comminution sample – Diorite PQ NMDD153

13.2.6.3 Sample spatial distribution indication

The spatial distribution of all the metallurgical sampling is shown in Figure 54 to Figure 58 in relation to the proposed Namdini Pit.

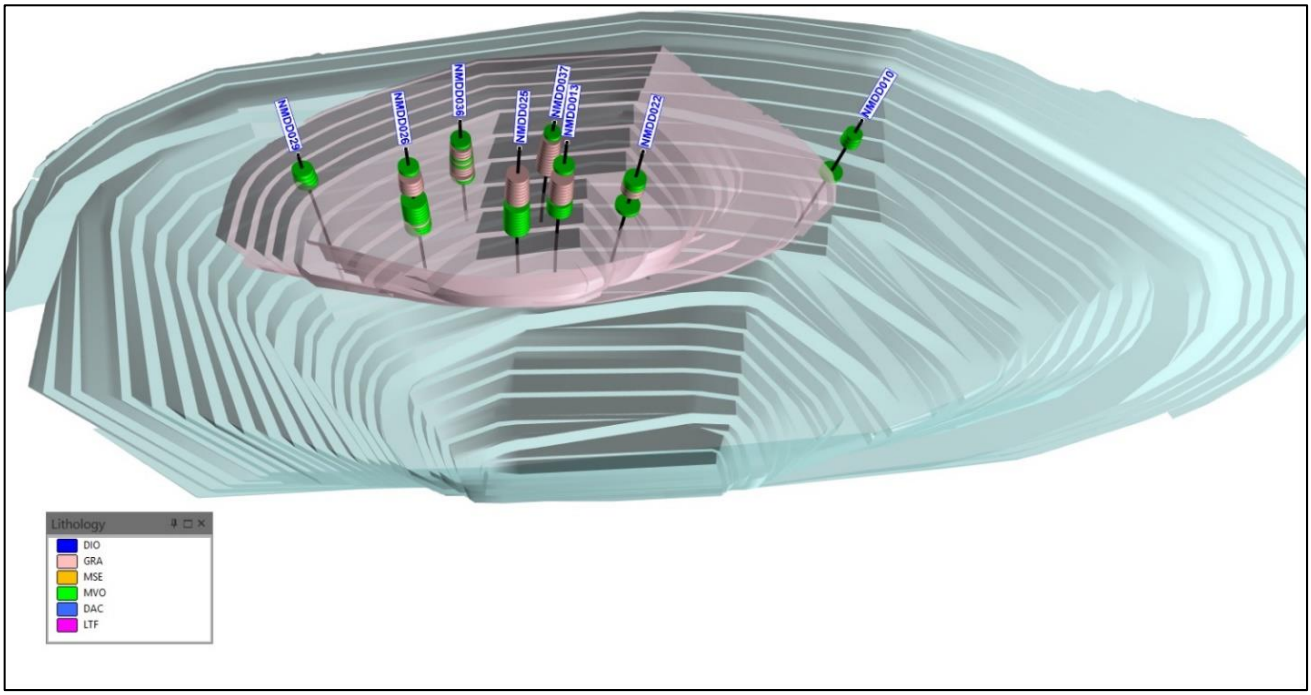


Figure 54: Provenance of composites in the Namdini Starter Pit

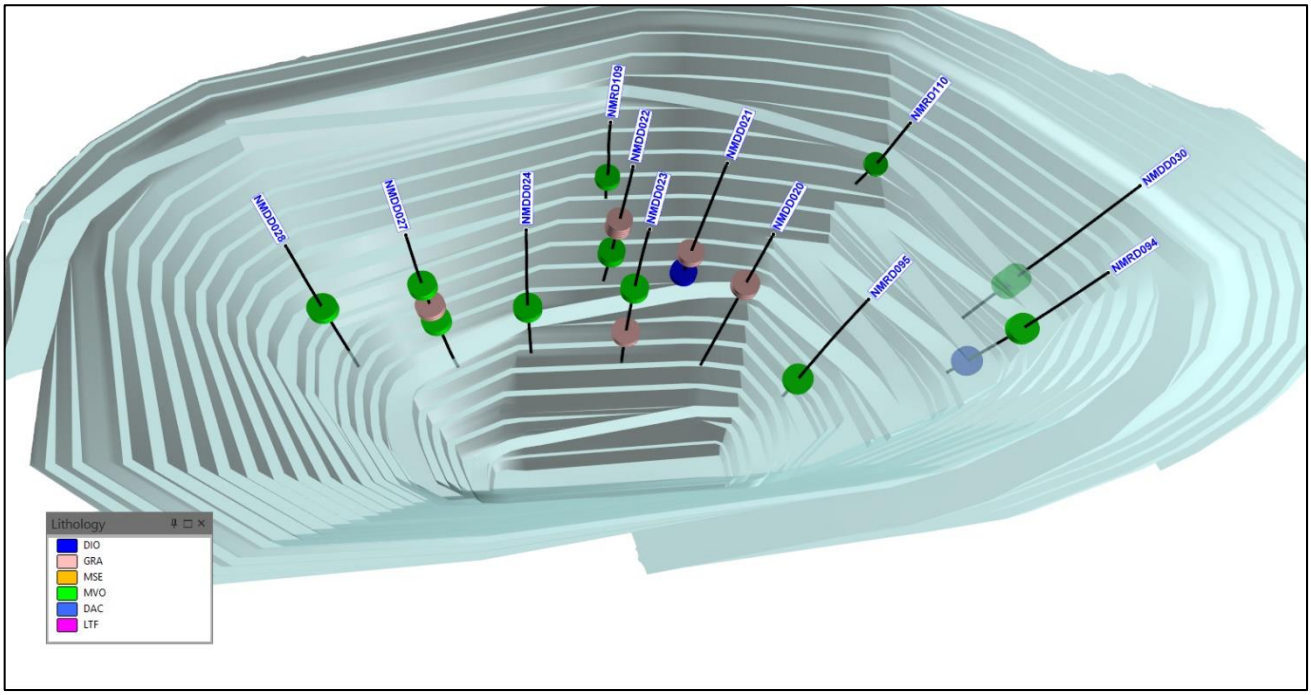


Figure 55: Provenance of the flotation composites in the Namdini Final Pit

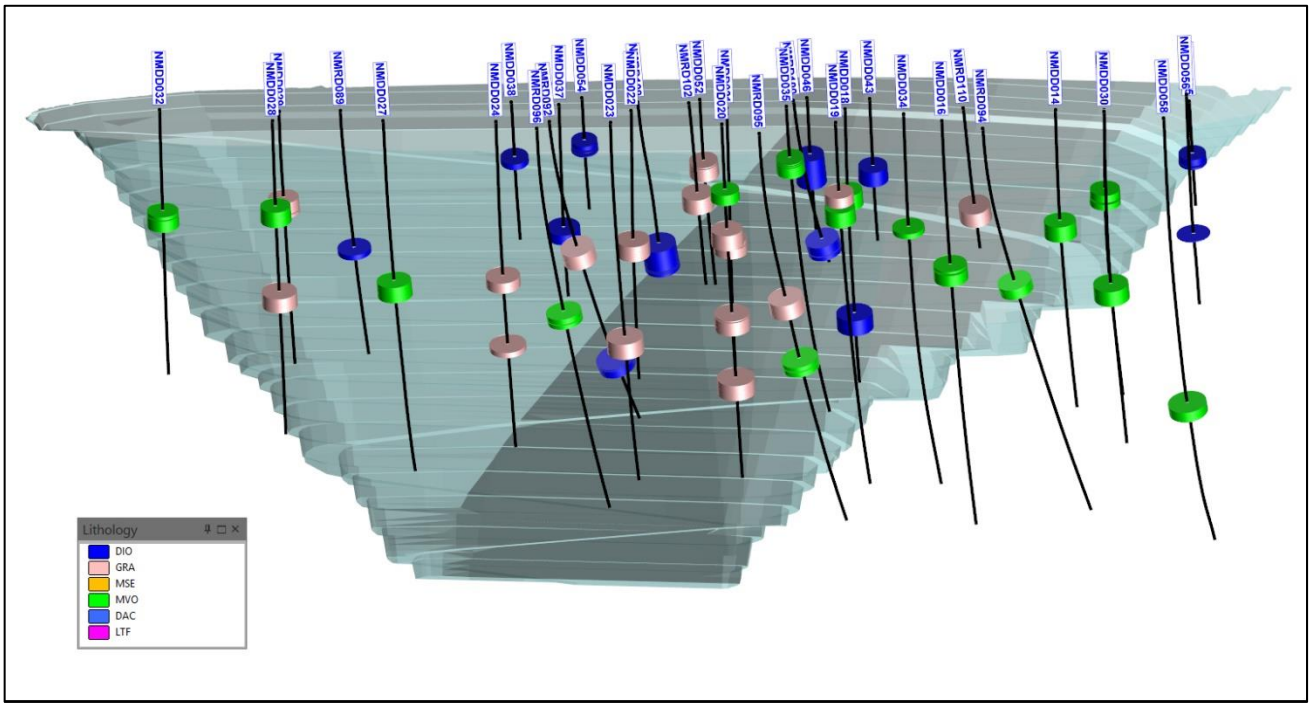


Figure 56: Provenance of the Life of Mine Composites in the Namdini Final Pit

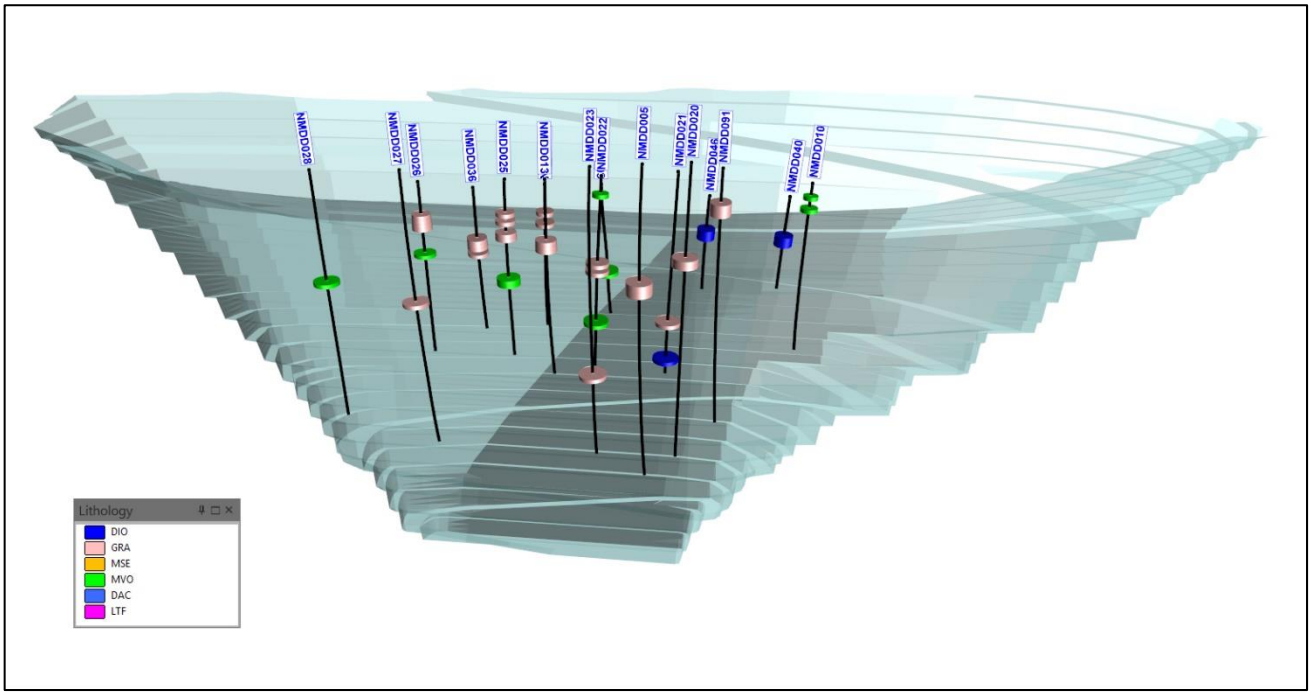


Figure 57: Provenance of the variability composites in the Namdini Final Pit

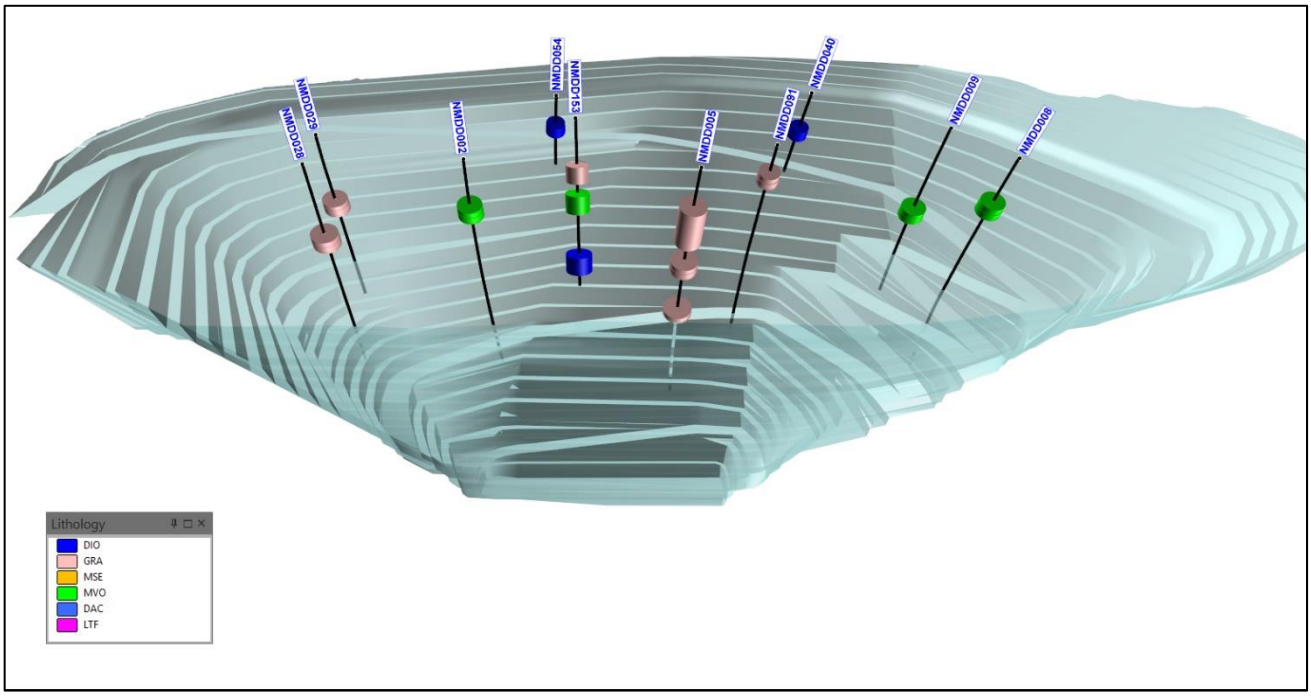


Figure 58: Provenance of the comminution composites in the Namdini Final Pit

13.2.7 Sample provenance (Feasibility Study)

In selecting representative samples for this FS and in addition to the PFS, 1 m intervals and 5 m downhole composite lengths were used to incorporate internal dilution that would occur during normal mining operations. DD core was used to produce the majority of the metallurgical composite samples of the different lithologies, with a small proportion of RC drill chips being selected.

13.2.8 ALS LOM lithology composites

The holes sampled to produce the LOM metallurgical sample, subdivided by dominant lithology as shown in Table 57, were:

- MVO: 25 drill holes comprising 67 composite samples, with an average grade of 1.00 g/t Au.
- GRA: 61 drill holes comprising 127 composite samples, with an average grade of 0.86 g/t Au.
- DIO: 1 drill hole comprising 3 composite samples, with an average grade of 1.50 g/t Au.

Table 57: ALS LOM Lithology composite selection

Element	Number Intervals	Total Weight (kg)	Mean	Standard Deviation	Minimum	Maximum
LOM Metavolcanics Composite (MVO)						
Au (g/t)	67	297	1.00	0.55	0.42	2.68
As (ppm)			735	876	68	6406
S (ppm)			1,071	685	190	3,170
LOM Granite Composite (GRA)						
Au (g/t)	127	353	0.86	0.48	0.39	4.30
As (ppm)			397	503	71	3,490
S (ppm)			878	202	450	1,790
LOM Diorite (DIO)						
Au (g/t)	3	26	1.50	0.58	0.69	2.03
As (ppm)			1,671	1,481	506	3,760
S (ppm)			1,707	0.05	1,630	1,750

13.2.9 ALS variability composites

The holes sampled to produce the LOM metallurgical sample, subdivided by dominant lithology as shown in Table 58, were:

- MVO: 4 drill holes comprising 4 composite samples, with an average grade of 2.20 g/t Au
- GRA: 4 drill holes comprising 4 composite samples, with an average grade of 1.31 g/t Au.

Table 58: ALS variability composite selection

Element	Number Intervals	Total Weight (kg)	Mean	Standard Deviation	Minimum	Maximum
LOM Metavolcanics Composite (MVO)						
Au (g/t)	4	33	2.20	0.49	1.56	2.69
As (ppm)			571	268	196	941
S (ppm)			1,005	606	450	1,950
LOM Granite Composite (GRA)						
Au (g/t)	4	31	1.31	0.22	1.05	1.56
As (ppm)			384	74	298	501
S (ppm)			933	184	800	1,250

13.2.10 MMSA LOM and variability composites

The holes sampled to produce the LOM metallurgical sample, subdivided by dominant lithology as shown in Table 59, were:

- MVO: 5 drill holes comprising 5 composite samples, with an average grade of 2.15 g/t Au
- GRA: 5 drill holes comprising 5 composite samples, with an average grade of 1.28 g/t Au
- DIO: 10 drill holes comprising 10 composite samples, with an average grade of 1.50 g/t Au.

Table 59: MMSA LOM and variability composite selection

Element	Number Intervals	Total Weight (kg)	Mean	Standard Deviation	Minimum	Maximum
LOM Metavolcanics Composite (MVO)						
Au (g/t)	5	666	2.15	2.32	1.03	7.52
LOM Granite Composite (GRA)						
Au (g/t)	5	242	1.28	1.14	0.33	3.36
LOM Diorite (DIO)						
Au (g/t)	10	346	1.50	0.74	0.52	3.14

13.2.10.1 Sample spatial distribution indication

The spatial distribution of all the FS metallurgical sampling is shown in Figure 59 to Figure 61 in relation to the proposed Namdini Pit. Refer to Cardinal's PFS NI 43-101 for the PFS sample provenance, the results of which have been carried into the FS due to their relevance to the overall testwork results and process design criteria basis.

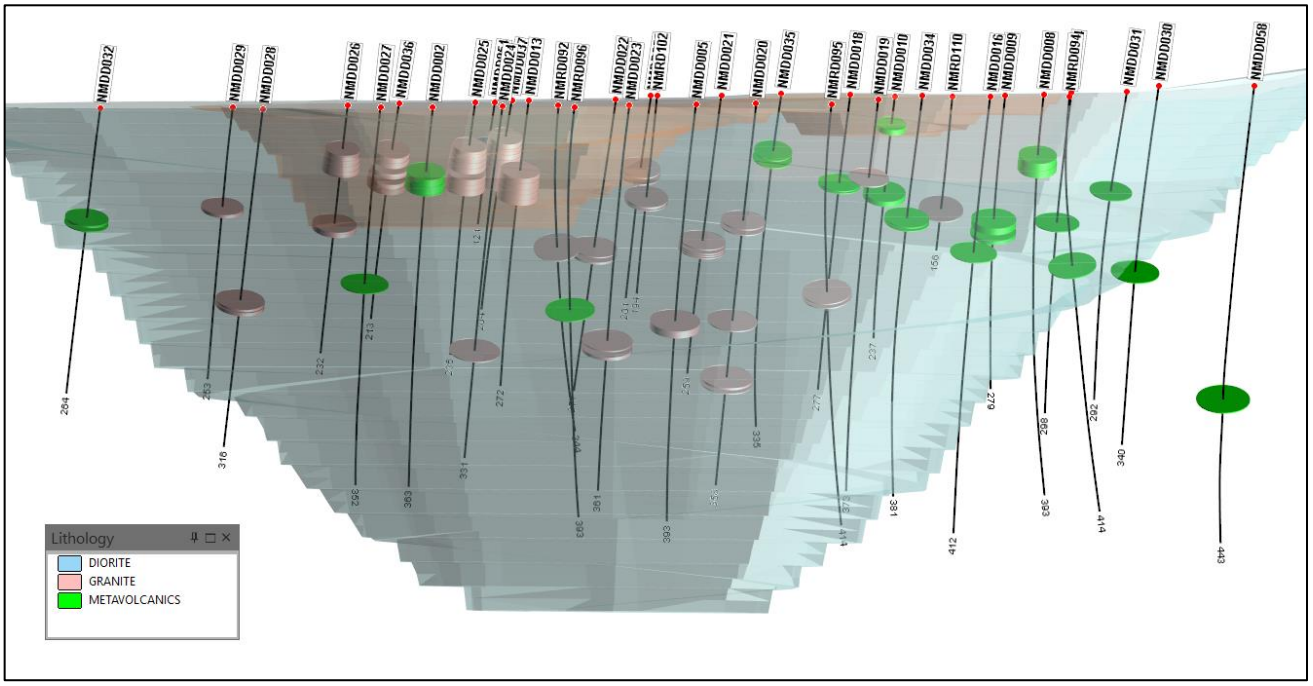


Figure 59: Provenance of the ALS Life of Mine composites in the Namdini Final Pit

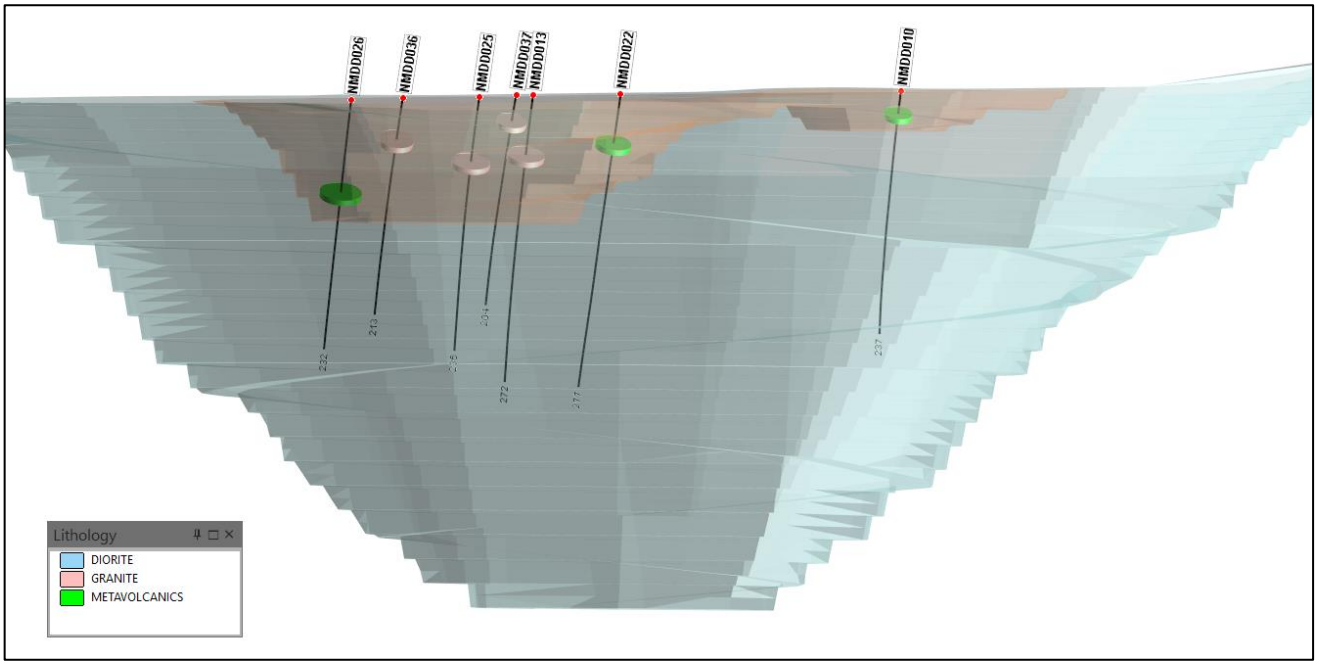


Figure 60: Provenance of the ALS variability composites in the Namdini Starter Pit

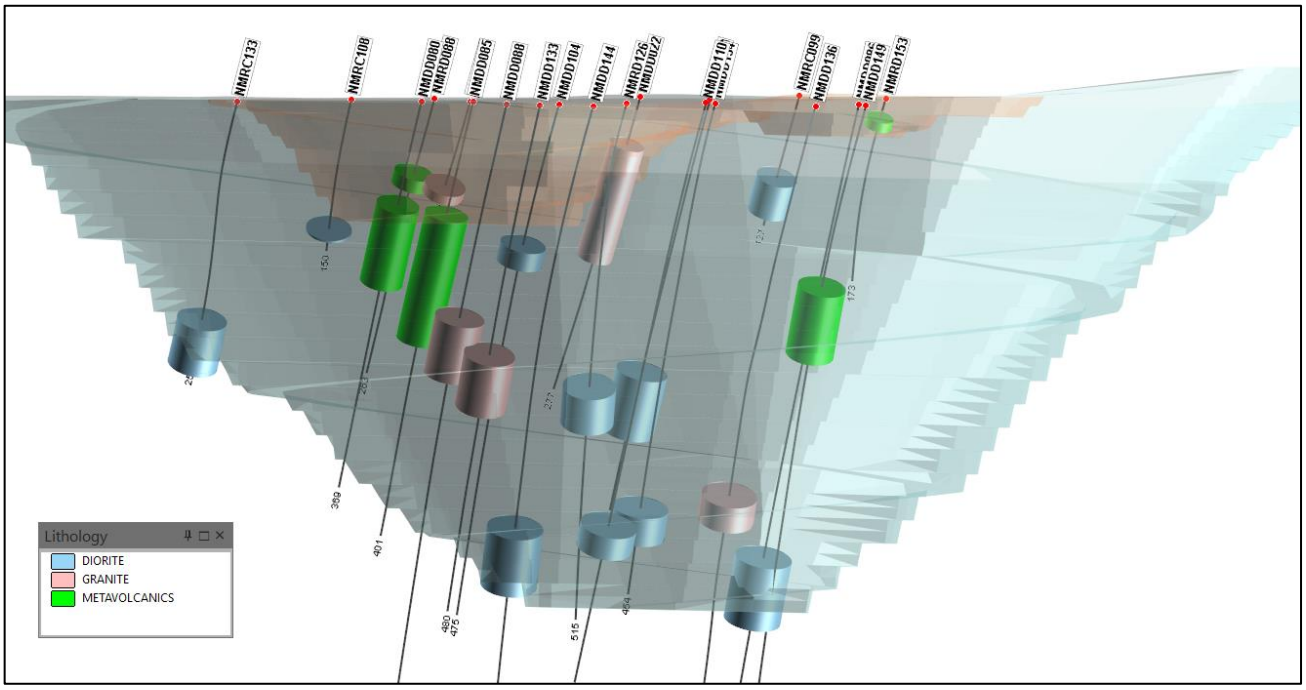


Figure 61: Provenance of the MMSA Life of Mine and variability composites in the Namdini Final Pit

13.3 Composite head analysis

13.3.1 ALS lithology composites

ALS composite head assaying was conducted based on:

- Duplicate gold fire assay
- Total sulphur and carbon by LECO combustion, including multi-element ICP assay.

Table 60: ALS lithology composite – head analysis

Analyte	Method	Unit	Composite				
			LOM-MVO-LC	LOM-GRA-LC	SP-MVO-LC	SP-GRA-LC	SP-DIO-LC
Au	Fire Assay 1	g/t	1.67	1.57	2.02	1.18	1.01
Au	Fire Assay 2	g/t	1.63	1.34	2.00	1.18	1.06
Average Au Head (Assay)		**g/t**	**1.65**	**1.46**	**2.01**	**1.18**	**1.04**
Ag	D3 digest/ICP	ppm	<2	<2	<2	<2	<2
As	D3 digest/ICP	ppm	970	330	720	280	790
Sb	D1 digest/ICP	ppm	2.2	4.4	5.2	4.1	1.6
Te	D1 digest/ICP	ppm	0.4	0.4	0.6	0.8	0.2
Hg	D1 digest/ICP	ppm	0.1	<0.1	0.1	0.2	0.1
Cu	D3 digest/ICP	ppm	54	24	46	16	88
Pb	D3 digest/ICP	ppm	15	10	20	25	20
Ni	D3 digest/ICP	ppm	135	15	125	15	80
Zn	D3 digest/ICP	ppm	80	42	86	56	102
Fe	D4Z digest/ICP	%	5.48	1.50	4.96	1.74	7.40
S Total	CS2000	%	0.82	1.10	0.88	0.94	0.96
Sulphide	CS2000-Sherritt	%	0.80	0.78	0.82	0.90	0.92
C Total	CS2000	%	2.88	0.90	2.55	0.90	2.01
Organic C	CS2000	%	0.03	0.03	<0.03	<0.03	0.06
Carbonate	Calculated	%	2.85	0.87	2.55	0.90	1.95

Key analytes are presented in Table 60 and summarised as follows:

1) Reasonable repeatability is indicated in duplicate gold fire assays. Head grade ranged from 1.04 g/t in SP-DIO to a maximum of 2.01 g/t in SP-MVO.

2) Silver assayed below a 2 ppm detection limit; later stage calculated silver assays indicate silver below 1 ppm.

3) Mercury assay is noted low, typically close to detection limit.

4) Total sulphur ranged from 0.82% to 1.10%; associated sulphide sulphur assays indicate amenability to recovery by flotation.

5) Arsenic ranged from 280 ppm in SP-GRA to 970 ppm in LOM-MVO.

6) Total carbon reported from 0.90% to 2.88% with low organic carbon, 0.03% to 0.06%. Organic carbon assay is consistent with early PFS composites that also reported a low Pre-Robbing Index of 0.06.

7) Cyanide consuming elements: Cu, Zn, Pb, and Ni reported within similar ranges to earlier PFS composites.

8) Later stage testing has indicated cyanide soluble copper above levels expected assuming copper predominantly as chalcopyrite. While a proportion of cyanide soluble copper is associated with copper sulphate dosed during flotation, cyanide solubilised copper may be present due to tennantite, a copper-arsenic sulphosalt identified in early PFS QEMSCAN mineralogy.

13.3.2 ALS variability composites

ALS conducted variability testwork on MVO and GRA composites derived within the Starter Pit. Head assays presented in Table 61 indicate reasonable comparability between duplicate gold fire assays with average grades for MVO ranging from 1.78 to 4.05 g/t; GRA composites reported lower ranging from 0.59 to 1.14 g/t.

Relative to LOM lithology composites in Table 60:

1) Arsenic and antimony were represented over a wider range with a maximum arsenic at 1,640 ppm and a maximum antimony assay of 12 ppm and tellurium at 1.6 ppm (MVO Composite V2).

2) Silver and mercury reported similarly with mercury below detection limit.

3) Copper and nickel reported higher with respective average grades of 75 ppm and 262 ppm. MVO variability composites tended higher in copper than GRA averaging 103 ppm.

4) Lead and zinc assay reported within similar ranges to LOM composites. Zinc was elevated in MVO variability composites relative to GRA.

Table 61: ALS Variability Composite – Head Analysis

Analyte	Unit	Starter Pit MVO				Starter Pit GRA			
		V1	V2	V3	V4	V5	V6	V7	V8
Au (Assay 1)	g/t	2.90	1.75	2.14	3.67	1.08	0.56	1.13	1.07
Au (Assay 2)	g/t	2.93	1.80	2.25	4.42	0.94	0.61	1.15	1.11
Au Average	**g/t**	**2.92**	**1.78**	**2.20**	**4.05**	**1.01**	**0.59**	**1.14**	**1.09**
Ag	ppm	<2	<2	<2	<2	<2	<2	<2	<2
As	ppm	620	1640	300	700	560	520	190	360
Sb	ppm	1.1	12.0	2.4	3.4	3.2	2.4	4.8	4.5
Te	ppm	0.4	1.6	0.4	0.6	0.4	0.4	0.6	0.4
Hg	ppm	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1
Cu	ppm	92	102	110	110	42	42	58	46
Pb	ppm	<5	30	10	<5	10	15	<5	20
Ni	ppm	265	295	320	280	230	240	230	235
Zn	ppm	100	76	128	80	48	50	40	42
Fe	%	5.52	5.20	5.92	4.42	1.88	1.82	2.58	1.86

13.3.3 MMSA LOM lithology composites

MMSA conducted Aachen reactor leach testing on LOM lithology composites; umpire assays reported by ALS are presented below in Table 62. Relative to ALS testwork composites:

1) Triplicate fire and screen fire assays demonstrate reasonable comparability.

2) Silver and mercury report similarly with mercury close to detection limit.

3) Arsenic ranged from 600 to 900 ppm typically above mine plan range (347-547 ppm) with maximal arsenic reported for GRA; antimony ranged from 2.7-4.6 ppm similarly to ALS testwork composites.

4) Copper, lead and zinc assayed within similar ranges to ALS LOM lithology composites.

5) Sulphide sulphur assayed from a minimum of 0.68% (LOM-MVO) to a maximum of 1.60% in LOM-DIO.

6) Organic carbon was typically higher, averaging 0.06% relative to 0.03% for ALS LOM composites.

Table 62: LOM lithology composite – head analysis

Analyte	Method	Unit	Composite		
			LOM-MVO	LOM-GRA	LOM-DIO
Au	Fire Assay 1	g/t	0.94	1.55	1.50
Au	Fire Assay 2	g/t	0.82	2.06	1.60
Au	Fire Assay 3	g/t	0.87	1.62	1.48
Average Au Head (Assay)		**g/t**	**0.88**	**1.74**	**1.53**
Au Screen Fire Assay		**g/t**	**0.91**	**1.62**	**1.39**

Analyte	Method	Unit	Composite		
			LOM-MVO	LOM-GRA	LOM-DIO
Ag	D3 digest/ICP	ppm	<2	<2	<2
As	D3 digest/ICP	ppm	600	900	600
Sb	D1 digest/ICP	ppm	2.70	4.60	2.00
Te	D1 digest/ICP	ppm	0.40	0.40	0.40
Hg	D1 digest/ICP	ppm	0.20	0.10	0.10
Cu	D3 digest/ICP	ppm	32.0	54.0	70.0
Pb	D3 digest/ICP	ppm	<5	<5	<5
Ni	D3 digest/ICP	ppm	95.0	135.0	100.0
Zn	D3 digest/ICP	ppm	60.0	70.0	78.0
Fe	D4Z digest/ICP	%	4.48	5.99	6.61
S Total	CS2000	%	0.70	0.94	1.86
Sulphide	CS2000-Sherritt	%	0.68	0.86	1.60
C Total	CS2000	%	1.44	2.37	2.40
Organic C	CS2000	%	0.06	0.06	0.06

13.3.4 MMSA grade-variability composites

MMSA performed Aachen reactor grade variability testing across lithology types based on the composites presented for MVO and GRA in Table 63 with DIO and an overarching LOM Blend composite reported in Table 64.

Table 63: MVO and GRA grade variability composite – head analysis

Analyte	Method	Unit	MVO			GRA		
			MVO-1	MVO-2	MVO-3	GRA-1	GRA-2	GRA-3
			NMDD085	NMDD080	NMDD149	NMDD088	NMDD136; NMDD022; NMRC085	NMDD133
Au	Fire Assay 1	g/t	2.07	1.02	1.69	2.24	1.35	0.45
Au	Fire Assay 3	g/t	2.21	0.98	1.91	1.97	2.15	0.43
Au	Fire Assay 3	g/t	2.33	1.01	1.70	2.00	1.96	0.41
Average Au Head (Assay)		**g/t**	**2.20**	**1.00**	**1.77**	**2.07**	**1.82**	**0.43**
Au Screen Fire Assay		**g/t**	**2.37**	**1.24**	**2.36**	**1.96**	**1.95**	**0.45**
Ag	D3 digest/ICP	ppm	<2	<2	<2	<2	<2	<2
As	XRF BM	ppm	2100	600	1200	400	200	200
Sb	D1 digest/ICP	ppm	2.20	1.70	3.50	4.40	3.95	3.90
Te	D1 digest/ICP	ppm	0.40	0.20	0.40	0.60	0.50	0.40
Hg	D1 digest/ICP	ppm	<0.1	0.10	0.10	0.10	0.20	0.20
Cu	D3 digest/ICP	ppm	53.0	68.0	44.0	48.0	20.0	20.0
Pb	D3 digest/ ICP	ppm	<5	<5	<5	<5	<5	<5
Ni	D3 digest/ ICP	ppm	145.0	150.0	105.0	15.0	30.0	30.0
Zn	D3 digest/ ICP	ppm	71.0	70.0	60.0	42.0	53.0	44.0
Fe	D4Z digest/ ICP	%	6.21	6.35	4.98	1.75	2.56	2.47
S Total	CS2000	%	0.94	0.56	0.86	0.92	1.19	0.56
Sulphide	CS2000-Sherritt	%	0.87	0.56	0.80	0.78	0.93	0.40
C Total	CS2000	%	2.01	1.77	3.48	0.99	1.41	1.29
Organic C	CS2000	%	0.03	0.03	0.03	0.06	<0.03	0.06

Table 64: DIO and LOM Blend – grade variability composite head analysis

Analyte	Method	Unit	DIO				LOM Blend
			DIO-1	DIO-2	DIO-3	DIO-4	
			NMDD098	NMDD110; NMRD126	NMDD144; NMDD104; NMDD134	NMRC133; NMDD086	
Au	Fire Assay 1	g/t	1.35	1.50	0.67	2.15	2.51
Au	Fire Assay 2	g/t	1.23	1.38	0.79	2.05	2.17
Au	Fire Assay 3	g/t	1.14	1.43	0.72	2.33	2.13
Average Au Head (Fire Assay)		**g/t**	**1.24**	**1.44**	**0.73**	**2.18**	**2.27**
Au Screen Fire Assay		**g/t**	**1.26**	**1.57**	**0.78**	**1.95**	**2.08**
Ag	D3 digest/ICP	ppm	<2	<2	<2	<2	<2
As	D3 digest/ICP	ppm	300	800	200	1100	900
Sb	D1 digest/ICP	ppm	1.90	1.70	1.10	1.60	2.60
Te	D1 digest/ICP	ppm	0.20	0.20	<0.2	0.80	0.60
Hg	D1 digest/ICP	ppm	0.20	0.20	0.20	0.20	0.10
Cu	D3 digest/ICP	ppm	64.0	76.0	68.0	78.0	44.0
Pb	D3 digest/ICP	ppm	<5	<5	<5	<5	<5
Ni	D3 digest/ICP	ppm	75.0	145.0	60.0	110	110.0
Zn	D3 digest/ICP	ppm	74.0	74.0	72.0	76.0	66.0
Fe	D4Z digest/ICP	%	6.53	7.21	6.01	7.03	5.17
S Total	CS2000	%	1.16	2.32	1.30	2.12	0.94
Sulphide	CS2000-Sherritt	%	1.40	1.94	1.02	1.80	0.88
C Total	CS2000	%	2.10	2.40	1.78	2.34	2.10
Organic C	CS2000	%	0.03	<0.03	0.03	0.06	0.06

13.4 Gravity recovery and intensive leach extraction

During the PFS gravity recovery was characterised by primary grinding targeting a P_{80} of 250 µm followed by single pass (3") Knelson Concentrator recovery and mercury amalgamation of Knelson concentrate. FS testwork incorporated identical grinding and Knelson recovery with concentrate beneficiation to present mass recoveries to intensive leaching more consistent with expectation at scale. Pre-beneficiated concentrate was subjected to intensive leaching and subsequent process streams recombined consistent with the intended process flowsheet as follows:

- Knelson gravity concentrator tailings recombined with Knelson concentrate beneficiated tailings, the combined stream reporting to rougher flotation feed.
- Beneficiated Knelson concentrate subjected to intensive leaching with intensive leach tailings deporting to rougher flotation concentrate for UFG milling and CIL. In the case of MMSA composites, flotation concentrate was subjected to prior pre-aeration and Aachen reactor oxidation ahead of CIL leaching.

13.4.1 Conclusions

The following conclusions are provided based on the results reported below:

1) ALS testwork indicates Starter Pit blend with an average head grade of 1.34 g/t delivers a recovered grade of 0.165 g/t representing a recovery of 14.4% at a nominal full-scale Knelson concentrator mass recovery of 0.025% into intensive leach.

2) Gravity recoveries estimated for the LOM Blend at a head grade of 1.14 g/t by separate testwork performed at ALS and MMSA indicate MVO recovery ranging from 4.4-9.0%; GRA 5.0-8.4% and DIO 1.0-1.8%.

3) The estimated gravity recovery for the LOM Blend with head grading 1.14 g/t is 5.0-6.4% based on separate testing conducted by ALS and MMSA.

4) Low gravity recoveries are primarily a result of the coarse nature of concentrate into intensive leach and expected mass recovery at scale (0.025%). Deferral of coarse gravity gold by intensive leach tailings to flotation concentrate, UFG and CIL leaching mitigates recovery loss.

5) The retention of gravity recovery by Knelson Concentrator and intensive leach is endorsed, given the impact of coarse gold deporting to the flotation circuit and elevation in flotation tailings grade.

6) Cyanide soluble copper was noted in leach liquors with extraction at 8.7% for Starter Pit blend composite and a maximum of 51.7% extraction reported for LOM GRA. Arsenic extraction was in the range 0.03% – 0.08% with a maximum of 0.23% reported for LOM DIO.

13.4.2 ALS lithology composite gravity recovery

13.4.2.1 Gravity recovery

ALS performed gravity and intensive leaching on the following lithology composites:

- Starter Pit MVO, GRA, and DIO
- LOM MVO and GRA
- A bulk composite comprising 58.8% MVO and 41.2% GRA Starter Pit lithologies.

Concentrate beneficiation was achieved by hand panning achieving a typical mass recovery, 0.09% of whole ore mass; subsequent intensive leaching was conducted under the following conditions:

- 10% Solids (w/w)
- NaOH dosing to maintain pH > 11
- NaCN concentration maintained at 2%
- Leach Well 2%
- Lead nitrate 2 kg/t
- Dissolved oxygen maintained > 20 ppm by O_2 sparging.

Table 65: ALS gravity concentrate intensive leach summary

Composite	Gold				Silver				Copper			
	Assay Head (g/t)	Con (g/t) Ore	Pan Con (g/t) (*)	Extn (%)	Assay Head (g/t)	Con (g/t) Ore	Pan Con (g/t) (*)	Extn (%)	Assay Head (g/t)	Con (g/t) Ore	Pan Con (g/t) (*)	Extn (%)
SP-MVO-LC	2.01	0.36	414	89.5	< 2	0.027	30.4	76.3	46	0.31	359	16.4
SP-GRA-LC	1.18	0.29	319	92.3	< 2	0.024	26.5	80.8	16	0.14	152	34.1
SP-DIO-LC	1.04	0.03	39	31.4	< 2	0.003	4.5	39.4	88	0.18	509	21.4
LOM-MVO-LC	1.65	0.14	149	74.7	< 2	0.009	10.1	70.4	54	0.23	246	18.8
LOM-GRA-LC	1.46	0.19	210	89.6	< 2	0.017	18.8	76.0	24	0.09	103	51.7
SP-BULK	1.67	0.34	290	91.3	< 2	0.026	22.5	80.0	34	1.47	329	8.7

Note: * Pan Con is panned gravity concentrate grade @ 0.09% concentrate mass.

Results in Table 65 are summarised as follows:

- Gold and silver extraction was proportional to gravity head grade up to a limit in the range 89.5-92.3%.
- Cyanide soluble copper was evidenced; extraction reduced with increasing copper deportment to concentrate; leach extraction ranged from 8.7% for SP-Bulk composite to 51.7% for LOM-GRA.
- PFS QEMSCAN mineralogy detected both chalcopyrite, which is sparingly cyanide soluble and tennantite, a copper/arsenic sulphosalt $Cu_{12}As_4S_{13}$. Tennantite exists in solid solution series with tetrahedrite $Cu_{12}Sb_4S_{13}$ where antimony substitutes for arsenic. Both forms exhibit significantly higher cyanide solubility than chalcopyrite and contain high molar ratios of copper.

Test results are presented below for Starter Pit and LOM composites including scaled values representing gravity recoveries expected at a nominal full-scale Knelson concentrate mass recovery of 0.025%:

- Starter Pit lithology results (Table 66) report gravity recovery as 17.7% based on the Starter Pit blend and composite head grading 1.65 g/t. Starter Pit gravity recovery at the mine plan head grade of 1.34 g/t is estimated as 14.4%.
- LOM lithology test data (Table 67) reported 7.0% gravity recovery based on the LOM Blend and a composite head grading 1.50 g/t. LOM gravity recovery is estimated at 5.0% based on the mine plan head grade of 1.14 g/t.

Table 66: ALS Starter Pit lithology composite – gravity recovery summary

Criteria	Unit	Value (Testwork)			Value (Design @ 0.025%)		
		MVO	GRA	DIO	MVO	GRA	DIO
Head Grade	Au g/t	2.01	1.18	1.04	1.41	1.26	1.08
Concentrate Mass	%	0.09	0.09	0.07	0.025	0.025	0.025
Concentrate Grade	Au g/t	414.5	318.8	39.2	752.1	943.0	69.4
Leach Residue	Au g/t	43.7	24.6	26.9	43.7	24.6	26.9
Extraction	%	89.5	92.3	31.4	94.2	97.4	61.3
Recovered Grade	Au g/t Con	370.8	294.2	12.3	708.4	918.4	42.5
Recovered Grade	**Au g/t Ore**	**0.325**	**0.264**	**0.009**	**0.177**	**0.230**	**0.011**
Recovery	**%**	**16.2**	**22.4**	**0.9**	**12.6**	**18.2**	**1.0**
SP Calculated Head	**Au g/t**	**1.65**			**1.34**		
SP Blend Recovery	**%**	**17.7**			**14.4**		

Table 67: ALS LOM lithology composite – gravity recovery summary

Criteria	Unit	Value (Testwork)			Value (Design @ 0.025%)		
		MVO	GRA	DIO	MVO	GRA	DIO
Head Grade	Au g/t	1.65	1.46	1.04	1.17	1.12	1.04
Concentrate Mass	%	0.09	0.09	0.07	0.025	0.025	0.025
Concentrate Grade	Au g/t	149.2	210.0	39.2	244.4	399.6	67.0
Leach Residue	Au g/t	37.7	21.9	26.9	37.7	21.9	26.9
Extraction	%	74.7	89.6	31.4	84.6	94.5	59.9
Recovered Grade	Au g/t Con	111.5	188.1	12.3	206.7	377.7	40.1
Recovered Grade	**Au g/t Ore**	**0.10**	**0.17**	**0.01**	**0.052**	**0.094**	**0.010**
Recovery	**%**	**6.2**	**11.6**	**0.9**	**4.4**	**8.4**	**1.0**
LOM Calculated Head	**Au g/t**	**1.50**			**1.14**		
LOM Recovery	**%**	**7.0**			**5.0**		

13.4.2.2 Intensive leach metal extraction

Calculated head grades and metal extractions are presented below in Table 68 for Au, Ag, Cu, As, and Fe, notably:

- Silver and gold head grades and extractions were closely correlated; silver head grade ranged from 4.5 ppm in LOM DIO to 30.4 ppm in Starter Pit MVO. Extraction varied from 39.4% in LOM DIO to 80.8% in Starter Pit GRA.
- Cyanide soluble copper was in evidence; minimum extraction was 8.7% for Starter Pit blend composite with a maximum of 51.7% reported in the case of LOM GRA.
- Arsenic leach extraction reported generally in the range 0.03-0.08% with a maximum of 0.23% for LOM DIO.

Table 68: ALS gravity concentrate – intensive leach metal extraction

Case	Starter Pit						LOM					
Lithology	MVO		GRA		SP-Blend		MVO		GRA		DIO	
Mass %	0.0877		0.0897		0.1179		0.0916		0.0898		0.0724	
Element	Head (ppm)	Ext (%)	Head (ppm)	Ext (%)	Head (ppm)	Ext (%)	Head (ppm)	Ext (%)	Head (ppm)	Ext (%)	Head (ppm)	Ext (%)
Au	414.5	89.5	318.8	92.3	290.4	91.3	149.2	74.7	210.0	89.6	39.2	31.4
Ag	30.4	76.3	26.5	80.8	22.5	80.0	10.1	70.4	18.8	76.0	4.5	39.4
Cu	358.8	16.4	151.7	34.1	328.7	8.7	246.4	18.8	103.5	51.7	508.9	21.4
As	29018	0.06	23418	0.08	22609	0.04	39318	0.05	15104	0.03	22853	0.23
Fe	361062	0.02	407036	0.01	291054	0.02	335045	0.01	313045	0.01	389079	0.02

Table 69: ALS gravity concentrate – intensive leach liquor ICP analysis

Case	Starter Pit			LOM		
Lithology	MVO	GRA	SP-Blend	MVO	GRA	DIO
Analyte	mg/L	mg/L	mg/L	mg/L	mg/L	mg/L
As	2.00	2.00	2.00	2.00	<1.0	6.00
Cd	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5
Co	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5
Cr	<1.0	<1.0	<1.0	<1.0	<1.0	<1.0
Cu	6.60	5.80	3.20	5.20	6.00	12.40
Fe	7.00	4.00	6.00	5.00	5.00	9.00
Mn	1.00	1.00	1.00	<0.5	0.50	<0.5
Mo	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5
Ni	<0.5	<0.5	<0.5	<0.5	<0.5	<0.5
Pb	2.50	2.00	5.50	2.00	2.00	2.50
Zn	1.00	0.40	0.40	1.40	1.20	4.00

13.4.2.3 Intensive leach cyanide and caustic consumption

Intensive leach reagent consumptions are presented below in Table 70.

- Starter Pit blend reported 17.13 kg/t NaCN and 1.30 kg/t NaOH on a concentrate feed basis.
- LOM composites and LOM Blend returned reduced NaCN and similar NaOH consumption relative to Starter Pit composites; respectively 12.89 kg/t NaCN and 1.10 kg/t NaOH on a concentrate feed basis at 10% solids (w/w).

Table 70: ALS Starter Pit and LOM lithology composite – intensive leach NaCN and NaOH consumption

Composite	Concentrate Mass (%)	Kg/t Concentrate	
		NaCN	NaOH
SP-MVO-LC	0.09	17.32	1.32
SP-GRA-LC	0.09	16.89	1.29
SP-DIO-LC	0.07	16.77	1.10
SP-Blend	**0.09**	**17.13**	**1.30**
LOM-MVO-LC	0.09	12.60	1.05
LOM-GRA-LC	0.09	13.53	1.28
LOM-DIO	0.07	12.82	0.98
LOM-Blend	**0.09**	**12.89**	**1.10**

Note: LOM-GRA consumption scaled based on SP lithology reagent consumption.

13.4.3 MMSA lithology composite gravity recovery

MMSA conducted gravity and intensive leaching in assessing the application of Aachen Leachox technology. Staged recovery and leaching were completed on LOM and variable grade composites representing MVO, GRA, and DIO lithologies by bulk Knelson recovery followed by table beneficiation and intensive leaching under identical conditions to ALS.

13.4.3.1 LOM lithology composites

Results of LOM composite gravity recovery testwork are presented in Table 71 with intensive leach kinetics and extraction illustrated in Figure 62; summarised as follows:

- LOM lithology composites reported 7.7% gravity recovery based on the LOM Blend and a composite head grading 1.03 g/t. Estimated gravity recovery based on the mine plan head grade of 1.14 g/t is 6.8% compared to the estimate based on ALS testwork in Table 67 of 5.0%.
- Gravity recoveries estimated at the LOM head grade by separate testwork performed at ALS (Table 67) and MMSA (Table 71) indicate MVO recovery ranging from 4.4-9.0%; GRA 5.0-8.4% and DIO 1.0-1.8%.
- Relativity in leach extraction shown in Figure 62 is a function of lithology type and the extent of liberation, given no additional milling of gravity concentrate. This aspect is mitigated given intensive leach tailings deports to rougher flotation concentrate for UFG milling and CIL leaching.

Table 71: MMSA LOM lithology composite – gravity recovery summary

Criteria	Unit	Value (Testwork)			Value (Design @ 0.025%)		
		MVO	GRA	DIO	MVO	GRA	DIO
Head Grade	Au g/t	0.87	1.30	1.11	1.17	1.12	1.04
Concentrate Mass	%	0.08	0.06	0.08	0.025	0.025	0.025
Concentrate Grade	Au g/t	141.9	194.3	52.2	448.2	283.0	94.6
Leach Residue	Au g/t	19.9	65.9	22.8	26.9	56.7	21.4
Extraction	%	86.0	66.1	56.2	94.0	80.0	77.3
Recovered Grade	Au g/t Con	122.0	128.4	29.3	421.3	226.3	73.1
Recovered Grade	**Au g/t Ore**	**0.100**	**0.071**	**0.025**	**0.105**	**0.057**	**0.018**
Gravity Recovery	**%**	**11.5**	**5.4**	**2.2**	**9.0**	**5.0**	**1.8**
LOM Calculated Head	**Au g/t**	**1.03**			**1.14**		
LOM Recovery	**%**	**7.7**			**6.8**		



Figure 62: MMSA LOM composite intensive leach kinetic profiles

13.4.3.2 Variable Grade Lithology Composites

MMSA conducted gravity recovery testwork on 12 variable grade composites across lithology types and including and a blended composite representing LOM Blend proportions. Gravity recoveries estimated based on the testwork and a nominal gravity concentrate mass recovery of 0.025% are presented in Table 72, summarised as follows:

- Lower recovery was reported for discrete variable grade composites than for LOM composites formed within the same drill-holes across all intervals.
- MVO ranged from 2.5 to 4.2%, averaging 3.5% over the grade range 0.90-2.01 g/t relative to LOM MVO composite; 4.4-9.0% based on separate ALS and MMSA testwork.
- GRA recovery varied from 2.7 to 9.4%, averaging 5.6% over the head grade range 0.60-1.33 g/t, compared to 5.0-8.4% by ALS and MMSA testwork.
- DIO trended higher, reporting 1.8 to 4.0%, averaging 3.0% relative to 1.0-1.8% reported for LOM DIO.
- The LOM Blend composite formed based on the separate variable grade lithology composites returned 4.0% recovery at a head grade of 1.71 g/t.
- LOM recoveries reported by ALS and MMSA are considered representative with LOM Blend recovery in the range 5.0-6.8%.

Table 72: MMSA variable grade lithology composite – gravity recovery summary

Composite	MVO				GRA				DIO				LOM Blend
	MVO-1	MVO-2	MVO-3	MVO-4	GRA-1	GRA-2	GRA-3	GRA-4	DIO-1	DIO-2	DIO-3	DIO-4	
Calc Head Au g/t	1.82	1.86	0.90	2.01	1.33	1.25	1.29	0.60	1.06	1.17	0.72	1.17	1.71
Con Mass %	0.025	0.025	0.025	0.025	0.025	0.025	0.025	0.025	0.025	0.025	0.025	0.025	0.025
Con Au g/t	192.9	216.6	199.1	360.3	194.2	491.2	331.6	128.8	145.7	105.5	129.2	180.6	380.9
Leach Residue g/t	13.1	24.1	29.9	25.1	48.8	20.3	16.6	28.0	15.5	23.2	14.0	33.7	26.9
Extraction %	93.2	88.9	85.0	93.0	74.9	95.9	95.0	78.3	859.4	478.0	89.2	81.3	92.9

Composite	MVO				GRA				DIO				LOM Blend
	MVO-1	MVO-2	MVO-3	MVO-4	GRA-1	GRA-2	GRA-3	GRA-4	DIO-1	DIO-2	DIO-3	DIO-4	
Recovered g/t con	179.8	192.5	169.2	335.2	145.4	470.9	315.0	100.8	130.2	82.3	115.2	146.9	354.0
Recovered g/t ore	**0.045**	**0.048**	**0.042**	**0.084**	**0.036**	**0.118**	**0.079**	**0.025**	**0.003**	**0.021**	**0.029**	**0.037**	**0.089**
Recovery (%)	**2.5**	**2.6**	**4.7**	**4.2**	**2.7**	**9.4**	**6.1**	**4.2**	**3.1**	**1.8**	**4.0**	**3.1**	**5.2**
Ave Recovery (%)	**3.5**				**5.6**				**3.0**				**4.**

13.5 Flotation

13.5.1 Scope of testwork

The Namdini Gold Project Feasibility Study gravity and flotation testwork was conducted by ALS (Perth) and MMSA (South Africa).

ALS scope targeted additional bulk concentrate for the Process Design Criteria supporting testwork including:

- Reduction of copper sulphate activator
- Concentrate leach extraction and impact of lead nitrate addition
- Generation of bulk flotation concentrate and tailings for physical testwork including static and dynamic concentrate and tailings thickening and slurry rheology
- Bulk concentrate leaching to generate liquor for carbon adsorption kinetic and equilibrium loading testwork
- Bulk concentrate leaching to generate liquor for cyanide detoxification testwork.

MMSA scope was specific to assessing Aachen shear reactor technology to:

- Increase leach extraction and overall recovery
- Reduce cyanide and lime consumption, targeting improved operating cost
- Assess the application of Aachen technology for the purpose of cyanide detoxification.

Flotation testwork conducted by ALS and MMSA is reported below; results derived for other sub-programs are reported in the following Sections.

13.5.2 Conclusions

The following conclusions are presented based on the results reported in this Section:

1) Reduction in copper sulphate dosage is possible to a minimum of 25 g/t.

2) Bulk flotation demonstrated high recovery. Consistent with the PFS, GRA and DIO lithologies deliver higher recovery than MVO. This is considered due to the extent of liberation at an identical primary grind size and the resulting proportion of fine sulphide relative to composite sulphide and non-sulphide gangue.

3) Flotation feed composites were prepared including blended Knelson Concentrator tailings; consistent high flotation recoveries indicate efficient flotation of fine gravity gold.

4) Assuming stable operation at nominal tonnage, acceptably low flotation tailings grades at and below 0.1 g/t are possible within average mine plan head grades for the Starter Pit and across the LOM. Given the circuit is configured Rougher only, tailings grades increase for MVO grading above ~2.15 g/t.

5) While acceptable recovery was reported by MMSA for MVO, lower recoveries were reported for GRA and DIO relative to ALS. Observational evidence indicates improved recovery could have been achieved by staging increased air during later stages of flotation with increased frother to assist injected air bubble size.

6) LOM flotation concentrate mass is expected to average ~7.5%. Approximate empirical mineralogy suggests the concentrate comprises 60-80% by weight non-sulphide (low iron-siliceous) gangue. Observations indicate the gangue occurs as composite with (predominantly) pyrite and arsenopyrite as opposed to liberated physically entrained gangue. Initial rapid flotation of fine liberated sulphide is typical in early stages of flotation with a persistent (composite) slower floating fraction, this is particularly the case for MVO.

7) An opportunity may exist to improve flotation recovery, reduce regrind power and HIGmill maintenance cost by employing a scavenger stage with slower floating (higher specific energy) siliceous composite ground utilising a small conventional (impact-style) regrind ball mill. Scavenger concentrate could then be recirculated to rougher feed for conditioning and rejection of siliceous gangue by rougher tailings, presenting softer liberated sulphide for HIGmilling at a lower overall specific energy. Adoption of this circuit configuration could also prove more operationally robust reducing the risk of a volumetric throughput bottleneck at the HIGmill stage.

13.5.3 Recommendations

The following recommendations are offered:

- ***The PDC be adjusted to reflect 25 g/t copper sulphate dosing into flotation and the Project operating cost updated accordingly.***
- ***The Project considers opportunity that may exist to improve flotation recovery, reduce regrind power and HIGmill maintenance cost by employing a scavenger stage with slower floating (higher specific energy) siliceous composite ground utilising a small conventional (impact-style) regrind ball mill (Section 13.5.2).***

13.5.4 ALS – copper sulphate optimization testwork

A series of 15 batch tests were conducted to assess the impact of reduced copper sulphate on recovery; as for other ALS tests reported in this Section, testing was performed based on the following conditions established during the PFS:

- Primary grind (P_{80}) size 106 µm.
- Natural pH, nominally 8.0-8.2.
- Copper sulphate staged maximum dosage 50 g/t.
- Collector (PAX) – staged 100 g/t dosage.
- Frother Huntsman H-27.

Head grades reported in Table 73 characterise the composites as follows:

- Gold ranging from a minimum of 1.08 g/t in SP-DIO to a maximum of 2.08 g/t for SP-MVO.
- Low silver with calculated head grades below 1 g/t.

- Wide ranging arsenic and sulphur grade; with approximately 0.07-0.19% Arsenopyrite and 1.46-1.87% Pyrite.

Concentrate grade and recovery, summarised in Table 74 indicates:

- Gold flotation recovery appears unaffected with reduction from 50 g/t to 25 g/t.
- As expected, arsenic recovery and flotation of liberated Arsenopyrite is impacted at nil dosage.
- Overall, reduction in dosage could be adopted to 25 g/t; staged dosing e.g. 10 g/t on milling and 15 g/t into flotation conditioning could be adopted to mitigate anticipated recovery loss.

Table 73: ALS – variable copper sulphate flotation – head grade summary

Composite	Calculated Head Grade					Approx. Mineral %	
	Au g/t	Ag g/t	As ppm	Fe %	Total S ppm	Apy	Py
SP-MVO-LC	2.08	0.89	704	5.67	7959	0.15	1.46
SP-GRA-LC	1.46	0.73	395	1.95	9187	0.09	1.70
SP-DIO-LC	1.08	0.69	646	7.52	10136	0.14	1.87
LOM-MVO-LC	1.99	0.37	897	5.70	8281	0.19	1.52
LOM-GRA-LC	1.34	0.41	328	1.78	8650	0.07	1.61

Note: Approximate mineral % assuming As grade as Arsenopyrite and nil sulphate S.

Table 74: ALS – variable copper sulphate flotation concentrate grade-recovery summary

Composite	$CuSO_4$ g/t	Concentrate Mass and Grade						Gravity + Flotation Recovery (%)					Tailing Au g/t
		Mass	Au	Ag	As	Fe	S	Au	Ag	As	Fe	S	
		%	g/t	g/t	%	%	%	%	%	%	%	%	
SP-MVO-LC	0	8.11	21.8	5.8	0.78	13.1	9.89	92.6	50.5	77.6	18.6	93.6	0.17
	25	9.10	14.0	4.0	0.60	10.5	6.93	93.3	44.2	92.3	17.2	93.3	0.12
	50	9.60	17.3	4.9	0.69	11.9	8.43	93.3	50.7	93.6	20.1	94.7	0.14
SP-GRA-LC	0	8.14	13.3	3.2	0.42	11.1	11.01	94.9	36.2	88.3	45.9	97.0	0.07
	25	7.51	14.8	3.7	0.47	11.9	11.93	95.0	37.7	88.4	46.4	98.0	0.07
	50	8.34	13.1	3.4	0.42	11.0	10.80	94.3	37.9	88.4	47.2	98.0	0.08
SP-DIO-LC	0	10.18	9.9	2.2	0.56	14.5	9.68	95.1	33.1	86.4	19.7	97.3	0.05
	25	9.27	10.5	2.6	0.61	15.4	10.58	94.1	34.8	92.6	19.0	97.3	0.06
	50	8.30	12.1	3.0	0.69	16.6	11.85	94.5	34.8	86.1	18.3	96.4	0.06
LOM-MVO-LC	0	8.09	21.0	2.9	0.91	12.9	9.54	93.9	62.9	80.0	18.4	94.4	0.12
	25	8.32	20.7	2.8	0.91	13.0	9.44	94.4	62.9	80.5	18.9	93.5	0.11
	50	8.55	20.5	2.8	0.87	12.7	9.13	92.0	63.6	89.0	19.1	94.5	0.16
LOM-GRA-LC	0	7.61	13.8	3.5	0.34	11.0	10.96	93.7	65.4	85.0	47.2	96.8	0.08
	25	7.40	15.1	3.7	0.36	11.5	11.42	95.9	66.5	85.2	47.6	97.9	0.06
	50	6.43	17.0	4.2	0.42	13.0	13.21	95.8	65.7	74.2	46.9	97.8	0.06

13.5.5 ALS – bulk flotation

A series of 5 (48 kg) flotation tests were performed to generate concentrate for thickening and slurry rheology physical testwork. A further bulk (198 kg) test was conducted for leaching and generation of liquor into carbon adsorption, kinetic and equilibrium loading testwork.

Bulk flotation was conducted utilising PFS conditions and a conservative reduction in activator dosage to 30 g/t was adopted for series of tests.

The results in Table 75 were achieved using prior gravity concentrate beneficiation and intensive leaching with deportment of gravity and intensive leach tailings consistent with the intended process flowsheet.

Recoveries were consistent with Starter Pit Master and LOM lithology recoveries reported at the close of the PFS. Observed consistently, while rougher flotation kinetics are rapid with the bulk of recovery achieved within eight minutes; there is a persistent slow floating fraction comprising composite sulphide blebs and coarser non-sulphide gangue. Bulk concentrate mass averaged 6.8% and calculated proportions of sulphide and non-sulphide gangue ("**NSG**") in Table 76 indicate MVO concentrate retains 77-80% siliceous fraction. Slightly lower proportions are indicated for DIO, with GRA lowest at 66%.

Typically, the incremental specific grinding energy for re-grinding sulphide would be expected lower than for siliceous mineral. The PDC references HIG Mill re-grind specific energy as 120 kWh/t, this implies a large energy impost is associated with grinding slow floating sulphide-siliceous composite. While the proposed rougher circuit delivers acceptable recovery, potential may exist to reduce UFG re-grind power and improve recovery. This opportunity could potentially be realised based on adopting a rougher-scavenger configuration with scavenger concentrate subjected to a conventional ball mill regrind. Internalising a proportion of the re-grinding effort within the flotation circuit potentially offers the means to recirculate milled scavenger concentrate to rougher feed, allowing rejection of siliceous gangue by rougher tailings and presentation of lower specific energy sulphide for final grinding preparation by the proposed HIGmills.

Table 75: ALS bulk flotation – head grade summary

Composite	Calculated Head Grade					Approx. Mineral %	
	Au g/t	Ag g/t	As ppm	Fe %	Total S ppm	Apy	Py
SP-MVO-LC	2.15	1.0	839	5.71	8481	0.18	1.56
LOM-MVO-LC	1.92	0.4	948	5.64	8146	0.21	1.49
SP-GRA-LC	1.41	0.4	398	1.92	9007	0.09	1.67
LOM-GRA-LC	1.38	0.4	309	1.82	8618	0.07	1.60
SP-DIO-LC	1.06	0.4	657	7.49	9979	0.14	1.84
SP-BULK-LC	1.93	0.5	568	4.15	8689	0.12	1.60

Note: Approximate mineral % assuming As is in arsenopyrite and nil sulphate S.

Table 76: ALS bulk flotation concentrate – approximate sulphide/non-sulphide content

Composite	% Mineral (w/w)		Concentrate Mass %				NSG % Fe	Mass % Sulphide	Mass % NSG
	FeAsS	FeS_2	FeAsS	FeS_2	NSG (*)	Total			
SP-MVO-LC	2.05	21.32	0.14	1.48	5.30	**6.92**	4.3	**23.4**	**76.6**
LOM-MVO-LC	2.06	17.66	0.17	1.43	6.49	**8.08**	4.6	**19.7**	**80.3**
SP-GRA-LC	1.51	32.50	0.08	1.64	3.32	**5.04**	1.5	**34.0**	**66.0**
LOM-GRA-LC	1.13	31.06	0.06	1.57	3.42	**5.04**	1.4	**32.2**	**67.8**
SP-DIO-LC	1.48	21.65	0.12	1.79	6.37	**8.29**	6.1	**23.1**	**76.9**
SP-BULK-LC	1.55	21.24	0.11	1.55	5.65	**7.32**	4.0	**22.8**	**77.2**

Note: NSG is non-sulphide gangue.

Table 77: ALS – bulk flotation concentrate grade-recovery summary

Composite	Head Au g/t	Concentrate Mass and Grade						Gravity + Flotation Recovery (%)					Tailing Au g/t
		Mass	Au	Ag	As	Fe	S	Au	Ag	As	Fe	S	
		%	g/t	g/t	%	%	%	%	%	%	%	%	
SP-MVO-LC	2.15	6.92	24.4	7.2	0.94	15.0	11.6	95.7	51.8	77.8	18.1	94.5	0.09
LOM-MVO-LC	1.92	8.08	20.7	3.2	0.95	13.5	9.6	94.2	64.9	80.6	19.4	95.5	0.11
SP-GRA-LC	1.41	5.04	21.2	5.9	0.70	17.1	17.5	96.0	67.4	88.1	45.0	97.9	0.06
LOM-GRA-LC	1.38	5.04	22.9	5.1	0.52	16.3	16.7	97.3	64.2	84.7	45.2	97.8	0.04
SP-DIO-LC	1.06	8.29	11.8	3.2	0.68	16.7	11.7	94.8	65.8	86.0	18.5	97.2	0.06
SP-BULK-LC	1.93	7.32	20.6	5.2	0.71	14.4	11.5	95.7	73.3	91.8	25.4	96.8	0.08

13.5.6 ALS – starter pit flotation variability testwork

A series of five (5) Starter Pit MVO samples were tested to assess flotation recovery and leach extraction variability.

13.5.6.1 Starter pit MVO flotation recovery variability

MVO results are tabled below with the overall recovery trend illustrated in Figure 63 and trending in flotation tailing grade depicted in Figure 64; the following key observations are presented:

- Recovery ranged from 92.7% (SP-MVO2-VC) to 95.3% (SP-MVO-LC) relative to the average for the PFS Starter Pit Master Composite, 95.5% at 1.90 g/t head grade.
- The trend in Figure 64 indicates tailings grade is maintained below 0.10 g/t up to a head grade of 2.15 g/t.
- Observational evidence on witnessing batch and bulk flotation tests indicates that the recovery variability shown in Figure 63 is a combination of variation in head grade as well as the proportion of finer sulphide recovered at early stages of flotation and sulphide (blebs) associated with coarser siliceous fraction reporting to rougher concentrate at later stages. Across all lithologies, MVO comprises a greater proportion of (slower floating) sulphide/siliceous composite; this aspect is supported based on the approximate sulphide/non-sulphide mass proportions shown in Table 76.
- Comparison with the average Bulk Flotation result in Figure 64 indicates acceptable recovery with tailings grades at and below 0.10 g/t at the expected average mine grade for Starter Pit MVO (1.41 g/t).

Table 78: ALS Starter Pit MVO flotation variability – head grade summary

Composite	Calculated Head Grade					Approx. Mineral %	
	Au g/t	Ag g/t	As ppm	Fe %	Total S ppm	Apy	Py
SP-MVO-LC	2.15	1.0	839	5.71	8481	0.18	1.56
SP-MVO2-VC	2.84	0.4	615	5.70	4791	0.13	0.87
SP-MVO12-VC	2.41	0.8	2066	5.32	16235	0.45	2.96
SP-MVO25-VC	2.44	0.5	312	6.11	13672	0.07	2.55
SP-MVO38-VC	3.51	0.6	867	4.50	13498	0.19	2.49

Table 79: ALS Starter Pit MVO flotation – grade-recovery variability

Composite	Head Au g/t	Concentrate Mass and Grade						Gravity + Flotation Recovery (%)					Tailing Au g/t
		Mass	Au	Ag	As	Fe	S	Au	Ag	As	Fe	S	
		%	g/t	g/t	%	%	%	%	%	%	%	%	
SP-MVO-LC	2.15	9.38	18.9	4.8	0.78	12.7	9.07	95.3	77.3	94.2	20.4	95.9	0.10
SP-MVO2-VC	2.84	5.62	36.8	3.9	1.01	12.5	8.19	92.7	64.7	92.3	12.3	96.1	0.21
SP-MVO12-VC	2.41	9.02	22.4	6.9	2.09	18.5	17.40	93.6	82.5	91.2	31.4	96.6	0.15
SP-MVO25-VC	2.44	8.32	23.0	3.9	0.32	17.6	16.00	94.7	71.6	85.3	23.9	97.3	0.13
SP-MVO38-VC	3.51	8.52	34.3	5.1	0.91	16.5	15.30	94.1	77.0	89.5	31.3	96.6	0.21
PFS Average	**1.90**	**7.58**	**18.3**	**-**	**0.63**	**13.1**	**10.72**	**95.5**	**-**	**88.0**	**26.8**	**97.3**	**0.09**

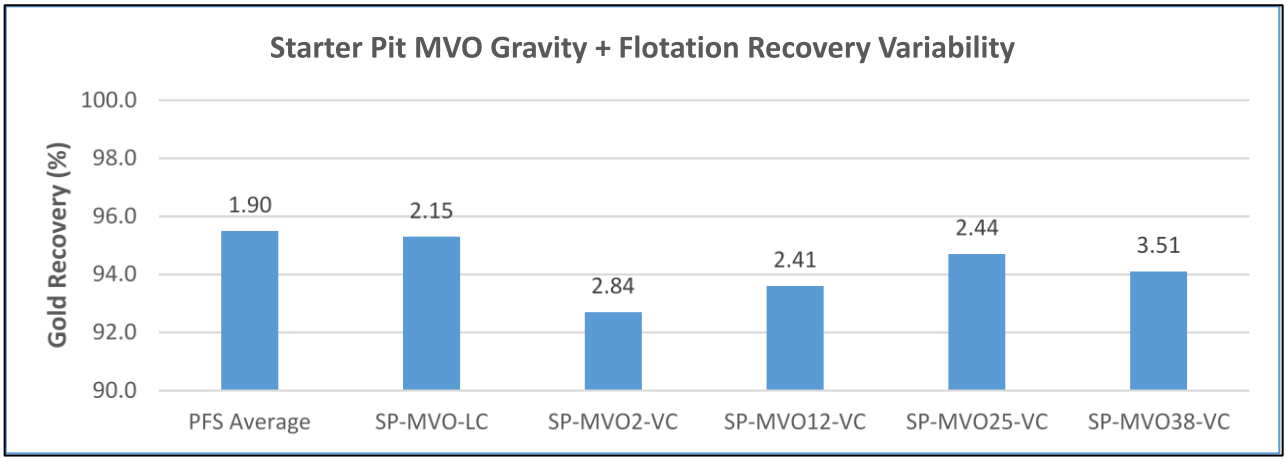


Figure 63: Starter Pit MVO – Gravity + Flotation Recovery Variability

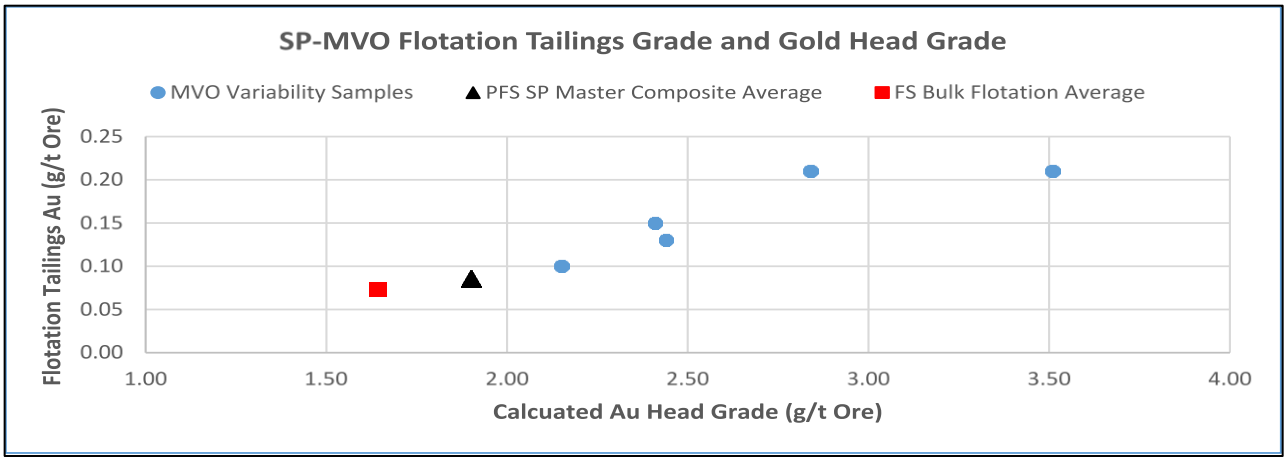


Figure 64: Starter Pit MVO – flotation tailings gold grade and head grade

13.5.6.2 Starter Pit GRA flotation recovery variability

Starter Pit GRA recovery (Table 81) ranged from 94.6% (SP-GRA1-VC and SP-GRA30-VC) to 97.0% in SP-GRA-LC) relative to the PFS Starter Pit Master Composite at 95.5%. Compared to SP-MVO the following points are noted:

- GRA (and DIO) delivers higher recovery given GRA comprises a higher proportion of fine sulphide relative to blebs of sulphide locked in coarser siliceous gangue. This is in evidence based on the lower proportion of non-sulphide gangue attributed to GRA (66%) relative to MVO (76.6%) in Table 76.
- Consistent with MVO, recovery variations shown in Figure 65 represent variations in the extent of sulphide liberation and proportions of (slower floating) sulphide blebs in coarser siliceous non-sulphide fraction.
- GRA also presents a similar increasing trend in tailings grade with head grade (Figure 66), albeit a lower level of variability and reduced average flotation tailings grade.
- Based on the expected Starter Pit GRA head grade of 1.26 g/t and including allowance for Knelson capture of coarse gravity gold, flotation tailings grades below 0.10 g/t are achievable with a likely average of ~0.06 g/t.

Table 80: ALS Starter Pit GRA flotation variability – head grade summary

Composite	Calculated Head Grade					Approx. Mineral %	
	Au g/t	Ag g/t	As ppm	Fe %	Total S ppm	Apy	Py
SP-GRA-LC	1.64	0.4	415	1.93	9655	0.09	1.79
SP-GRA1-VC	1.19	0.4	563	1.87	10052	0.12	1.86
SP-GRA14-VC	0.66	0.3	568	1.82	8683	0.12	1.60
SP-GRA23-VC	1.15	0.6	270	2.44	12560	0.06	2.34
SP-GRA30-VC	1.02	0.4	360	1.87	8916	0.08	1.65

Table 81: ALS Starter Pit GRA flotation – grade-recovery variability

Composite	Head Au g/t	Concentrate Mass and Grade						Gravity + Flotation Recovery (%)					Tailing Au g/t
		Mass	Au	Ag	As	Fe	S	Au	Ag	As	Fe	S	
		%	g/t	g/t	%	%	%	%	%	%	%	%	
SP-GRA-LC	1.64	9.02	13.9	3.0	0.41	10.8	10.50	97.0	68.1	89.0	50.5	98.1	0.05
SP-GRA1-VC	1.19	7.71	11.9	3.3	0.67	12.8	12.80	94.6	65.9	91.8	52.7	98.2	0.06
SP-GRA14-VC	0.66	7.80	7.2	2.1	0.61	11.1	10.90	95.8	54.9	83.8	47.7	97.9	0.03
SP-GRA23-VC	1.15	8.30	13.2	5.1	0.27	14.6	14.80	96.8	75.5	83.0	49.7	97.8	0.04
SP-GRA30-VC	1.02	7.66	11.8	3.6	0.41	11.5	11.40	94.6	66.9	87.2	47.1	97.9	0.06
PFS Average	**1.90**	**7.58**	**18.3**	**-**	**0.63**	**13.1**	**10.72**	**95.5**	**-**	**88.0**	**26.8**	**97.3**	**0.09**

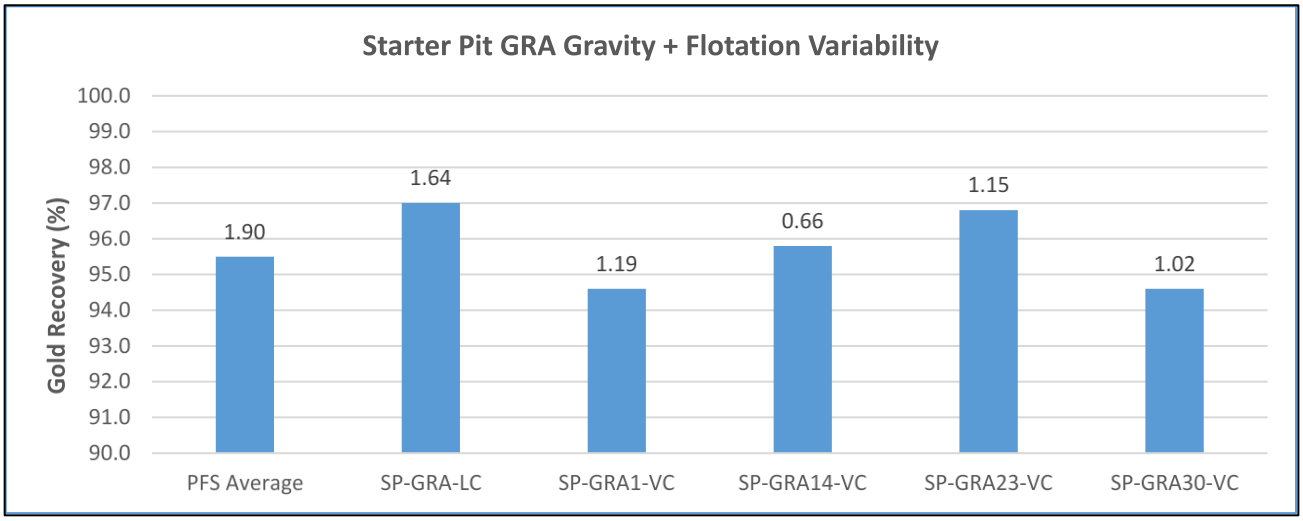


Figure 65: Starter Pit GRA – gravity + flotation recovery variability

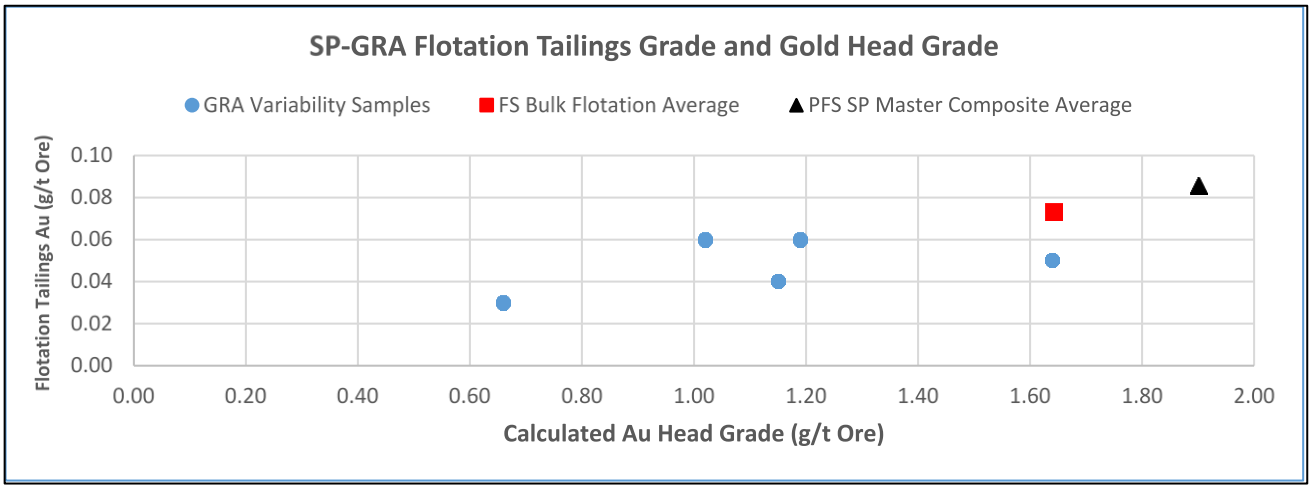


Figure 66: Starter Pit GRA – flotation tailings gold grade and head grade

13.5.7 MMSA flotation

MMSA generated bulk flotation concentrate for assessing Aachen leach extraction relative to Baseline CIL. LOM bulk samples were subjected to crushing and primary milling with Knelson gravity and intensive leaching conducted as reported in Section 13.4. Knelson gravity and intensive leach tailings were blended with respective bulk flotation feed and concentrate consistent with the preparation employed by ALS and the intended process flowsheet. Bulk flotation was conducted on 75 kg charges in a 200L Denver flotation cell at natural pH, stage dosing 50 g/t copper sulphate and 100 g/t PAX. Results of combined gravity and flotation recovery are summarised in Table 82 indicating respective gold recoveries for MVO, GRA, and DIO at 91.2%, 88.4%, and 88.2%. Recoveries recorded for GRA and DIO are noted lower than achieved historically by ALS.

Table 82: MMSA LOM lithology bulk flotation – results summary

Composite	Head Au g/t		Concentrate Mass and Grade				Gravity + Flotation Recovery			Tailing Au g/t Ore
	Assay	Calc	Mass	Au	As	S	Au	As	S	
			%	g/t	%	%	%	%	%	
LOM MVO	0.88	0.85	6.2	10.93	0.449	11.20	91.2	79.8	97.4	0.075
LOM GRA	1.77	1.26	8.2	12.77	0.703	8.28	88.4	87.1	94.9	0.147
LOM DIO	1.53	1.10	8.2	11.50	0.425	12.50	88.2	92.4	96.5	0.129

13.5.8 Flotation tailings gold distribution

13.5.8.1 ALS flotation tailings gold distribution

The flotation tailings gold distribution for key ALS composites is summarised below in Table 83 and as follows:

- Tailings calculated and assay heads compare well across all composites.
- Flotation tailings P_{80} sizes are noted typical relative to earlier ALS-PFS results at a primary grind (P_{80}) size of 106 µm. MVO ranges from 93 to 99 µm, DIO at 104 µm and GRA noted coarser, ranging from 110 to 113 µm.
- Gold distribution is typically higher in coarser +106 µm fraction with relatively even distribution through mid-range size fractions down to 38 µm.
- Increased distribution attributed to coarser (+106 µm) fraction may be a result of the slower floating nature of coarser composites. This was noted previously based on visual observations during flotation and particularly the case for MVO (0.11 g/t) relative to lower tailings grades achieved for GRA (0.04-0.06 g/t) and DIO (0.05 g/t).

Table 83: ALS flotation tailings elemental distribution

Size (µm)	Mass %	Gold		Arsenic		Iron		Sulphur	
		g/t	% Dist	%	% Dist	%	% Dist	%	% Dist
SP-MVO-LC; P_{80} = 93 µm									
-212/+106	12.5	0.17	18.6	0.020	14.1	4.90	12.2	0.08	20.6
-106/+75	17.4	0.16	24.3	0.020	19.6	4.69	16.2	0.06	21.5
-75/+38	23.0	0.14	28.2	0.010	13.0	4.53	20.7	0.04	19.0
-38	47.2	0.07	28.9	0.020	53.3	5.42	50.9	0.04	38.9
Calculated	**100.0**	**0.11**	**100.0**	**0.018**	**100.0**	**5.02**	**100.0**	**0.05**	**100.0**
Assay		**0.10**		**0.020**		**5.02**		**0.05**	
LOM-MVO; P_{80} = 99 µm									
+212	0.5	0.26	1.2	0.040	1.01	5.18	0.5	0.11	1.4
-212/+106	15.9	0.18	26.9	0.020	15.8	4.80	15.4	0.07	27.9
-106/+75	15.2	0.14	20.0	0.020	15.1	4.53	13.9	0.05	19.1
-75/+38	20.8	0.15	29.4	0.020	20.7	4.51	19.0	0.03	15.7

Size (µm)	Mass %	Gold		Arsenic		Iron		Sulphur	
		g/t	% Dist	%	% Dist	%	% Dist	%	% Dist
-38	47.6	0.05	22.4	0.020	47.4	5.31	51.2	0.03	35.9
Calculated	**100.0**	**0.11**	**100.0**	**0.020**	**100.0**	**4.94**	**100.0**	**0.04**	**100.0**
Assay		**0.12**		**0.020**		**4.95**		**0.04**	
SP-GRA-LC; P_{80} = 113 µm									
+212	1.1	0.12	2.0	0.010	1.6	1.29	1.2	0.05	3.1
-212/+106	24.0	0.07	26.8	0.005	18.6	0.81	17.8	0.02	28.7
-106/+75	23.1	0.07	25.9	0.005	17.9	0.78	16.5	0.02	27.6
-75/+38	23.8	0.06	22.9	0.005	18.5	0.88	19.2	0.01	7.1
-38	28.0	0.05	22.4	0.010	43.4	1.77	45.3	0.02	33.4
Calculated	**100.0**	**0.06**	**100.0**	**0.006**	**100.0**	**1.09**	**100.0**	**0.02**	**100.0**
Assay		**0.06**		**0.005**		**1.11**		**0.02**	
LOM-GRA; P_{80} = 110 µm									
+212	0.7	0.04	0.6	0.010	1.3	1.41	0.9	0.04	1.3
-212/+106	21.7	0.06	29.1	0.005	21.6	0.75	16.3	0.02	21.6
-106/+75	20.8	0.03	14.0	0.005	20.7	0.63	13.1	0.02	20.7
-75/+38	24.1	0.05	27.0	0.005	23.9	0.76	18.4	0.02	23.9
-38	32.7	0.04	29.3	0.005	32.5	1.56	51.2	0.02	32.5
Calculated	**100.0**	**0.04**	**100.0**	**0.005**	**100.0**	**1.00**	**100.0**	**0.02**	**100.0**
Assay		**0.04**		**0.005**		**1.05**		**0.02**	
SP-DIO-LC; P_{80} = 104 µm									
+212	0.8	0.10	1.5	0.010	1.0	6.88	0.8	0.09	1.9
-212/+106	18.0	0.11	34.7	0.020	46.5	6.24	16.9	0.04	19.6
-106/+75	15.7	0.07	20.1	0.005	10.1	6.05	14.2	0.03	12.8
-75/+38	20.5	0.05	18.9	0.005	13.3	6.31	19.5	0.03	16.8
-38	45.0	0.03	24.8	0.005	29.1	7.18	48.6	0.04	49.0
Calculated	**100.0**	**0.05**	**100.0**	**0.008**	**100.0**	**6.65**	**100.0**	**0.04**	**100.0**
Assay		**0.06**		**0.010**		**6.65**		**0.03**	

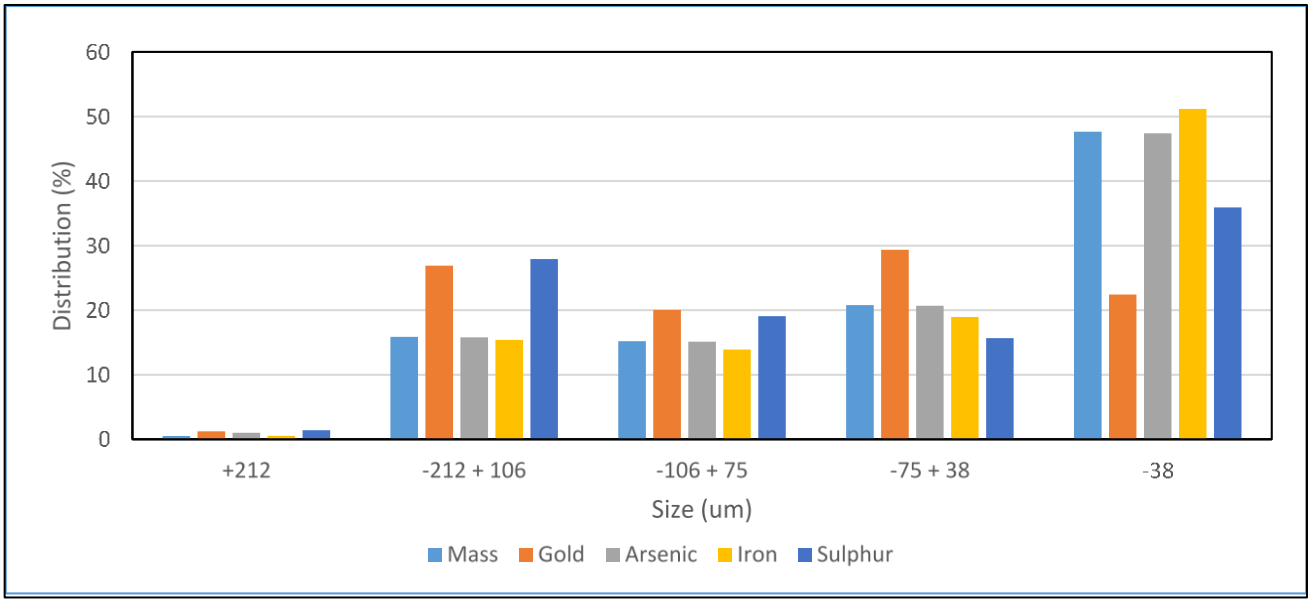


Figure 67: ALS flotation tailings elemental distribution: LOM-MVO-LC

13.5.8.2 MMSA flotation tailings gold distribution

The MMSA flotation tailings gold distribution for LOM lithology composites is summarised in Table 84:

- MMSA flotation tailings reported P_{80} sizes for MVO at 133 µm, GRA (98 µm), and DIO (133 µm). Except for GRA, P_{80} sizes for MVO and DIO indicate coarser tailings than reported by ALS (93 µm to 113 µm).
- Historically ALS test data has proven MVO is more difficult to float targeting low flotation tailings grades below 0.10 g/t. This is highlighted based on a persistent coarse siliceous fraction that tends slower floating than the bulk of the flotation feed, also containing blebs of sulphide. Relative to ALS at 0.10-0.11 g/t, MMSA reported calculated MVO flotation tailings grading 0.07 g/t at a P_{80} size of 133 µm.
- The calculated tailings grade for GRA (P_{80} 98µm) is noted at 0.17 g/t relative to ALS (0.04-0.06 g/t). MMSA reported DIO (P_{80} 133 µm) flotation tailings grading 0.16 g/t relative to ALS at 0.05 g/t. Elevation in GRA and DIO tailings grades across all size fractions indicate that improved recoveries, consistent with ALS could have been achieved with increased air flow and frother at later stages of flotation and/or increased flotation time.

Table 84: MMSA flotation tailings elemental distribution

Size (µm)	LOM-MVO; P_{80} = 133 µm			LOM-GRA; P_{80} = 98 µm			LOM-DIO; P_{80} = 133 µm		
	Mass %	Au g/t	Dist %	Mass %	Au g/t	Dist %	Mass %	Au g/t	Dist %
212	2.29	0.15	4.78	2.75	0.21	3.31	5.75	0.31	11.14
106	23.79	0.07	23.14	14.22	0.20	16.59	19.04	0.21	25.00
75	14.55	0.08	16.17	11.90	0.19	13.19	14.13	0.20	17.67
53	12.95	0.06	10.80	8.36	0.17	8.30	7.55	0.20	9.44
38	7.71	0.12	12.86	6.50	0.16	6.07	3.31	0.41	8.49
-38	38.70	0.06	32.25	56.27	0.16	52.54	50.22	0.09	28.26
Calculated	**100.0**	**0.07**	**100.0**	**100.0**	**0.17**	**100.0**	**100.0**	**0.16**	**100.0**
Assay		**0.08**			**0.16**			**0.14**	

13.6 Flotation concentrate and tailings dewatering

Thickening testwork was conducted by Outotec on samples of flotation concentrate and tailings derived from Starter Pit composite (SP-Bulk). The scope included static thickening, flocculant screening and dynamic thickening based on a selected flocculant to determine optimum dilution, flocculant dosage, flux rate and overflow clarity. Detailed results are available in Outotec report "*319020 T2.pdf*" and "*319020 T2_B.pdf*", key outcomes are discussed below.

13.6.1 Conclusions

The following conclusions are presented based on the testwork reported in this Section:

- Dynamic thickening testwork highlighted BASF Magnafloc 155 as optimal for flotation concentrate and tailings thickening duty.
- Flotation concentrate thickening is viable with acceptable overflow clarity based on a flux rate of 0.25 t/m^2.h dosing 15 g/t Magnafloc M155 to achieve an underflow density in the range 55-57% solids (w/w).
- Flotation tailings dynamic testwork indicates acceptable performance based on a flux rate in the range 1.25-1.50 t/m^2.h dosing Magnafloc 155 at 10 g/t.

13.6.2 Flotation concentrate

Flocculant screening was conducted based on static settling tests at 10% solids (w/w) over the range of BASF Magnafloc products sighted in Figure 68. All flocculants delivered reasonable settling rates; Magnafloc 155 was selected based on highest settling rate and cleanest supernatant liquor, albeit BASF M5250 performed similarly.

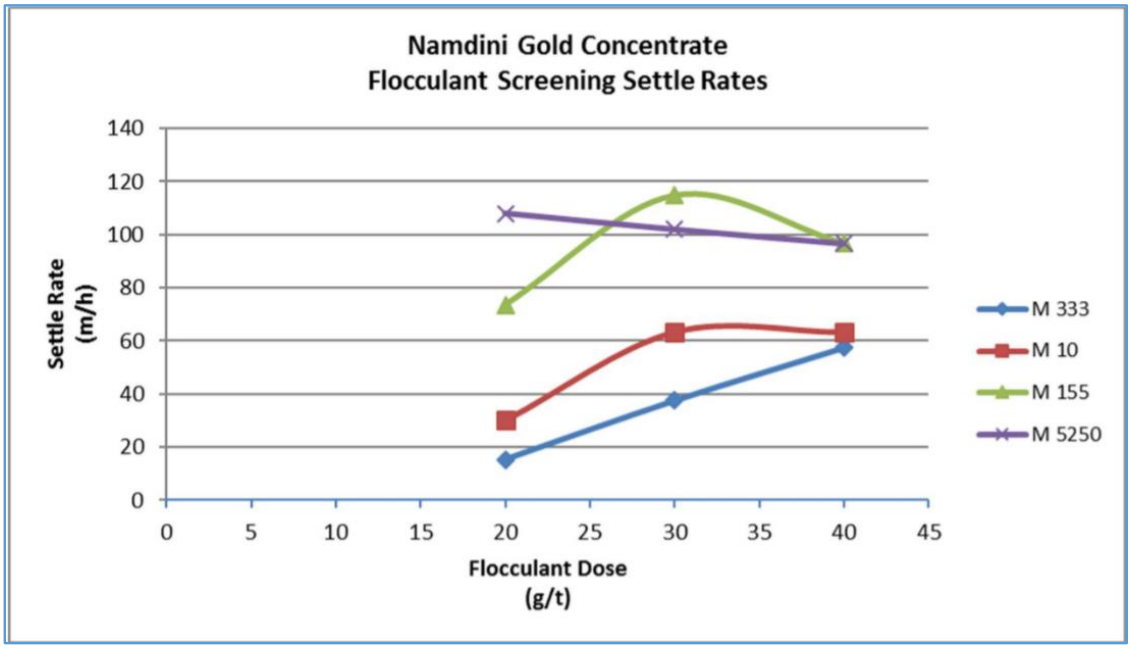


Figure 68: Flotation concentrate – flocculant screening test summary

Dynamic thickening tests were conducted utilising BASF M155 flocculant targeting a flux rate of 0.25 t/m^2.h, consistent with the Project PDC. Results are tabled below at 15 g/t and 10 g/t dosages. Acceptable overflow clarity at an underflow density of 55.8% solids and slurry (Bingham) yield stress of 16 Pa is indicated based on a flux rate of 0.25 t/m^2.h with BASF M155 dosed at 15 g/t. Rise rate is noted well below a maximum limit for high-rate thickening at 10 m/h.

The PDC references an operating flocculant dosage of 40 g/t at an identical flux rate and underflow solids density; this could be reduced consistent with the results reported below, reducing operating cost.

Table 85: Flotation concentrate – dynamic test summary

Feed		Flocculant		Underflow		Overflow Clarity mg/L
Flux Rate t/m^2.h	Rise Rate m/h	Type	Dose g/t	% Solids (w/w)	Yield Stress Pa	
0.25	1.40	M155	15	55.8	16	100
0.25	1.40	M155	10	55.9	15	320

13.6.3 Flotation tailings

Flotation tailings was subjected to flocculant screening utilising the same flocculant range applied for concentrate thickening. Results presented in the figure below indicate BASF M155 as optimal based on settling rate (and observed supernatant clarity), albeit as was the case for concentrate, BASF M5250 performed similarly.

Dynamic test data presented in Table 86 and depicted in Figure 70 indicates operation based on flux rate in the range 1.25-1.50 t/m^2.h with BASF M155 flocculant dosed at 10 g/t. The PDC references a flux rate of 1.0 t/m^2.h and operating flocculant dose of 30 g/t. Potential exists to reduce the current thickener diameter (41 m) with operation at a higher flux rate and lower flocculant dosage. Revision of the design would need to evaluate the effect on underflow density, allowing for compaction at scale and resulting rheology and underflow pump selection.

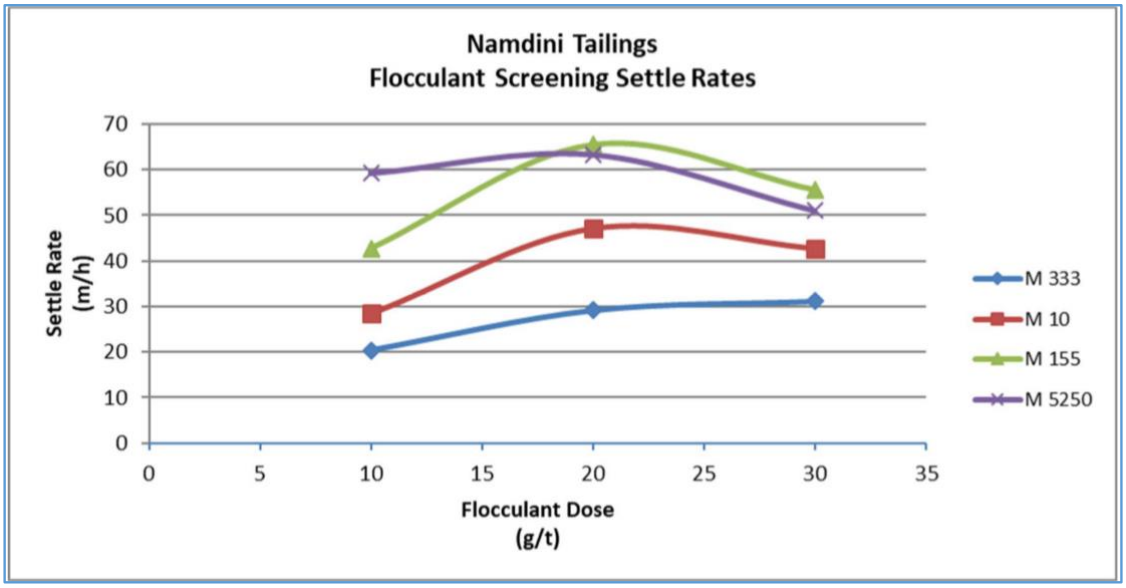


Figure 69: Flotation tailings – flocculant screening test summary

Table 86: Flotation tailings – dynamic test summary

Feed		Flocculant		Underflow		Overflow Clarity mg/L
Flux Rate t/m².h	Rise Rate m/h	Type	Dose g/t	% Solids (w/w)	Yield Stress Pa	
0.5	2.47	M155	20	69.1	42	< 100
0.5	2.47	M155	10	69.6	28	< 100
1.0	4.93	M155	10	65.2	19	100
1.5	7.40	M155	10	62.0	18	210

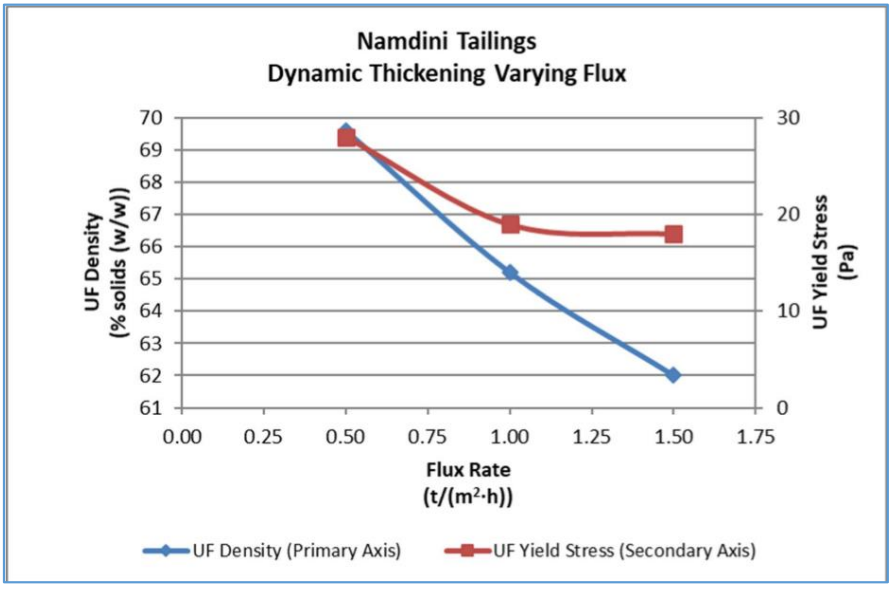


Figure 70: Flotation tailings – variable flux rate and underflow characteristics

13.7 Slurry rheology

ALS performed slurry rheology characterisation on a range of samples representing:

- Flotation Concentrate;
- Flotation Concentrate – (Dynamic) Thickener Underflow;

- HIG Pilot (UFG) Flotation Concentrate;
- CIL Leach Tailings based on HIG (UFG) Flotation Concentrate; and,
- Flotation Tailings – (Dynamic) Thickener Underflow.

13.7.1 Conclusions

Testwork was conducted utilising a Bolin Visco 88 Viscometer at ambient temperature; key conclusions are as follows:

- Rheology testing was conducted at an ambient temperature of 24°C; lower slurry viscosity is expected in operation, given typically higher slurry temperatures occurring at scale.
- Flotation concentrate centrifugal pump transfer to the Flotation Concentrate Thickener at 30% solids (w/w) is supported based on the rheology test data reported in Section 13.7.2.
- Positive displacement pumping is indicated for Flotation Concentrate Thickener Underflow based on the results reported in Section 13.7.3.
- CIL feed and tailings slurry rheology test data reported respective Section 13.7.4 and Section 13.7.5 indicates the slurry to be of medium to moderate viscosity at a temperature of 24°C. Nominally the test data does not indicate issues with inter-tank screen slurry transfer up to a density of ~37% solids (w/w).
- Flotation Tailings Thickener Underflow transfer via centrifugal pumping is endorsed based on the intended underflow density of 55% solids (w/w) and the results reported in Section 13.7.6.

13.7.2 Flotation concentrate rheology

Rheology testwork was performed on bulk flotation concentrate derived from the PFS Starter Pit Master Composite (SP-MC). Slurries were prepared by repulping and diluting in Perth tap water targeting: 55%, 45%, and 35% solids density (w/w). The flowsheet calls for centrifugal pumping of concentrate to the concentrate thickener ahead of UFG milling at a nominal density of 30% solids. The rheology data presented below indicates the slurry to be low to medium viscosity at 35% solids.

Table 87: Flotation concentrate – slurry rheology

Pulp Density % (w/w)	Shear Rate (s-1)						
	24.19	42.2	73.6	129.1	225.7	3094.7	690.4
Shear Stress (Pa)							
55	6.0	6.6	8.0	9.1	11.0	13.5	18.5
45	1.4	1.7	1.7	2.2	3.1	3.9	8.0
35	0.0	0.0	0.0	0.0	1.1	2.5	4.7
Viscosity (cP)							
55	247	156	109	71	49	34	27
45	59	41	23	17	14	10	12
35	0	0	0	0	5	6	7

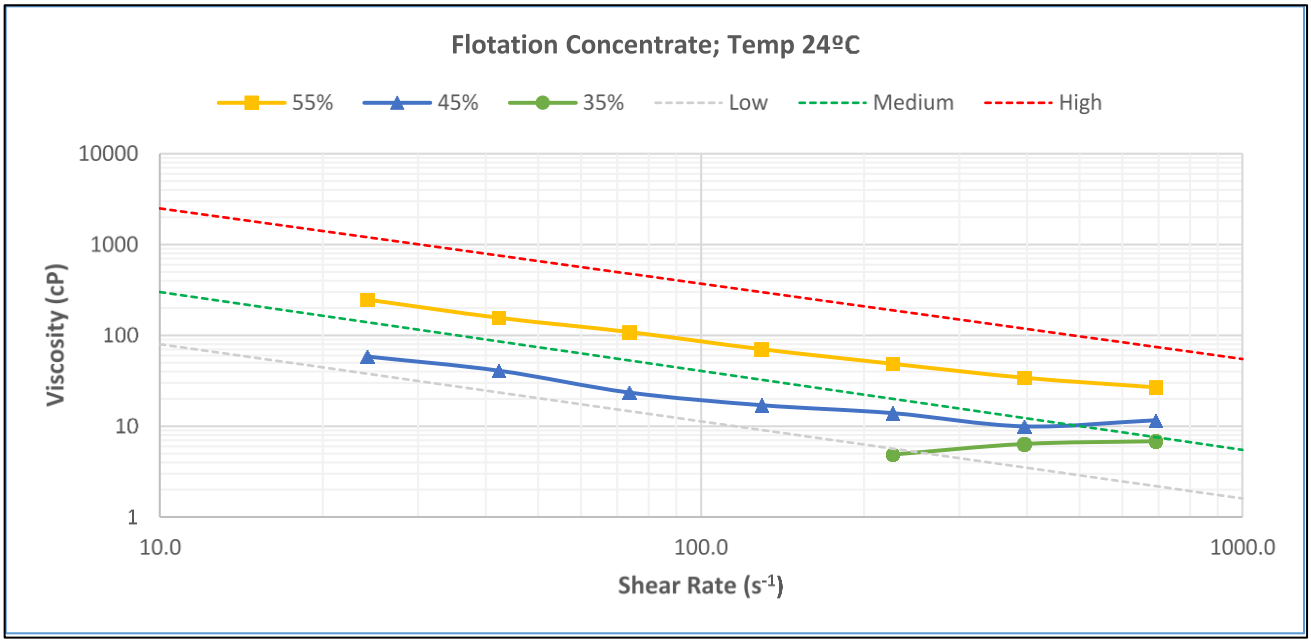


Figure 71: Flotation concentrate – slurry viscosity

13.7.3 Concentrate thickener underflow rheology

Rheology characterisation was undertaken on thickener underflow derived from dynamic thickening testwork. Slurries were prepared, repulping and diluting in Perth tap water to 60%, 55%, 45% and 35% solids density (w/w).

Thickening testwork reported in Section 13.6.1 indicated 55% underflow density, albeit an operating underflow density of 57% could be realised allowing for compaction at scale. Generally, issues with centrifugal pumping may occur where slurry viscosity exceeds 100 cP at a shear rate of 118.5 s^{-1}. Referencing the viscosity data in Table 88 a slurry viscosity of ~87 cP is indicated at 55% solids and ~400 cP at 60% solids with a shear rate at 118.5 s^{-1}. The PDC references application of peristaltic pumps for underflow pumping duty, this selection is endorsed given the viscosity recorded in the range 55% to 60% solids and allowing for possible underflow density excursions as may occur on coming off recycle.

Table 88: Flotation concentrate thickener underflow – slurry rheology

Pulp Density % (w/w)	Shear Rate (s^{-1})							
	24.19	42.2	73.6	129.1	225.7	3094.7	690.4	1207.0
Shear Stress (Pa)								
60	33.3	39.8	42.6	46.2	52.3	59.7	70.1	86.8
55	7.2	7.5	9.0	10.2	12.1	14.5	19.0	35.2
45	4.2	5.7	6.0	6.8	8.6	8.8	11.8	22.5
35	0.0	0.0	0.0	0.0	0.0	2.5	4.7	10.7
Viscosity (cP)								
60	1381	941	579	358	232	151	102	72
55	300	179	122	79	54	37	28	29
45	176	134	81	52	38	22	17	19
35	0	0	0	0	0	6	7	9

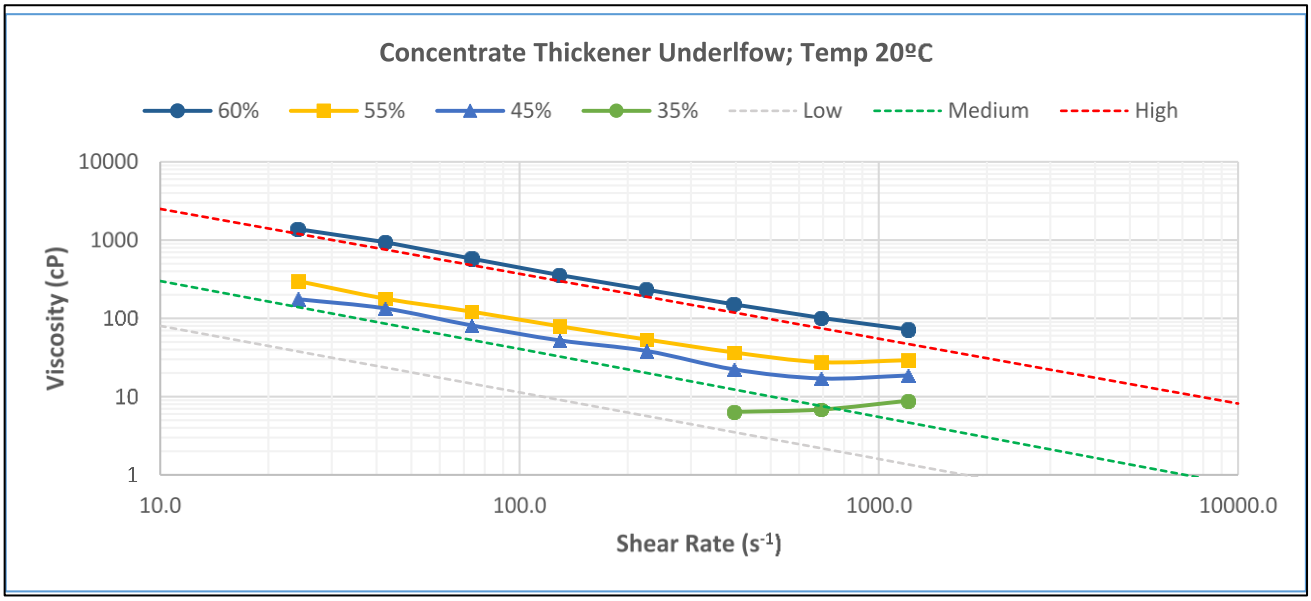


Figure 72: Flotation concentrate thickener underflow – slurry viscosity

13.7.4 HIG (UFG) concentrate rheology

Testwork was conducted on pilot UFG ground HIG flotation concentrate. Composite was prepared by blending LOM lithologies in proportions: 56% MVO, 28% GRA, and 16% DIO. Test slurries were prepared by repulping and diluting in Perth tap water targeting: 42%, 40%, 35%, and 30% solids density (w/w).

The test data indicates the slurry to be medium to moderate viscosity based on the expected UFG mill circuit product density of ~37% solids (w/w). Notably lower viscosity would be expected in practice, given the data was obtained at an ambient temperature of 24°C relative to the anticipated CIL slurry feed temperature of ~40°C.

Table 89: HIG (UFG) pilot flotation concentrate LOM Blend – slurry rheology

Pulp Density % (w/w)	Shear Rate (s^{-1})						
	24.19	42.2	73.6	129.1	225.7	3094.7	690.4
Shear Stress (Pa)							
42	11.3	12.1	13.2	14.5	15.9	17.4	20.3
40	9.6	10.2	10.4	10.8	12.1	13.2	14.8
35	6.0	7.4	6.9	7.9	8.8	10.1	12.1
30	3.0	3.1	3.3	3.6	4.4	5.3	8.2
Viscosity (cP)							
42	469	287	179	112	70	44	29
40	397	242	141	84	54	33	21
35	247	175	94	61	39	25	18
30	124	74	45	28	20	14	12

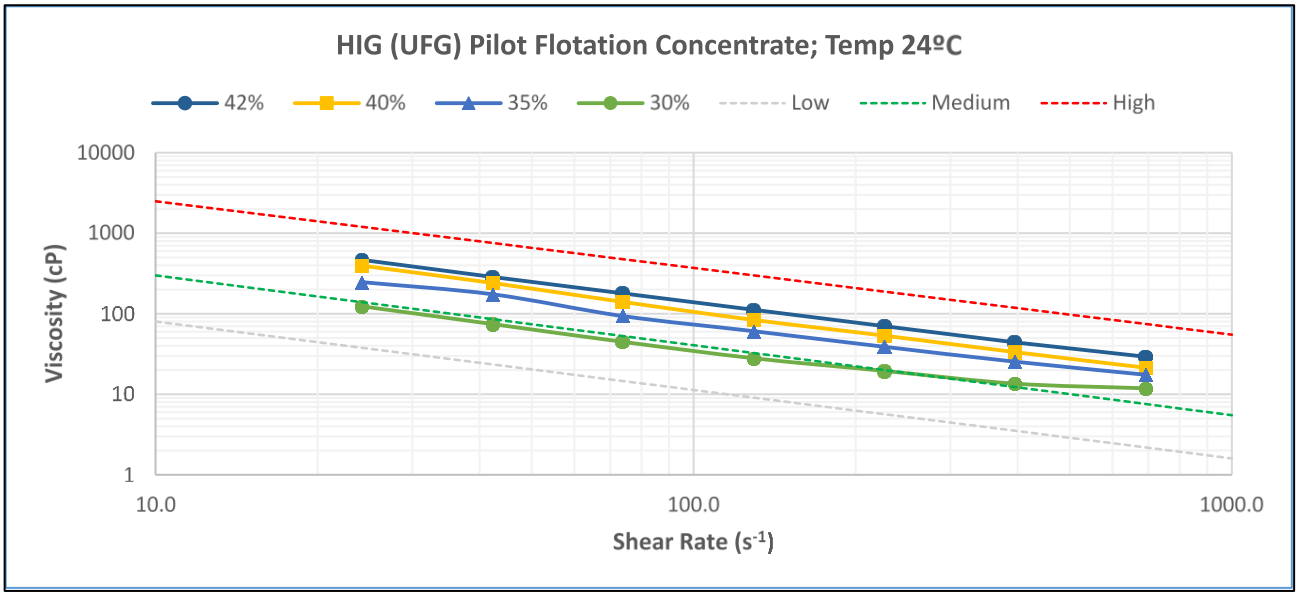


Figure 73: HIG (UFG) pilot flotation concentrate LOM Blend – slurry viscosity

13.7.5 CIL tailings rheology

HIG (UFG) pilot blend composite was subjected to CIL leaching to prepare CIL leach tailings for rheology characterisation. Test slurries were prepared by repulping and diluting in Perth tap water to 42%, 40%, 35%, and 30% solids density (w/w).

The slurry is characterised as medium to moderate viscosity based on the plot shown in Figure 74, relative to the UFG CIL feed slurry illustrated in Figure 73. CIL tailings viscosity is slightly lower based on a CIL feed density of 35% solids.

Slurry viscosity at a CIL density of 35% solids is noted as 241 cP at a shear rate of 24.2 s^{-1}, nominally issues with high viscosity and reduced slurry flow through inter-tank screens could be expected if slurry viscosity exceeds 3,000 cP at a shear rate of ~2.5 s^{-1}. The viscosity trend based on the test data in Table 90 at 35% solids indicates a slurry viscosity of ~1,430 cP at 2.5 s^{-1} also indicating no material flow issues at this density and up to a maximum density of ~37% solids.

Table 90: LOM Blend CIL tailings – slurry rheology

Pulp Density % (w/w)	Shear Rate (s^{-1})						
	24.19	42.2	73.6	129.1	225.7	3094.7	690.4
Shear Stress (Pa)							
42	17.0	18.2	19.5	20.9	23.3	26.1	29.9
40	13.0	14.3	15.4	16.8	18.5	20.8	24.2
35	5.8	6.3	6.8	7.4	8.3	9.4	11.5
30	2.7	2.8	3.3	3.6	4.2	4.9	7.9
Viscosity (cP)							
42	703	432	265	162	103	66	43
40	540	339	209	130	82	53	35
35	241	149	92	57	37	24	17
30	111	67	45	28	19	12	11

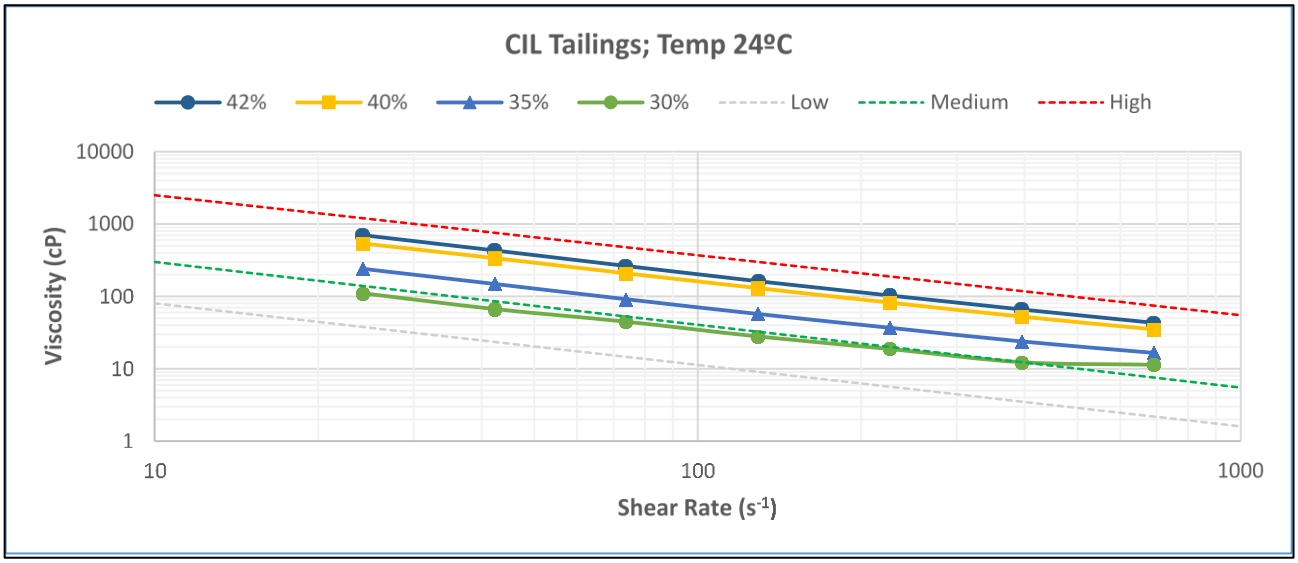


Figure 74: LOM Blend CIL tailings – slurry viscosity

13.7.6 Flotation tailings thickener underflow rheology

Flotation Tailings Thickener Underflow sample derived from dynamic thickening testwork was utilised for slurry rheology characterisation based on solids densities at 65%, 60%, 55%, and 45% solids (w/w). The data presented in Table 91 and illustrated in Figure 75 does not indicate any issues with pumping based on the expected thickener underflow density of 55% solids (w/w).

Table 91: Flotation tailings thickener underflow – slurry rheology

Pulp Density % (w/w)	Shear Rate (s^{-1})							
	4.2	7.3	12.7	22.4	39.1	68.3	119.5	209.0
Shear Stress (Pa)								
65	3.9	4.6	5.5	7.2	9.0	13.2	21.9	37.7
60	2.2	2.5	3.1	3.8	5.0	7.4	11.5	24.5
55	0.0	0.0	0.0	1.3	2.0	3.5	7.7	17.6
45	0.0	0.0	0.0	0.0	1.3	2.0	5.2	11.8
Viscosity (cP)								
65	940	623	432	323	229	193	183	181
60	527	344	247	169	129	108	96	117
55	0	0	0	56	52	51	64	84
45	0	0	0	0	32	30	43	56

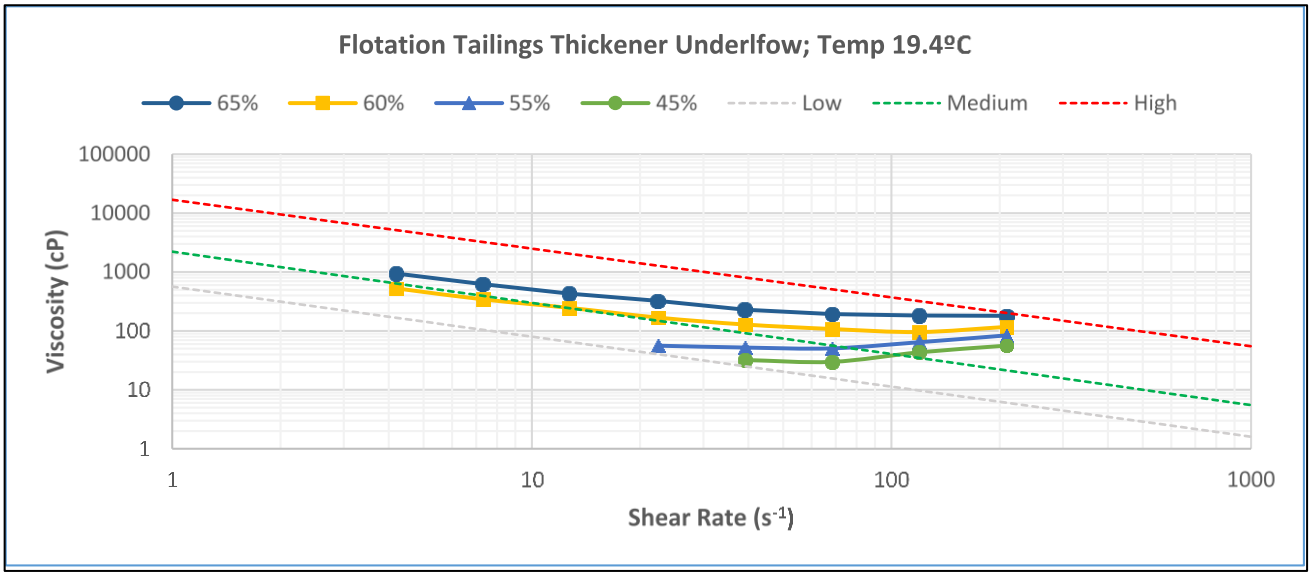


Figure 75: Flotation tailings thickener underflow – slurry viscosity

13.8 Flotation concentrate leaching

13.8.1 Conclusions

The following conclusions are presented following the testwork data analysis reported in this Section:

1) **Lead nitrate dosing** Leach extraction improves with lead nitrate dosing. MVO extraction increased in the range 0.7-4.1%; GRA 0.4-2.3% and DIO +0.4%. Improved extraction is as a result of reduced chemical passivation, likely due to the presence of antimony. Whether Aachen pre-treatment negates the improvement recorded based on lead nitrate dosing is unknown.

2) **Arsenic solution tenor** ranged from a minimum of 1.7mg/L in Starter Pit MVO to a maximum of 52.3 mg/L for Starter Pit DIO based on a leach density of 35% solids.

3) **Cyanide soluble copper** was identified in leach liquors; concentration ranged from 74 mg/L to a maximum of 488 mg/L, averaging 193 mg/L over a total of 20 tests across all lithologies based on a leach density of 35% solids. PFS whole ore mineralogy indicated copper as chalcopyrite and tennantite; tennantite is indicated as the possible source of elevated copper tenor.

4) **Copper and CIL** adsorption of copper (di-cyanide) complex onto activated carbon reduces gold adsorption kinetics and gold loading. Mitigation is by operation at high pH (> 10.5-11.0), maintaining a minimum molar free cyanide concentration four (4) times that of copper to promote copper tri and tetra cyanide complexation. On this basis a maximum copper solution tenor at 488 mg/L requires terminal CIL free cyanide above a minimum of 1,505 ppm. To date standard leach conditions adopted for the purpose of testwork have targeted 1,500 ppm.

5) **Thiocyanate** Based on the cyanide dosing during leach tests and resulting leach liquor ICP analysis, the predominant form of cyanide consumption is by active sulphides and the formation of thiocyanate (SCN).

6) **Starter Pit leach extraction variability** Baseline testing performed by ALS indicates leach extractions consistent with the PFS. MVO gold extraction is predicated on associated arsenic grade with lower arsenic producing lower leach residues and higher extraction. Similar trending was noted for DIO during the PFS.

7) **Aachen sighter testing** demonstrated improved leach extraction. Based on a regrind (P_{90}) size of 9 µm and 40-Pass treatment, respective increases in leach extraction for MVO, GRA, and DIO were 7.1%, 8.0%, and 5.1%.

8) **Aachen definitive testing** demonstrated increase in leach extraction of 4.7-5.9% over Baseline on a LOM Blend basis. The associated increase in total gold extraction (including gravity recovery) reported as +4.6% based on the composites tested.

9) **Baseline cyanide consumption** on a LOM Blend basis net consumption averaged 4.8 kg/t Con with a maximum of 6.2 kg/t based on ALS testwork. Similar testing by MMSA reported 5.2 kg/t (net).

10) **Aachen cyanide consumption** comparison between Baseline (without Aachen) and Aachen pre-treatment was complicated given seemingly anomalously high consumption rates reported for initial LOM composite tests. Baseline consumption was 5.2 kg/t relative to lower post Aachen consumption at 2.91 kg/t (excluding high-end consumption rates). Reduction in cyanide consumption by Aachen oxidation of cyanide consumers is indicated based on a separate LOM Blend composite tested by MMSA and reporting 2.29 kg/t.

11) **Lime (CaO) consumption** average gross consumption across ALS and MMSA tests ranged from 2.2 kg/t to a maximum of 4.3 kg/t; average consumption reported at 3.4 kg/t on a LOM Blend basis.

13.8.2 Recommendations

- ***Incorporation of Aachen shear reactor pre-oxidation is supported.***
- ***Lead nitrate dosing is not assumed within the flowsheet, given recent testing of MMSA Leachox technology. Sighter Aachen testing is recommended, including lead nitrate dosing to assess the possibility of realising any residual improvement in leach extraction, mitigating the impact of chemical passivation.***
- ***The process flowsheet includes a standard ferric sulphate arsenic precipitation circuit. The requirement for, and specification of this circuit will need to be assessed based on the expected maximum arsenic solution tenor.***
- ***Increase in the terminal cyanide operating set-point is recommended from 1,500 ppm to 1,650 ppm to maintain a 10% margin based on maximum cyanide soluble copper solution tenor.***
- ***Current results indicate potential for reduction in cyanide consumption from a Baseline of ~5 kg/t to an average in the range 2.3-2.9 kg/t. Confirmation regarding any expected reduction in cyanide consumption post Aachen treatment is recommended.***

13.8.3 ALS flotation concentrate leach – lead nitrate and leach extraction

During the PFS improved leach extraction was achieved by lead nitrate dosing; MVO reported + 4.1% and GRA + 0.6% extraction uplift. Following the PFS ALS performed a series of ten (10) paired leach tests with and without 500 g/t lead nitrate. Tests were performed under the following conditions:

- Flotation concentrate UFG grind (P_{90}) size of 9 -10 µm
- Pulp density 35% solids (w/w); pH > 10.5 (lime); DO > 20 ppm sparging oxygen and standard cyanide conditions.

Consistent with the PFS the results presented in Table 92 confirm an overall improvement in leach extraction:

- MVO averaged + 0.7% based on separate SP and LOM composites
- GRA generated +2.3% based on SP composite with LOM-GRA reporting + 0.4%
- DIO reported + 0.4% based on SP-DIO composite.

Based on PFS results and the results presented below, a minimum improvement in leach extraction of + 0.4% is expected across lithology types. Reduction in dosage, optimization and impact in conjunction with MMSA Aachen technology should be considered.

Table 92: ALS lead nitrate dosing leach assessment – results summary

Composite	P_{90} (µm)	Lead Nitrate (g/t)	Gold Leach Feed		Gold Residue Grade and Extraction		Silver Leach Feed		Silver Residue Grade and Extraction	
			Assay g/t Con	Calc g/t Con	Residue g/t Con	Ext %	Assay g/t Con	Calc g/t Con	Residue g/t Con	Ext %
SP-MVO-LC	9.9	0	24.4	25.4	4.44	82.5	7.2	8.41	1.20	85.7
		500		24.2	4.07	83.2		7.78	1.20	84.6
LOM-MVO-LC	10.2	0	20.7	19.9	3.02	84.8	3.2	3.39	0.60	82.3
		500		21.7	3.15	85.5		3.61	0.60	83.4
SP-GRA-LC	9.9	0	21.2	23.8	3.54	85.1	5.9	6.92	1.20	82.6
		500		23.1	2.92	87.4		6.36	0.90	85.8
LOM-GRA-LC	9.3	0	22.9	21.6	3.34	84.6	5.1	6.51	0.90	86.2
		500		21.3	3.19	85.0		6.46	0.90	86.1
SP-DIO-LC	10.5	0	11.8	12.7	4.50	64.6	3.2	3.90	0.90	76.9
		500		12.6	4.40	65.0		3.61	0.60	83.4

Leach liquor ICP assays for each leach test are presented in Table 93; key comments include the following:

- Arsenic solution tenor ranged from a minimum of 7-12 mg/L in SP-GRA; maximal in SP-DIO at 39-52.3 mg/L.
- Maximum copper solution tenor was reported for LOM-GRA at 488 mg/L; solubilised copper impacts gold-activated-carbon adsorption kinetics due to preferential adsorption of di-cyanide copper complex $Cu(CN)_2$. Typically maintaining pH at 10.5-11.0 and a 4:1 molar ratio of free cyanide to copper alleviates this effect with solubilised copper existing as tetra and tri-cyanide complex. The reported maximum copper tenor at 488 mg/L is equivalent to a requirement to maintain a minimum leach terminal free cyanide concentration of ~1,505 ppm.
- Nickel, lead and zinc reported in relatively low ranges, typically below 20 mg/L.
- Sulphur ranged from a minimum of 769 mg/L (SP-MVO) to a maximum of 2,401 mg/L in LOM-GRA. A predominant form of cyanide consumption is by active sulphides and conversion to thiocyanate (SCN). Notably this series of tests excluded any potential reduction offered by Aachen pre-oxidation of leach feed.

Table 93: ALS lead nitrate dosing leach assessment – leach liquor ICP assay (key analytes)

Analyte	48-Hour Leach Liquor ICP Assay (mg/L) (With/Without 500 g/t Pb $(NO_3)_2$ Dosing									
	SP-MVO	SP-MVO	LOM-MVO	LOM-MVO	SP-GRA	SP-GRA	LOM GRA	LOM GRA	SP-DIO	SP-DIO
	0 g/t	500 g/t	0 g/t	500 g/t	0 g/t	500 g/t	0 g/t	500 g/t	0 g/t	500 g/t
As	46.5	44.9	14.4	13.7	11.6	7.0	11.2	21.8	52.3	39.0
Ca	333	229	610	735	613	478	1361	564	268	297
Co	1.85	2.05	0.70	1.05	0.50	0.55	1.70	1.80	1.95	2.20
Cu	277	250	156	163	152	143	419	488	192	188
Fe	51.4	41.7	60.2	65.3	55.8	49.7	32.5	38.9	57.9	45.5
Hg	0.004	0.004	<0.002	<0.002	<0.002	0.002	<0.002	0.002	0.022	0.024

Analyte	48-Hour Leach Liquor ICP Assay (mg/L) (With/Without 500 g/t Pb $(NO_3)_2$ Dosing									
	SP-MVO	SP-MVO	LOM-MVO	LOM-MVO	SP-GRA	SP-GRA	LOM GRA	LOM GRA	SP-DIO	SP-DIO
	0 g/t	500 g/t	0 g/t	500 g/t	0 g/t	500 g/t	0 g/t	500 g/t	0 g/t	500 g/t
K	27	27	42	62	43	43	40	36	27	29
Mg	5.4	5.6	8.4	9.8	9.0	6.0	15.8	7.8	6.0	7.2
Mo	0.10	0.10	0.35	0.50	0.20	0.30	0.15	0.20	0.10	0.15
Na	1042	971	1171	1117	1110	1115	1378	1352	968	856
Ni	16.40	17.80	6.25	9.35	5.40	6.70	5.50	7.95	17.40	19.70
Pb	0.10	0.10	0.10	0.15	<0.05	0.15	0.20	0.20	0.05	0.10
S	933	769	1402	1589	1419	1251	2401	1518	793	818
Sr	2.32	2.10	2.44	3.22	3.12	2.26	1.70	1.00	2.48	3.44
Zn	21.0	21.7	13.5	17.1	14.0	16.0	19.6	15.9	13.3	15.1

13.8.4 ALS – starter pit gold extraction variability

ALS conducted a series of tests to assess leach extraction variability of composites derived within the Starter Pit based on a regrind (P_{90}) size of 9 µm. Test scope focussed on key Starter Pit lithologies MVO and GRA. Results for MVO are presented in Table 95 with head assay analytes reported in Table 94; observations include the following:

Starter Pit extraction exhibits a clear trend relative to head arsenic concentration, illustrated in Figure 76.

- Composite SP-MVO12-VC reported minimum extraction of 70.1%, also reporting highest arsenic and antimony, respectively 1,640 ppm and 12 ppm.
- Composite SP-MVO25-VC returned the highest extraction at 89.8% assaying lowest in terms of arsenic and antimony, respectively 300 ppm and 2.4 ppm.
- The results indicate based on arsenic more representative of the mine plan, 500-700 ppm, Starter Pit MVO extraction in the range 83.1-85.8% at an average MVO head grade of 2.65 g/t and excluding any benefit offered by Aachen pre-oxidation.

Table 94: Starter Pit MVO – variability composite head assays

Comp	Au Assay	Au Calc	Ag	As	Cu	Hg	Pb	Sb	Te	Zn
	g/t	g/t	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm
SP-MVO-LC	2.01	2.15	<2	720	46	0.1	20	5.2	0.6	86
SP-MVO2-VC	2.92	2.90	<2	620	92	<0.1	<5	1.1	0.4	100
SP-MVO12-VC	1.78	2.39	<2	1640	102	<0.1	30	12.0	1.6	76
SP-MVO25-VC	2.20	2.43	<2	300	110	<0.1	10	2.4	0.4	128
SP-MVO38-VC	4.05	3.87	<2	700	110	<0.1	<5	3.4	0.6	80
LOM-MVO-LC	1.65	1.85	<2	970	54	0.1	15	2.2	0.4	80

Table 95: Starter Pit MVO – grade and extraction variability summary

Comp	Calc Head Au g/t Ore	Flotation			Leach Residue Au g/t Con	Total Residue Au g/t Ore	Extraction			
		Mass %	Con Au g/t Con	Tail Au g/t Ore			Gravity	Flot	Leach	Total
SP-MVO-LC	2.07	9.38	18.52	0.10	2.89	0.37	11.25	83.93	84.40	**82.09**
SP-MVO-LC	2.22	6.92	25.41	0.09	4.44	0.40	16.41	79.38	82.53	**81.93**
SP-MVO2-VC	2.90	5.62	37.98	0.21	6.31	0.56	19.33	73.52	83.39	**80.63**
SP-MVO12-VC	2.39	9.02	22.20	0.15	6.22	0.72	9.78	83.75	71.98	**70.06**
SP-MVO25-VC	2.43	8.32	22.84	0.13	1.44	0.25	16.56	78.16	93.70	**89.79**
SP-MVO38-VC	3.87	8.52	38.58	0.21	4.25	0.57	9.72	84.96	88.98	**85.32**
LOM-MVO-LC	1.85	8.08	19.89	0.11	3.02	0.35	7.36	86.69	84.82	**80.89**

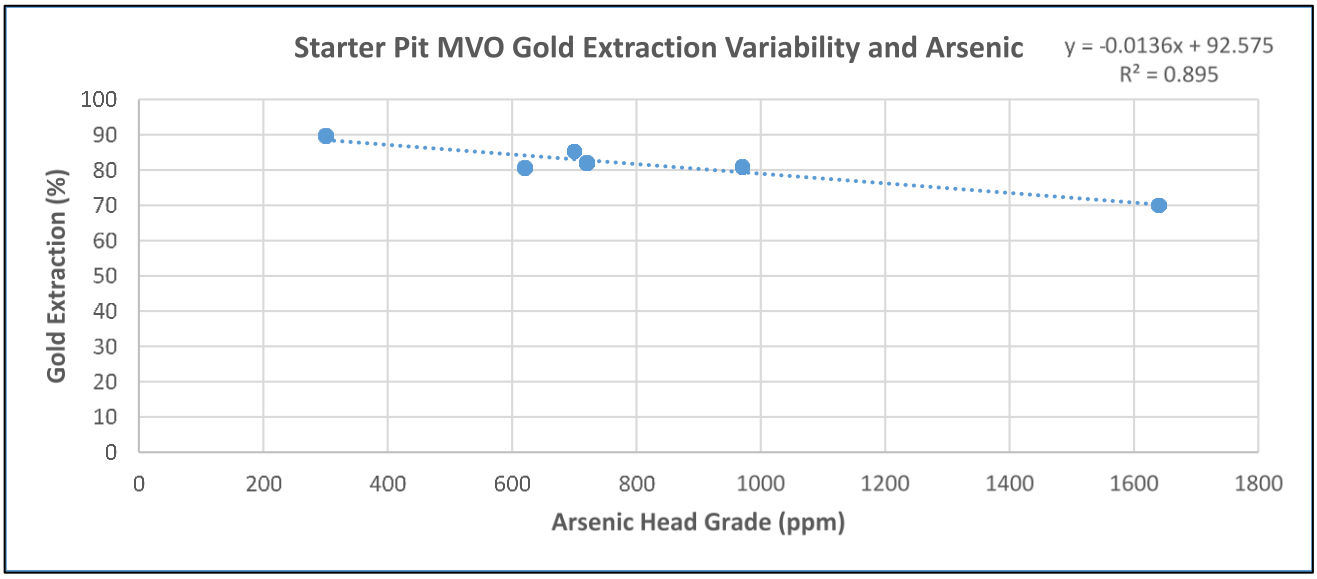


Figure 76: Starter Pit MVO – gold extraction variability and arsenic head grade

Results for GRA variability composites are shown below in Table 97 with key assay analytes presented in Table 96:

- GRA extraction does not appear as sensitive to arsenic and antimony as evidenced for MVO.
- Extraction ranged from a minimum of 77.2% (SP-GRA23-VC) to a maximum of 86.7% (SP-GRA-LC).

Table 96: Starter Pit GRA – variability composite head assays

Comp	Au Assay	Au Calc	Ag	As	Cu	Hg	Pb	Sb	Te	Zn
	g/t	g/t	ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm
SP-GRA-LC	1.18	1.58	<2	280	16	0.2	25	4.1	0.80	56
SP-GRA1-VC	1.01	1.24	<2	560	42	<0.1	10	3.2	0.40	48
SP-GRA14-VC	0.59	0.75	<2	520	42	<0.1	15	2.4	0.40	50
SP-GRA23-VC	1.14	1.16	<2	190	58	<0.1	<5	4.8	0.60	40
SP-GRA30-VC	1.09	1.06	<2	360	46	<0.1	20	4.5	0.40	42
LOM-GRA-LC	1.46	1.32	<2	330	24	<0.1	10	4.4	0.40	42

Table 97: Starter Pit GRA – grade and extraction variability summary

Comp	Calc Head Au g/t Ore	Flotation			Leach Residue Au g/t Con	Total Residue Au g/t Ore	Extraction			
		Mass %	Con Au g/t Con	Tail Au g/t Ore			Gravity	Flot	Leach	Total
SP-GRA-LC	1.61	9.02	13.49	0.05	1.82	0.21	21.05	75.83	86.51	**86.66**
SP-GRA-LC	1.54	5.04	23.79	0.06	3.54	0.24	18.55	77.75	85.12	**84.73**
SP-GRA1-VC	1.24	7.71	12.63	0.06	1.37	0.17	16.48	78.32	89.16	**86.31**
SP-GRA14-VC	0.75	7.80	8.42	0.03	1.31	0.13	9.14	87.19	84.43	**82.75**
SP-GRA23-VC	1.16	8.30	13.34	0.04	2.75	0.26	1.40	95.44	79.39	**77.17**
SP-GRA30-VC	1.06	7.66	12.31	0.06	1.74	0.19	5.89	88.89	85.87	**82.22**
LOM-GRA-LC	1.32	5.04	21.63	0.04	3.34	0.21	14.32	82.80	84.56	**84.33**

Leach liquor ICP assays for Starter Pit MVO and GRA are provided in Table 98:

- Arsenic ranged from 1.7 mg/L in SP-MVO25 with a maximum of 36.2 mg/L reported for SP-MVO12; maximum solubilised arsenic tenor is noted approximately 70% of that reported in Table 93 for SP-DIO (52.3 mg/L).
- Copper ranged from 74 mg/L in SP-GRA14 with a maximum of 236 mg/L reported for SP-MVO2; overall average Copper tenor was ~144 mg/L relative to an average of ~243 mg/L for composites reported in Table 93.
- Mercury ranged from below detection limit to a maximum of ~0.8 mg/L in SP-GRA1.
- Nickel, lead and zinc reported low, typically at and below ~60 mg/L.
- Sulphur varied from a minimum of 1,186 mg/L in SP-GRA30 through to a maximum of 2,707 mg/L in SP-MVO38.

Table 98: ALS Starter Pit MVO and GRA variability – leach liquor ICP assays (key analytes)

Analyte	48-Hour Leach Liquor ICP Assay (mg/L)									
	SP-MVO	SP-MVO2	SP-MVO12	SP-MVO25	SP-MVO38	SP-GRA	SP-GRA1	SP-GRA14	SP-GRA23	SP-GRA30
As	26.6	17.7	36.2	1.7	16.3	25.7	30.0	22.9	3.9	30.9
Ca	611	759	1069	1412	1442	733	717	728	932	381
Co	2.35	9.20	2.62	3.51	2.35	0.60	0.76	0.45	0.85	0.76
Cu	210	236	132	144	200	83	83	74	150	128
Fe	42.8	20.1	80.5	7.6	43.6	48.5	61.3	37.4	16.0	46.6
Hg	0.01	0.138	<0.002	0.58	0.004	0.242	0.788	0.66	0.628	0.092
K	48.7	68.5	49.5	56.0	68.2	71.5	69.3	67.4	56.1	64.9
Mg	5.3	2.9	7.9	7.4	7.2	7.7	7.6	10.4	5.5	5.7
Mo	0.14	<0.05	0.21	0.12	0.21	0.42	1.22	0.36	0.42	1.20
Na	1258	1496	1421	1268	1425	1319	1396	1194	1523	1261
Ni	23.3	60.5	31.6	26.6	29.9	7.3	9.1	5.3	12.4	8.3
Pb	0.05	0.12	0.17	0.12	0.19	0.05	0.07	0.05	0.12	<0.05
S	1458	1711	2419	2286	2707	1681	1748	1580	2038	1186
Sr	2.84	1.56	1.90	3.86	2.99	2.94	2.90	3.50	3.09	1.38
Zn	15.3	11.0	13.7	11.0	15.1	6.2	7.7	4.8	13.8	9.5

13.8.5 Cyanide and lime consumption

Net cyanide and Lime consumption rates are presented in Table 99 including combined results derived during the PFS and current feasibility testwork. Results based on maintaining a minimum of 1,500 ppm free cyanide and pH 11 indicate:

- LOM Blend cyanide consumption ranging from 3.34 to 6.20 kg/t with an average of ~4.8 kg/t concentrate.
- Lime (CaO) consumption in the range 3.14 - 3.92 kg/t with average consumption at ~3.5k g/t concentrate on a LOM Blend basis.
- Highest cyanide (and lime) consumer content is indicated for GRA with an average cyanide consumption of 7.65 kg/t concentrate, approximately double that of MVO and DIO. GRA lime consumption also reported highest at 4.40 kg/t concentrate.

Table 99: ALS concentrate leach – cyanide and lime consumption rates

Lithology	Mass %	P_{90} (µm)	Reagent Regime		NaCN (kg/t) Concentrate			CaO (kg/t) Concentrate		
			NaCN (ppm)	pH	Ave	Min	Max	Ave	Min	Max
MVO	56.1	9	1500	11.0	3.69	3.31	4.07	3.26	2.86	3.66
GRA	27.7				7.65	4.55	10.75	4.40	3.86	4.93
DIO	16.1				3.58	1.38	5.78	2.97	2.86	3.07
LOM	**100.0**	**9**	**1500**	**11.0**	**4.77**	**3.34**	**6.20**	**3.53**	**3.14**	**3.92**

Note: Lime consumption based on 69% available CaO.

13.9 MMSA Aachen Reactor – flotation concentrate leaching

13.9.1 Sighter stage leach testwork

Testwork employing MMSA Aachen Leachox technology was implemented to assess the impact of high shear pre-oxidation on leach extraction. Seven (7) bulk samples derived within the Starter Pit North/South and separate lithology composites representing the LOM case were dispatched to MMSA for sighter stage assessment.

Bulk samples were subjected to Knelson gravity recovery and intensive leaching ahead of primary milling and flotation to generate bulk flotation concentrate for comparative leach tests. Separate bulk flotation concentrates were homogenised, subsampled and ground targeting a range of regrind (P_{90}) sizes from 75 µm through to 5 µm. At each grind size samples were split to provide subsample for Baseline direct cyanidation leaching (without Aachen) and a separate sample for Aachen pre-oxidation and leaching.

13.9.1.1 LOM composite head assay

Key head assay analytes for LOM composites are summarised in Table 100 and as follows:

- Gold head grades reported higher than mine plan average for LOM GRA (1.76 g/t relative to 1.12 g/t) and DIO (1.84 g/t compared to 1.04 g/t). LOM-MVO assayed lower at 0.75 g/t relative to 1.17 g/t.
- Organic carbon assays tended within the range indicated for historical testwork samples.

Table 100: MMSA composite head assays – key analytes

Comp	Au (FA)	Ag	As	C(Org)	Cu	Pb	S^{2-}	Sb	Te	Zn
	g/t	ppm	ppm	ppm	ppm	ppm	%	ppm	ppm	ppm
LOM-MVO	0.75	0.05	411	0.060	25.5	15.9	0.73	4.4	0.6	40
LOM-GRA	1.76	0.05	802	0.025	44.6	6.7	0.73	1.8	0.3	70
LOM-DIO	1.84	0.05	363	0.025	82.5	11.2	1.44	0.9	0.4	83

13.9.1.2 LOM composite – Baseline and Aachen leach extraction

Comparative leach extractions are presented in Table 101 including Baseline (0-Pass) and Aachen results for 5 and 40 passes through the pilot shear reactor. All composites demonstrated increase in leach extraction, notably:

- LOM-MVO extraction increased by an average of 3.5-5.3% across the grind size range from a P_{90} size of 75 µm through 5 µm. Based on a regrind P_{90} size of 9 µm extraction increased by 4.6-7.1% over Baseline.
- LOM-GRA improved by an average 5.3-6.7% across grind sizes, reporting +8.0-8.6% at a 9 µm grind.
- LOM-DIO demonstrated a 6.5-7.7% increase over the full grind size range and 1.5-5.4% at 9 µm.

Table 101: MMSA LOM composite – Baseline and Aachen Leachox summary

P_{90} (µm)	Calc Head Au (g/t)	48 hr Leach Residue Au (g/t)			48 hr Au Extraction (%)			Extraction Delta (%)		
		0-Pass	5-Pass	40-Pass	0-Pass	5-Pass	40-Pass	5-Pass	40-Pass	Average
LOM-MVO; Au Assay Head = 14.1 g/5										
75	14.12	4.40	4.25	3.80	68.1	71.3	72.4	3.3	4.3	**3.8**
45	13.89	2.60	2.05	1.75	81.7	85.2	87.1	3.5	5.4	**4.5**
9	14.81	1.68	0.93	0.60	88.9	93.5	96.0	4.6	7.1	**5.8**
5	14.16	1.42	1.05	0.73	90.1	92.7	94.7	2.6	4.6	**3.6**
							Average	**3.5**	**5.3**	**4.4**
LOM-GRA; Au Assay Head = 19.0 g/t										
75	19.07	5.17	4.16	3.52	72.1	77.2	82.7	5.1	10.7	**7.9**
45	18.05	1.20	0.86	0.62	93.4	95.2	96.6	1.9	3.2	**2.5**
9	18.57	2.55	0.79	0.91	87.0	95.6	95.0	8.6	8.0	**8.3**
5	18.02	1.80	0.80	0.91	89.9	95.5	95.0	5.7	5.1	**5.4**
							Average	**5.3**	**6.7**	**6.0**
LOM-DIO; Au Assay Head = 13.8 g/t										
75	13.93	3.99	2.42	2.11	70.4	82.2	85.6	11.9	15.2	**13.6**
45	13.64	1.11	0.47	0.78	91.8	96.6	94.4	4.8	2.6	**3.7**
9	12.98	1.66	1.39	0.91	87.5	89.0	93.0	1.5	5.4	**3.5**
5	13.43	1.61	0.55	0.52	88.3	96.0	96.0	7.7	7.7	**7.7**
							Average	**6.5**	**7.7**	**7.1**

Note: 0-Pass = Baseline (without Aachen oxidation).

13.9.2 Stage 2 Aachen testwork

13.9.2.1 Gold extraction

On completion of sighter testing MMSA completed a second stage utilising separate lithology composites as well as a blended composite (LOM Blend Comp) prepared based on homogenised subsamples of separate lithology composites.

The scope was consistent with sighter testing and included front-end gravity and intensive leaching, flotation and ultrafine grinding targeting regrind (P_{90}) sizes at 45 µm, 20 µm, and 9 µm. Separate sub-splits of ground flotation concentrate were subjected to comparative leaching with and without prior Aachen pre-oxidation. Subsequent CIL leaching was conducted under standard conditions with 20 g/L activated carbon, pH > 10.5 and maintaining 1,500 ppm free cyanide for a duration of 48 hours.

Key results are shown in Table 102; results achieved at a P_{90} size of 9 µm demonstrate the following:

- Increased gold extraction across all lithologies; MVO reporting + 2.1%; GRA +6.0% and DIO +7.2%.
- LOM Blend Baseline extraction at 73.7% based on the specific composites tested.
- Aachen extraction reported at 77.7% on a LOM basis representing an increase of +4.0%.
- GRA Baseline extraction reported at 68.4%, low relative to historical results due to elevated arsenic noted in this specific composite relative to expected mine plant average.
- Flotation recovery was 88.0-90.0% based on separate lithologies and 88.8% based on the LOM Blend composite. Historical average recoveries have ranged from 93 to 95%; lower recoveries were due to elevated flotation tailings grades reported for GRA (0.147 g/t) and DIO (0.129 g/t). The historical average across 72 tests is 0.083 g/t. The result achieved for GRA at 0.147 g/t is compared to the historical average at 0.053 g/t and DIO at 0.129 g/t relative to 0.061 g/t.
- MVO has proven the most difficult lithology to achieve flotation tailings grading below 0.1 g/t, notably the tailings grade achieved by MMSA for MVO at 0.075 g/t is actually lower than the historical average at 0.09-0.10 g/t. Lower flotation recoveries reported by MMSA for GRA and DIO lithologies are suspected to be a function of technique.

Table 102: MMSA LOM composite – gravity and intensive leach results summary

Head Au (g/t)	Gravity and Flotation				Regrind P_{90} (µm)	CIL Au Extraction (%)			Total Au Extraction (%)		
	Gravity Rec (%)	ILR Ext (%)	Gravity Ext (%)	Flotation Rec (%)		Baseline	Aachen	Delta	Baseline	Aachen	Delta
MVO											
0.88	13.4	86.0	11.5	90.0	45	73.7	79.2	5.4	70.2	74.6	**4.3**
					20	72.3	83.4	11.1	69.1	78.0	**8.9**
					9	**84.7**	**87.4**	**2.7**	**79.0**	**81.1**	**2.1**
GRA											
1.77	6.5	66.1	4.3	87.7	45	63.7	78.0	14.3	57.8	69.8	**12.0**
					20	70.2	79.8	9.6	63.2	71.2	**8.1**
					9	**76.3**	**83.5**	**7.2**	**68.4**	**74.4**	**6.0**
DIO											
1.53	3.1	56.2	1.8	88.0	45	50.3	66.2	15.9	45.2	59.0	**13.8**
					20	64.5	73.1	8.6	57.5	65.0	**7.4**
					9	**71.4**	**79.7**	**8.3**	**63.5**	**70.7**	**7.2**
LOM Blend: 56% MVO; 28% GRA; 16% DIO											
1.23	9.8	75.7	7.9	89.0	45	67.2	76.8	9.6	63.0	70.9	**7.9**
					20	70.5	80.8	10.3	65.7	74.1	**8.4**
					9	**80.3**	**85.1**	**4.8**	**73.7**	**77.7**	**4.0**

13.9.2.2 Reagent consumption

Net cyanide and gross CaO consumption rates are reported in Table 103; key comments include:

- Average (Net) NaCN consumption per tonne concentrate across all tests ranged from 4.18 kg/t for DIO to 5.33 kg/t for MVO. Calculated average consumption on a LOM Blend basis was 5.07 kg/t. Reference to ALS results in Section 13.8.5 (Table 99) shows an equivalent LOM net consumption of 4.77 kg/t.

- Average (Gross) CaO consumption ranged from 3.04 kg/t in MVO to 3.83 kg/t in DIO with a calculated LOM Blend consumption rate of 3.26 kg/t. Reference to separate ALS results indicates a LOM Blend consumption of 3.53 kg/t.
- The data does not indicate a clear differential between rates reported for Baseline tests (without Aachen treatment) and with Aachen pre-treatment. This is due to the possible bias introduced by large consumptions reported for Aachen tests on LOM lithologies; these range from ~11.6 to 15.0 kg/t relative to lower rates reported for separate lithology variability tests. MMSA completed Aachen treatment on a separate LOM Blend composite formed from sub-splits of the separate LOM lithologies, this test reported net cyanide consumption at 2.29kg/t consistent with the results tabled for lithology variability composites.
- Comparative results are presented in Table 104 on a LOM Blend basis, excluding high-end cyanide consumptions reported for Aachen testing on LOM composites. The average consumption rate for the Baseline (without Aachen) is consistent with earlier ALS results at 5.20 kg/t and consumption with Aachen treatment averages 2.91 kg/t, this is also noted more consistent with the actual test result reported for the separate LOM Blend composite at 2.29 kg/t. In overall terms, the test data indicates potential for reduced cyanide consumption by Aachen pre-oxidation of sulphides, minimising consumption by formation of thiocyanate.

Table 103: MMSA Baseline and Aachen cyanide and lime consumption – all data

Lithology	Composite	Case	Net Consumption (kg/t Con)	
			NaCN	CaO
MVO	LOM-MVO	Baseline	5.19	1.10
	LOM-MVO	Aachen	15.02	1.72
	MVO NMDD085 (1)	Aachen	3.57	4.25
	MVO NMDD085 (2)	Aachen	3.69	3.71
	MVO NMDD080	Aachen	2.43	3.36
	MVO NMDD149	Aachen	2.06	4.09
		Average	**5.33**	**3.04**
		Minimum	**2.06**	**1.10**
		Maximum	**15.02**	**4.25**
GRA	LOM-GRA	Baseline	6.33	1.21
	LOM-GRA	Aachen	11.75	3.07
	GRA NMDD088	Aachen	3.73	3.56
	NMDD136/NMDD022/NMRC085 (1)	Aachen	3.34	4.63
	NMDD136/NMDD022/NMRC085 (2)	Aachen	2.90	4.27
	NMDD133	Aachen	2.37	3.43
		Average	**5.07**	**3.36**
		Minimum	**2.37**	**1.21**
		Maximum	**11.75**	**4.63**
DIO	LOM-DIO	Baseline	3.30	1.62
	LOM-DIO	Aachen	11.57	3.02
	DIO NMDD098	Aachen	2.94	6.70
	DIO NMDD110/NMRD126	Aachen	3.52	5.35
	DIO NMDD144/NMDD104/NMDD134	Aachen	2.14	3.61
	DIO NMRC133/NMDD086	Aachen	1.59	2.69
		Average	**4.18**	**3.83**
		Minimum	**1.59**	**1.62**
		Maximum	**11.57**	**6.70**
LOM Blend	**Average**	LOM	**5.07**	**3.26**
	Minimum		**2.07**	**1.22**
	Maximum		**13.56**	**4.75**

Table 104: MMSA Baseline and Aachen cyanide and lime consumption – trimmed data

Basis	Case	NaCN (kg/t Con) Net			CaO (kg/t Con) Gross		
		Average	Min	Max	Average	Min	Max
LOM Blend	Baseline	5.20			1.22		
LOM Blend	Aachen	2.91	2.07	3.68	4.00	3.27	4.75
LOM Blend Composite	Aachen	2.29			2.23		

Note: excludes high end consumption rates

13.10 Metallurgical recovery

13.10.1 Baseline recovery – excluding Aachen treatment

On completion of the PFS, Baseline LOM gold recovery was estimated at 79.3% at a regrind (P_{90}) size of 9µm and LOM Blend representing: 60% MVO, 30% GRA and 10% DIO. Subsequent to the PFS changes in the mine plan resulted in a LOM Blend comprising: 56.1% MVO, 27.7% GRA and increased DIO at 16.2%. Relative changes in separate lithology gold head grade and arsenic were also noted.

LOM Baseline gold recovery is shown in Table 105 as 78.6% after adopting revised blend proportions, including variation in gold and arsenic head grade and with allowance for nominal CIL solution and attritioned carbon loss.

13.10.2 Metallurgical recovery – including Aachen treatment at regrind P_{90} 9 µm

The current process flowsheet incorporates Aachen reactor pre-oxidation ahead of CIL; Aachen pilot test data demonstrated an uplift of +4.0% on a LOM Blend basis as shown in Table 105 and Figure 77.

Analysis of the variable regrind test data presented in Section 13.9 (Table 102) indicates that with incorporation of Aachen pre-oxidation, a total recovery in the range 82 – 83% can be achieved from Namdini Oxide, Transition and Fresh ore feed at a regrind (P_{90}) size of ~9 µm.

Table 105: Baseline and Aachen – estimated Fresh ore gold recovery at regrind P_{90} 9 µm

Lithology	Blend %	Head Grade		Recovery (%)		
		Au (g/t)	As (ppm)	Baseline	Aachen Lower	Delta
MVO	56.1	1.17	547	77.2	79.3	2.1
GRA	27.7	1.12	463	85.0	91.0	6.0
DIO	16.2	1.04	347	72.3	79.5	7.2
LOM (Fresh)	**100.0**	**1.13**	**491**	**78.6**	**82.6**	**4.0**

Given existing Baseline gold recovery at 78.6%, LOM Fresh gold recovery including Aachen pre-treatment is estimated at 82.6% based on revised blend proportions, LOM head grading 1.13 g/t and a regrind P_{90} size of 9 µm.

Table 106 indicates the calculated overall recovery with both Oxide and Fresh ore for the Namdini Gold Project process flow sheet which includes Aachen technology. The calculations are rounded to reflect their precision and may include rounding errors.

Table 106: Overall estimated gold recovery – Fresh plus Oxide ore

Ore Type	Tonnes (Mt)	Gold Grade (g/t)	Contained Gold (Moz)	Recovered Gold (Moz)	Recovery (%)
Oxide	4.1	1.11	0.15	0.13	90.0
Fresh	134.5	1.13	4.89	4.05	82.6
Overall	**138.6**	**1.13**	**5.05**	**4.18**	**82.9**

The overall recovery for the Namdini Gold Project LOM Fresh plus Oxide ore is ~83%.

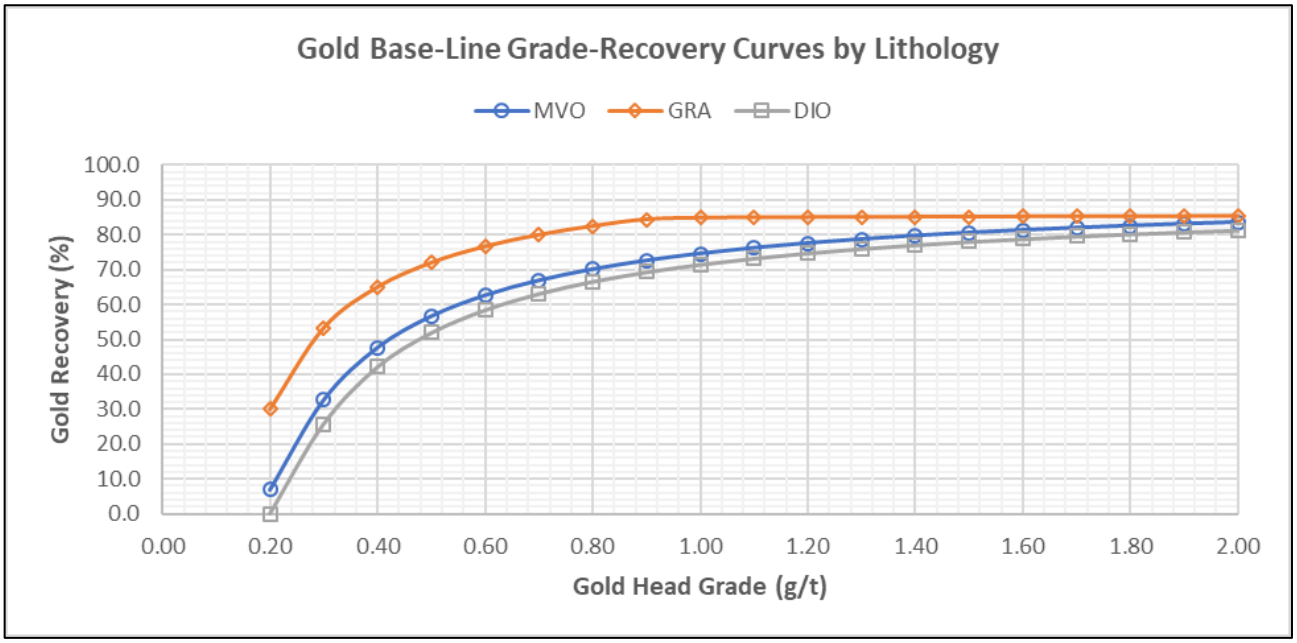


Figure 77: Baseline gold grade-recovery curves by lithology type P_{90} 9 µm

13.11 ALS carbon adsorption and cyanide detoxification

13.11.1 Carbon adsorption – kinetics and equilibrium loading

13.11.1.1 Bulk leach extraction and solution tenor

Carbon adsorption testwork was performed on LOM Blend bulk sample prepared utilising reserve samples derived from the HIG pilot and maintained in cold storage by ALS prior to this phase of testwork.

The bulk composite reported respective gold and silver assay head grades of 20.6 g/t and 5.2 g/t. Bulk leaching was conducted at 35% solids (w/w) over 48 hours targeting pH >10.5, dosing lime and maintaining 1,500 ppm free cyanide with oxygen sparged to maintain DO > 20 ppm.

Results are presented in Table 107 at 36 and 48-hours retention; final gold extraction was 85.3% and Silver, 73.5%. Final 48-hour leach liquor contained 83.8 mg/L arsenic and 411 mg/L copper. Based on the level of cyanide soluble copper (411 mg/L) a minimum terminal free cyanide concentration of ~1,268 ppm (4:1 molar ratio) would be required to maximise the proportion of $Cu(CN)_3^{-2}$ and $Cu(CN)_4^{-3}$ over $Cu(CN)_2^{-}$, given the propensity of di-cyanide copper complex to adsorb onto activated carbon taking up active sites for the absorption of gold. In overall terms the results support maintaining terminal free cyanide at or greater than 1,500 ppm dependent on terminal leach liquor copper tenor.

Table 107: LOM bulk composite – leach extraction summary

Retention (hours)	Reagent (kg/t)		Extraction (%)		Solution Tenor (mg/L)			
	NaCN	CaO	Au	Ag	Au	Ag	As	Cu
36	6.14	2.62	85.0	69.7	9.06	2.56	78.7	391
48	6.21	3.82	85.3	73.5	9.10	2.70	83.8	411
Assay Head (g/t)			**20.6**	**5.2**				
Calculated Head (g/t)			**19.8**	**6.8**				

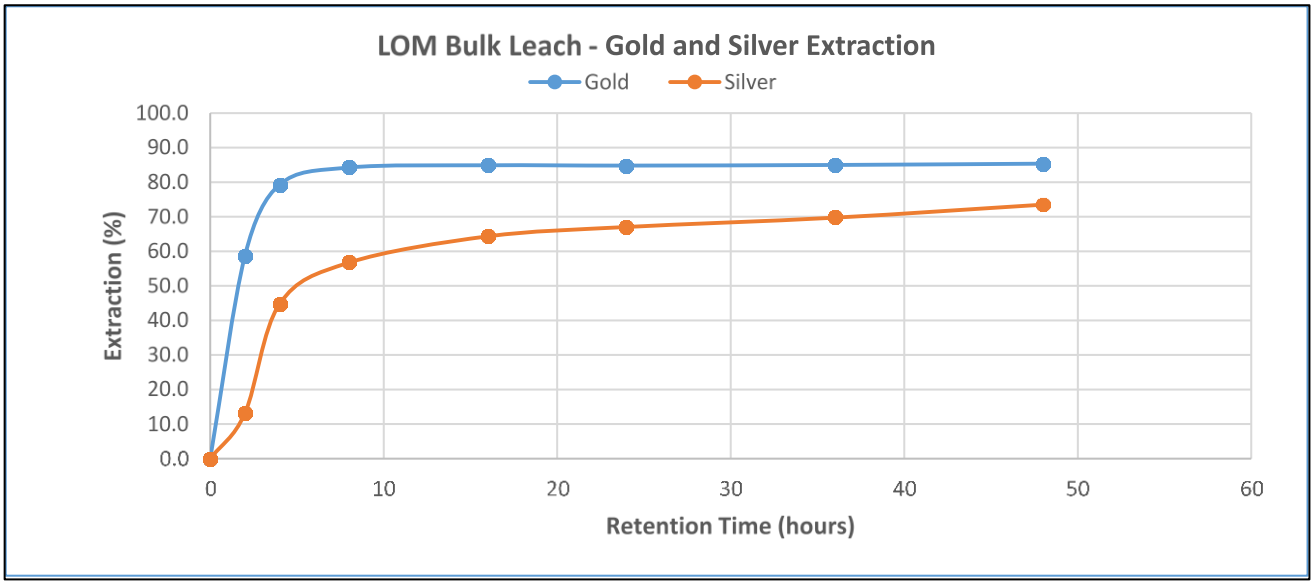


Figure 78: LOM bulk leach – gold and silver kinetic profile

13.11.1.2 Carbon adsorption kinetics and equilibrium loading

Adsorption testwork used the 48-hour leach liquor presented in Table 105 and activated carbon (HATCARB YAO 6 × 12). Kinetic profiles for gold and silver loading are illustrated in Figure 78 and Fleming kinetic constants presented in Table 108. Gold adsorption kinetics are noted low range based on a Fleming (K) value of ~63 hr^{-1} relative to industry typical values in the range 150 – 250.

Carbon equilibrium loadings are presented in Table 110; gold loading reported at 9,583 g/t and silver, 1,170 g/t based on an equilibrium solution tenor of 1 mg/L.

Throughout the testwork it was noted that copper continued to slowly leach after 48-hours. Cyanide consumption also persisted due to active sulphides and consumption as thiocyanate (SCN). These two effects required the free cyanide concentration to be maintained at 1,500 ppm to prevent preferential copper adsorption onto the carbon.

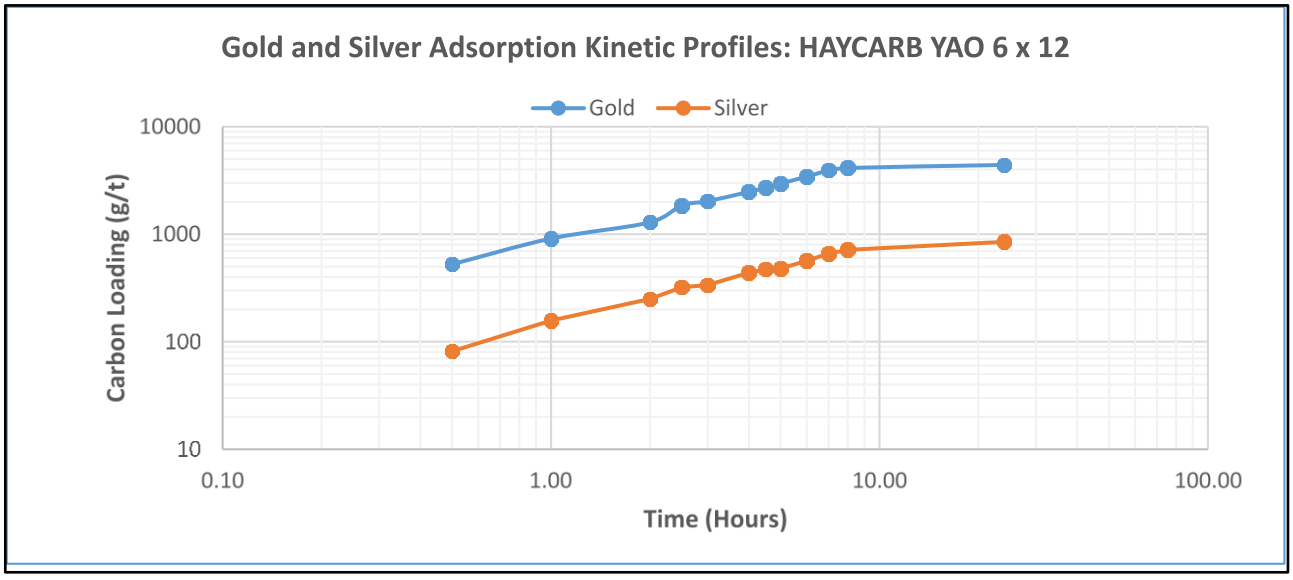


Figure 79: Gold and silver – carbon adsorption kinetic profiles

Table 108: Fleming kinetic constants

Element	K (hr^{-1})	n
Au	62.94	0.753
Ag	32.32	0.765
Cu	9.95	0.482

Table 109: Triple contact kinetic test – loaded carbon assay

Au	Ag	Cu	As	Fe	Co	Ni	Zn
ppm	ppm	ppm	ppm	ppm	ppm	ppm	ppm
3,576	696	744	120	1,924	< 5	1,215	102

Table 110: Gold and silver – equilibrium carbon loading

Element	Equilibrium Solution Tenor (mg/L)		
	0.20	0.50	1.00
Au	2929	5752	9583
Ag	197	543	1170

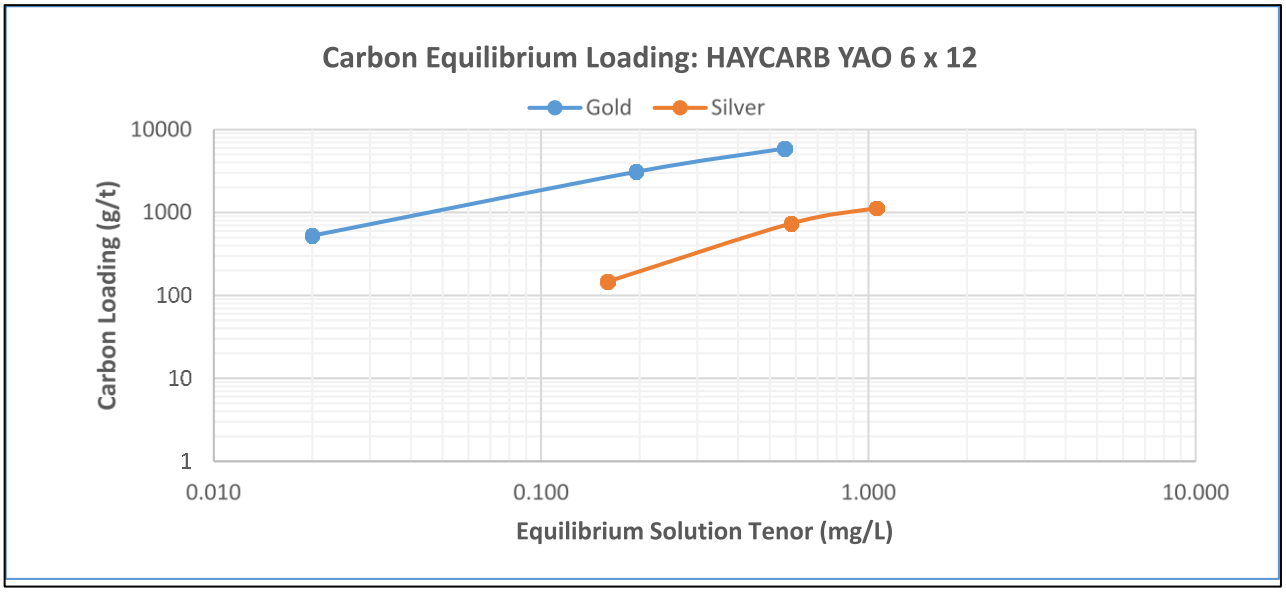


Figure 80: Gold and silver – carbon equilibrium loading

13.11.2 Cyanide detoxification

Cyanide detoxification by SMBS/Air was conducted on leach liquor generated by a bulk leach conducted Starter Pit bulk composite. Leaching was performed at 35% solids (w/w) maintaining 1,500 ppm free cyanide and a pH target > 10.5.

The CIL tailings liquor assay is presented below; free cyanide on commencement was 796 mg/L, WAD cyanide reported at 1,100 mg/L based on determination by Chemistry Centre of Western Australia ("**CCWA**") and copper assayed 411 mg/L. Copper liquor assays throughout this stage of feasibility testwork averaged 193 mg/L, ranging from a minimum of 74 mg/L to maximum at 488 mg/L. The copper liquor assay tabled below is noted high at approximately 85% of the maximum recorded across a total of 20 leach tests.

Table 111: Leach tailing liquor ICP assay (mg/L)

Cu	Fe	Ni	Zn	CN_{WAD}*	CN_{WAD}**	CN_{FREE}	CN_{TOTAL}*	SCN	OCN
411	254	11.7	16.4	1352	1100	796	1700	920	32.0

Notes: * WAD cyanide by ALS picric acid method.

** WAD cyanide by CCWA.

Results of SMBS/Air cyanide detoxification are presented in Table 112; key comments include the following:

- Testwork was conducted over a range of SMBS dosages representing 100% to 250% of the stoichiometric requirement with weak acid dissociable cyanide (CN_{WAD}) determined by picric acid method. Independent CN_{WAD} determination was conducted by CCWA based on the final test at 250% SMBS dose.
- The independent CCWA CN_{WAD} assay for the 250% test is noted at 0.3 mg/L relative to ALS at 44.1 mg/L by picric acid method. This result was confirmed by repeat assay at CCWA and is low as a result of decay following completion of the test at ALS, transfer and storage prior to assay at CCWA.
- Targeting a ceiling limit of 50 mg/L WAD cyanide a CN_{WAD}: SO_2 ratio of 1:4.76 is indicated based on results confirmed by picric acid method. Operation based on these criteria results in an SMBS consumption of 9.56 kg/m^3 liquor, copper sulphate consumption of 0.65 kg/m^3 liquor, requiring Lime at 2.46 k g/m^3 liquor. Overall this represents SMBS at 194% of the Stoichiometric requirement relative to current PDC at 200% stoichiometric.

Table 112: Cyanide detoxification (SMBS/Air) reagent dosage and consumption

Mass Ratio CN_{WAD}:SO_2	Dosage % (Note 1) SO_2:CN_{WAD}	Reagent Dosage (kg/m^3 Liquor)			Cyanide Speciation (mg/L)				
		SMBS	$CuSO_4$ (Note 2)	Lime	CN_{WAD} (Note 3)	CN_{WAD} (Note 4)	CN_{Total} (Note 4)	SCN (Note 4)	OCN (Note 4)
1:2.46	100	4.93	0.67	3.05	89.2				
1:3.79	154	7.61	0.71	3.38	79.4				
1:4.76	194	9.56	0.65	2.46	48.4				
1:6.17	251	12.38	0.58	2.65	44.1	0.3	310	690	780

Notes: 1) Dosage stated as % stoichiometric addition.
2) Dosage stated as $CuSO_4.5H_20$.
3) WAD cyanide determination by picric acid method.
4) Determination by CCWA.

Table 113: Cyanide detoxification (SMBS/Air) – terminal liquor assay

Mass Ratio CN_{WAD}:SO_2	Final Liquor Assay (mg/L)						Final pH and DO	
	Cu	Fe	Ni	Zn	CN_{Wad}	CN_{Total}	pH	DO (ppm)
1:2.46	2.72	182	0.05	<0.02	89.2	590	8.51	8.15
1:3.79	2.30	103	<0.05	<0.02	79.4	363	8.59	8.24
1:4.76	1.88	146	<0.05	<0.02	48.4	450	8.53	8.71
1:6.17	1.20	99	<0.05	<0.02	44.1	316	8.52	8.74

13.12 Metallurgical testing summary

Key aspects supporting selection of the process flowsheet include:

- Average recovery is dependent on head grade and the ratio of the different lithologies.
- Recovery was calculated by adding the gold recovered by gravity, flotation and leaching gold and allowing for gravity gold loss at scale and CIL carbon and solution loss. This is effectively gravity recovery plus flotation recovery multiplied by leach recovery.
- The Oxide results reported in the PFS have been retained as 90% recovery. This is based on whole ore CIL processing.
- The Transition material at Namdini is approximately 3% of the ore feed. It is only partially weathered (oxidized) and behaves in metallurgical test work similar to the Fresh material, with similar recoveries.

- Metallurgical testwork carried out to date indicates that the Namdini Gold Project can deliver a standard gold recovery process plant design with no unproven technology required.
- The metallurgical process includes 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.

13.13 Conclusions – metallurgy

Early testwork established that Namdini Oxide ore is free milling with an average recovery of 90%. The Transition material at Namdini is only partially weathered (oxidized) and behaves in metallurgical test work similar to the Fresh material, with similar recoveries. The Transition material is to be processed with the Fresh material as part of the mill feed and thus these are combined in all the mine and mill schedules.

The metallurgical work carried out to date indicates that gold can be recovered from Namdini Transition and Fresh ore feed using conventional flotation, regrind and Carbon In Leach (CIL) cyanidation techniques, and from Oxide ore feed using conventional CIL cyanidation techniques. The work is considered sufficient to define a technically and economically viable gold mining project.

14.0 MINERAL RESOURCE ESTIMATES

The information in this Section is derived from MPR (2019) and Golder (2019a). It is presented in a similar format to that in the PFS (Golder, 2018b) which is a comprehensive report provided on Sedar (https://www.sedar.com).

14.1 Introduction

Independent mining industry consultant, MPR Geological Consultants Pty Ltd (MPR) was commissioned by Cardinal to estimate the Mineral Resources of the Namdini deposit. The resulting block model was used for the mining planning studies leading to the Ore Reserve (Section 15.0).

This FS Report discusses the conversion of the Mineral Resource to an Ore Reserve in Section 16.0, and in so doing considers the Modifying Factors specifically addressed by international reporting codes (JORC, 2012; CIM, 2014). Consequently there are no known environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that materially impact on the Mineral Resource.

The Mineral Resources for the Namdini Gold Project were 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 4 February 2019. Relative to drilling information available for the March 2018 resource estimates, the dataset available for the current estimates includes an additional 4,271 m of drilling. This additional drilling is dominated by infill holes central and northern portions of the deposit as discussed in Section 10.2.

Micromine software was used for data compilation, domain wire-framing and coding of composite values and GS3M 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 from 11 to 14 January 2017.

The Mineral Resource estimates have been classified and reported in accordance with NI 43-101 guidelines and classifications adopted by CIM (2014).

14.2 Estimation dataset

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

- 14 diamond holes drilled for geotechnical or metallurgical purposes for which routine down-hole assays are un-available,
- 10 unassayed RC holes drilled for hydrological investigations,
- 196 sterilisation and exploratory RC holes from outside the resource area, and
- 317 closely spaced holes, drilled as part of a grade control trial program.

The complied resource dataset comprises 41,475 composites with gold grades from 0.001 to 242.05 g/t and averaging 0.57 g/t. Additional information compiled since construction of the March 2018 resource estimates (Golder, 2018b; MPR, 2018) represents around 3% of the dataset.

14.3 Modelling domains

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

The current estimates are based on a mineralized domain interpreted on the basis of composited gold grades. Domain boundaries were digitised on cross sections, snapped to drill hole traces where appropriate, then wire-framed into three-dimensional solid designated Domain 2. Domain 1 represents a background domain capturing generally unmineralized 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 degrees and is interpreted to a constant elevation of -650 mRL, which represents an average depth of around 860 m.

Figure 81 presents a plan-view of the surface expression of the mineralized domain relative to drill hole traces.

Logging, interpretation and modelling undertaken by Cardinal's technical staff and specialist structural consultants Orefind Pty Ltd (Davis & Cowan, 2016) produced a three-dimensional model of key rock types, structures and oxidation 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 partitioning final resources by oxidation type.

Depth to the interpreted base of complete oxidation ranges from locally 1 to 2 m up to 20 m, averaging 10 m. Interpreted depth to Fresh rock ranges from around 8 to 30 m and averages 18 m.

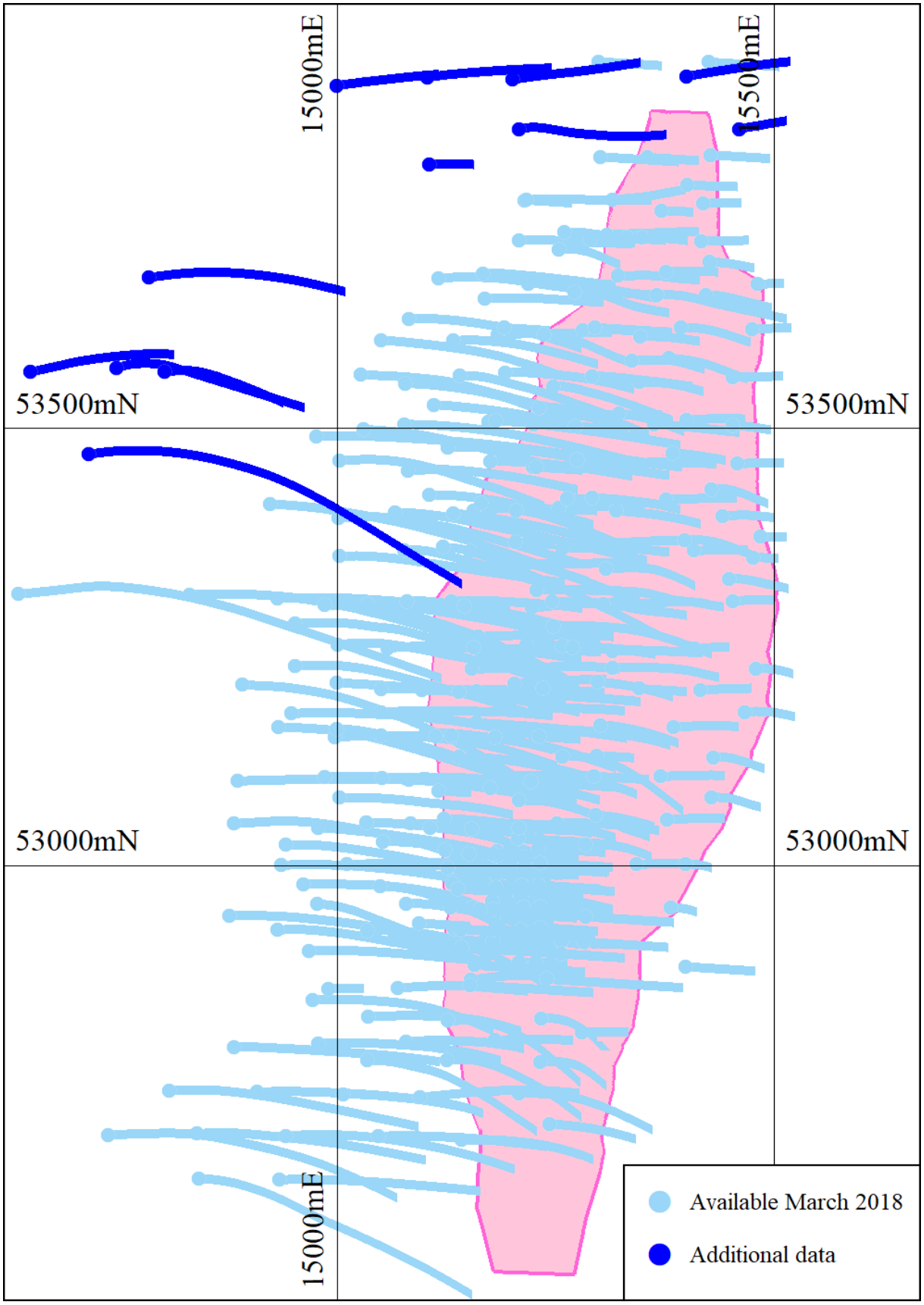


Figure 81: Mineralized domain and drill hole traces

14.4 Exploratory data analysis

Table 114 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 0.04 g/t, the mean gold grade for Domain 1 composites is notably lower than for Domain 2 demonstrating that the domaining has been 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 coefficients of variation of 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 114: Composite statistics

Au g/t	Oxide	Transition	Fresh	Combined
Domain 1				
Number	561	867	11,435	12,863
Mean	0.05	0.03	0.04	0.04
Variance	0.02	0.01	0.96	0.86
Coef. Var.	3.02	3.36	24.7	23.4
Minimum	0.005	0.005	0.001	0.001
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	1.87	2.11	103.09	103.09
Domain 2				
Number	1,297	1,057	26,258	28,612
Mean	0.83	0.74	0.81	0.81
Variance	2.24	2.35	11.1	10.4
Coef. Var.	1.81	2.06	4.12	3.99
Minimum	0.001	0.005	0.001	0.001
1st Quartile	0.120	0.070	0.045	0.050
Median	0.37	0.29	0.26	0.27
3rd Quartile	0.91	0.76	0.78	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 from the bin mean or median, with or without exclusion of high-grade composites, or rarely the upper bin threshold. 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 115 presents the grade thresholds and class mean gold grades, with the value assigned to the upper bin shown in brackets along with the 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 99.9 percentile of this subset.

Table 115: Indicator thresholds (Base) and class mean grades for percentiles

Percentile	Oxide		Transition		Fresh	
	Base	Mean	Base	Mean	Base	Mean
	Au g/t	Au g/t	Au g/t	Au g/t	Au g/t	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%	0.007	0.006	0.005	0.005	0.005	0.005
50%	0.013	0.011	0.007	0.006	0.006	0.006
60%	0.020	0.017	0.013	0.012	0.007	0.006
70%	0.027	0.024	0.018	0.015	0.013	0.011
75%	0.040	0.033	0.020	0.019	0.018	0.015
80%	0.050	0.045	0.025	0.024	0.020	0.019
85%	0.065	0.056	0.035	0.029	0.030	0.025
90%	0.090	0.077	0.052	0.042	0.043	0.035
95%	0.180	0.132	0.095	0.069	0.085	0.060
97%	0.230	0.203	0.190	0.141	0.145	0.108
99%	0.740	0.328	0.320	0.254	0.400	0.237
100%	1.865	1.260 (0.740)	2.110	0.815(0.658)	103.091	2.063(0.73)
		Threshold		Median		Median
Domain 2 (Mineralized)						
10%	0.040	0.019	0.015	0.007	0.007	0.005
20%	0.090	0.062	0.050	0.030	0.030	0.019
30%	0.165	0.123	0.105	0.073	0.070	0.046
40%	0.250	0.208	0.195	0.148	0.147	0.104
50%	0.370	0.308	0.290	0.245	0.260	0.201
60%	0.495	0.429	0.415	0.352	0.410	0.333
70%	0.735	0.598	0.620	0.515	0.625	0.510
75%	0.910	0.818	0.760	0.682	0.780	0.699
80%	1.145	1.017	0.995	0.867	0.995	0.882
85%	1.410	1.284	1.260	1.121	1.315	1.142
90%	1.980	1.707	1.710	1.454	1.835	1.547
95%	3.060	2.450	2.742	2.150	2.935	2.298
97%	4.190	3.545	3.985	3.379	3.945	3.395
99%	7.370	5.372	7.705	5.365	7.205	5.190
100%	21.200	11.138(9.925)	22.490	11.401(8.955)	46.185	12.790 (9.75)
		Median		Median	Mean excluding > 50 g/t	

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 the combined oxidation subdomains (Table 116). 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 82 presents a three-dimensional variogram surface map of the median indicator variogram model at variogram ranges at 90% of the sill.

Table 116: Variogram models

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)
Rotation: Z+34,Y+41,X+77							
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

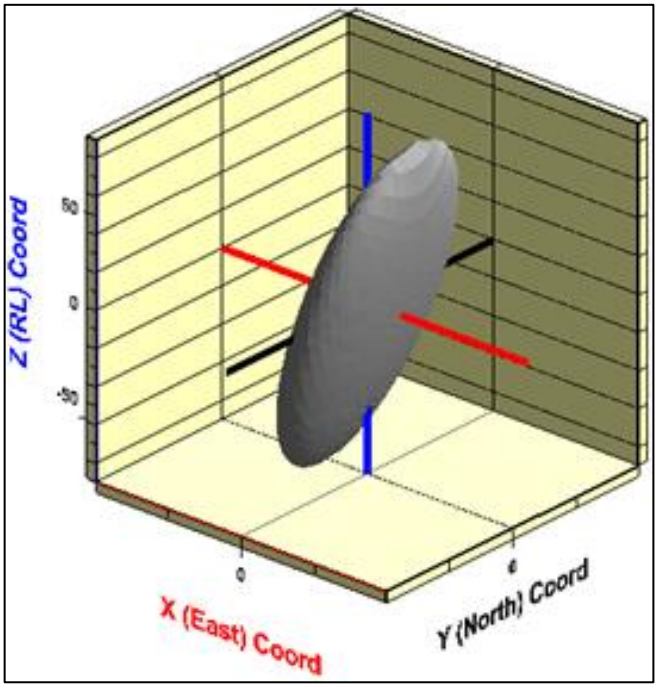


Figure 82: Three dimensional variogram plot

14.7 Estimation parameters

The block model framework used for MIK modelling covers the full extent of the composite dataset (Figure 38). It includes panels with dimensions of 12.5 m east-west by 25 m north-south by 5 m vertical. The plan-view panel dimensions are consistent with the drill hole spacing in more closely drilled portion of the deposit.

The three progressively more relaxed search criteria used for MIK estimation are presented in Table 117. The search ellipsoids were aligned with the general mineralization orientation.

Table 117: 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 selective mining unit (SMU) dimensions of 5 mE by 10 mN by 2.5 m RL with high quality grade control sampling on an 8 m by 12 m by 1.25 m pattern (see Section16.16).

The variance adjustments were applied using the Direct Lognormal Correction method with the adjustment factors listed in Table 118. 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.

Table 118: Variance adjustment factors

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

Bulk densities were assigned to the block model by rock type and oxidation domain. The assigned values (Table 119) were derived from the average of the available measurements (Table 49).

Table 119: 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 Resource classification

Resource model blocks were classified as Measured, Indicated or Inferred on the basis of search pass and three wire-frames outlining more closely drilled portions of the mineralization (Figure 83) comprising the following:

- A broad envelope, constrained to the mineralized domain outlining the extents of generally 50 by 50 m spaced drilling, which in the following description is defined as the classification wire-frame.
- A smaller triangulation, designated as classification wire-frame 2, encapsulating the volume tested by trial grade control drilling.
- A triangulation designated as classification wire-frame 3 outlining a zone of 25 by 25 m, with local infill to 12.5 by 12.5 m drilling centred around 52,960 mN.

Resource classifications were assigned to model panels 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 classification wire-frame 1 informed by search pass 3 were re-assigned to the Indicated category. These panels represent around 0.2% of the Indicated Mineral Resource.
- Proportionally few search pass 3 panels outside classification wire-frame 1 surrounded by Indicated panels were re-classified as Indicated. These panels represent around 0.4% of the Indicated Mineral Resource.
- Panels within the area of grade control sampling as defined by classification wire-frame 2 informed by searches 1 or 2 were classified as Measured Mineral Resource.
- Panels within the southern zone of closely spaced drilling outlined by classification wire-frame 3 informed by search pass 1 were assigned to the Measured Mineral Resource category.

The classification approach assigns estimates tested by generally 50 by 50 m and closer spaced drilling to the Indicated category, with estimates for zones with more closely spaced drilling classified as Measured. Estimates for panels not informed consistently by 50 by 50 m drilling are assigned to the Inferred category. Classification of the area of grade control sampling as Measured is warranted by the close agreement between resource and grade control estimates (Table 121).

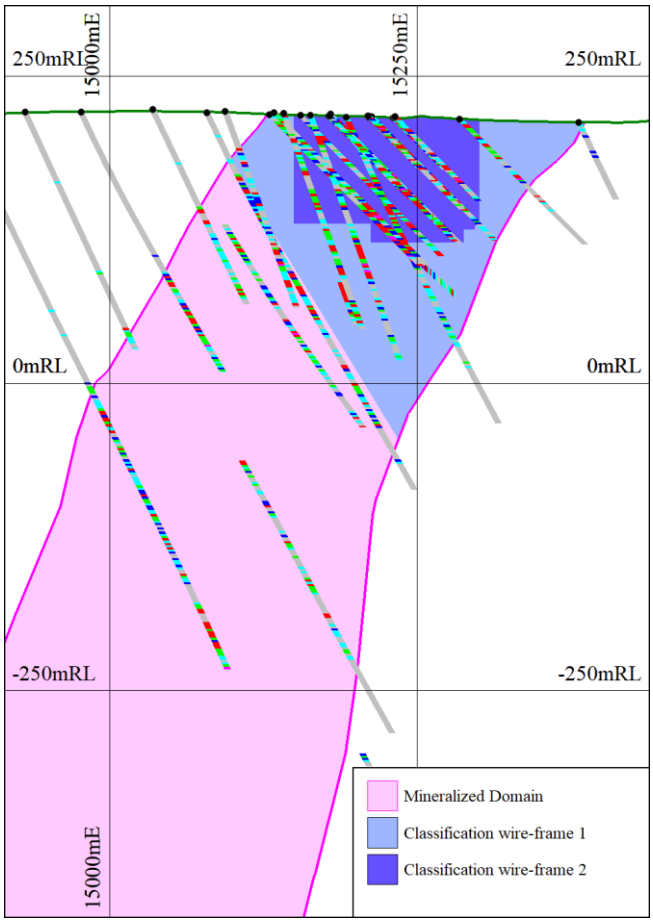


Figure 83: Classification wire-frame relative to mineralized domain, section at 52,945 mN

14.10 Pit shell constraint

To provide estimates with reasonable prospects for eventual economic extraction (RPEEE), 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 120. Also shown are the parameters used to constrain the previous resource modelling (MPR, 2018). Note that the costs used in Table 120 are not related to the current mine plan and financial analysis. They were used only to define the RPEEE envelope and the values selected were derived from current studies at the time.

Table 120: Resource pit shell optimization parameters

Description		Value used for previous model (MPR, 2018)	Value used for current model (MPR, 2019)
Gold price		US$1,500/oz	US$1,950/oz
NSR royalty		5.00%	5.00%
Pit slope angle		45º	50º
Average ore and waste mining cost		US$2.88/t	US$3.87/t
Average mill recovery	Oxide	90.0%	90.0%
	Transition and Fresh	86.0%	78.1%
Mill processing cost	Oxide	US$11.61/t	US$10.97/t
	Transition and Fresh	US$12.08/t	US$13.15/t

14.11 Model reviews

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, and showed no significant issues.

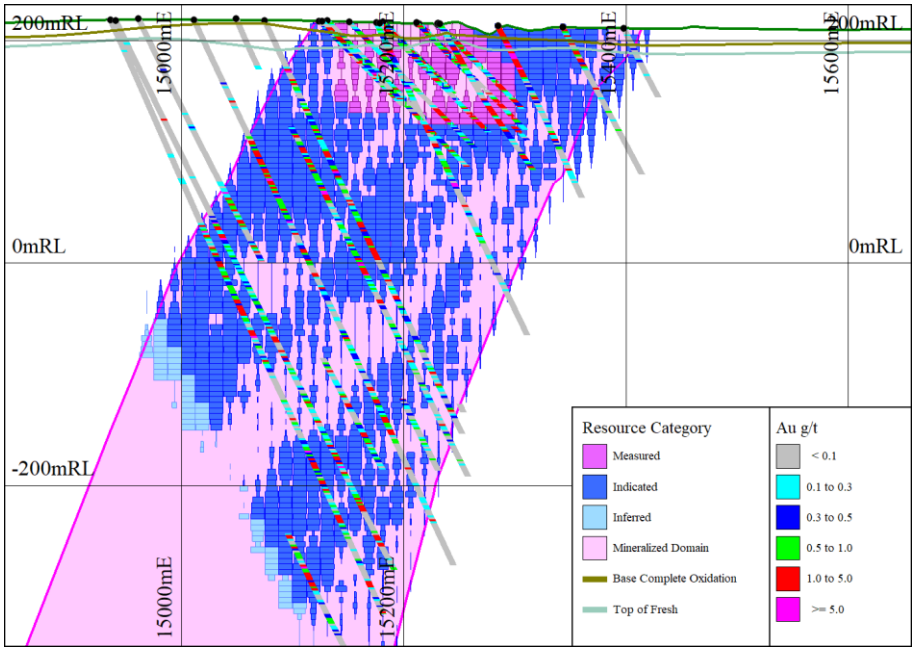
Figure 84 shows a representative cross section of the Namdini resource model. The resource model panels are scaled by the estimated proportion above 0.5 g/t gold cut-off and coloured by resource category and the average estimated gold grade at 0.5 g/t cut-off relative to resource domains, with holes traces coloured by 2 m composited gold grades.

It should be noted that when viewing the vertical sections through the resource model there are situations where the model blocks appear to be uncorrelated to the mineralized intercepts in the neighbouring drill holes. This is because of the way the resource model blocks have been presented. The model blocks plotted are only those that contain an estimated resource above 0.5 g/t cut-off and the proportion above cut-off has been used to scale the east and north dimensions of the model block for presentation purposes. The scaling occurs about the model block centroid co-ordinate and therefore introduces the apparent mismatch between data and the resource model blocks.

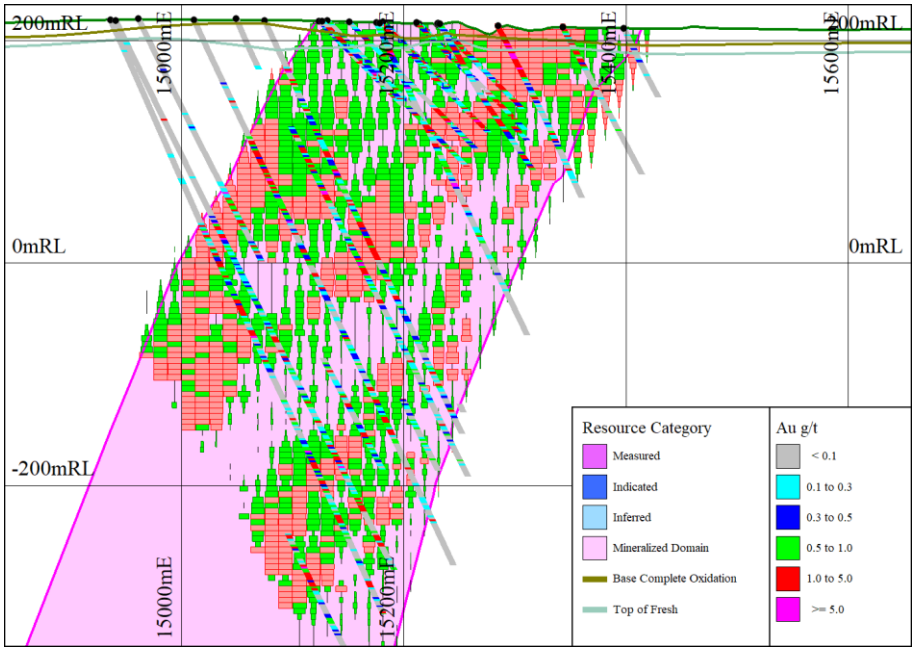
The swath plots in Figure 85 compare average estimated panel grades for Indicated Resources and average composite grades by easting, northing and elevation. The average composite grades include an upper cut of 50 g/t which is consistent to the upper limit used for generating the MIK domain statistics for Domain 2 and reduces the impact of a small number of outlier composite grades (maximum 242.05 g/t Au).

The plots in Figure 85 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 with the exception of areas of variably spaced or limited sampling. There are minor local deviations between the model and composite trends seen on the plots due to the following features:

- The use of an octant search strategy in the MIK estimation has a de-clustering effect on the estimates.
- The data used in the estimation of the MIK block grades are coming from a greater volume than the vertical or horizontal slices being compared which are consistent with model panel dimensions.
- Areas of variable spacing, with drilling preferentially clustered in higher, or lower-grade mineralization causes apparent inconsistencies between average composite and model grades as presented in the swath plots.



Resource categories



Gold grade at 0.5 g/t cut-off grade

53,300 mN blocks scaled to 0.5 g/t Au cut-off

Figure 84: Block model coloured by Resource categories and gold grade

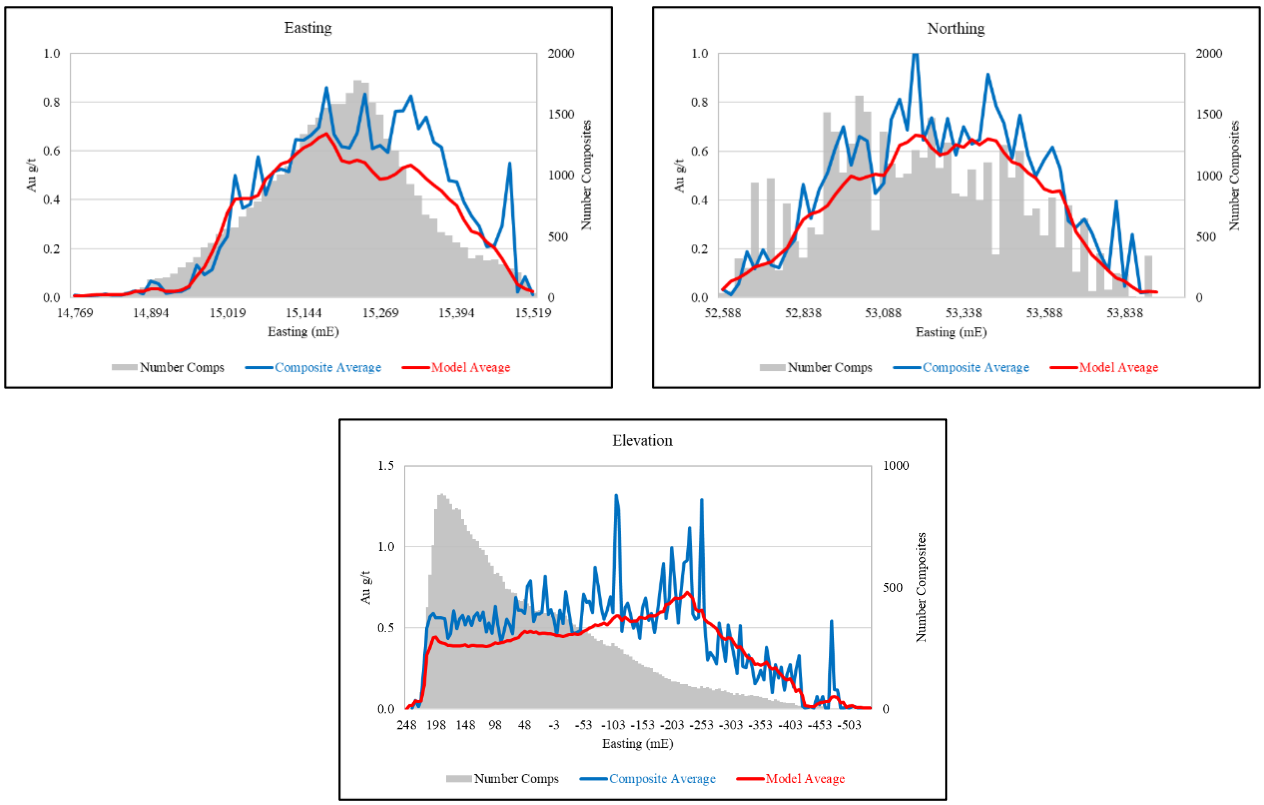


Figure 85: Average estimated panel grades versus composite grades

14.11.2 Comparison with grade control estimates

Cardinal reported on 317 trial RC grade control holes (Cardinal, 2017) at a spacing of 10 m east-west by 15 m north-south, drilled to a depth of around 40 m vertical, in the central-northern parts of the deposit which were not included in the resource estimation dataset. Comparison with estimates from a conditional simulation grade control model constructed from these data provides an independent check of the resource model estimates.

Table 121 compares estimates from the grade control model with the current 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.

Table 121 demonstrates that estimates from the grade control model, closely match resource model estimates for the area, that was drilled at grade control spacing, providing an independent check of the resource estimate.

Table 121: Grade control versus resource model estimates

Cut-off grade	Resource			Grade Control			Difference		
Au g/t	Mt	Au g/t	Au koz	Mt	Au g/t	Au koz	Mt	Au g/t	Au koz
0.3	4.12	1.40	185	4.12	1.41	187	0%	1%	1%
0.5	3.50	1.58	178	3.50	1.59	179	0%	1%	1%
0.7	3.00	1.75	169	2.94	1.78	168	-2%	2%	-1%

14.12 Mineral Resource estimate

14.12.1 Estimates at a range of cut-off grades

Table 122 shows the Mineral Resource Estimate for Namdini at a range of cut-off grades. The estimates are rounded to reflect their precision, and may include rounding errors.

The Mineral Resources are reported within the resource pit shell and extend from natural surface to the bit base at around 580 m depth.

Measured Mineral Resources extend to a maximum depth of approximately 100 m. Although Indicated Mineral Resources extend to the pit base, around 90% are at less than 400 m depth.

Most of the resources at comparatively shallow depths are classified as Measured or Indicated, with comparatively few Inferred Mineral Resources from depths of less than 250 m. Around 90% of Inferred resources are from depths of between 250 and 500 m (Figure 86).

The estimates make no allowance for depletion by currently active artisanal mining, which does not significantly impact the reported estimates. The area targeted by the artisanal miners covers a very small portion within the area of the Starter Pit. All digging activities were by hand, only within the Oxide zone.

Oxidized and Transition material types each host around 2.5% of the combined Measured and Indicated resources, with the remainder (95%) from Fresh rock. Around 0.3% and 0.1% of Inferred resources are hosted by the Oxide and Transition material respectively with the remainder (99.5%) from Fresh rock.

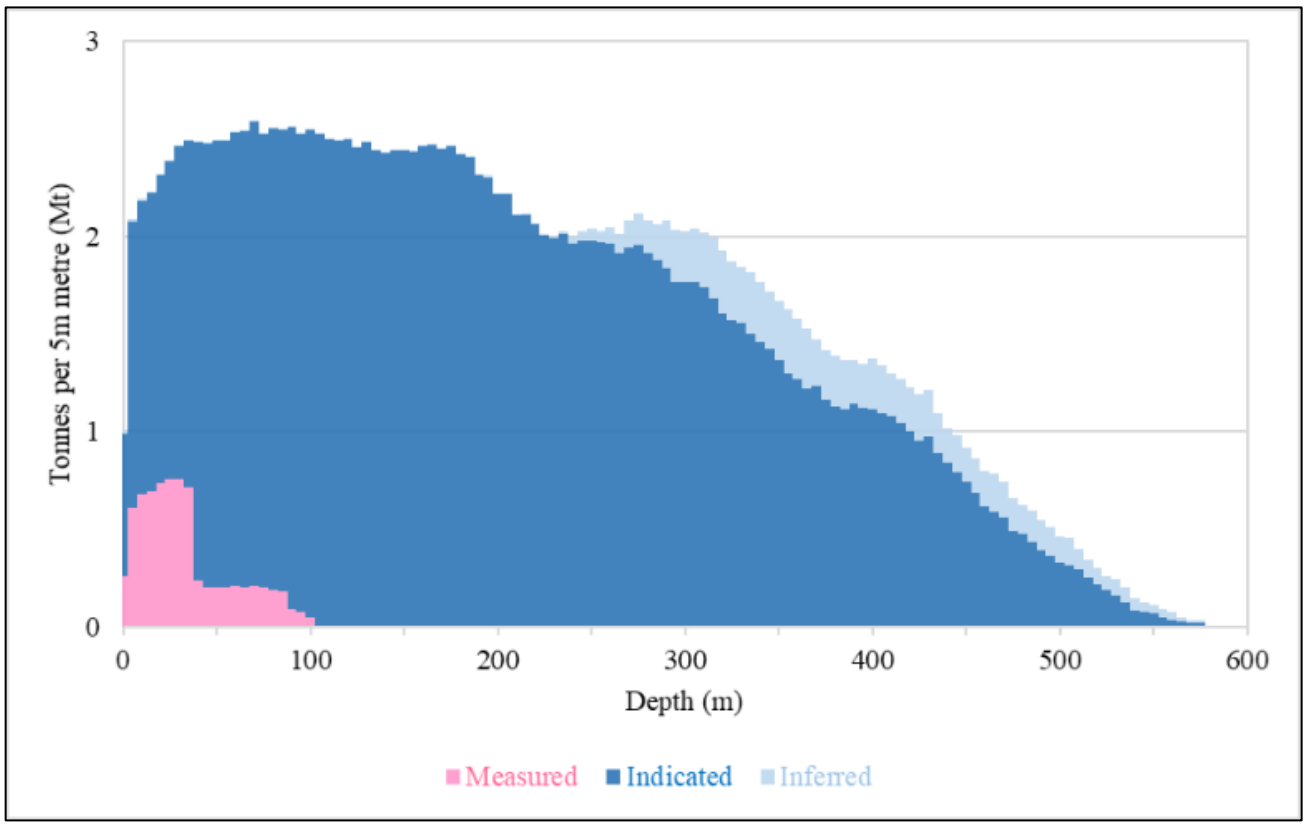


Figure 86: Mineral Resource estimates at 0.5 g/t cut-off grade by depth

Table 122: March 2019 Namdini Mineral Resource estimates at various cut-off grades

Cut-off (Au g/t)	Tonnes (Mt)	Grade (Au g/t)	Metal (Au Moz)
Measured Mineral Resources			
0.2	10.4	1.05	0.35
0.3	9.6	1.11	0.34
0.4	8.5	1.20	0.33
0.5	**7.5**	**1.31**	**0.32**
0.6	6.5	1.42	0.30
0.7	5.7	1.53	0.28
0.8	5.0	1.64	0.26
0.9	4.5	1.74	0.25
1.0	4.0	1.84	0.24

Cut-off (Au g/t)	Tonnes (Mt)	Grade (Au g/t)	Metal (Au Moz)
Indicated Mineral Resources			
0.2	262	0.85	7.16
0.3	234	0.93	7.00
0.4	204	1.01	6.62
0.5	**174**	**1.11**	**6.21**
0.6	147	1.21	5.72
0.7	123	1.32	5.22
0.8	103	1.43	4.74
0.9	86	1.55	4.29
1.0	72	1.66	3.84
Combined Measured and Indicated Mineral Resources			
0.2	272	0.86	7.51
0.3	244	0.94	7.34
0.4	213	1.02	6.95
0.5	**182**	**1.12**	**6.53**
0.6	154	1.22	6.02
0.7	129	1.33	5.50
0.8	108	1.44	5.00
0.9	91	1.56	4.54
1.0	76	1.67	4.08
Inferred Mineral Resources			
0.2	19	0.9	0.5
0.3	17	1.0	0.5
0.4	15	1.1	0.5
0.5	**12**	**1.2**	**0.5**
0.6	11	1.3	0.5
0.7	9	1.5	0.4
0.8	8	1.6	0.4
0.9	7	1.7	0.4
1.0	6	1.9	0.4

14.12.2 Public reporting of Mineral Resource at 0.5 g/t Au cut-off grade

The effective date of this Mineral Resource estimate is 3 April 2019 when it was first publicly reported (Cardinal, 2019a) in accordance with the JORC Code (2012) along with JORC Table 1. The Mineral Resource estimate, summarised in Table 123 and Table 124, reports the Mineral Resources by category and material type (weathering) above a 0.5 g/t Au cut-off grade. The classification categories of Measured, Indicated and Inferred Mineral Resources under the JORC Code (2012) are equivalent to the CIM categories of the same name (CIM, 2014).

Table 123: Measured and Indicated Mineral Resource estimate at 0.5 g/t Au cut-off, 3 April 2019

Mineral Resource category	Type	Tonnes (Mt)	Gold grade (g/t Au)	Contained gold (Moz)
Measured	Oxide	1.1	1.23	0.04
Measured	Fresh	6.4	1.33	0.27
Measured Resource	**Total**	7.5	1.31	0.32
Indicated	Oxide	3.3	1.08	0.11
Indicated	Fresh	171	1.11	6.10
Indicated Resource	**Total**	174	1.11	6.21
Measured and Indicated	Oxide	4.40	1.12	0.16
Measured and Indicated	Fresh	177	1.12	6.38
Measured and Indicated	**Total**	182	1.12	6.53

Table 124: Inferred Mineral Resource estimate at 0.5 g/t Au cut-off (Cardinal, 2019a), 3 April 2019

Mineral Resource category	Type	Tonnes (Mt)	Gold grade (g/t Au)	Contained gold (Moz)
Inferred	Oxide	0.04	1.0	0.001
Inferred	Fresh	12	1.2	0.46
Inferred Resource	**Total**	12	1.2	0.46

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).

Mineral Resource estimates follow the ASX listing rule LR 5.8.1 guidelines and a JORC Table 1 was provided with the ASX release.

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

14.13 Summary

The current estimates are based on a mineralized domain interpreted on the basis of composited gold grades. The mineralized 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 degrees and is interpreted to a constant elevation of -650 mRL, which represents an average depth of around 860 m, around 175 m below the base of drilling.

Namdini Mineral Resources were estimated by Multiple Kriging of 2 m down-hole composited gold grades from RC and diamond drilling. Estimated resources include a variance adjustment to give estimates of recoverable resources for selective mining dimensions of 5 m by 10 m by 2.5 m assuming high quality RC grade control drilling, and are reported within an optimal pit shell generated at a gold price of US$1,950/oz.

Resource model blocks have been classified as Measured, Indicated or Inferred by search pass and several wire-frames outlining more closely drilled portions of the mineralization:

- Estimated Measured Mineral Resources at a cut-off grade of 0.5 g/t are 7.5 Mt at 1.31 g/t for 0.32 Moz of gold
- Estimated Indicated Mineral Resources at a cut-off grade of 0.5 g/t are 174 Mt at 1.11 g/t for 6.21 Moz of gold
- Estimated Inferred Mineral Resources at a cut-off grade of 0.5 g/t are 12 Mt at 1.2 g/t for 0.5 Moz of gold.

Cardinal's exploration and resource drilling activities have defined a substantial Measured and Indicated Mineral Resource. Inferred Mineral Resources, which represent a small proportion of model estimates within the RPEEE pit shell, are generally at depths below 250 m.

14.14 Review of the Mineral Resource by Golder

As part of the mine planning studies leading to the Ore Reserve in this FS, Golder reviewed the Mineral Resource model developed by MPR (2019). This Section summarises a review by Golder (2019a) of the changes to the Mineral Resource based on the additional drilling and re-modelling.

14.14.1 Additional drilling data

An additional 49 drill holes (9,955 m) were added to the previous dataset (MPR, 2018) to produce the current model. The majority of new holes were away from the main deposit (Figure 87) including those drilled to test the deeper central part of the deposit and the northern limit.

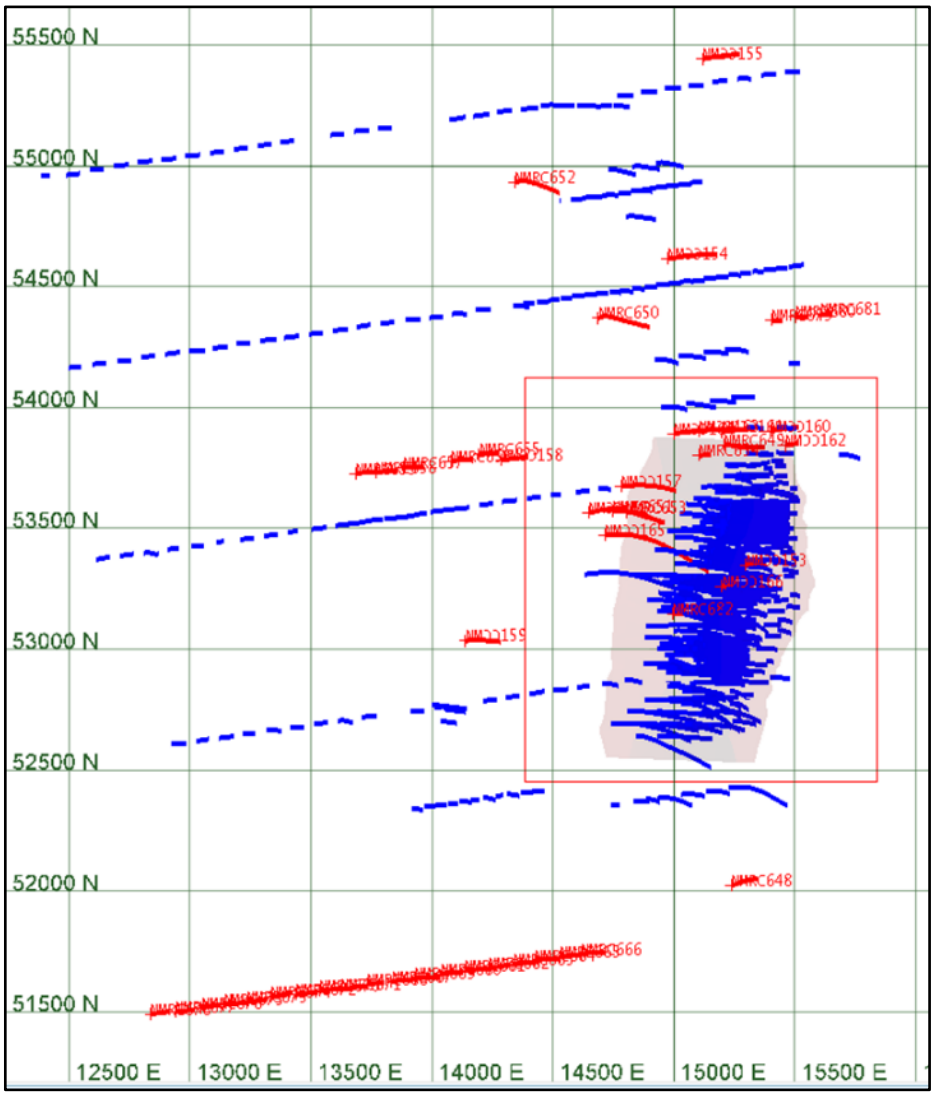


Figure 87: Plan showing drill hole samples in red since the PFS (Golder, 2018a)

14.14.2 Global model comparison

Cardinal supplied tabulations of the global grade and tonnage for the 2018 and 2019 models. The change between the models is summarised in Table 125. These figures are not constrained by a pit shell. The global differences are within the error of the estimates indicating that the additional drilling has not materially changed the deposit. Similar results were obtained at higher and lower cut-off grades.

The most significant change in the 2019 model is the change in classification of a part of the deposit to a higher resource confidence of Measured Resources as described further below.

Table 125: Comparison of the 2018 and 2019 global Mineral Resource at 0.5 g/t Au cut-off

Model (Unconstrained)	Measured			Indicated			Inferred		
	Tonnes (Mt)	Au (g/t)	Au (k oz)	Tonnes (Mt)	Au (g/t)	Au (k oz)	Tonnes (Mt)	Au (g/t)	Au (k oz)
2018			0	192	1.11	6,843	27	1.15	993
2019	7	1.31	315	169	1.10	6,606	27	1.13	995
Difference (%)	**-**	**-**	**-**	**-3%**	**-1%**	**-3%**	**0%**	**-2%**	**0%**

The estimation process did not materially change from that documented previously (MPR, 2018) and reviewed by Golder in the PFS (Golder, 2018b). Golder reviewed this methodology in detail for the PFS (Golder, 2018b) and found it acceptable. While Golder expressed reservations about the use of the MIK method to define recoverable tonnes and grades at the planned level of mining selectivity it accepted that the grade control drilling and modelling substantially validated the estimation methodology. This analysis has not been repeated for the FS.

Golder imported the supplied 2019 resource block model and verified the global unconstrained tonnes and grade supplied by Cardinal for the 0.5 g/t Au cut-off grade. The MIK values reported in Table 126 show the selective mining case (see Section 14.7). For comparison, the whole panel (e-type estimate) grade and tonnages were also reported by Golder. The whole panel estimates (12.5 m by 25 m by 5 m panels), yield 7% higher tonnes and 7% lower grade compared to the SMU recoverable case, reflecting the implied lower mining selectivity.

Note that these are global numbers and do not represent the reportable resource estimates which are further constrained by a resource pit shell.

Table 126: Comparison of 2019 MIK and e-type (whole panel) estimates above 0.5 g/t Au

Classification	MIK Tonnes (Mt)	MIK Grade (g/t Au)	e-type Tonnes (Mt)	e-type Grade (g/t Au)
Measured	7	1.31	8	1.24
Indicated	187	1.10	200	1.02
Inferred	27	1.13	29	1.06

14.14.3 Resources constrained by RPEEE shell

Use of an RPEEE shell is an acceptable standard approach used in industry for resource reporting purposes to ensure that the Mineral Resource is tested for ‘reasonable prospects of eventual economic extraction’. MPR used an optimistic gold price of US$1,950 /oz and a metal recovery factor of 78% for the 2019 RPEEE pit shell, compared to 86% for the 2018 RPEEE shell (see Table 120).

There is 15% of the 2019 modelled mineralization outside the RPEEE shell, which is excluded from the 2019 reported Mineral Resource, as shown in Table 127. Also shown is the corresponding RPEEE constrained resource from the 2018 resource estimate. The additional drilling has resulted in a negligible change to the overall RPEEE-constrained Mineral Resource, although some of the Inferred Mineral Resource has been converted to Indicated Mineral Resource, reflecting the increased confidence from the additional drilling.

Table 127: Comparison of constrained 2019 and 2018 Mineral Resources (0.5 g/t Au cut-off)

Classification	2019 Global		2019 RPEEE		2018 RPEEE	
	Tonnes (Mt)	Grade (g/t) Au	Tonnes (Mt)	Grade (g/t) Au	Tonnes (Mt)	Grade (g/t) Au
Measured	7	1.31	7	1.31		
Indicated	187	1.10	173	1.11	180	1.12
Inferred	27	1.13	12	1.22	13	1.25

14.14.4 Mineral Resource classification

The most significant change to the model has been the definition of Measured Resource material in two regions of the deposit, as illustrated by the areas in red in Figure 88. These areas correspond to the location of grade control drilling to the north of the deposit, drilled at 10 mE by 15 mN spacing, and an area of the model to the south that was drilled at an approximate spacing of 30 mE by 10 mN.

For the northern grade control area, the grade control holes were excluded from the grade interpolation process. Golder agreed that this is good practice to test and reconcile the grade estimation process based on the wider spaced exploration drilling in this area with an independent estimate based on only the grade control holes.

However, in Golder's view, to support the Measured classification, the closer spaced grade control drilling should be used in addition to the exploration holes in the grade interpolation process. This is to help achieve the local estimation accuracy implied by the Measured classification. The risk is somewhat mitigated by the use of the grade control data in an independent model that reportedly yielded similar tonnages and grade to the exploration-based grade estimate and has been used as the basis to justify the Measured classification.

For the southern area, the closer-spaced drilling was used in the grade estimation process and Golder considers it reasonable to apply a Measured classification in this area.

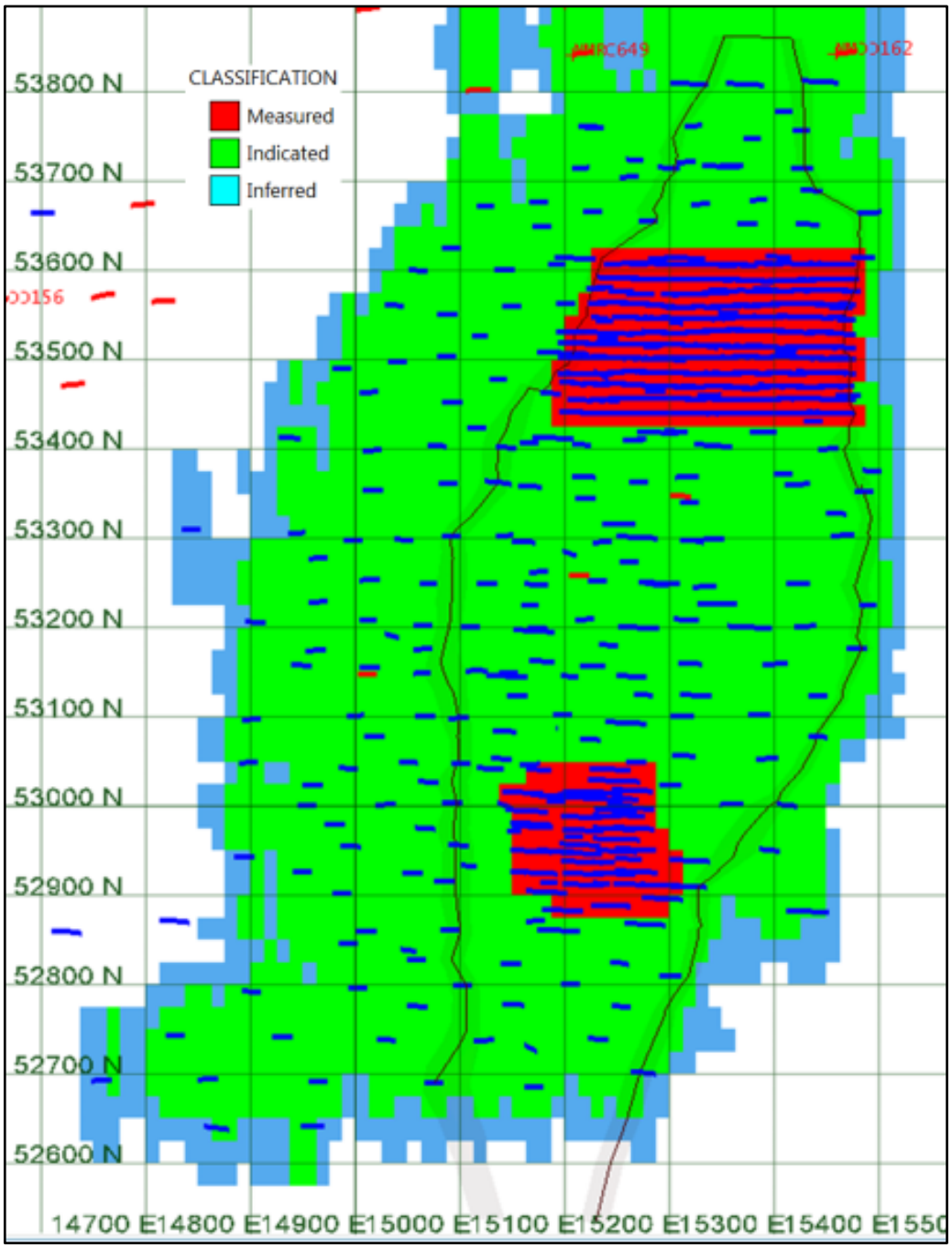


Figure 88: Bench 108 RL showing Mineral Resource Classification and drill hole traces

14.14.5 Local changes from new drilling

Changes in the resource estimate resulting from additional drilling were assessed using the change in gold grade and change in resource classification, as illustrated in Figure 89.

There was only limited additional drilling in the mineralized portion of the deposit. Deeper drilling in the vicinity of 53350N, shown in Figure 89, demonstrates that the new deeper hole (coloured red) has led to a change in the grade estimation in the vicinity of the new hole. Elsewhere, there are some minor grade changes near the mineralized domain boundary, likely due to slight changes in the interpretation and modelling. The new deeper hole shown in Figure 89 has resulted in additional Indicated and Inferred resources at depth. The new drill hole nearer the surface in Figure 89 has not materially changed the grade or classification.

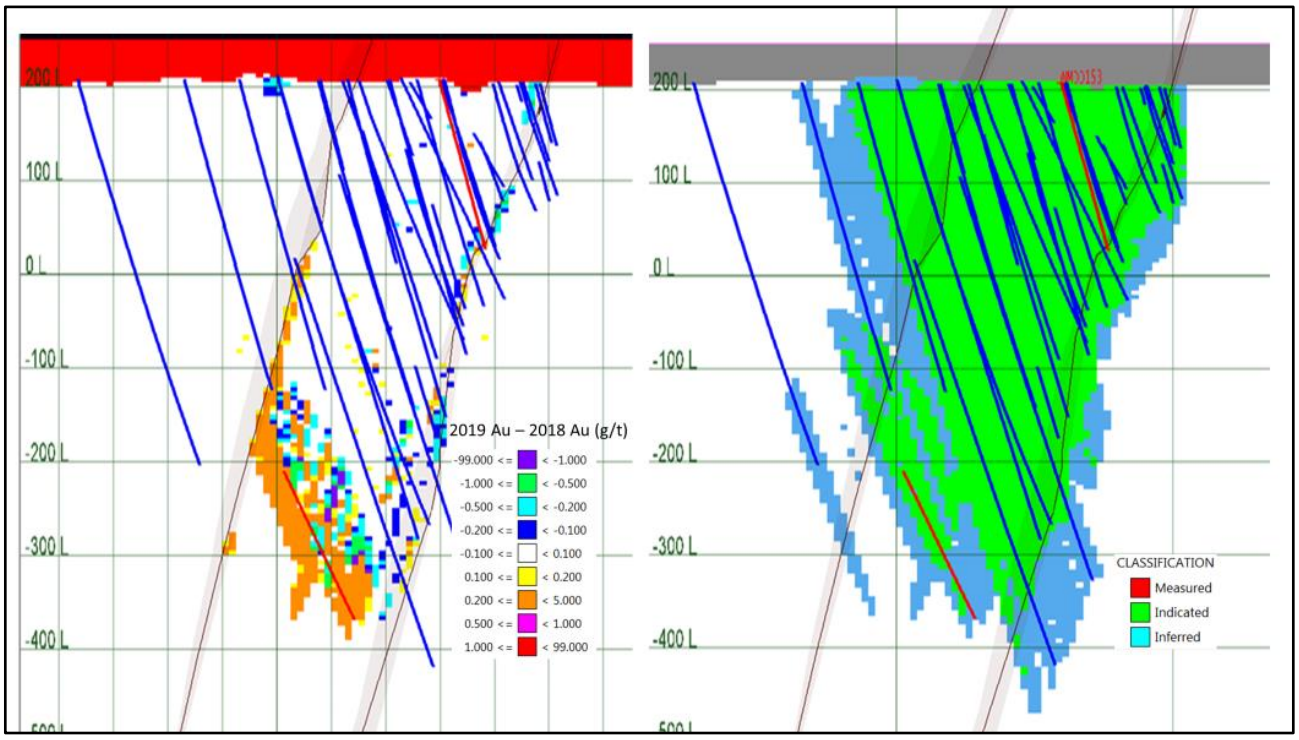


Figure 89: Change in gold grade (LHS) and Resource classification (RHS) on cross section 53350N

Additional drilling to the north of the deposit, shown in Figure 90, had very little impact on the gold grade in the vicinity but has increased the resource classification confidence in this area.

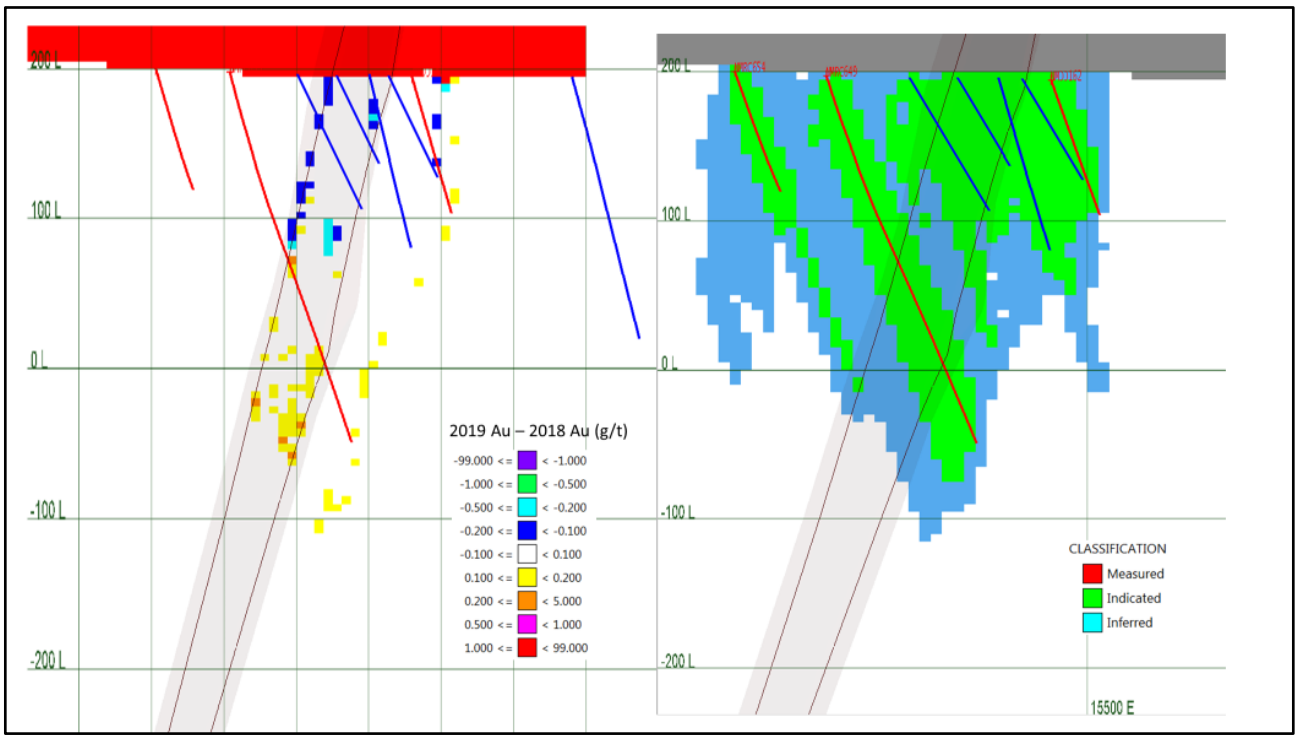


Figure 90: Change in gold grade (LHS) and Resource classification (RHS) on cross section 53825N

In Figure 90 isolated thin zones of Indicated Mineral Resource are seen to be supported by individual drill holes. These regions do not represent material proportions of the resource estimate.

14.14.6 Resource model review conclusions

Based on an external review, Golder concluded that the additional drilling has not materially changed the tonnes and grade in the Mineral Resource. The most significant change is to the reclassification of 7 Mt from Indicated to Measured Resource.

The 2019 resource estimate is supportable as an appropriate input to mining studies at the FS level and is a reasonable development from the previously reviewed 2017 and 2018 resource estimates.

While not material for this FS the following refinements should be considered for future resource estimation:

- Close-space grade control drilling, such as that in the northern region, can be used to refine the tonnes and grade estimates where they support the Measured Mineral Resource classification.
- Indicated Mineral Resource defined by isolated drill holes should be downgraded to Inferred.

14.15 Infill drilling test

In 2019 Cardinal completed a 3,640 m test infill drilling programme which comprised 30 RC and diamond holes infilling earlier drilling to a grid pattern of approximately 25 m (E) by 25 m (N), within the proposed Starter Pit. The drilling comprised three 25 m spaced traverses in an area of approximately 300 m (E) by 75 m (N) to approximately 140 m vertical (Figure 91). The drilling was inclined at around -65° to the east in the Namdini local grid, consistent with drilling undertaken for previous Mineral Resource models (Figure 92 to Figure 94). .

The close spaced infill drill pattern demonstrated continuity of the mineralized zones within the infill drilling area. The results provide additional confidence that the spatial distribution and tenor of gold within this test area are in line with the Mineral Resource expectations.

A test comparative block model generated from the combined drilling dataset was compared to the current Mineral Resource model (announced 3 April 2019) which, in the test area, was based upon approximately 50 m by 50 m spacing. The comparative block model was generated consistent with the 3 April 2019 model and was intended for comparative purposes only. Table 128 and Table 129 present the results of this work.

Table 128: Comparison between 3 April 2019 Resource Model and Infill Drilling Model by cut-off grade

Mineral Resource Model (April 2019)				Infill Drilling Model (June 2019)			Delta
Cut-off	Tonnes	Grade	Metal	Tonnes	Grade	Metal	Metal
(g/t Au)	(Mt)	(g/t Au)	(koz Au)	(Mt)	(g/t Au)	(koz Au)	(koz Au)
0.3	7.07	1.34	305	6.87	1.44	317	13
0.5	**6.05**	**1.50**	**291**	6.01	**1.58**	**306**	**15**
0.7	5.00	1.69	271	5.13	1.75	289	18
1.0	3.72	1.98	236	3.93	2.03	257	20

Note: The nominal reporting cut-off grade is 0.5 g/t Au

Table 129: Comparison between 3 April 2019 Mineral Resource Model and Infill Drilling Model

Model	Measured			Indicated			Measured + Indicated		
Estimate	Tonnes (Mt)	Grade (g/t Au)	Metal (koz Au)	Tonnes (Mt)	Grade (g/t Au)	Metal (koz Au)	Tonnes (Mt)	Grade (g/t Au)	Metal (koz Au)
Apr-2019 (50 × 50 m)	1.15	1.73	64	4.90	1.44	227	6.05	1.50	291
Jun-2019 (25 × 25 m)	6.01	1.58	306				6.01	1.58	306
% Diff							**-1%**	**+6%**	**+5%**

Notes: By Mineral Resource classification, reported above 0.5 g/t Au.

The 3 April 2019 Mineral Resource model has drill holes at approximately 50 m × 50 m grid spacing. The Infill Drilling increased the drill spacing to 25 m × 25 m over the selected test area.

The classification of Measured and Indicated Mineral Resources reflects the level of confidence in the geological continuity. The classification approach assigns estimates tested by generally 50 m by 50 m and closer spaced drilling to the Indicated category, with estimates for zones with more closely spaced drilling, approximately 25 m × 25 m or closer, classified as Measured Mineral Resources.

Mineral Resource classification was also informed by the Cardinal drilling database. The drilling, surveying, sampling, sample preparation, analytical techniques and database validation, are consistent with the drilling used for the Mineral Resource estimate.

No Inferred Mineral Resources were defined in the area tested.

Numbers are rounded to reflect the precision of the estimates.



Figure 91: Plan view of Namdini deposit showing infill drill hole locations and interpreted geology

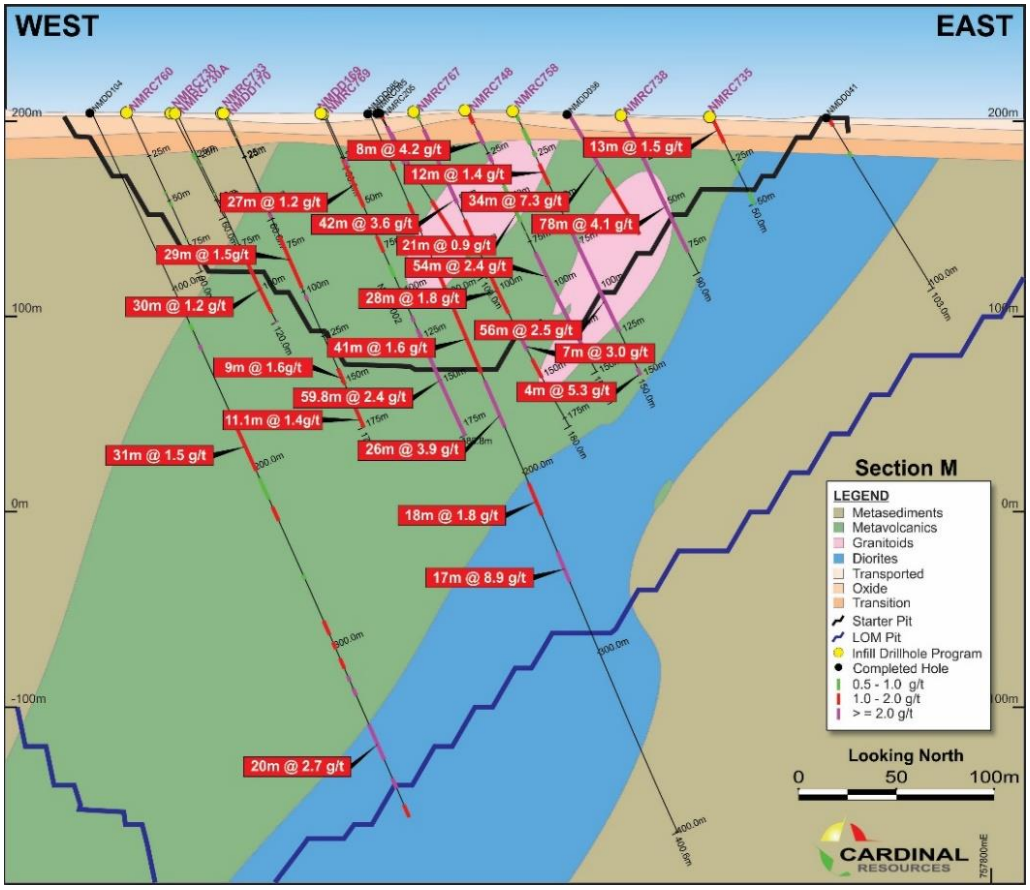


Figure 92: Cross section showing down hole mineralized intersections of infill holes

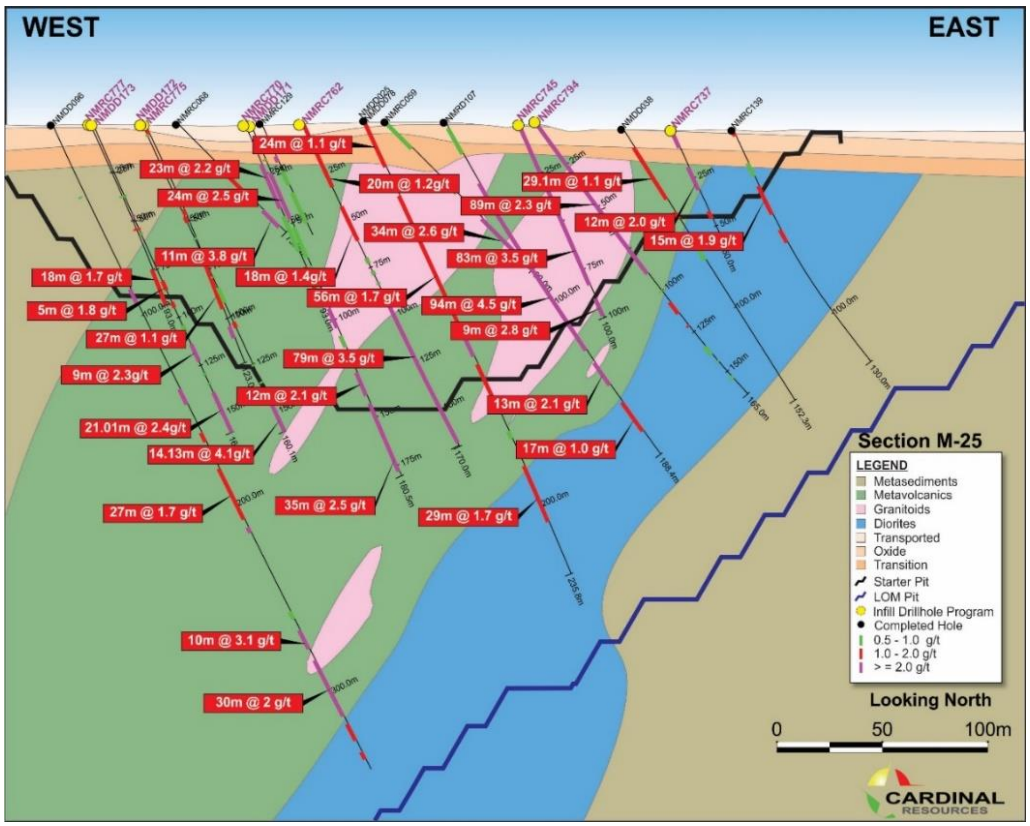


Figure 93: Cross section showing down hole mineralized intersections of infill holes

15.0 MINERAL RESERVE ESTIMATES

The information in this Section is derived from Golder (2019b). It is presented in a similar format to that in the PFS (Golder, 2018b) which is a comprehensive report provided on Sedar (https://www.sedar.com).

15.1 Introduction

The maiden Ore Reserve (see Section 6.6) for Namdini was announced on 19 September 2018 (Cardinal, 2018c) and documented in detail in the PFS (Golder, 2018b). In February 2019 Cardinal engaged Golder to estimate the updated Ore Reserve based on the latest available resource model.

Golder has 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 Proved Ore Reserve estimate is based on Measured Mineral Resources. The Probable Ore Reserve estimate is based on Indicated Mineral Resources. The Ore Reserve represents the economically mineable part of the Measured plus Indicated Mineral Resources. Mineral Resource estimates are reported inclusive of those Mineral Resources converted to Ore Reserves. No Inferred Mineral Resource has been included in the Ore Reserve.

This updated Ore Reserve estimate was released on 18 April 2019 (Cardinal, 2019b). It was established based on the outcomes of this present Feasibility Study prepared by Cardinal, Lycopodium and external consultants (the Authors). The Ore Reserve estimate includes Proved Ore Reserve based on that portion of the Mineral Resource now classified as Measured.

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.

15.2 Mining model

The Mineral Resource model produced by MPR (2019) discussed in Section 14.0 was used to develop a mining model. The mining model was subjected to pit optimization using FS mining and processing costs, processing recoveries, recommended geotechnical slope angles and a gross gold price of US$1,300/oz, 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 Project high-level financial model were developed on the basis of the mineral inventory contained within the open pit design. The mining model within the Final Pit is reported as the Ore Reserve.

Cardinal is considering contract mining, for which expressions of interest and indicative costs have been sourced. The formal tender process will commence once the decision to proceed is finalised. A comprehensive cost model has been developed from first principles to assist Cardinal in adjudicating the tenders. This will also assist the Cardinal board to evaluate an owner-based mining alternative to contractor-based mining.

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 three flitches in the ore within 10 m benches. The use of the 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.

A marginal cut-off grade ("**COG**") was established for the project using estimated mining costs provided in November 2018 and compared against similar projects in the region. The processing costs have been supplied by Lycopodium. Royalties, TC/RC and G&A costs have been provided by Cardinal. Table 130 documents the parameters used to define the COG

Table 130: Namdini Project marginal cut-off grade (COG) estimation

Item	Units	Value
Process plant	Throughput capacity	9.5 Mtpa
Gold price	$/oz	1,300.00
TC/RC	$/oz	1.10
Royalties (5%)	$/oz	65.00
Gold price after royalties and TC/RC	$/oz	1,233.90
Net gold price	$/g Au	39.67
Process plant metallurgical recovery	LOM average[1]	82%
Process costs	/t	14.30
Stockpile reclaim costs	/t	1.50
Total process costs	/t	15.80
Marginal COG (used 0.5 g/t Au for reporting)	g/t Au	0.49

Note: Source: 2019-03-10_DFS_Golder_Mine_Inputs_RevG.xlsx

[1] Excludes Oxide ore.

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 Summary of the Ore Reserve estimate

Using a cut-off grade of 0.5 g/t Au the Measured and Indicated Mineral Resources within the final stage mine design indicate a total Proved and Probable Ore Reserves estimate of 139 Mt at 1.13 g/t Au for a contained 5.05 M ounces. Table 131 shows the breakdown of the estimate Ore Reserves for the Namdini Project at April 2019. Ore Reserves stated here are in accordance with JORC 2012 reporting standards after the application of appropriate Modifying Factors.

Table 131: Ore Reserve estimate for the Namdini Project – April 2019

Class	Type	Ore Tonnes (Fully Diluted) (Mt)	Diluted Contained Ounces (koz)	Grade (Au g/t)
Proved	Oxide	1.03	40.0	1.21
Proved	Fresh	6.35	270.9	1.33
Proved Reserves	**Total**	**7.38**	**310.8**	**1.31**
Probable	Oxide	3.02	104.6	1.08
Probable	Fresh	128.19	4,638.3	1.13
Probable	**Total**	**131.21**	**4,743.0**	**1.12**
Proved and Probable	**Oxide**	**4.04**	**144.6**	**1.11**
Proved and Probable	**Fresh**	**134.54**	**4,909.2**	**1.13**
Proved and Probable	**Total**	**138.59**	**5,053.8**	**1.13**

Notes: Source: namdini_2019_03_28_pb04_v7_3B_scheduler_rev_a.xlsx

The Ore Reserve conforms with and uses the JORC 2012 Code definitions

The Ore Reserve is evaluated using a gold price of US$1,300/oz

The Ore Reserve is 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 that has been 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.

For the purposes of the mining study, it was assumed that a ‘dry pit’ will be operated with sufficient dewatering and control for the geotechnical design of 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 pit optimization used FS level mining and processing costs, processing recoveries, recommended geotechnical slope angles and a gold price of US$1,300/oz 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 Ore Reserve. All pit designs have been carried out in accordance with the recommended mining geotechnical slope parameters for the Namdini Project.

Based on the information presented in this FS, the Ore Reserve estimation process has converted approximately 80% of the combined Measured and Indicated Mineral Resource to Proved and Probable Ore Reserve.

The proposed mine plan is technically achievable. All technical proposals made for the operational phase involve the application of conventional technology that is widely used in the gold industry. Financial modelling completed as part of the FS 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.

16.0 MINING METHODS

The information in this Section is derived from Golder (2019b). It is presented in a similar format to that in the PFS (Golder, 2018b) which is a comprehensive report provided on Sedar (https://www.sedar.com).

16.1 Approach

For the purposes of the mining study, the mine design and costing assessment was based on the Mineral Resource block model for the Namdini Project supplied by MPR as discussed in Section 14.0. Measured 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 used as provided in the Local grid system developed by Cardinal.

The Local grid transformation comprises an 8° rotation from UTM coordinates (Table 132), with no elevation change. The transformation rotates the obliquely (WGS grid) trending drill lines to east-west (Local grid).

Table 132: Transformation of UTM Grid (WGS 84 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

Optimization modelling and pit designs were done on the geology model supplied in Local grid format and then the designs were converted to UTM grid. The UTM (WGS 84 Zone 30P) coordinate system was used for final mine planning and design to ensure pit layouts, waste rock dump ("**WRD**") locations and roads were in the survey system used for the plant layout and all project infrastructure (Section 18.0).

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 from top down of the following material types:

- Oxide zone, comprising:
  - strongly oxidized upper weathered zone
  - moderately oxidized zone with total oxidization of sulphides extending to approximately 10 m average depth from surface
- Transition zone with partial oxidization of sulphides extending to approximately 18 m average depth from surface
- 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 and is of limited thickness throughout the remainder of the planned mining area. Figure 94 shows the Oxide, Transition and Fresh zones within the planned Final Pit extent.

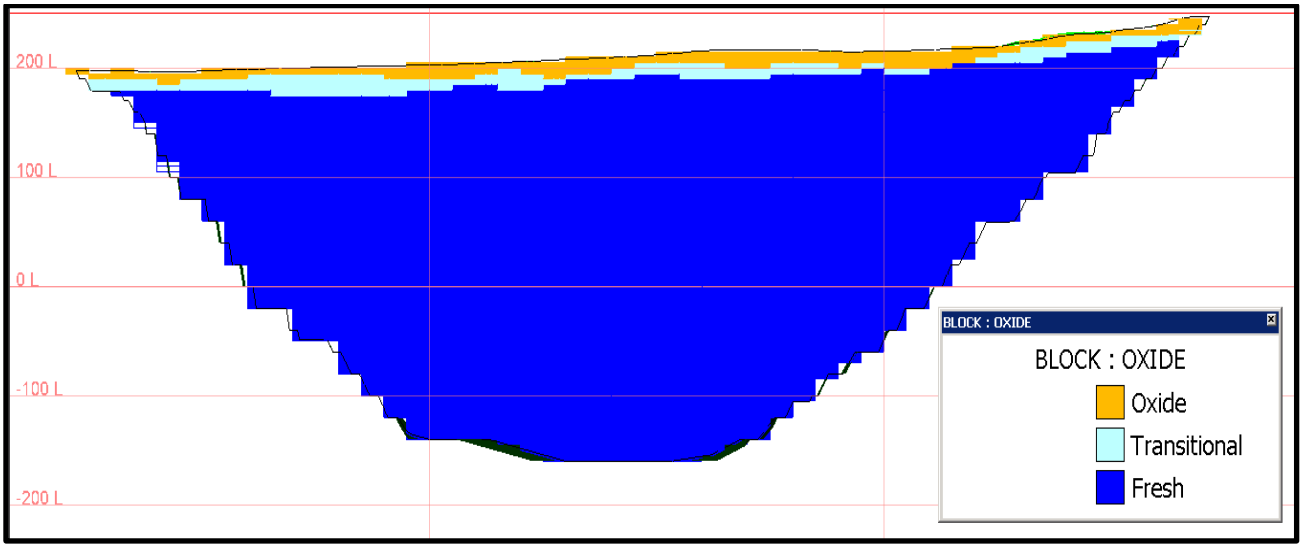


Figure 94: Oxide, Transition and Fresh material within the Final Pit – April 2019

The main orebody consists of a combination of granite, metavolcanic and diorite hosted within metasediments. A cross section from west to east through the pit (Figure 95) shows the main ore zone dipping within the metasediment regions.

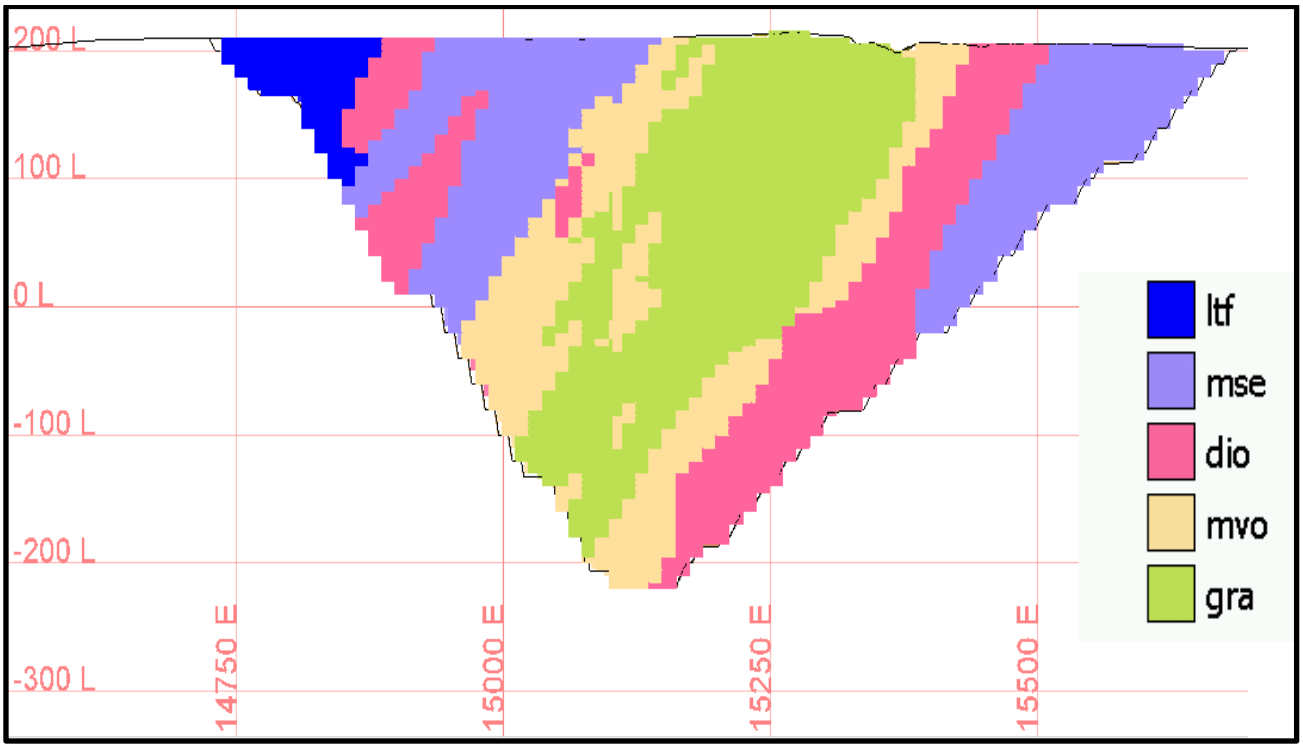


Figure 95: Main lithologies within the Namdini pit

Mining will consist of a conventional hydraulic shovel operation typically utilising 400 t class excavators in a combination of face-shovel and back-hoe configuration and 134 t (Cat 785 class or similar) rigid body dump trucks hauling on the designed access roads. An auxiliary mining fleet of dozers, graders, water carts and utility vehicles will support the mining operation. The excavator back-hoe configuration will be used to allow selective mining of the ore zone in three flitches within the 10 m bench height. The face shovel configuration will be used for bulk mining of the waste on benches, where vertical selectivity is not required.

No consideration has been given to underground extensions of the operation in the FS. No underground extensions are contemplated at this time. Accordingly, any consideration or discussion of underground development is not relevant for the purposes of this Report.

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 30 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 design can be seen in Figure 96.

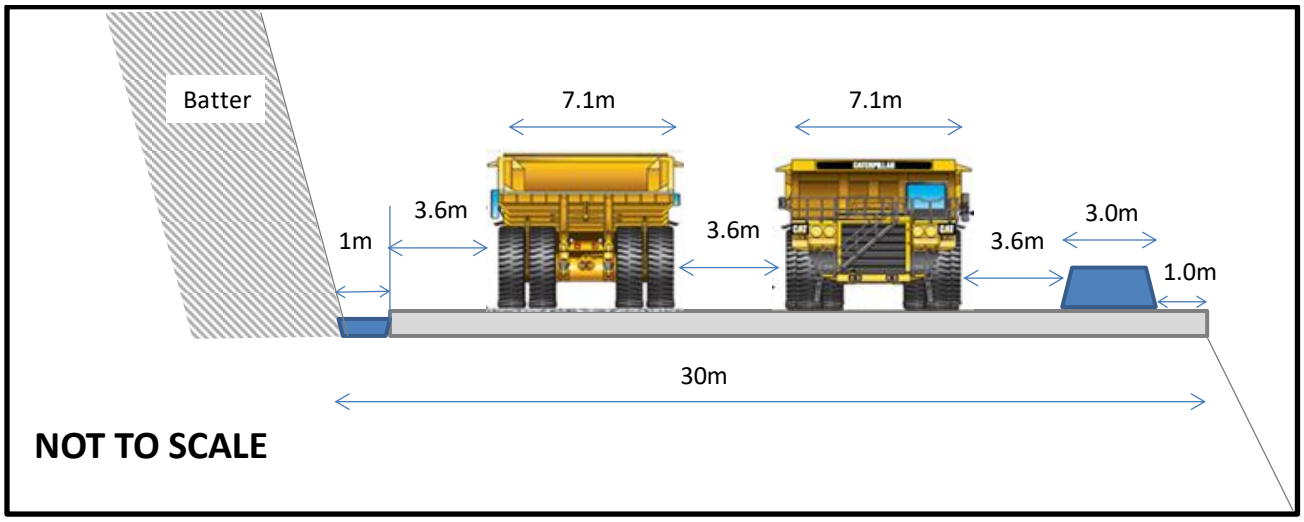


Figure 96: Road width design parameters for Cat 785 size haul trucks

16.3 Hydrogeology and mine groundwater

A comprehensive hydrogeology field program was completed and reported in the PFS (Golder, 2018b) based on field studies (Golder Africa, 2018b). Further recommendations on groundwater management were made to Cardinal for this FS in March 2019 (Golder, 2019f).

16.3.1 Pit inflows

The groundwater system in the mining area comprises two key units being an upper saprolite and saprock layer which overlies weathered to Fresh Metasediments and Metavolcanics. The transmissivities of these units, which are directly proportional to groundwater flow rates, are low.

There are two important issues:

- The volume of groundwater stored in the upper saprolite/saprock layer is high and will fluctuate more strongly due to seasonal cycles (dry season to wet season) than groundwater volumes in the deeper hydrogeological units. These materials can be prone to failure in pits as a result of these seasonal cycles. Flows into the pit from this unit will be from benches that are located at or immediately below the base of this unit. Improved drainage of this water is recommended.
- There are a large number of lineament structures (i.e. dykes, faults or fractures) that have been mapped that intersect the open pit footprint area.

Given the generally low permeability conditions, it was not possible to undertake tests that would provide estimates of groundwater storage. Golder's experience in this region is that the saprolite layer may contain significant volumes of groundwater storage.

Using the conceptual hydrogeological model developed, as well as data obtained from field testing and groundwater modelling (including analytical estimates), groundwater inflows of between 33 and 51 L/s are estimated, excluding elevated inflows that may arise from time-to-time when intersecting the above, unmapped lineaments. Provision of continual sump pumping arrangements has been catered for in the mining tender requirements.

Ongoing monitoring of near pit groundwater levels and pit pumping rates will be required to validate the above estimates and the dewatering strategy will necessarily need to be primarily focussed on in-pit dewatering from sumps. However, due to the size of the pit and the nature of the geological structures present, additional targets may be required at various times to provide adequate dewatering of the pit. These may consist of pit perimeter dewatering wells or even inclined boreholes within the pit along contact zones.

Groundwater level monitoring data collected from wells installed in and around the pit area, should be used to undertake further validation of the groundwater model.

16.3.2 Pit dewatering and wall depressurisation

Saprolite acts as a sponge which stores large amounts of groundwater, but only releases it slowly under a typical hydraulic gradient that would develop during mining. The hydraulic conductivity distribution range of the saprolite typically plots at the lower end of the hydraulic conductivity scale and therefore this unit does not favour an active dewatering system. Dewatering can be achieved by means of passive sub-horizontal drains or low-flow ejector pumping systems.

Golder has estimated groundwater inflow rates and proposed appropriate drain holes and monitoring, based on the results of pit slope Slide™ modelling.

Based on recommendations in Golder (2019f), the following strategy will be adopted:

- Following mining commencement, visual monitoring of the seepage from the saprolites will be done over the initial 3 to 6 months. In the event that seepage persists after 6 months and/or the crest of the saprolite benches show evidence of instability, then installation of sub-horizontal drains will be considered.
- Prior to the commencement of mining, vertical Vibrating Wire Transducers will be installed into the hanging wall Metasediments and Metavolcanics, to monitor groundwater (pore) pressures before and during mining.
- If the pore pressures remain high in the hanging walls during mining, and there is evidence of slope instability, then it is advised that sub-horizontal drain holes are installed to relieve these pressures, especially within the more permeable Fresh Bedrock zones.

Specifications were supplied by Golder (2019f) for subhorizontal drain holes with an option to install a pressure gauge and a shutoff /flow control value on the outside of the drain pipe. The potential overall Nandini open pit depressurisation system is illustrated in Figure 97. Specific recommendations for the western, eastern, northern and southern walls of the Namdini pit were based on Slide modelling. All pit slope geotechnical guidance has been based on the achievement of a dry pit for mining purposes.

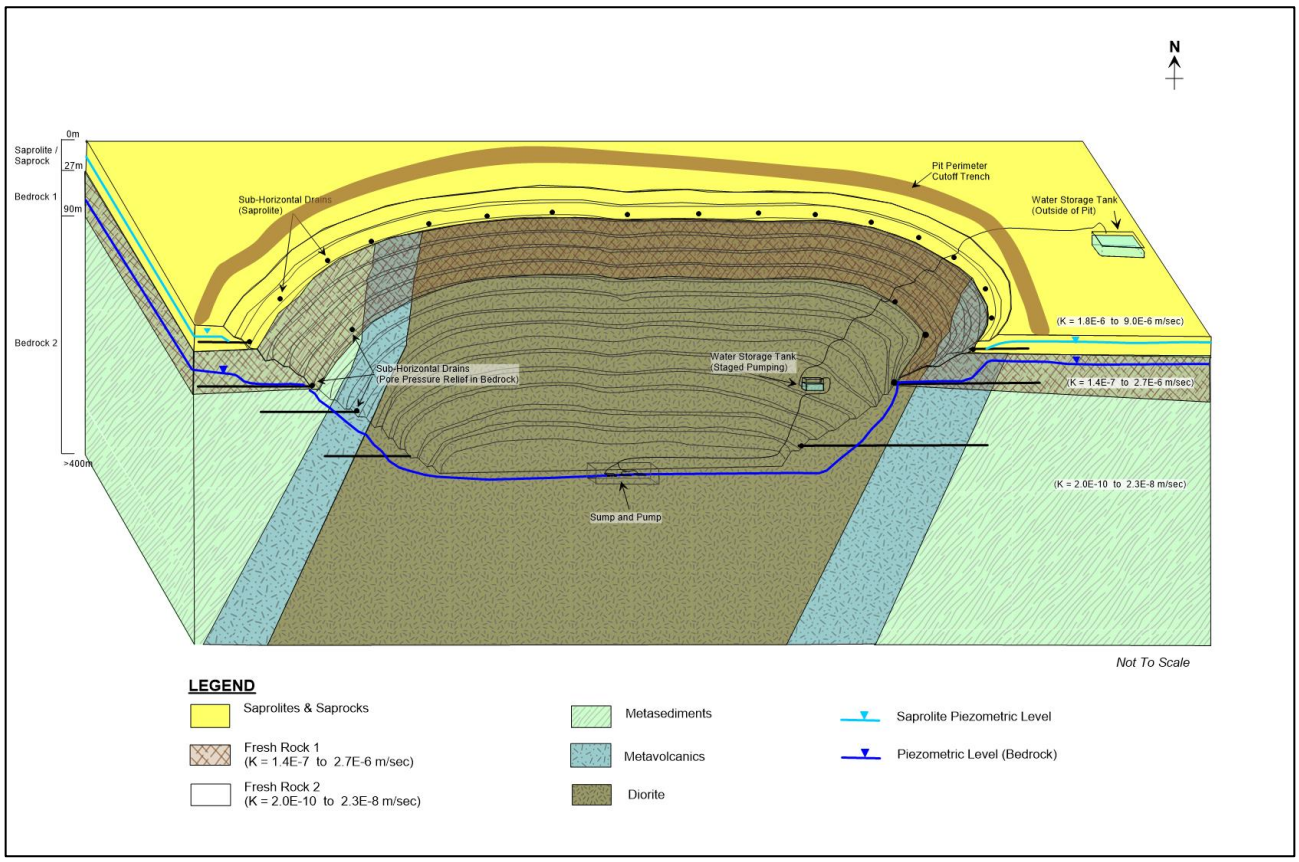


Figure 97: Overview of the potential depressurisation requirements for the Nandini open pit

16.3.3 Mine water management

For the PFS (Golder, 2018b), it was noted from studies done by (Golder Africa, 2018b) that:

- Water with different water qualities should be separated (i.e. clean water and dirty water streams), to allow optimization of reusing water across the mine site
- A pit perimeter cut-off trench be installed around the open pit, to intercept any rainfall run-off before entering the open pit. It was recommended that the cut-off trench be installed ~15 m away from the crest of the top bench, to avoid compromising the pit slope stability
- Water entering the open pit (i.e. direct rainfall recharge and groundwater inflow) should be considered as dirty water and should be channel to the in-pit sump
- All pit perimeter and inter-bench trenches be lined to avoid seepage and re-saturation of the pit slopes
- Clean water collected by the pit-perimeter trenches be released back into the environment, while the dirty water collected in the sump be either re-used for dust suppression and process water make up
- The in-pit dewatering system be designed to deal with a 1:20 year storm event, which is based on a receiving 103 mm rainfall per day, lasting for 7 consecutive days.

In a recent update to the hydrogeology and groundwater studies Golder (2019f) recommended:

> “To the extent possible, all clean water collected from the pit perimeter trenches should be discharged to downstream areas via lined channels for a minimum distance of 250 m from the pit crest, to minimise the risk of raising groundwater pressures in the shallow saprolite units near the pit wall (crest) areas.

The in-pit dewatering system should be designed to accommodate a 1:20 year storm event, with a specified sump and pumping system suggested. A practical approach to in-pit dewatering could be to install a smaller "operational" sump and use the pit floor and bottom bench as a larger 'emergency' sump."

16.4 Rock mechanics and slope angles

For the mine design, Golder Ghana (2018) carried out a study of existing geotechnical information, reviewed information on Mineral Resource estimates, conducted a detailed pit geotechnical drilling campaign, 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 and data and reports made available by Cardinal Resources.

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

For the final wall, bench stacks will consist of four double-benches (i.e. 80 m high) in Fresh material. Geotechnical criteria have allowed up to 120m, however the placement of haul ramps has resulted in a reduced total stack-height throughout most of the pit design. These will be separated by either 25 m wide catch-berms or ramps. Some flexibility in height is used where a ramp is close to a catch berm.

16.4.1 Background

The potential for steepening the mine design compared to the slopes used in the PFS was evaluated by Golder (2019d). The pit design criteria are for a dry pit.

For open pit mine design, the inter-ramp angle ("**IRA**") is the toe-to-toe slope angle (Figure 98). It can be achieved in a variety of ways, depending on bench height ("**BH**"), bench face angle ("**BFA**"), and catch bench or berm width ("**BW**"). Different combinations of these can be used to develop the same inter-ramp angle.

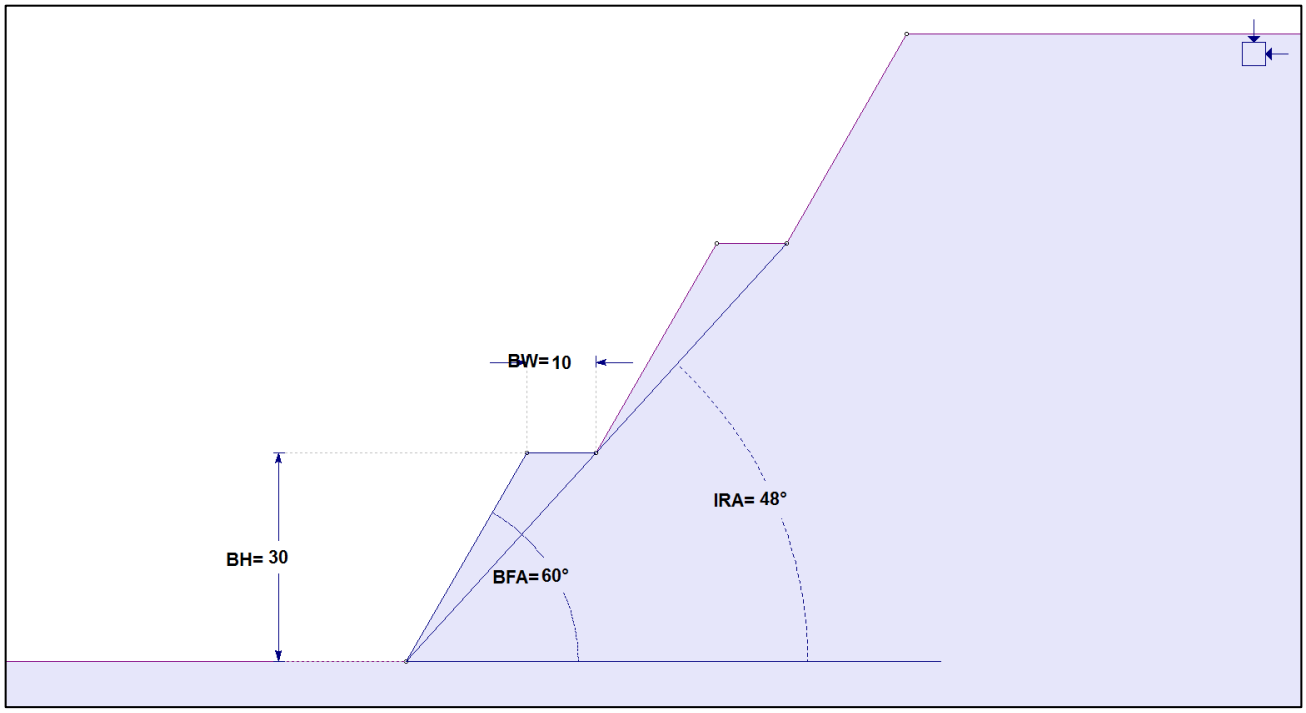


Figure 98: Schematic of open pit mine design components in cross section of the pit wall

16.4.2 Namdini rock types

For the generally strong rock types encountered at Namdini, the BFA is dictated primarily by the structural fabric and blasting (which can be optimized to minimise damage to the walls). The BFA for a rock type varies around the pit based on the relative orientation of the wall to the orientations of the structures in the rock mass; the relative orientations lead to variations in the failure mechanisms that can form in different wall sectors.

When footwall slope angles are steeper than the dips of persistent structures such as foliations (see Figure 99), they result in “daylighted” planar wedges. Due to the low natural strength of discontinuities and blast damage that further weakens this strength (by reducing cohesion), such wedges have significant probability of failure.

Planar failure mechanisms dominate in pit footwall sectors where walls strike sub-parallel to dominant structures. In the case of the Namdini pit, the critical structural control for the footwall slopes are foliations in the metasediments that dip at an average angle of 60° into the pit. In the diorites, the mean dip of the dominant orientations is about 55°. The structures in the diorite are not expected to be as persistent as the foliations in the metasediments.

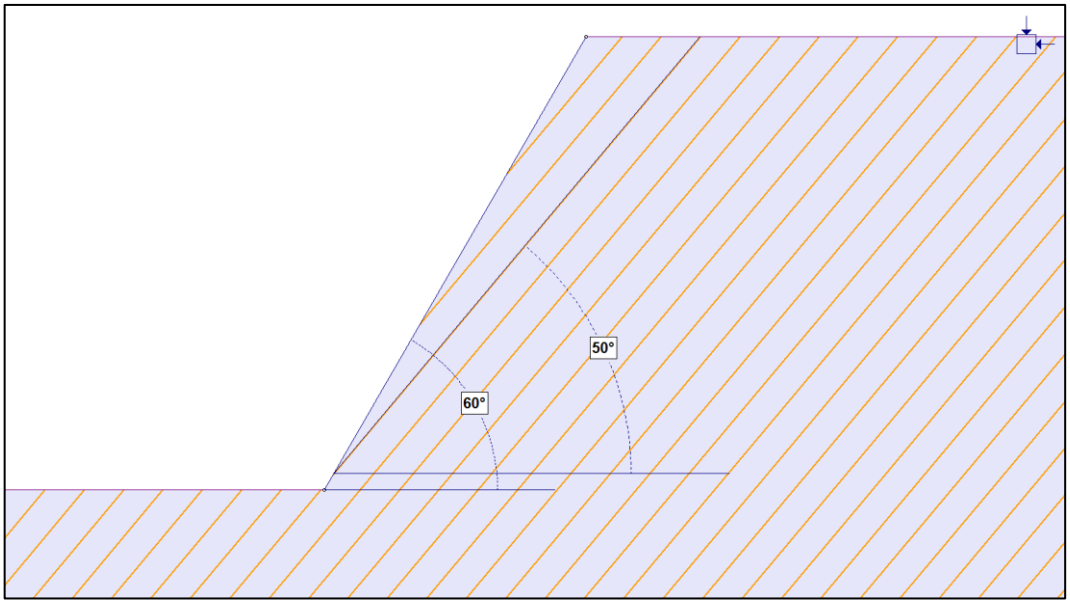


Figure 99: Daylighting of persistent structures into footwall bench faces

16.4.3 Implications for wall-steepening

The widths of catch berms are typically bracketed by empirical guidelines related to bench height. Using a vertical bench separation in the hard rock of 10 m, and double benching, gives bench designs for the footwall (Sector 5) shown in Table 133. These designs for Fresh rock are predicated upon the use of good drilling and blasting practices (such as trim blasting or pre-split blasting) and scaling operations to mitigate the risk of rock falls.

Table 133: Bench designs for footwall of Namdini Pit

Rock Mass Domain	Bench Height (m)	Bench Face Angle (°)	Spill Berm Width (m)	Limiting Bench Stack Height (m)	Inter-Ramp Angle (°)
Fr FWMSE	20	60	9	120	44.2
Fr DIO	20	60	9	120	44.2

Since a significant percentage of the orientations in the metasediments (MSE) dip flatter than 60° and would daylight into bench faces, the current design can be already considered to be sufficiently aggressive.

16.4.4 Modelling of footwall slopes and factors of safety

Using the currently available strength data, the performance of two different bench face angles, 60° and 65°, were analysed for stability at the bench scale and overall slope scale. In the analysed models 20 m bench heights, 9 m berm widths, and persistent discontinuities dipping at 60° were applied.

The assumption of discontinuities (foliation) remaining planar and extending across the entire height of slopes, particularly at the overall slope scale, may be quite conservative. However, given the present lack of data on the persistence of foliations, this conservatism still helps gauge the risks and merits of steepening of the slopes. It was assumed that the discontinuities had a spacing of 5 m, which was considered sufficient to provide insight on the expected behaviours of benches.

The bench-scale mechanisms were studied using the Rocscience 2D finite element program, *RS2*. The analysis aimed at estimating the drop in the factors of safety from a 60° BFA configuration to a BFA of 65° under the above-outlined assumptions. Factors of safety were evaluated for the two different BFA configurations assuming two different thicknesses of rock slabs/wedges (the slab or wedge thicknesses depended on the proximity of the primary sliding plane to the bench toe).

For the two different slab thicknesses, the results show a sharp drop in factor of safety from a 60° BFA to 65° BFA. At the bench scale, the sharp drop in factor of safety is due to the change in failure mechanism from one involving both sliding along weak discontinuity planes and shearing through intact rock bridges to a mechanism which involves only sliding along the weak planar discontinuities (planar wedge failure).

16.4.5 Overall slope failure and factors of safety

The overall slope behaviours under anisotropic conditions were studied using the 2D limit equilibrium program, *Slide* using two different sets of groundwater conditions, the second involving the application of 100 m horizontal drain holes.

The application of an anisotropic model for the footwall units seemed to cause the Cuckoo Search method to produce uncharacteristically low factors of safety. The Path Search method provided more realistic factors of safety and was therefore applied.

Factor of safety values were calculated using a number of different techniques. For each analysis scenario (combination of groundwater conditions and bench face angle), while there were substantial differences in the factors of safety computed from the various techniques, the variation was consistent.

Like the results for the bench-scale analyses, the factors of safety sharply drop from the 60° BFA configuration to the 65° BFA case. The results indicate that, for the average discontinuity dip of 60°, a 60° BFA provides an overall slope with more stable factors of safety than the slopes of a 65° BFA.

The assumption of discontinuities being planar and having no intact rock bridges across hundreds of metres may be quite conservative. However, the analysis still facilitates highly useful comparisons of the stability of the different designs under similar assumptions, even if those differ from planar weak planes that extend (without rock bridges) for hundreds of metres. For example, it is possible that in the real world the weakness planes will not be so planar and extensive as currently assumed, which has caused the factors of safety to drop by 30% or more from slopes with 60° BFA to ones with 65° BFA.

16.4.6 Potential steepening of footwall sector

It is possible for BFAs slightly steeper than foliation dip to be achieved in pit slopes and may be established from field performance of benches. At Namdini, inter-ramp angles steeper than those recommended in Table 133 may be possible since the rocks are generally strong and the foliation sets may not be as persistent as assumed in the study. However, this cannot be established from the currently available data.

Once temporary slopes are excavated, and specific measures that improve slope performance are taken, it would be then feasible to consider footwall slope steepening. This would be facilitated using measures such as smooth blasting, geotechnical controls, data collection, pit depressurisation, dewatering, and/or triple benching.

Based on the available data and analysis, Golder recommends that the original footwall slope designs summarised in Table 133 be applied. During operations, steepening measures outlined elsewhere (Golder, 2019d) may be applied on temporary walls and monitored for performance. The footwall slopes should be steepened only when geotechnical/geological measurements and observed performance of temporary walls support that.

Given the already aggressive footwall bench design, it would be prudent to not steepen the slopes at the current stage of design.

16.4.7 Verification

On 24 April 2019 Golder (2019e) provided written verification that based on cross sections through the pit shell for all design sectors, it satisfies the geotechnical designs for the various domains and sectors.

16.5 Mining options

Conventional mine design methods were used for the generation of open pit layouts (mine design) and production scheduling (mine planning). The mining study is intended to determine the overall mining rate and estimated ore delivery in each period. The optimization strategy included testing a range of mining rates and stockpiling options to give an indication of maximum potential value from the 9.5 Mtpa process plant feed requirements. The discounted net present value (NPV) from the optimization was used to assess the relative value of each option.

16.6 Mining resource model

A resource model was provided to Golder by Cardinal in February 2019. The resource model was created by MPR and is an MIK estimated block model (see Section 14.0) for the Namdini deposit in Local grid coordinates. The block model framework dimensions and block sizes are shown in Table 134.

Table 134: MPR Namdini February 2019 Mineral Resource block model framework

	Easting	Northing	Elevation
Start	14,375	52,450	-550
End	15,850	54,125	250
Block Size	12.5	25.0	5.0
No Blocks	117	67	160
Dimensions	1,475	1,675	800
Block Model name = lg_2019resmod_RevA_MM_v4.bmf			

The model file was supplied in Comma Separated Value ("**CSV**") format and was imported into the Vulcan™ mine modelling software package for subsequent mine modelling and design options. The supplied model included grade and resource variables defining the proportion of potentially mineable material at varying gold cut-off grade values, by resource category. The model data as received (*201902 Namdini MIK Model.csv*) was statistically investigated, checked and corrected for anomalous values, for example, proportion values greater than 100, or density values outside the expected range. Once satisfied that the raw data appeared logical the CSV file was then imported into Vulcan to create a mining resource model for the Namdini Project. The supplied data file included variables that are shown in Table 135.

Table 135: Mineral resource model variable names and default values (Feb 2019 model)

Variables	Default	Type	Description
Lith	0	name	lith_code
SG	0	float	density in t per bcm
Weath	0	integer	weathering code 1=Oxide 2=Transition 3=Fresh
Remprop	0	float	Proportion of block remaining below topography
HLprop	0	float	Proportion of Oxide block remaining below topography
Rockprop	0	float	Proportion of rock in block below topo
PanAve	0	float	Au Grade used for Optimization Au00 for LUC and Au0
TonsProp0.200	0	float	Proportion of Block with Grade above 0.2 g/t Au
PGrade0.200	0	float	Average Grade of Block above 0.2 g/t Au
TonsProp0.300	0	float	Proportion of Block with Grade above 0.3 g/t Au
PGrade0.300	0	float	Average Grade of Block above 0.3 g/t Au
TonsProp0.400	0	float	Proportion of Block with Grade above 0.4 g/t Au
PGrade0.400	0	float	Average Grade of Block above 0.4 g/t Au
TonsProp0.500	0	float	Proportion of Block with Grade above 0.5 g/t Au
PGrade0.500	0	float	Average Grade of Block above 0.5 g/t Au
TonsProp0.600	0	float	Proportion of Block with Grade above 0.6 g/t Au
PGrade0.600	0	float	Average Grade of Block above 0.6 g/t Au
TonsProp0.700	0	float	Proportion of Block with Grade above 0.7 g/t Au
PGrade0.700	0	float	Average Grade of Block above 0.7 g/t Au
TonsProp0.800	0	float	Proportion of Block with Grade above 0.8 g/t Au
PGrade0.800	0	float	Average Grade of Block above 0.8 g/t Au
TonsProp0.900	0	float	Proportion of Block with Grade above 0.9 g/t Au
PGrade0.900	0	float	Average Grade of Block above 0.9 g/t Au
TonsProp1.000	0	float	Proportion of Block with Grade above 1.0 g/t Au
PGrade1.000	0	float	Average Grade of Block above 1.0 g/t Au
Rescat	0	integer	Resource Category
As	0	float	Arsenic grade in ppm
S	0	float	Sulphur grade in %

The imported Mineral Resource model was then used to create a mining resource model with the inclusion of additional variables, retaining the original as-supplied variable values. For example, the resource category values were copied to an Integer type value field to permit category reporting within the Vulcan software. A list of the additional mining variables included within the mining resource model is shown in Table 136.

Table 136: Additional variables added to the mining 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

Variables	Default	Type	Description
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 cu. m
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)
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 phases
period	0	integer	Period when mined from Back Schedule 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 potential Reserves using Measured plus Indicated ONLY

Mineral Resource estimation has used MIK approach which stores proportion and grade of ‘ore’ at various cut-off grades. The proportion of ‘ore’ at the selected cut-off grade (XX) per block is stored in PrXX and AuXX fields respectively. The estimated applicable cut-off grade for the Namdini Project derived from input gold pricing estimated mineral processing costs and estimated processing recoveries taken from the Namdini scoping study.

The estimated marginal cut-off grade of 0.5 g/t was determined from an input gold price of US$1,300/oz and a metallurgical recovery for the Fresh rock using a derived recovery estimate from testwork supplied by Cardinal. A summary of the criteria used to estimate the applicable marginal cut-off grade is provided in Table 130.

Using the cut-off grade of 0.5 Au g/t, scripting was then used to estimate the ‘ore’ and ‘waste’ tonnes on a block by block basis within the resource model for material within the Measured, Indicated or Inferred Resource categories. It is important to note that the Inferred Resource was only used to provide estimates of where the potential ultimate pit shell could be. No Inferred Resource material was considered when determining the pit shell that provides the basis for Ore Reserve reporting.

The mining resource model was then used to create a mining model for import into Whittle 4X™ software to enable pit optimization of the model and assessment of the potential final pit. The mining resource model exported from Vulcan was compared with the Whittle import model (*.mod*) file to ensure that no export or import errors had occurred during the conversion process. The check on the tonnage imported into Whittle 4X showed that the material above cut-off was 222 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 4X.

Table 137 and Table 138 show the comparison check between the Vulcan mining model and the exported Whittle 4X™ optimization model for the Namdini deposit. 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 137: Comparison of supplied MIK resource model to Whittle 4X™ mining model

>0.5 g/t Au	Indicated and Inferred		
Model Type	Vulcan	Whittle 4X	**Comparison**
Model Name	*lg_2019resmod_RevA_MM_v4.bmf*	*NAM_LGrid_RevA_v3_Model2019_AachenProcess_MREC_20190315.mod*	
Classification	Indicated + Inferred	Indicated + Inferred	**Variance**
Tonnes (M)	221.9	221.9	0.0%
Contained Au (Moz)	7.917	7.917	0.0%

Table 138: Vulcan and Whittle 4X comparison by ore type of contained gold above 0.5 g/t cut-off grade

Rock Type	RWIT Code	Vulcan Contained Au (g)	Whittle Contained Au (g)	Variance
Meas+Ox	11	1,382,688	1,382,688	0.0%
Meas+Trans	12	1,278,849	1,278,849	0.0%
Meas+Fresh	13	7,146,151	7,146,151	0.0%
Ind+Ox	21	3,567,946	3,567,946	0.0%
Ind+Trans	22	3,288,605	3,288,605	0.0%
Ind+Fresh	23	198,626,471	198,626,471	0.0%
Inf+Ox	31	41,776	41,776	0.0%
Inf+Trans	32	11,824	11,824	0.0%
Inf+Fresh	33	30,900,842	30,900,842	0.0%
	Grand Total	***246,245,152***	***246,245,152***	***0.0%***

Note: Source: lg_whittle_check_20190315

The Vulcan export routine was used to create the Whittle 4X model from the mining model. Ore types were created based on weathering and class. Table 139 summarises the model origin and dimensions of the Whittle 4X model. The block model was exported to Whittle 4X as a parcelled model retaining the 'ore' and 'waste' fractions defined by the MIK resource model.

Table 139: Whittle 4X parameters

Item	X (Easting)	Y (Northing)	Z (RL)
Block size (m)	12.5	25	5
Model Origin	14375	52425	-550.0
Extent (m)	1475	1700	800
Number Parent Blocks	117	67	160

Note: Source: NAM_LGrid_RevA_v3_Model2019_AachenProcess_MREC_20190315.mod

Table 140 presents the parameters used in the Whittle 4X™ pit optimization analysis.

Table 140: Technical and economic parameters used in Whittle 4X™ optimization studies

Parameter	Units	Value
Gold Price	US$/oz	1,300
Ore Lost during Mining*	%	incl in MIK
Dilution Added during Mining*	%	incl in MIK
Mining Rate	Mtpa	35
Plant Recovery – Oxidized Zone	%	90
Plant Recovery – Transitional Zone	%	82
Plant Recovery – Fresh Zone	%	82
Annual Discount Rate	%	10
Cut-off Grade**	g/t	0.5

Notes: * An estimate for ore loss and dilution has been allowed for in the resource estimation process; no further ore loss and dilution has been considered in the pit optimization process.

** Cut-off grade has been estimated using an estimated metallurgical recovery of 82% and an input gross gold price of US$1,300/oz. The recovery fractions are simplifications of the applied recovery formula derived from metallurgical testwork.

The cut-off grade (COG) has been estimated as follows:

Marginal COG = (Process + Reclaim Cost)/(US$1,300/oz – 5% Royalties – $1.10/oz TC/RC)/31.1034768 g/oz) × 82% (met recovery LOM average)

Marginal COG = (US$14.30/t + US$1.50/t)/US$39.67 per gram × 0.82

Where US$14.30 is the process plant plus general and administration cost per tonne of ore processed and US$1.50 is the stockpile reclaim cost per tonne ore at a design throughput of 9.5 Mtpa.

Mining costs were split into the fixed and variables costs with the fixed annual cost being allocated to the process plant throughput on a cost per tonne basis. The variable mining cost component was then used within the pit optimization with a cost increment added for depth.

The base mining variable cost at surface equates to $2.07 per tonne mined, with a cost increment with depth equivalent to 6c/t per 10 m bench from surface. The fixed mining cost of $15.47 m was applied to the process plant annual throughput equating to an additional $1.63/t milled.

For the purposes of the mining study, it has been planned that a ‘dry pit’ will be operated and that sufficient dewatering will be provided to ensure dry slopes (considered in the geotechnical design parameters in Section 16.3).

All tonnage is estimated on a dry basis with expected surface moisture associated with the hauled rock being less than 3% on average.

16.7 Pit optimization

The initial Whittle pit optimizations were carried out covering options with and without Inferred Resource material. The inclusion of the Inferred Resource material in the pit optimization is to assess likely areas that could provide possible future growth opportunities and to limit infrastructure and permanent waste dumps (WRD) to outside any potential future footprint. The optimization used a revenue factor (“**RF**”) range from 0.25 through to 1.0, equivalent to an input gold price range of US$325 to US$1,300. The low price (0.25 RF) indicating the target area for commencement of the Starter Pit, whilst the high price pit shell (1.0 RF) would indicate the likely maximum potential pit limit.

Whittle produces a Best case and Worst case schedule that guides the user to determine a practical solution that offers a balance of high financial value and practical mining constraints such as minimum mining width and bench advance (sink rate). The Whittle optimization 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.

The pit optimization was tested against a range of primary value drivers including mining cost, processing cost, capital cost and gold price as well as overall geotechnical slope angle ("**OSA**"). The geotechnical changes are considered modest in that the east wall is essentially fixed and only the west wall has been flexed (by ±5^{o} in 2.5^{o}intervals). As previously seen during the PFS stage of the project, the project value is most sensitive to gold price.

Figure 100 shows the sensitivity results for the pit optimization flexing. A 10% increase in the forecast LOM gold price changes the NPV by +36.6%, whilst a 10% decline in the forecast gold price results in a -36.5% drop in the NPV.

The project is next most sensitive to processing cost with a 10% increase in processing cost resulting in a reduction in project NPV of -13.9%, conversely a 10% reduction in total processing cost would see the project value improve by +16.1%.

The Namdini Project is only moderately sensitive to both capital and mining costs, and relatively insensitive to the overall geotechnical slope angle of the pit as a result of the relatively low stripping ratio for a pit of this size.

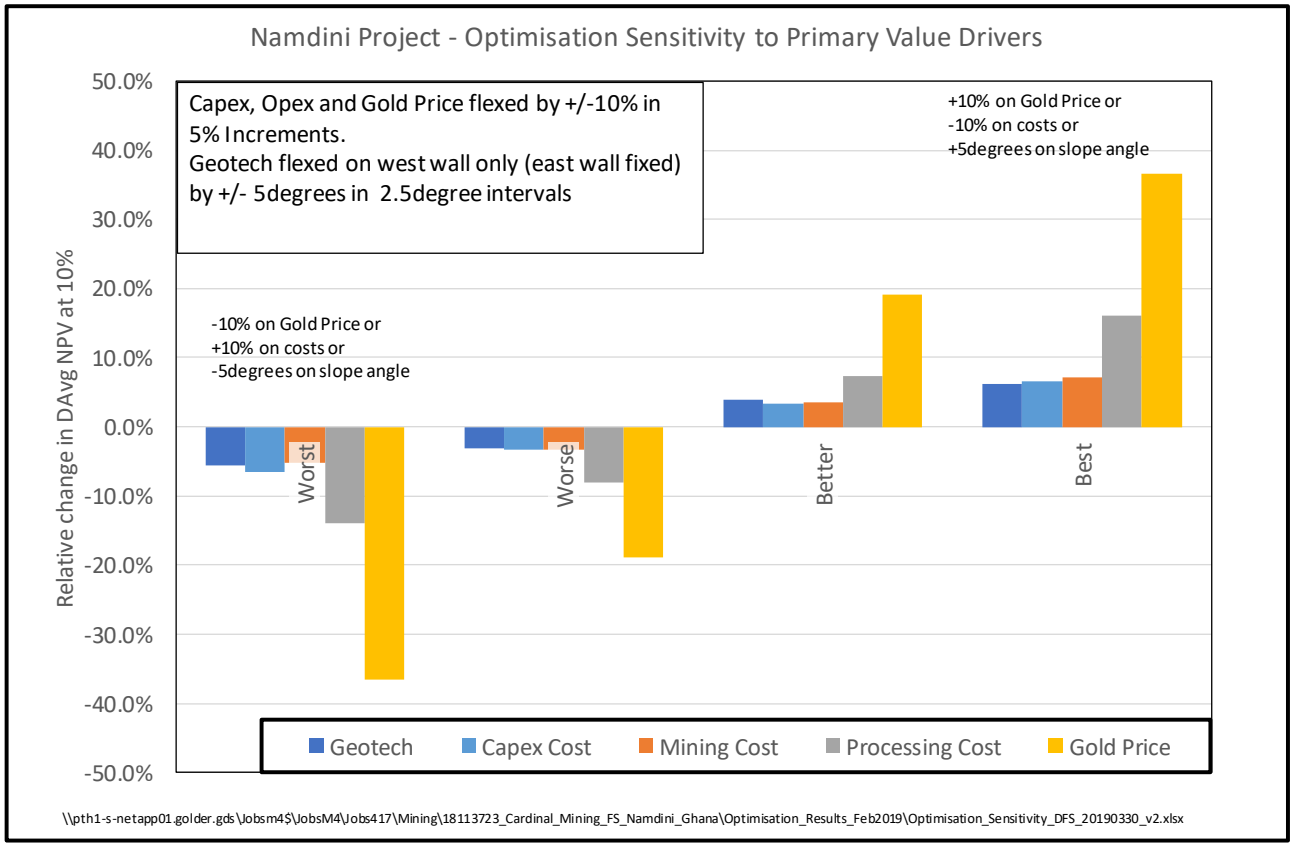


Figure 100: Whittle 4X pit optimization sensitivity analysis (M+I)

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 analysis of the Whittle optimization results shows that for the given costs and pricing assumptions, the discounted cashflow begins to erode marginally from about the 0.9 RF and more noticeably above 0.95RF indicating that any pit shell above 0.95RF would no longer be value accretive. The graph in Figure 101 shows the undiscounted and discounted cashflows, against the Revenue Factor for the M+I Resource. The reduction in value above 0.9 RF is however considered modest against the total Davg value.

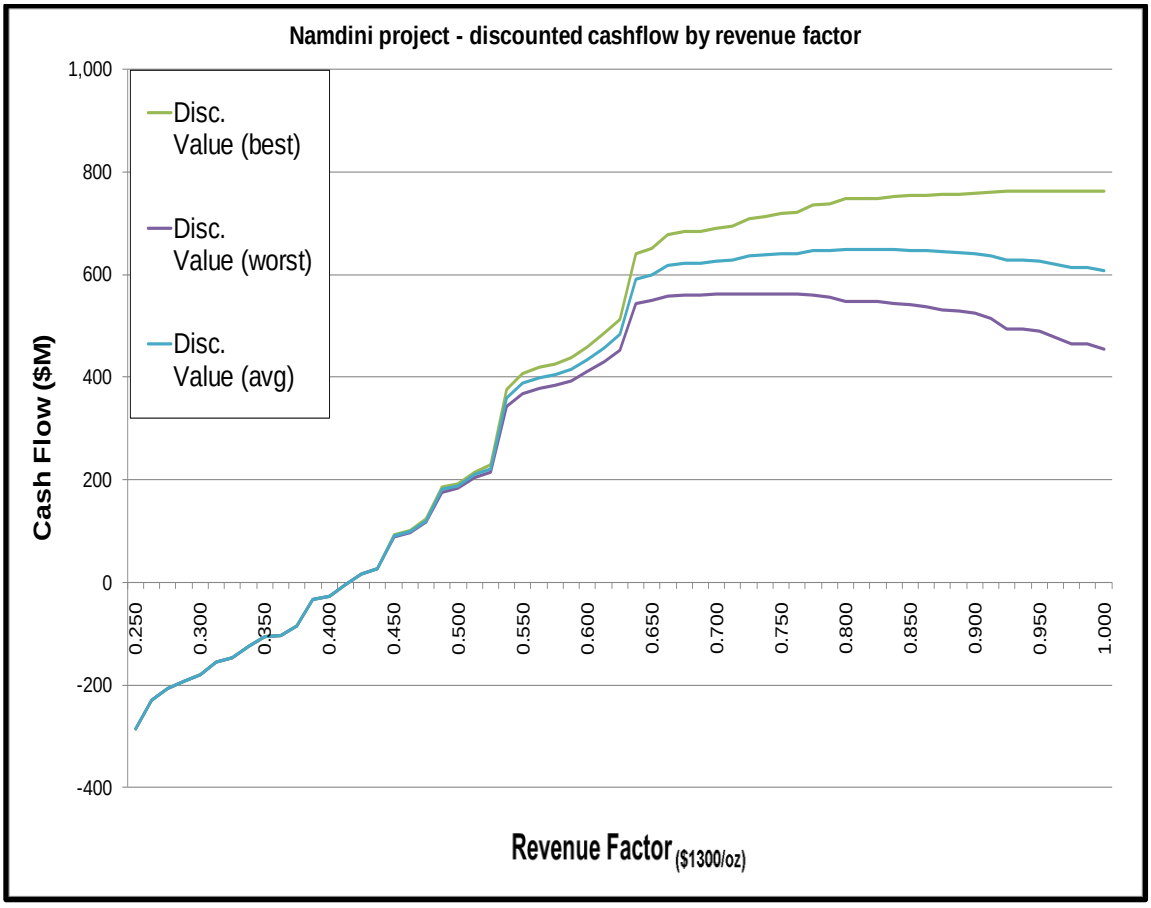


Figure 101: Whittle 4X pit optimization, discounted cash flow by RF

The ore and waste by tor within the pit shells for the Namdini Project can be seen in Figure 102.

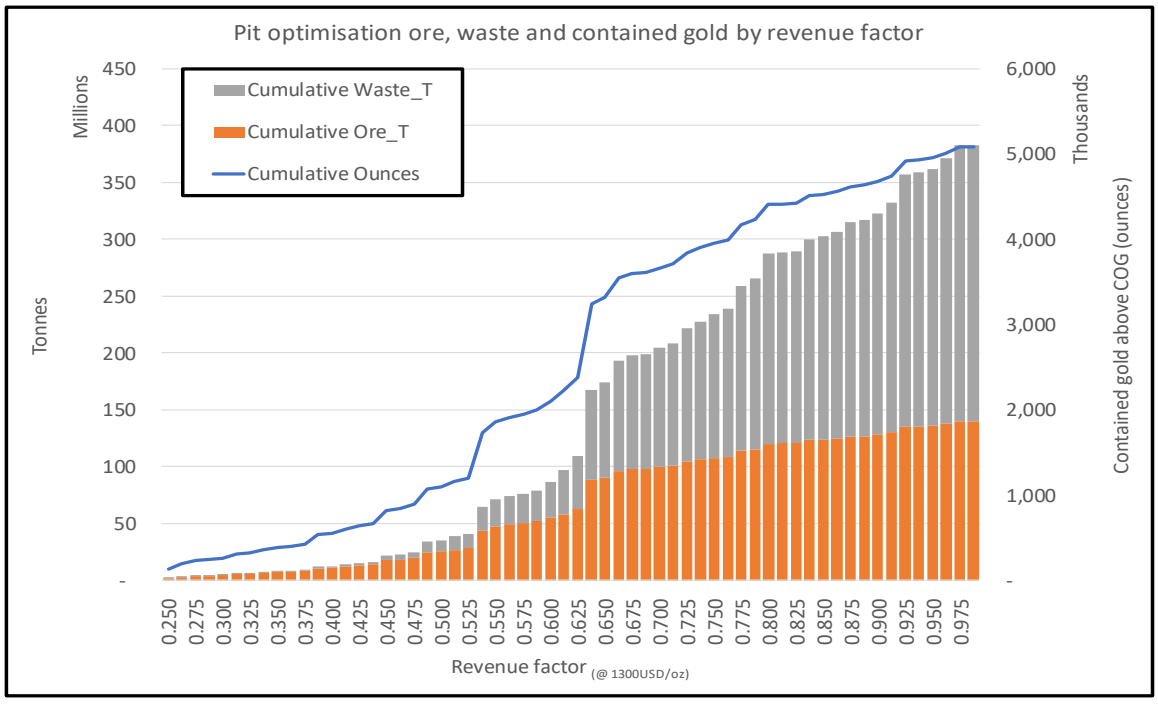


Figure 102: Pit optimization ore, waste and contained gold by RF

A visual inspection of the pit shells in the Whittle viewer shows the progression of the pit from east to west as the gold price is increased. This progression is to be expected as the ore has a general dip of some 70° from east to west. The pits have been shaded by group with the first pit group representing the RF range from 0.5 RF to 0.7 RF. Table 141 shows the Whittle 4X™ shell range grouped in the figures showing the pit shells in Figure 103, Figure 104 and Figure 105.

Table 141: Namdini Project Whittle 4X™ Pit Shells (M+I)

Pit Group	Revenue Factor Range	Comment
Pits 1 to 21	0.25 to 0.5	Initial target pit areas
Pits 21 to 37	0.5 to 0.7	Area for second pushback
Pits 37 to 45	0.7 to 0.8	Area for third pushback
Pits 45 to 53	0.8 to 0.9	Area for fourth pushback
Pits 53 to 61	0.9 to 1.0	Limits for fourth pushback

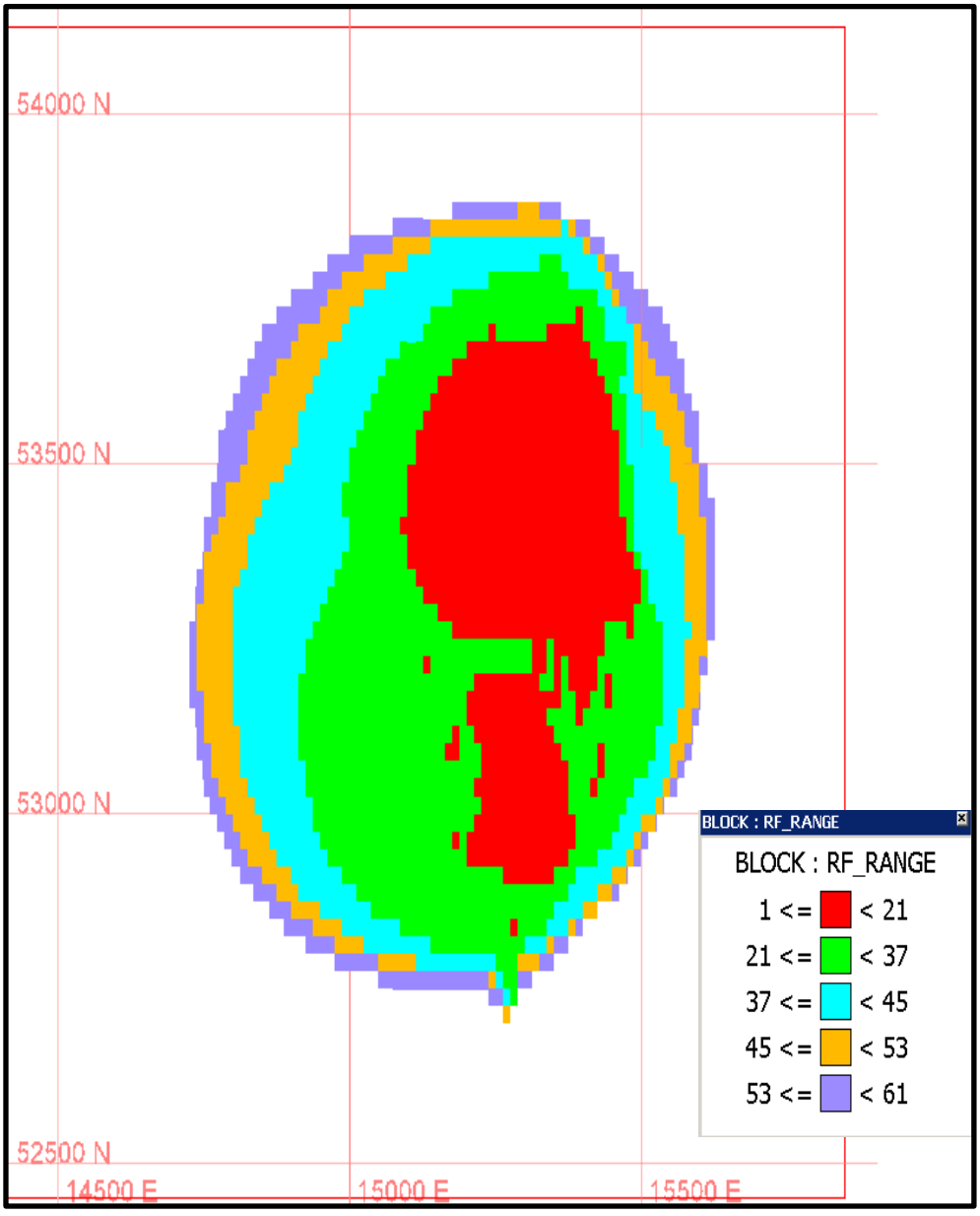


Figure 103: Whittle 4X pit shells (RF1.0 = US$1,300/oz), plan view

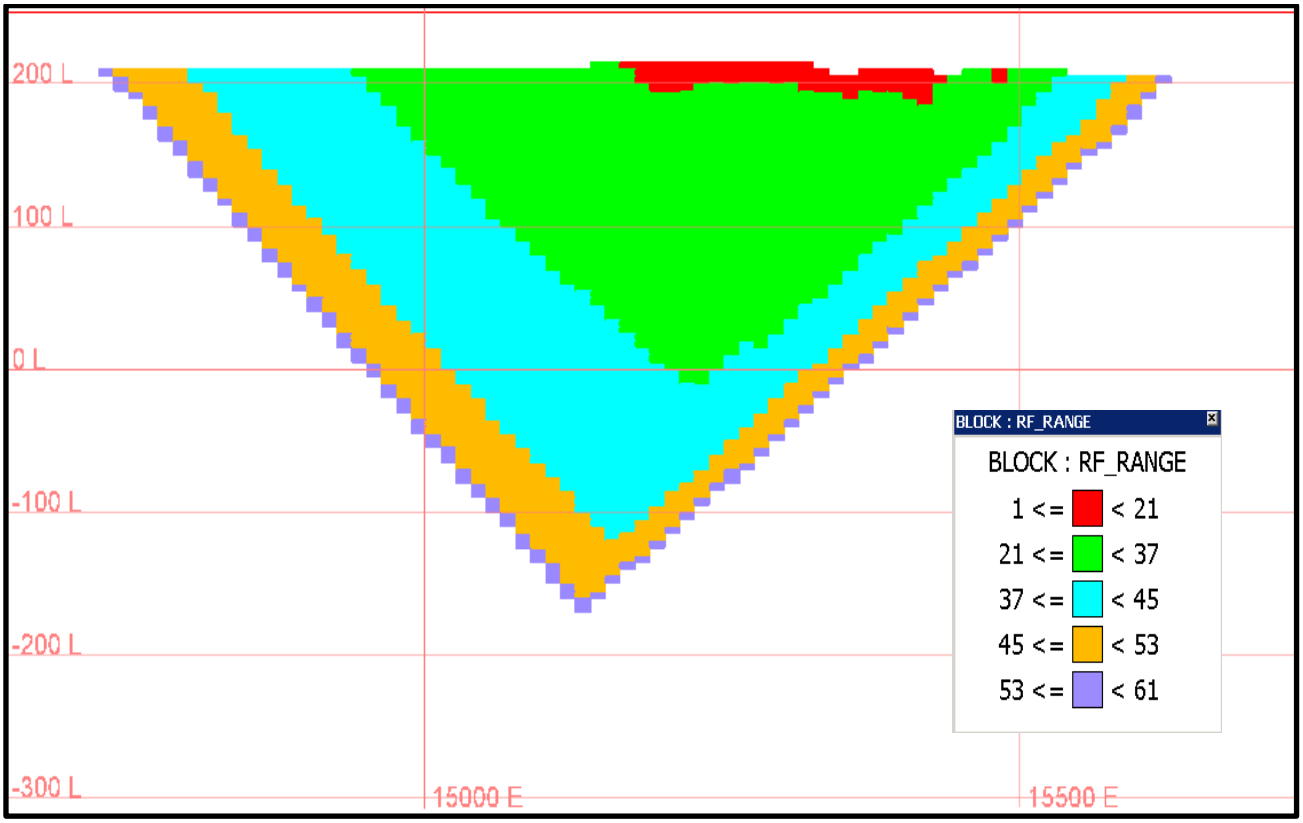


Figure 104: Whittle 4X pit shells (RF1.0 = US$1,300/oz), facing north

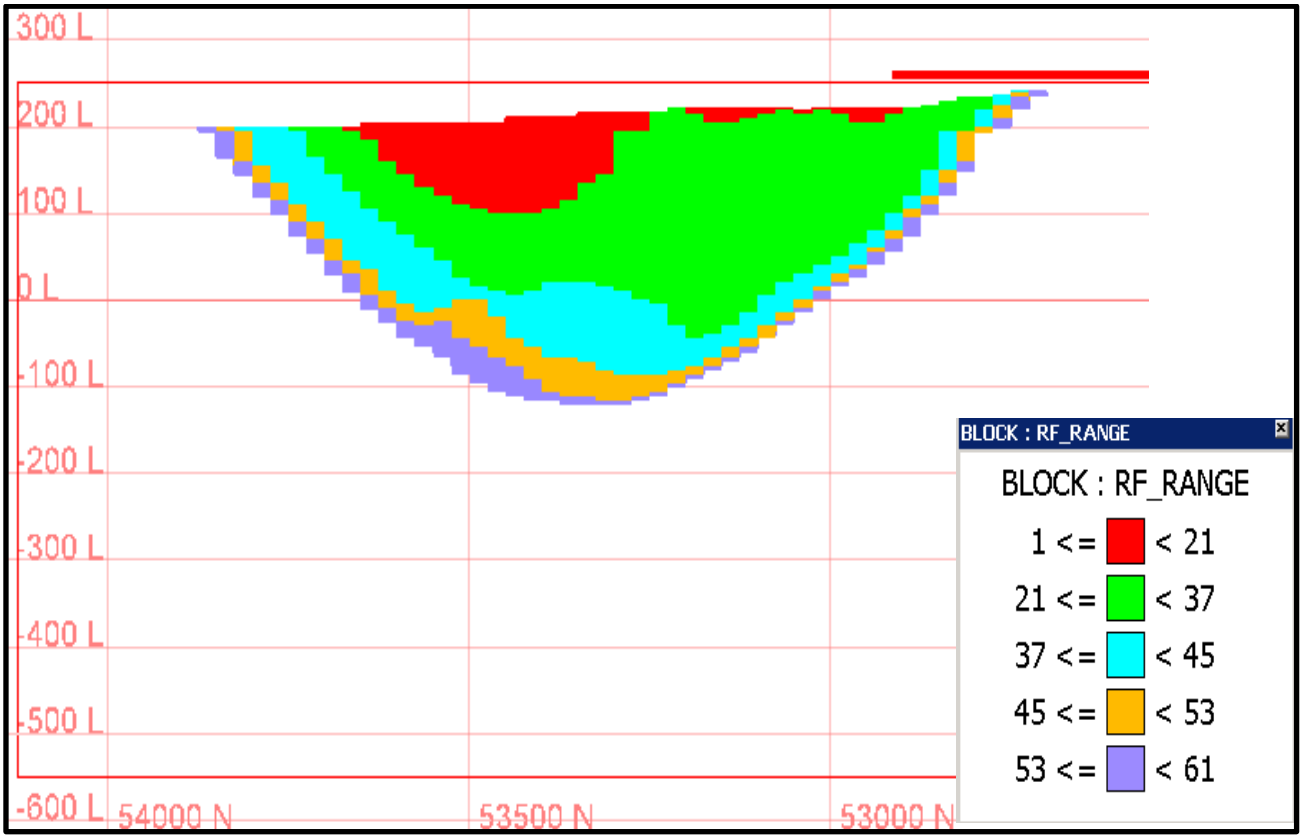


Figure 105: Whittle 4X pit shells (RF1.0 = US$1,300/oz), facing east

A view through the mineralized ore zone (Figure 106) in the block models shows the general dip of the ore zone and the pockets of higher-grade material (>2 g/t Au) that drive the initial pit selection. A plan view of the ore zone shown in Figure 107 clearly shows the two higher-grade ore pockets near surface.

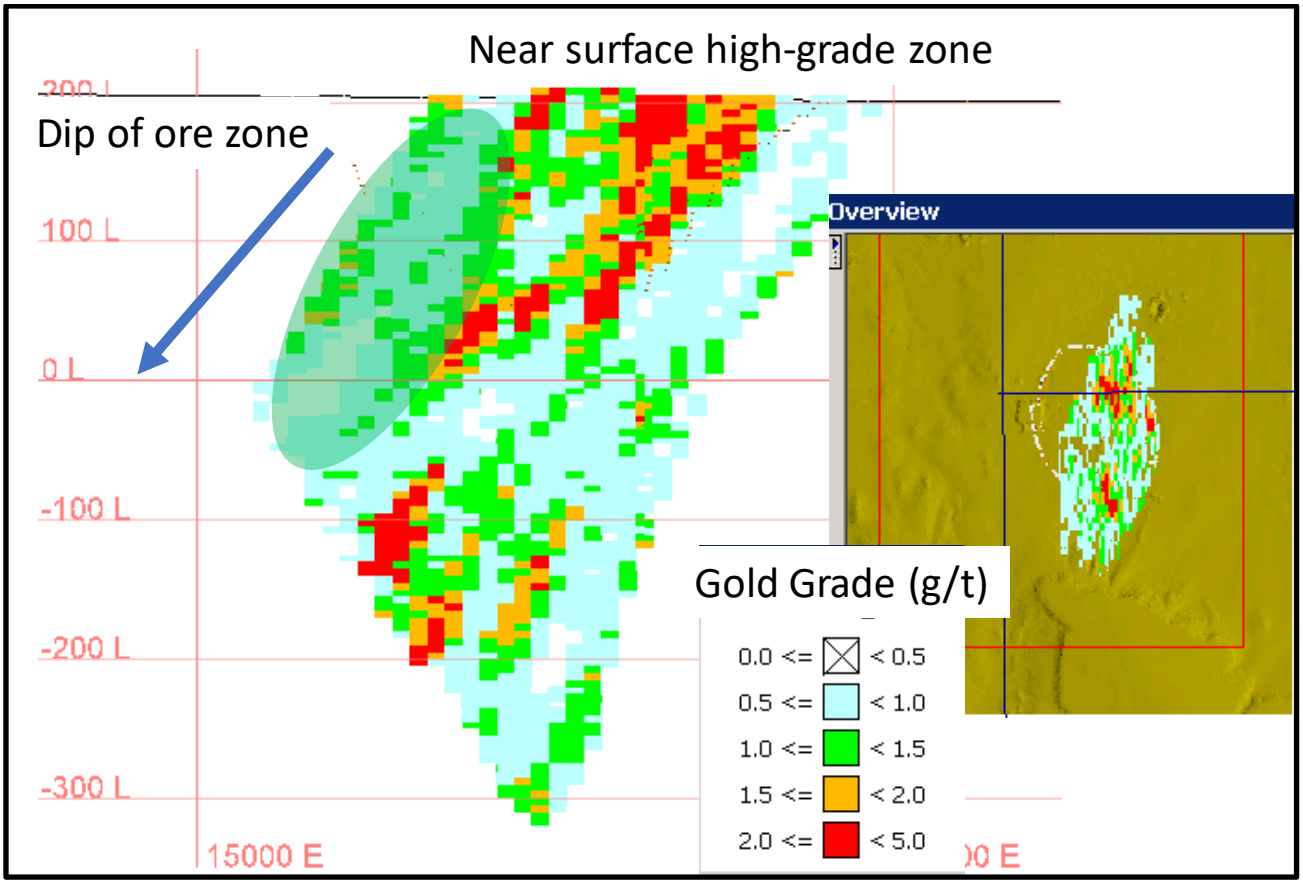


Figure 106: Grade >0.5 g/t Au (M+I) and general dip ~70°, facing north

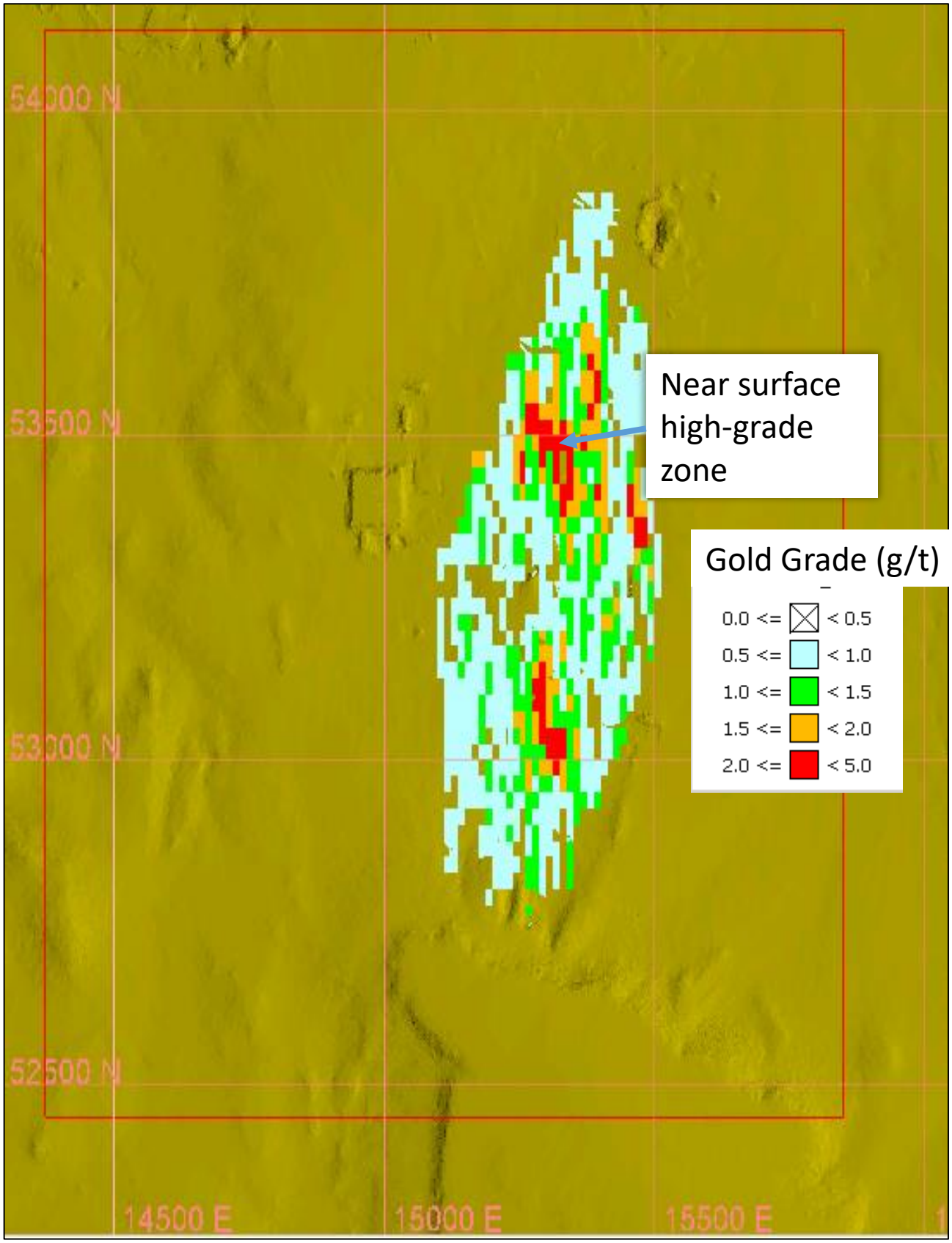


Figure 107: Grade >0.5 g/t Au, plan view

16.8 Pit selection

Pit design has largely focussed upon targeting maximum value change-points within practical mining constraint limits, such as the minimum mining width (80 m) for the pushbacks.

As an aid to identifying the higher value Starter Pit region, the pit optimization used a low revenue factor commencement point of 0.25. The range of shells indicated by the optimizer was compared for total rock content and ore feed material. An initial pit was selected that provided a minimum of two years mill feed as the commencement region for mining. The Starter Pit will be followed by a series of pushbacks or Phases that expand the mining area up to the Final Pit limit for the LOM planned production. The total waste and ore tonnes along with the contained gold within the pit shells are shown in Figure 108.

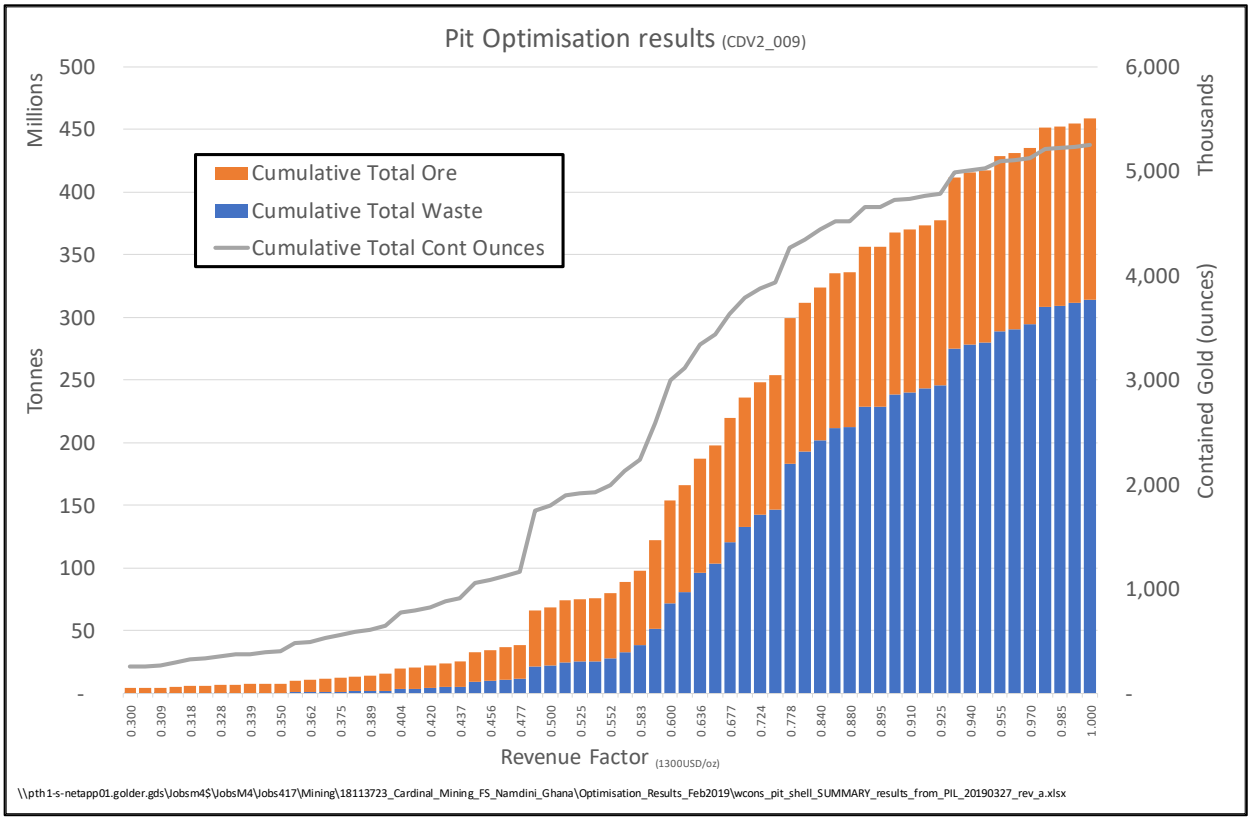


Figure 108: Close-spaced Whittle 4X ore, waste and contained gold by Revenue Factor

The pit shells from the Whittle 4X optimizer were then exported and imported into the Vulcan block model to enable Phase selection based upon 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 five Phases within the final pit. The final pit has been selected at an RF of ~0.95, this being the point where the resource is suitably maximised and the discounted average value is close to the peak, enabling a balance between ore volume and financial return.

Figure 109 shows the undiscounted cashflow as a proportion of the maximum by RF. The indicated value remains relatively flat across the range from 0.8 RF to 0.95 RF. The final pit was selected based upon a combination of indicated value and practical mining considerations such as potential bench turnover rate to ensure adequate mill feed to the process plant. The Final Pit contains an estimated 138 Mt of ore and 263 Mt of waste for a combined total rock of 401 Mt. During the design process, as was evident during the PFS, there is an opportunity to establish a goodbye cut that allows access to deeper ore, using a narrow access ramp (18 m single lane) and narrow mining widths for single excavator operation.

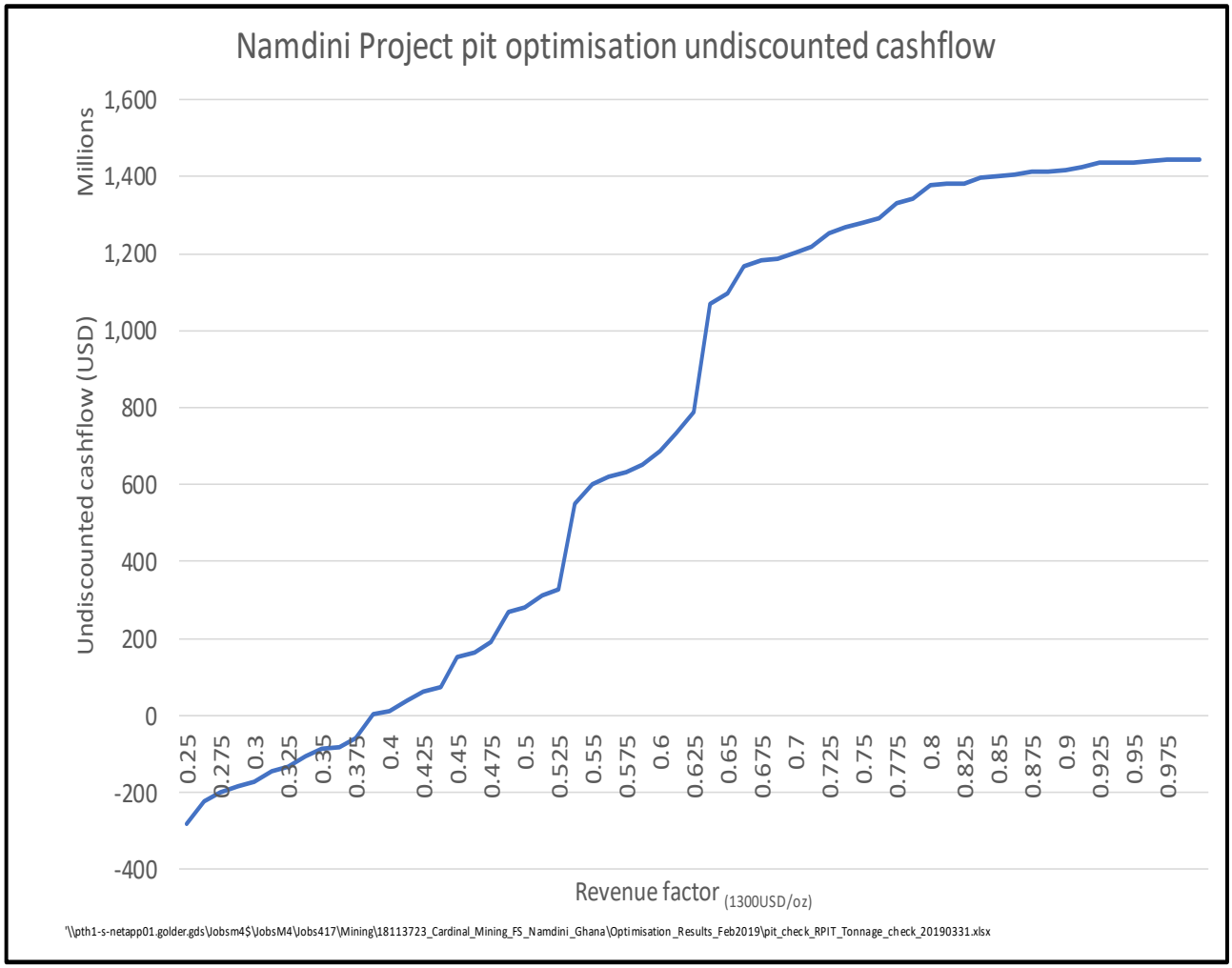


Figure 109: Undiscounted cashflow by RF

The pit Phase selection identified five potential Phases including the initial Starter Pit. The initial Starter Pit (Phase 1 and 2) has targeted an area that contains some 32 Mt of ore feed, with only 18 Mt of waste, this low strip ratio aids in bringing value forward for the project. The subsequent pushbacks (Phase 3 to Phase 5) have targeted areas that allow for a minimum mining width for each Phase, whilst maximising value. The Final Pit defines an Ore Reserve of 138.6 Mt at an average undiluted head grade of 1.13 g/t Au for a contained 5.1 M oz of gold. Table 142 summarises the material quantities at each Phase from the pit design process.

Table 142: Pit Phase selection

Stage	Total Rock (Mt)	Ore (Mt)	Waste[1] (Mt)	Contained Au (koz)	Strip Ratio (W:O, t:t)	Avg Grade (g/t Au)
Phase 1	15.7	12.3	3.4	564	0.28	1.43
Phase 2	34.8	19.9	14.9	634	0.75	0.99
Phase 3	65.7	34.6	31.2	1,251	0.90	1.13
Phase 4	103.9	35.6	68.3	1,271	1.92	1.11
Phase 5	181.5	36.3	145.2	1,334	4.00	1.14
Total	**401.6**	**138.6**	**263.0**	**5,054**	**1.90**	**1.13**

Note: [1] Inferred material has been included in with the waste tonnes.

16.9 Pit design

Pit design criteria have been based on geotechnical recommendations where the deposit has been 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. Table 143 shows the geotechnical design criteria for the Namdini pit.

The Final Pit was reviewed by the responsible geotechnical engineer and verified as being in accordance with the geotechnical recommendations for dry conditions at the Namdini Project (see Section 16.4).

Table 143: Recommended slope configurations for Namdini Pit

Design Sector	Wall Dip Direction	Rock Mass Domain	Bench Height (m)	Bench Face Angle (°)	Spill Berm Width (m)[1]	Limiting Bench Stack Height (m)	Inter-Ramp Angle (°)[1]
1	110°	*Saprolite & Saprock*[1]	10	60	6 (5)	30	40.3 (42.9)
		MW HWMSE[1]	10	60	6 (5)	30	40.3 (42.9)
		Fr HWMSE	20	70	9	120	50.9
		Fr MVO	20	80	9	120	57.9
		Fr DIO	20	80	9	120	57.9
2	145°	*Saprolite & Saprock*[1]	10	60	6 (5)	30	40.3 (42.9)
		MW HWMSE[2]	10	60	6 (5)	30	40.3 (42.9)
		Fr HWMSE	20	70	9	120	50.9
		Fr MVO	20	80	9	120	57.9
		Fr DIO	20	80	9	120	57.9
3	175°	*Saprolite & Saprock*[1]	10	60	6 (5)	30	40.3 (42.9)
		MW HWMSE[2]	10	60	6 (5)	30	40.3 (42.9)
		Fr HWMSE	20	80	9	120	57.9
		Fr MVO	20	80	9	120	57.9
		Fr DIO	20	80	9	120	57.9
4	235°	*Saprolite & Saprock*[1]	10	60	6 (5)	30	40.3 (42.9)
		MW DIO[2]	10	60	6 (5)	30	40.3 (42.9)
		Fr DIO	20	80	9	120	57.9
		Fr MVO	20	80	9	120	57.9
5	285°	*Saprolite & Saprock*[1]	10	60	6 (5)	30	40.3 (42.9)
		MW FWMSE[2]	10	60	6 (5)	30	40.3 (42.9)
		Fr FWMSE	20	60	9	120	44.2
		Fr DIO	20	60	9	120	44.2
6	315°	*Saprolite & Saprock*[1]	10	60	6 (5)	30	40.3 (42.9)
		MW FWMSE[2]	10	60	6 (5)	30	40.3 (42.9)
		Fr FWMSE	20	60	9	120	44.2
		Fr DIO	20	60	9	120	44.2
7	350°	*Saprolite & Saprock*[1]	10	60	6 (5)	30	40.3 (42.9)
		MW FWMSE[2]	10	60	6 (5)	30	40.3 (42.9)
		Fr FWMSE	20	80	9	120	57.9
		Fr DIO	20	80	9	120	57.9
8	050°	*Saprolite & Saprock*[1]	10	60	6 (5)	30	40.3 (42.9)
		MW HWMSE[2]	10	60	6 (5)	30	40.3 (42.9)
		Fr HWMSE	20	60	9	120	
		Fr MVO	20	80	9	120	57.9
		Fr DIO	20	80	9	120	57.9

Design Sector	Wall Dip Direction	Rock Mass Domain	Bench Height (m)	Bench Face Angle (°)	Spill Berm Width (m)[1]	Limiting Bench Stack Height (m)	Inter-Ramp Angle (°)[1]
9	080°	*Saprolite & Saprock*[1]	10	60	6 (5)	30	40.3 (42.9)
		MW HWMSE[2]	10	60	6 (5)	30	40.3 (42.9)
		Fr HWMSE	20	70	9	120	50.9
		Fr MVO	20	80	9	120	57.9
		Fr DIO	20	80	9	120	57.9

For practical pit design purposes, the berm widths have been rationalised to an 8 m wide berm to avoid having multiple berm widths required on the same mining bench. Catch-berms have been placed every 80 to 100 m in Fresh rock areas where there is no ramp system. The eastern side of the pit abuts the metasediments down to some 200 m below surface (Figure 95). Protection will be maintained by a combination of pre-splitting and controlled blasting to minimise possible damage to this wall unit.

The Phase designs have been created for optimal ore delivery placement for the first two stages, due to their low strip ratio, and total waste rock movement. The third and fourth stages contain a greater proportion of waste rock.

A minimum mining width of 80 m has been targeted between stages where practicable.

16.9.1 Stage designs

The pit designs for the Namdini Gold Project have targeted a pit shell at around 0.95 RF using a US$1,300/oz gold price. The pit optimization using the Whittle 4X™ software was used to identify the optimum pit shells with the Inferred Resource material considered as waste rock. The identified pit was then considered for practical staging to minimise waste movement and improve the cashflow. The analysis allowed the selection of five Phases with the Starter Pit targeting a relatively high-grade area of ore at near surface. The access to all mining Phases has been designed to incorporate a dual-ramp system to enhance safety and ensure ready access. The ramps exit from the northern side of the pit. The final Phase 5 has provided for a main access ramp on the western side of the pit linking to the eastern ramp system at 80 m RL (140 m from surface) to ensure a shorter haul to the waste rock dump. In particular having a ramp access on the western side of the pit for the final Phase reduces waste rock haulage costs on this higher strip ratio Phase and thus improves the overall value.

Phase 1 contains an estimated 12.3 Mt of ore and a low strip ratio of 0.28 (W:O, t:t) giving a total rock tonnage of 15.7 Mt within the Phase. The Oxide ore, predominantly mined in Phases 1 and 2, will be stockpiled and processed at the end of the mine life through the existing process plant after some minor plant modifications. Required modifications to the process plant design will be effected towards the end of the mine life prior to commencement of treatment of the Oxide ore.

Phase 2, which targets an extension of Phase 1, contains a further 19.9 Mt of ore at a relatively low strip ratio of 0.75:1 (W:O, t:t), resulting in a total rock movement of 34.8 Mt within the Phase. The total combined contained gold ounces above COG within the first and second mining Phases is 1,198 koz in 32.2 Mt of ore at an average head grade of 1.16 g/t Au.

The remaining three Phases follow a traditional pit expansion with the pits pushing out towards the dip of the ore and the Phases deepening with each expansion. The progression of all five Phases is shown in Figure 110 through to Figure 114.

A summary of the ore for each Phase is shown in Table 142 along with the gold grade and contained metal by Phase.

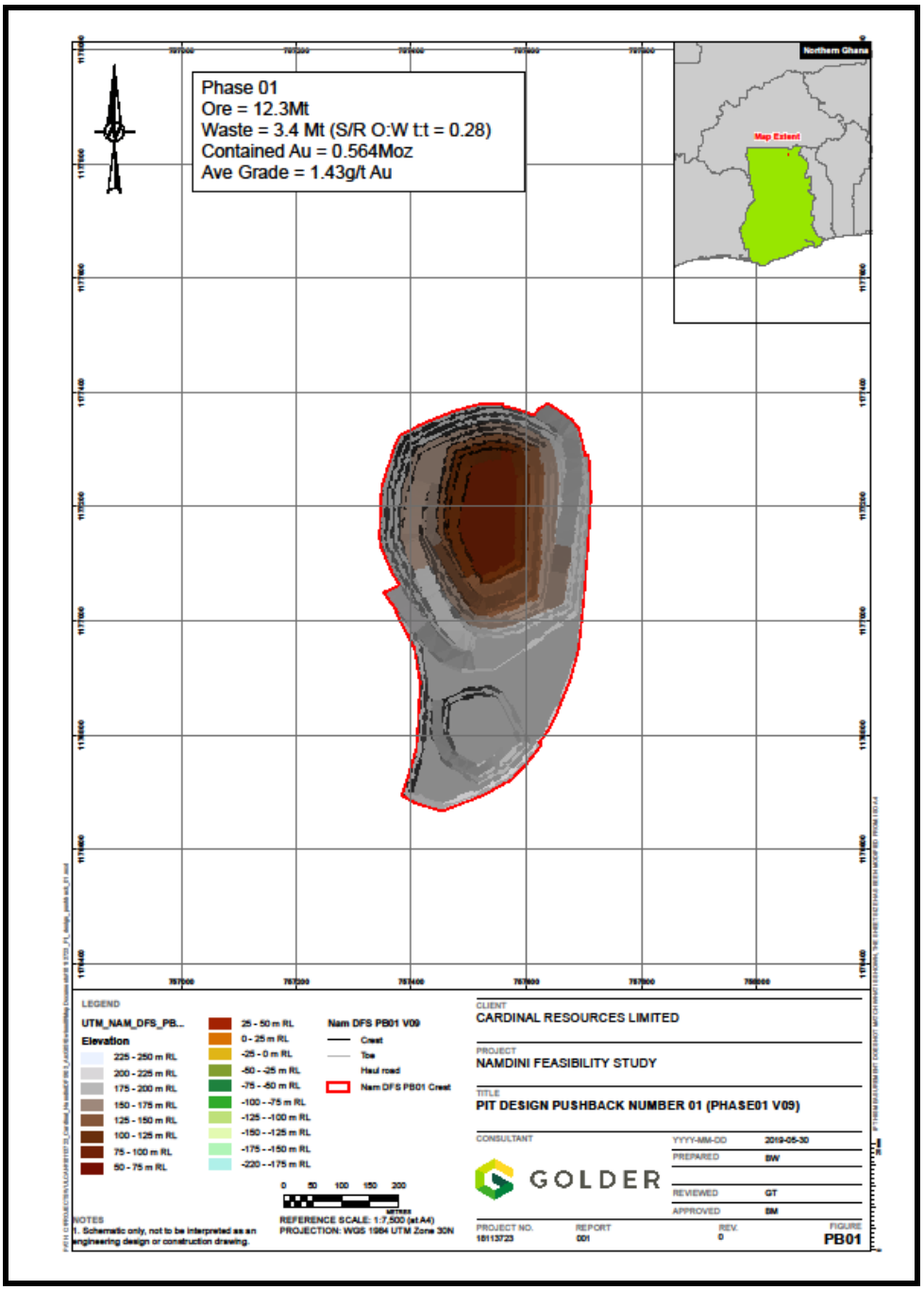


Figure 110: FS pit design Phase1 *utm_nam_dfs_pb01_v09_cl.00t* – May 2019

The initial Phases (Phase 1 Figure 110 and Phase 2 Figure 111) target 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 2 reaches a pit depth of 160 m from surface with a long axis (N-S) maximum dimension of 710 m and a short axis (E-W) maximum dimension of 520 m.

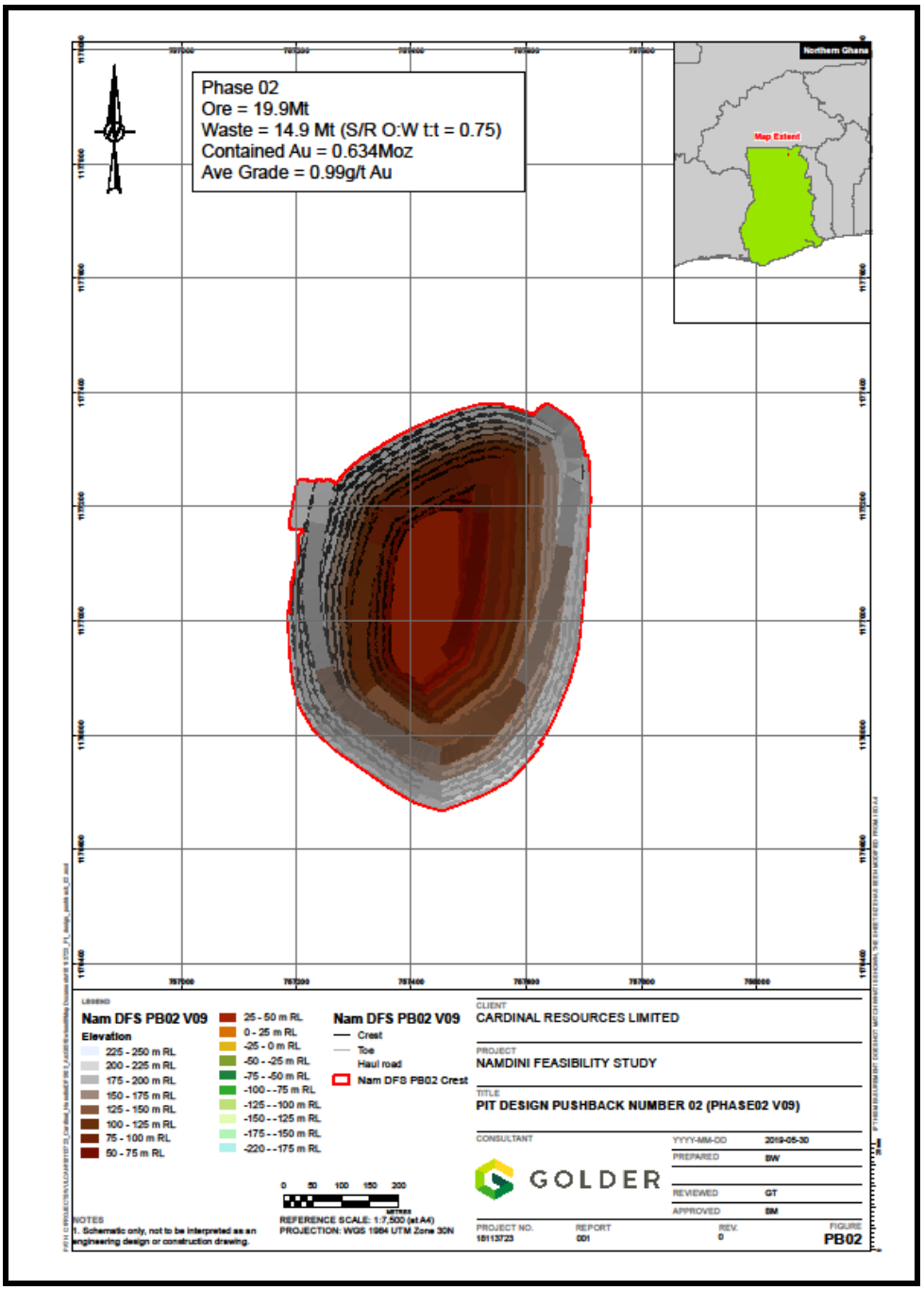


Figure 111: FS pit design Phase 2 *utm_nam_dfs_pb02_v09_1_cl.00t* – May 2019

Phase 3 (Figure 112) effectively expands on the initial first two Phases, targeting a further 35 Mt of ore at an average head grade of 1.13 g/t for a contained gold content of some 1,251 koz. Phase 3 reaches a pit depth of 250 m from surface with a long axis (N-S) maximum dimension of 870 m and a short axis (E-W) maximum dimension of 660 m.

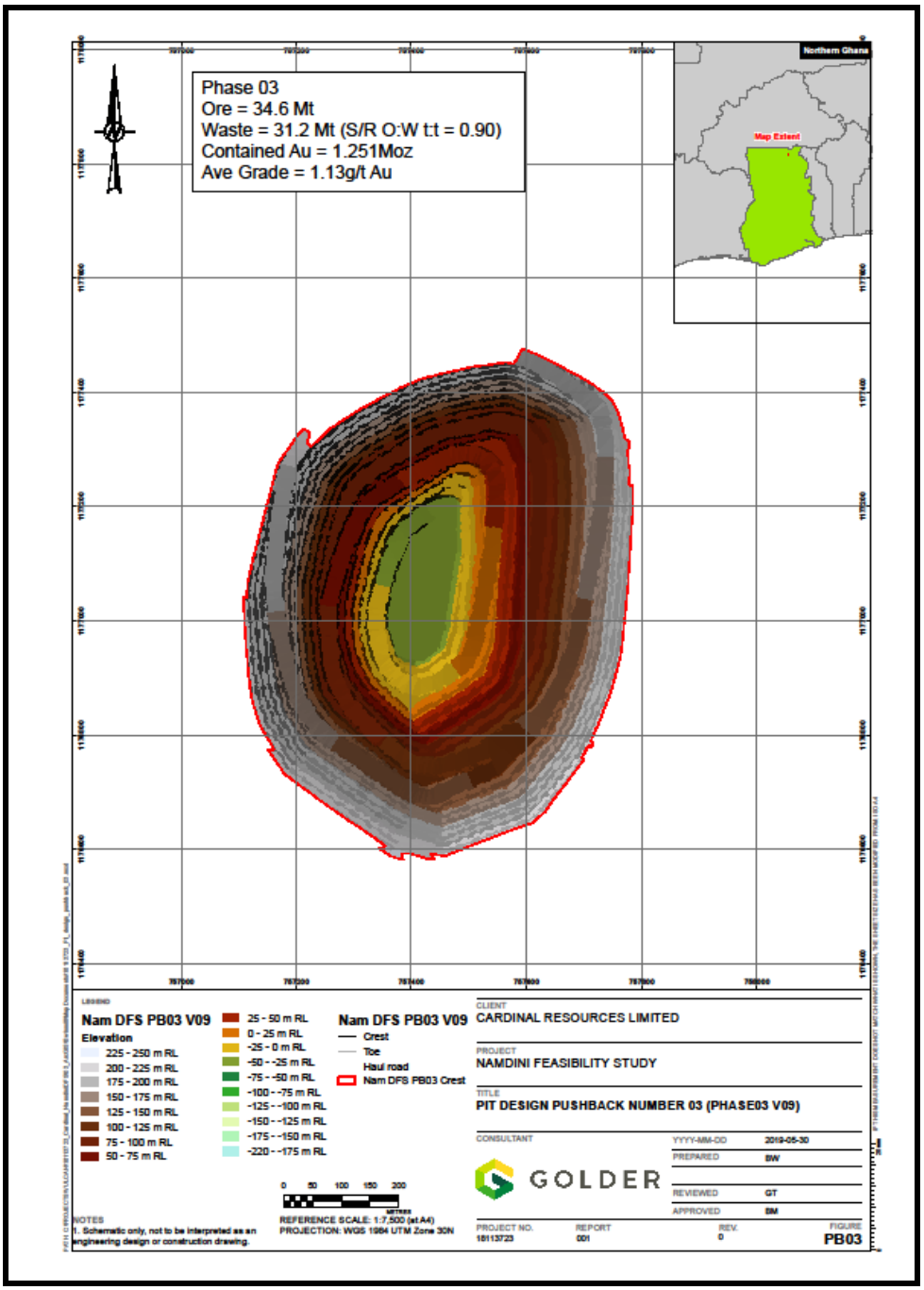


Figure 112: FS pit design Phase 3 *utm_nam_dfs_pb03_v9_cl.00t* – May 2019

Phase 4 (Figure 113) also targets a similar volume of ore (36 Mt) whilst requiring the movement of 68 Mt of waste rock. The contained gold is some 1,271 koz with an average grade of 1.11 g/t. Phase 4 reaches a pit depth of 340 m from surface with a long axis (N-S) maximum dimension of 1060 m and a short axis (E-W) maximum dimension of 810 m.

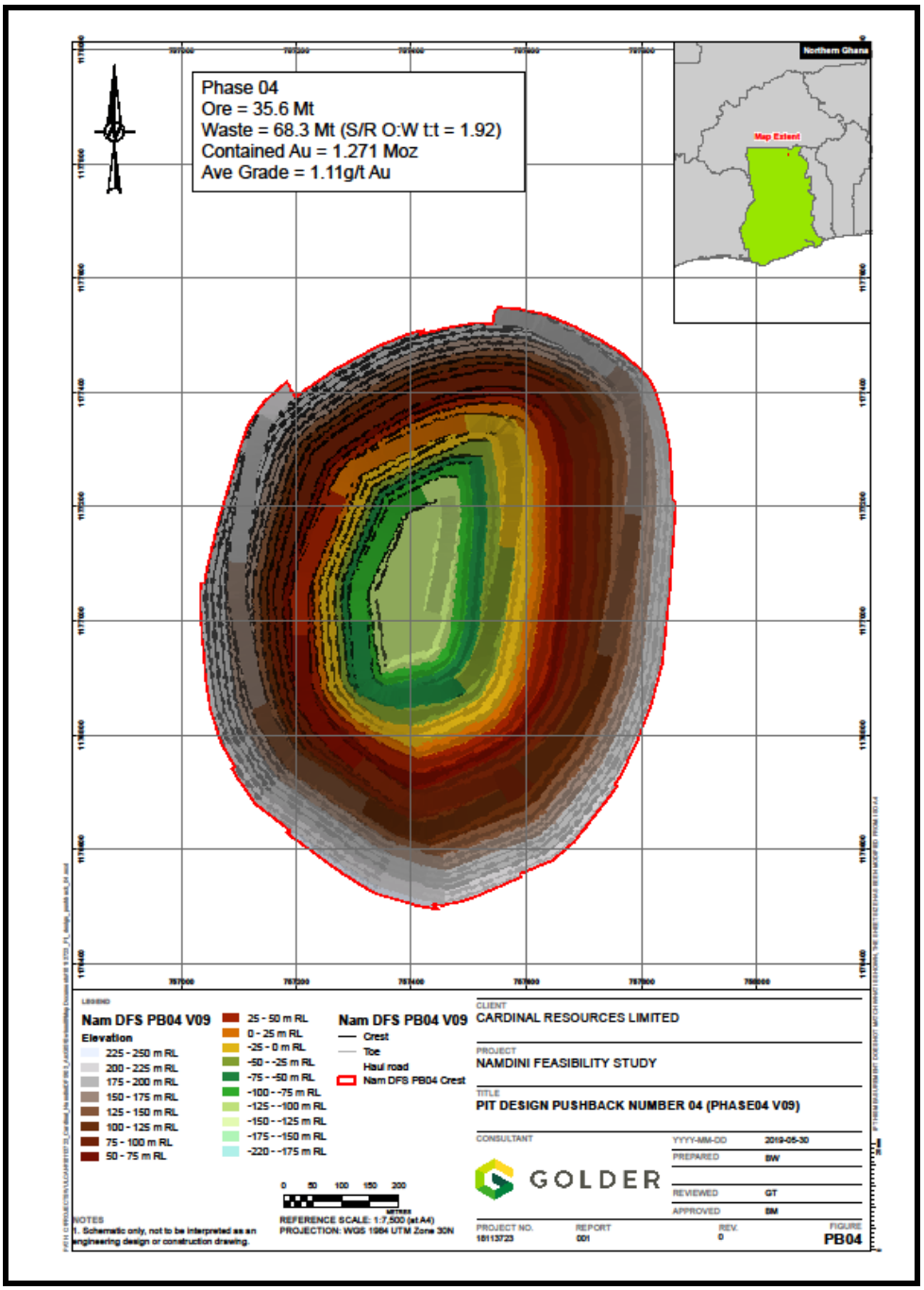


Figure 113: FS pit design Phase 4 *utm_nam_dfs_pb04_v09_1_cl.00t* – May 2019

The final Phase 5 (Figure 114) targets some 36 Mt of ore requiring the movement of an additional 145 Mt of waste rock. The increase in strip ratio is to be expected given the overall dip of the orebody to the west. The contained gold content is some 1,334 koz with an average grade of 1.14 g/t. Phase 5 reaches a Final Pit depth of 440 m from surface with a long axis (N-S) maximum dimension of 1230 m and a short axis (E-W) maximum dimension of 960 m.

The main waste rock access ramp for Phase 5 has been planned to exit on the western side of the pit, close to the waste rock dump to reduce overall haulage costs, whilst maintaining the eastern ramp system for ore delivery and as an alternative haul route for added security of operation.

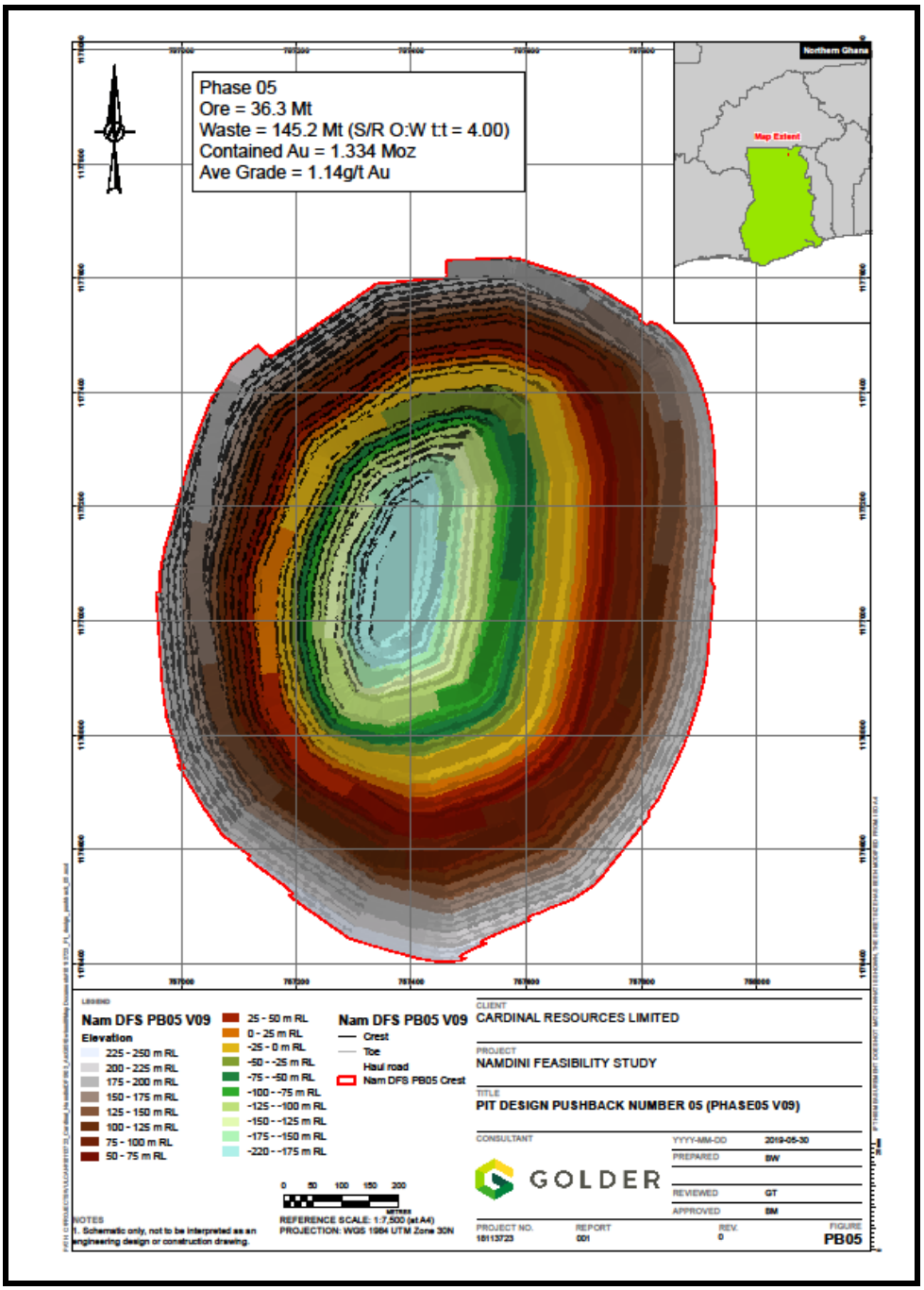


Figure 114: FS pit design Phase 5 Final Pit *utm_nam_dfs_pb05_v09_3_cl.00t* – May 2019

16.9.2 Ore selectivity within LOM Pit

The Namdini deposit can be regarded as a largely diffuse orebody with limited pockets of higher-grade zones. An analysis of the grade-tonnage curve shown in Figure 115 for the LOM Pit (Phase 5), indicates that there is a relatively low proportion of higher-grade material above 1.5 g/t Au. This type of grade-tonnage curve with a very flat curve for grades higher than 1.5 g/t Au shows that the ore attainable from selective mining would be minimal compared to the total available ore within the pit.

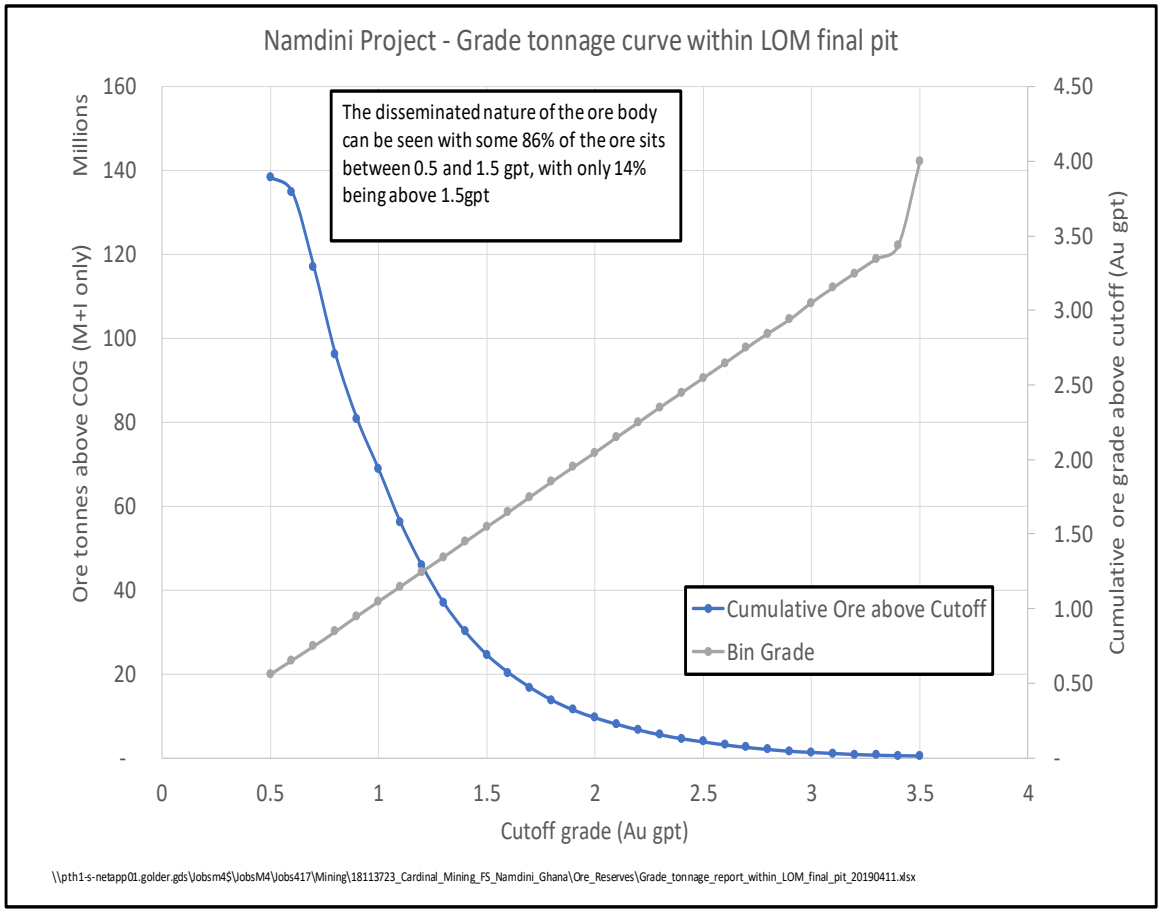


Figure 115: Grade-tonnage curve for Phase 5 Final Pit

The grade-tonnage profile for the Ore Reserve within the Final Pit design shows that some 86% of the Ore Reserve sits within a gold grade range of 0.5 to 1.5 g/t Au. Of the 138 Mt of ore within the Final Pit design, only 20 Mt of the total is above a gold grade of 1.5 g/t. The nature of the ore lends itself to bulk mining, with minimal edge dilution and ore loss expected at the edges of the demarcated ore zones during mining. The largely disseminated nature of the ore zone within the LOM Pit can be seen in Figure 116.

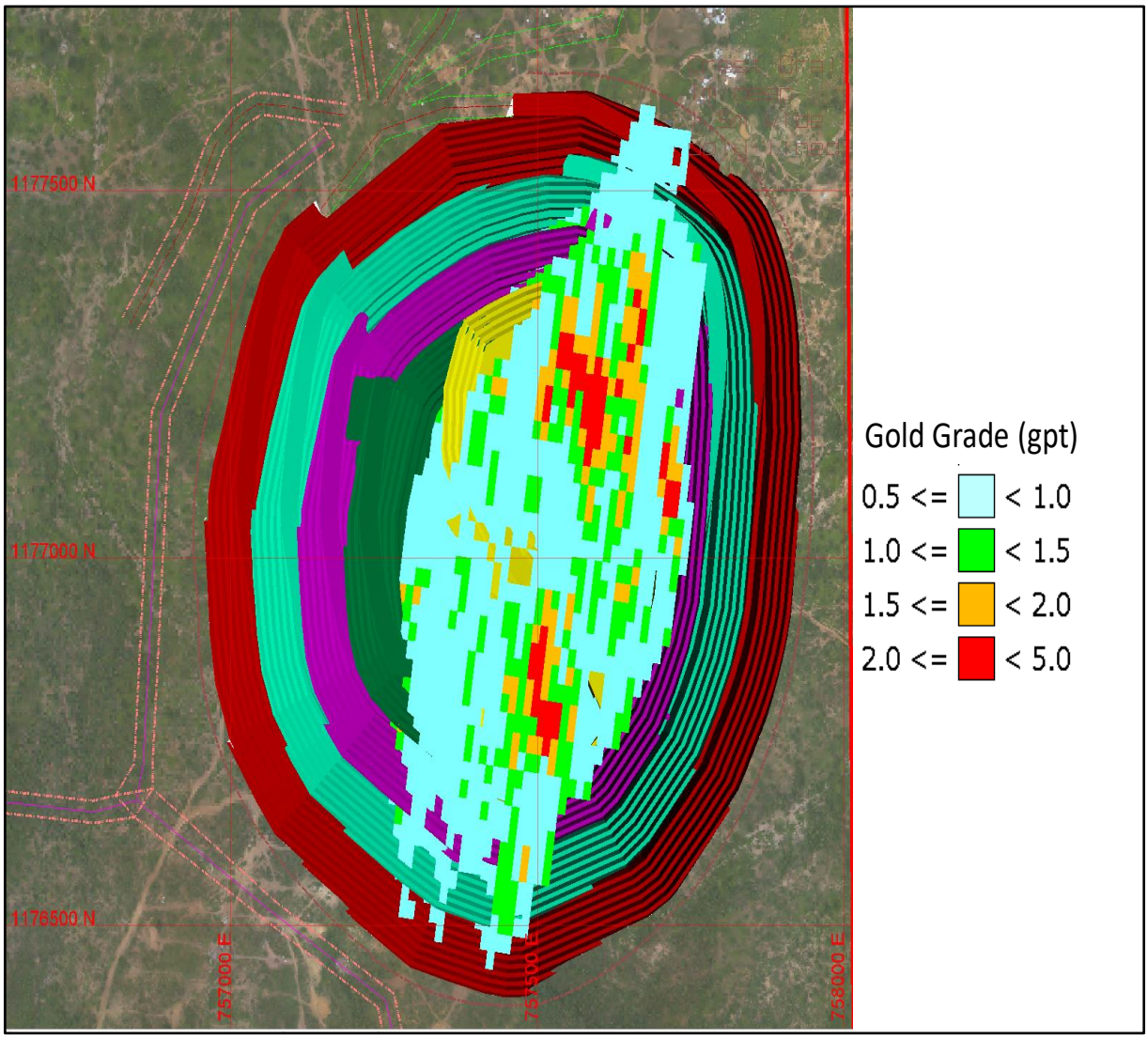


Figure 116: Plan view of Final Pit with grade >0.5 g/t Au (M+I only)

Figure 117 and Figure 118 show the position of the ore within the stages and the progression of the stages along the N-S axis and E-W axis.

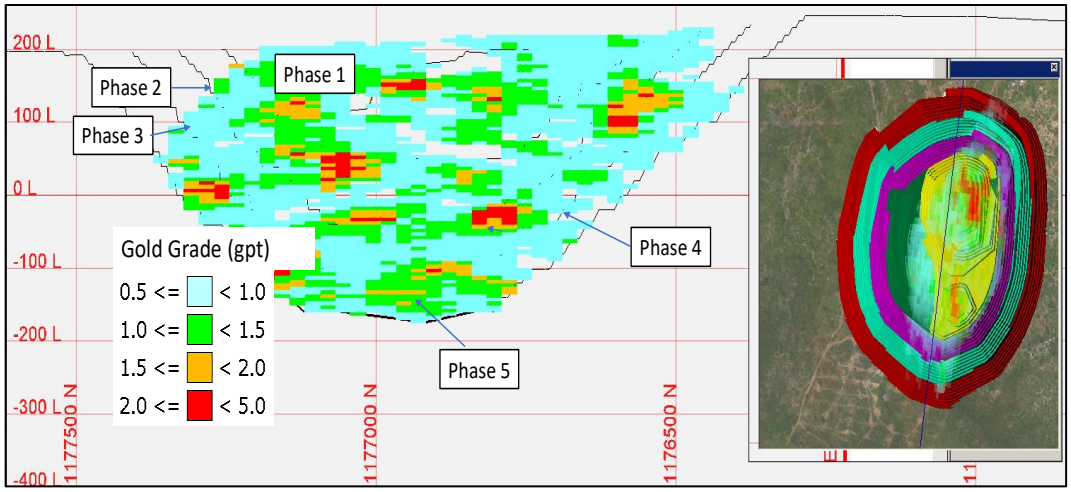


Figure 117: Section view of LOM Pit along N-S line looking to east – May 2019

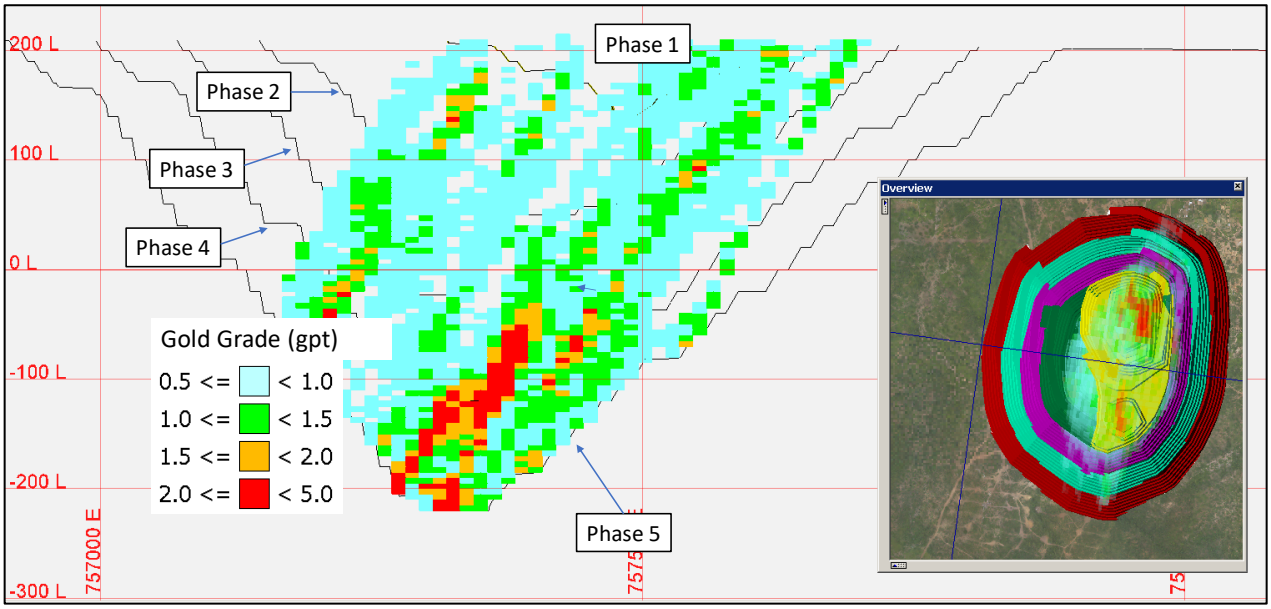


Figure 118: Section view of Final Pit along W-E line looking north, with each stage – May 2019

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 119.

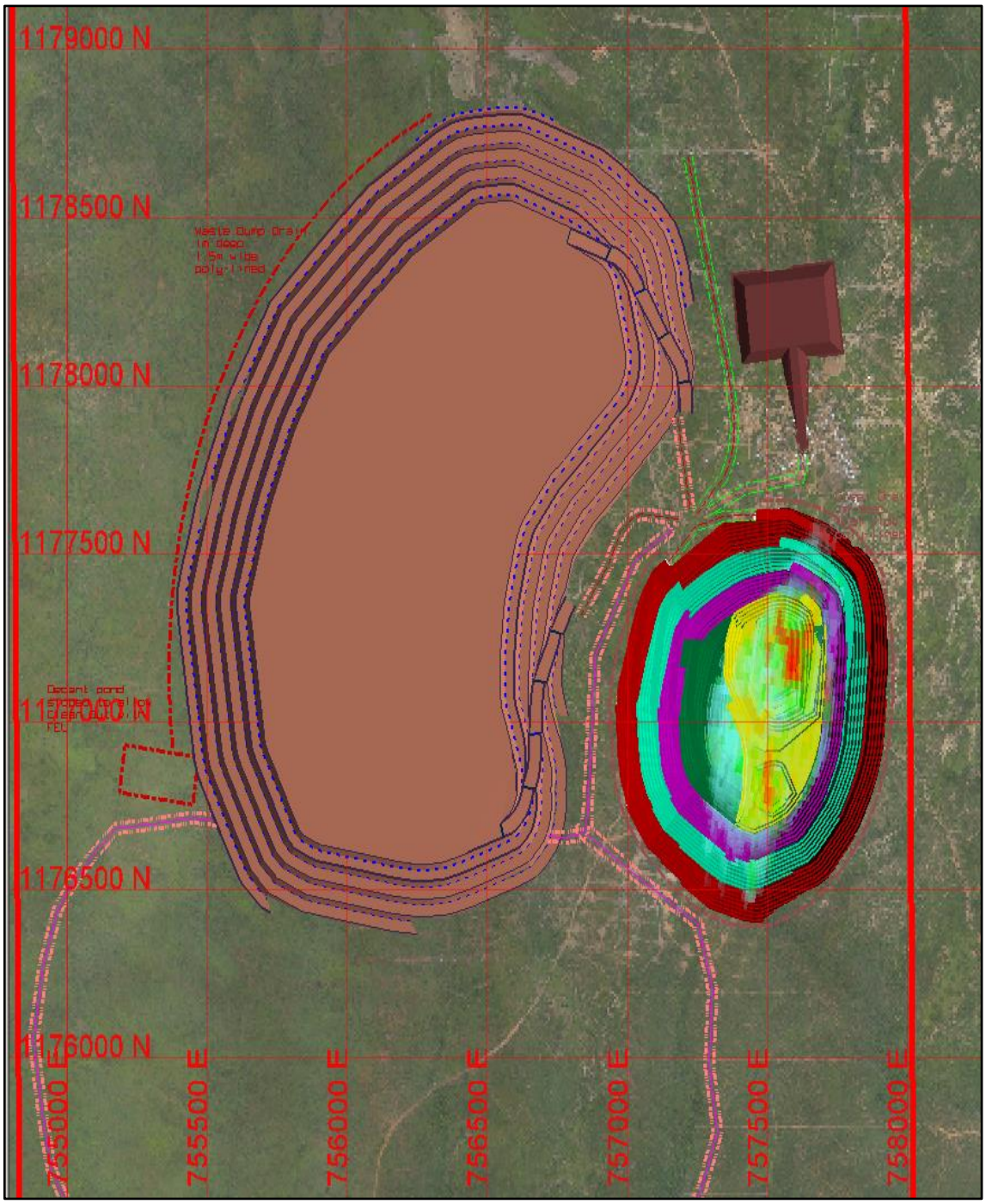


Figure 119: Final planned pit waste rock dump location – May 2019

16.9.3 Haul roads, stockpiles and waste dumps

Haul roads on the Namdini site comprise surface roads, in-pit ramps, stockpile ramps and waste storage access ramps. All roads are designed to the same two basic parameters: 1) the road width, which must allow safe passage of two haul trucks travelling in opposite directions on the road, and 2) the gradient, which must be shallow enough to allow the largest of vehicles to traverse them safely.

The operation will employ 134 t class rigid body dump trucks (Cat' 785 class or equivalent). All roads on site are designed with a width of 30 m 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).

16.9.4 Excavation assumptions

To date no specific trials have been completed to determine the large-scale amenability to bulk excavation of the material types found in the Namdini deposit.

In the absence of large-scale excavation results, the following assumptions are applied:

- In the Oxide and Transition zones, a mixture of free digging, ripping, drilling and blasting methods will be employed
- In the Fresh, competent material at depth, conventional pre-splitting, drill and blasting methods will be used to extract the ore and waste.

Topsoil quantities have been estimated for the areas that have to be cleared for mining and waste dump placement. The volumes and costs for removal are included in clearing and grubbing cost estimates within the mining cost model. An allowance of 0.3 m of topsoil thickness across the site has been assumed, with the terrain being relatively flat and of low agricultural quality. Clearing and grubbing will precede mining and waste dumping operations with the topsoil and seeds stored in controlled stockpiles to be re-used for progressive rehabilitation over the life of the operation.

16.10 Mining schedule

The mining resource model is an MIK model that contains an estimate at a number of cut-off grades (the indicators) of the proportion of ore and waste within each block.

The MIK resource model has included an allowance for some selective mining by adjusting the variance within the resource estimate (see Section 14.7).

During mine scheduling it is normal practice to add additional factors to account for unavoidable ore loss and dilution during ore block design and for mining recovery. This approach is not practical with the MIK method of estimation. The MIK resource model estimates the proportion of ore above a cut-off grade within a block, but not the spatial position of the ore within the block. Thus to attempt any edge dilution, which would normally be used on regular block models, the ‘edge dilution’ may not be honouring the movement of the ore and waste effectively.

In November 2017, Cardinal undertook a close-spaced drilling programme at Namdini to assess the representativity of the Mineral Resource Model and to trial the Grade Control Model being developed for the project. Cardinal drilled close spaced Reverse Circulation holes over the area that is targeted as the initial mining area. The RC drilling programme consisted of 317 holes (for 13,271 m) drilled at a 10 m × 10 m spacing covering an area of 350 m (E) by 200 m (N) to a vertical depth of approximately 40 m. The holes were drilled dipping 65^{o} to the east consistent with the resource drilling. A summary of the comparison (Cardinal, 2017) is reproduced in Table 144. This 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 Namdini deposit.

Table 144: Comparison between Mineral Resource Model and Grade Control Model

Resource Model (September 2017)				Grade Control Model			Delta
Cut-off grade (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

Note: Source: Cardinal, 2017

Given the close correlation between the Mineral Resource Model estimate and the Grade Control Model results, the Resource Model is considered a valid representation of an equivalent mining model for the proposed class and style of mining at the Namdini Project. The assumed SMU size represented by the resource model is 5 m east by 10 m north by 2.5 m deep.

The mine scheduling was completed using the Minemax Scheduler software using quarterly (Qtr) periods. The mining sequence has been designed to incorporate five mining Phases. The initial mining Phase targets a relatively small initial pit of higher-grade ore with minimal waste stripping requirements. The overall mining schedule is targeted towards the early generation of cashflow to assist in repayment of the project capital and to maximise project discounted value. The Minemax schedule was then used to back-flag the mining model to the quarterly periods mined. Subsequent to the flagging of the period mining sequence in the block model, the model was then used in the Alastri Tactical Scheduler (ATS) to identify bottlenecks and to allocate resources for detailed down-stream cost modelling.

The process plant will accommodate an annual feed rate of 9.5 Mtpa over a 15-year life (13 years of direct feed and two years of reclaimed stockpiles) including ramp-up. The ramp-up has been assumed to cover a three-month period within the first year. It is planned to treat 90% of annual throughput during that first year of plant operation. An initial pre-strip period has been 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.

The schedule was constrained to a maximum vertical sink rate of 80 m per year, with the mining Phases set to honour a minimum bench advance lead of three benches during any period.

The schedule targets the ore above the marginal cut-off grade, but uses the maximum recoverable value as a determination of process plant feed. Lower-grade material is stockpiled as long as the mill feed requirements for deliverable ore can be met.

The scheduler was set up to maximise net present value (NPV) whilst maintaining a reasonable smoothing of tonnage movement per year. The peak tonnage requirement over the period 2026 to 2033 is maintained at 36 Mtpa to allow mainly medium-grade and high-grade ore to be directly fed to the process plant, with the low-grade ore being stockpiled for treatment after primary mining has completed. Figure 120 shows the mining by Phase for ore and waste over the quarterly LOM schedule.

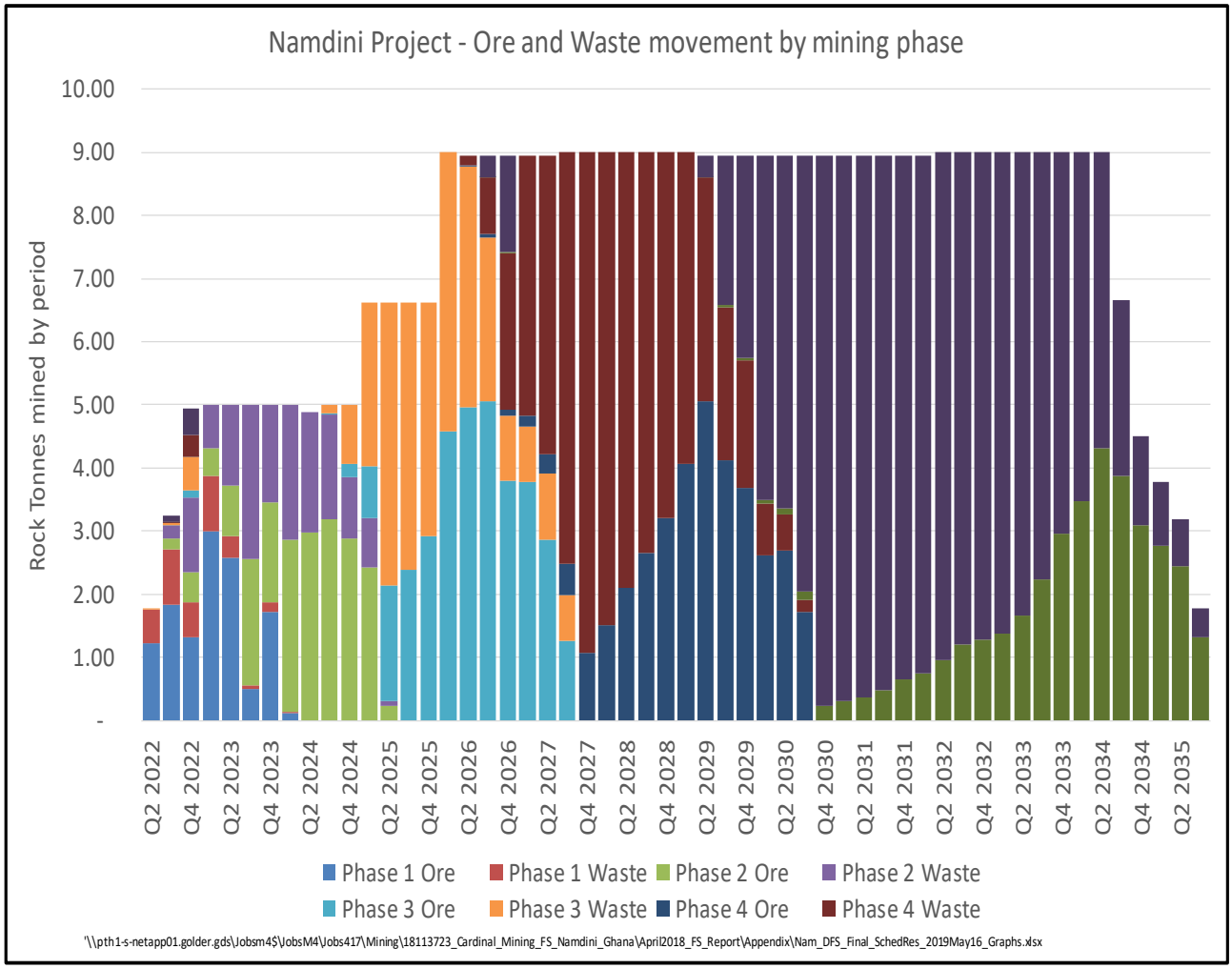


Figure 120: Rock movement by mining Phase (April 2019)

Table 145 shows the periods during which each mining Phase is able to provide ore to the process plant, with Figure 120 showing this graphically with the ore tonnes mined in each period.

Table 145: Ore access by Phase

	Phase 1	Phase 2	Phase 3	Phase 4	Phase 5
From	2022	2022	2024	2026	2030
To	2023	2025	2027	2030	2035

Figure 121 shows the ore mined by Phase over the LOM.

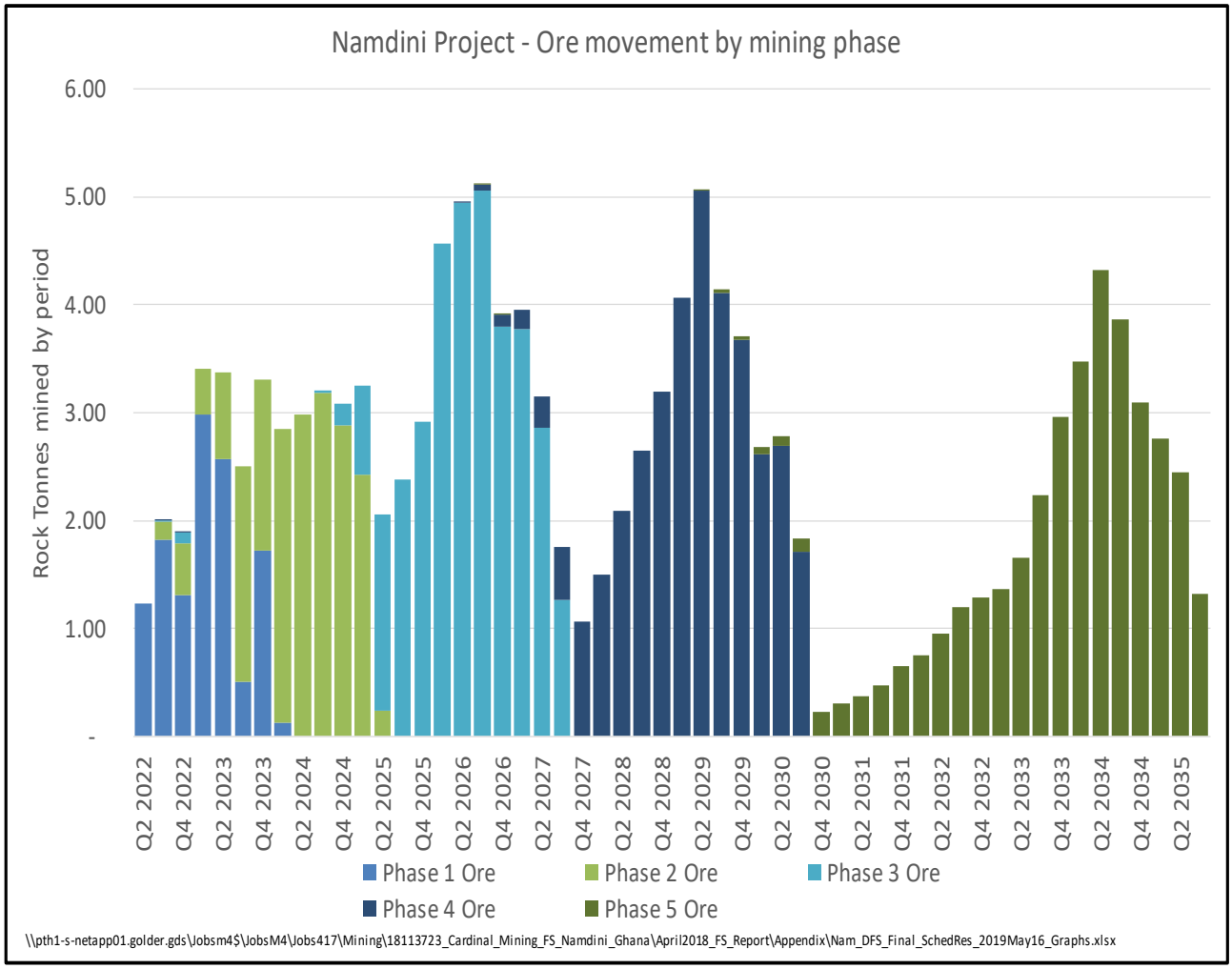


Figure 121: Ore mined by Phase over the LOM

The ore process plant feed by period can be seen in Figure 122, with the Oxide material being fed into the process plant at the end of LOM after the Transitional and Fresh material have been exhausted. There is sufficient stockpile capacity and no significant degradation of recovery efficiency for the material stockpiled is anticipated.

Relative throughput equivalent rates ("**TPEq**") were used in the scheduler to control the rate at which material could be fed to the process plant using a normalised 9.5 Mtpa rate. This ensures an optimal use of the mill capacity and maximises the throughput per period at the rated capacity. Figure 123 shows the constant nature of the mill feed using this method of constraint.

The granite and metavolcanics provide the bulk of the mill feed over the life of mine.

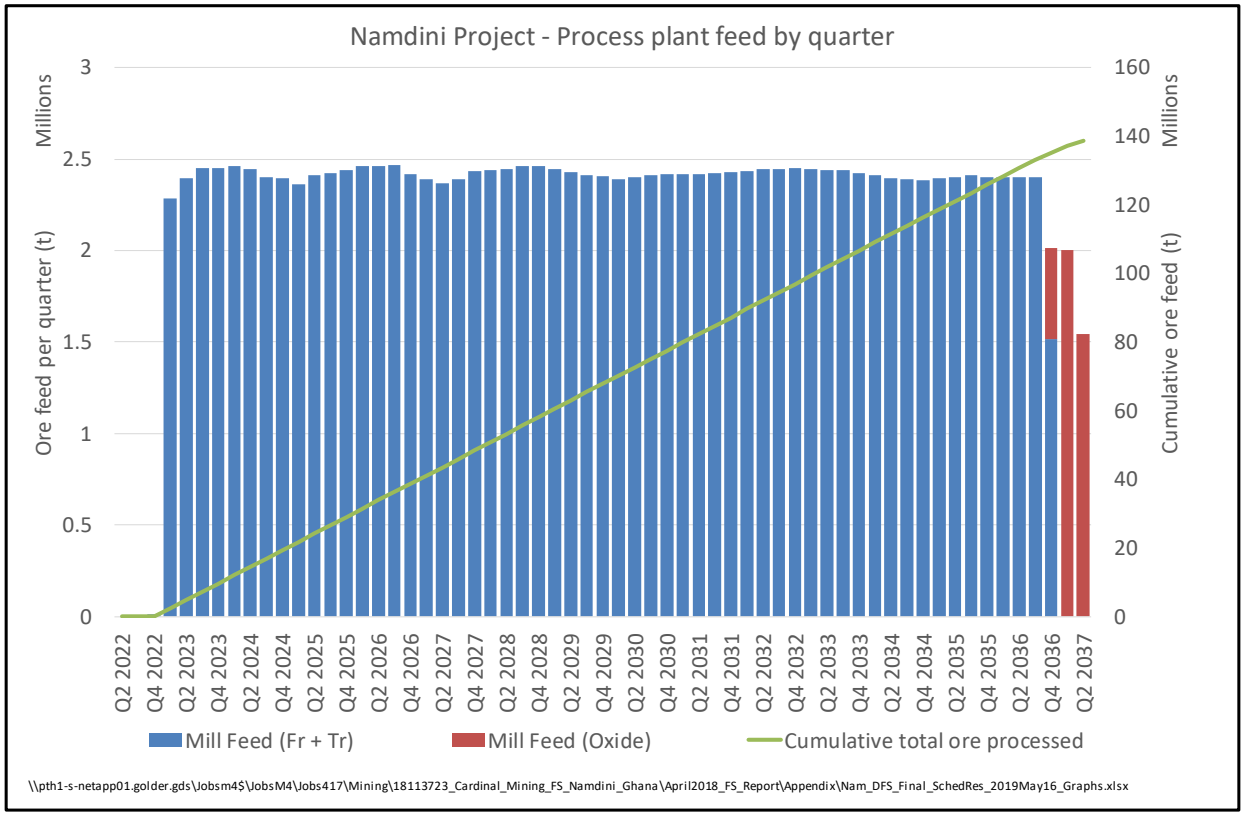


Figure 122: Mill feed by ore type over the LOM

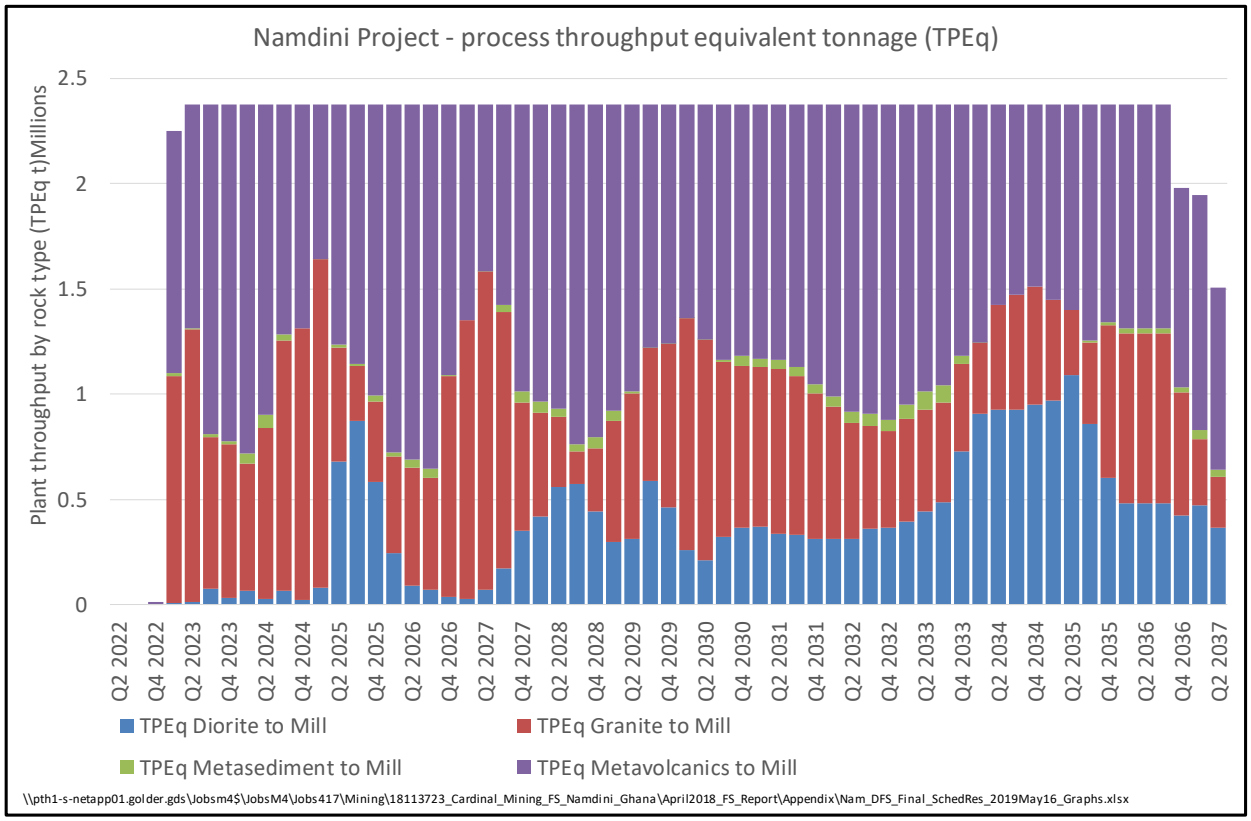


Figure 123: Mill feed by throughput equivalent feed rate per period

The Transition material at Namdini is only partially weathered (oxidized) and behaves in metallurgical test work similar to the Fresh material, with similar recoveries. The Transition material is to be processed with the Fresh material as part of the mill feed and thus these are combined in all the mine and mill schedules. The 15 year mine life (13 years of mining and then 2 years of mill feed from stockpiles) is the Life of Mine. The 4 Mt of Oxide feed will probably be milled in the last six months of those two years (Figure 125). The LOM excludes the Project build phase and closure.

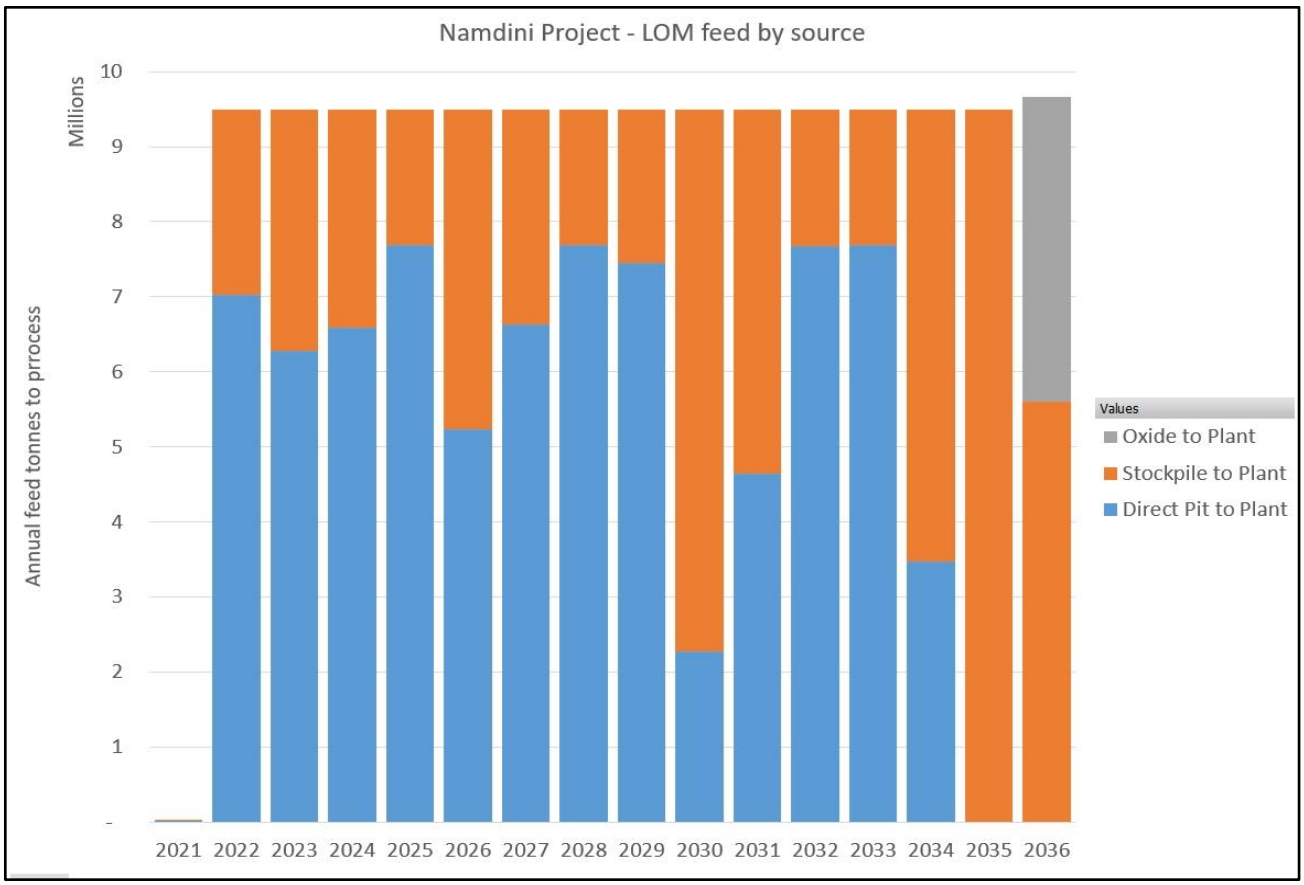


Figure 124: Mill feed by source and year, showing Oxide feed at end of LOM

16.11 Mining sequence and period progress

The mining sequence at Namdini consists of a relatively short period of higher-grade low strip ratio mining in the Phase 1 and Phase 2 pits. The Oxide ore is removed from most of the mining Phases during the early part of the schedule as this material is stockpiled for processing at the end of the LOM following process plant modifications. The sequence of the Phases is shown in Figure 125. Phase 1 commences operation in Qtr 2 of 2022 with some waste removal of Phase 2 commencing in Qtr 4 of the same year. Phase 3 mining operations will commence in late 2024 and last until 2027. Phase 4 mining will commence in 2026 and run through to 2030. Finally Phase 5 will commence in 2029 and continue until 2035. The low-grade ore is then reclaimed from the long-term stockpiles and finally the Oxide material will be processed from 2035 to the end of life of mine in Qtr 2 of 2037.

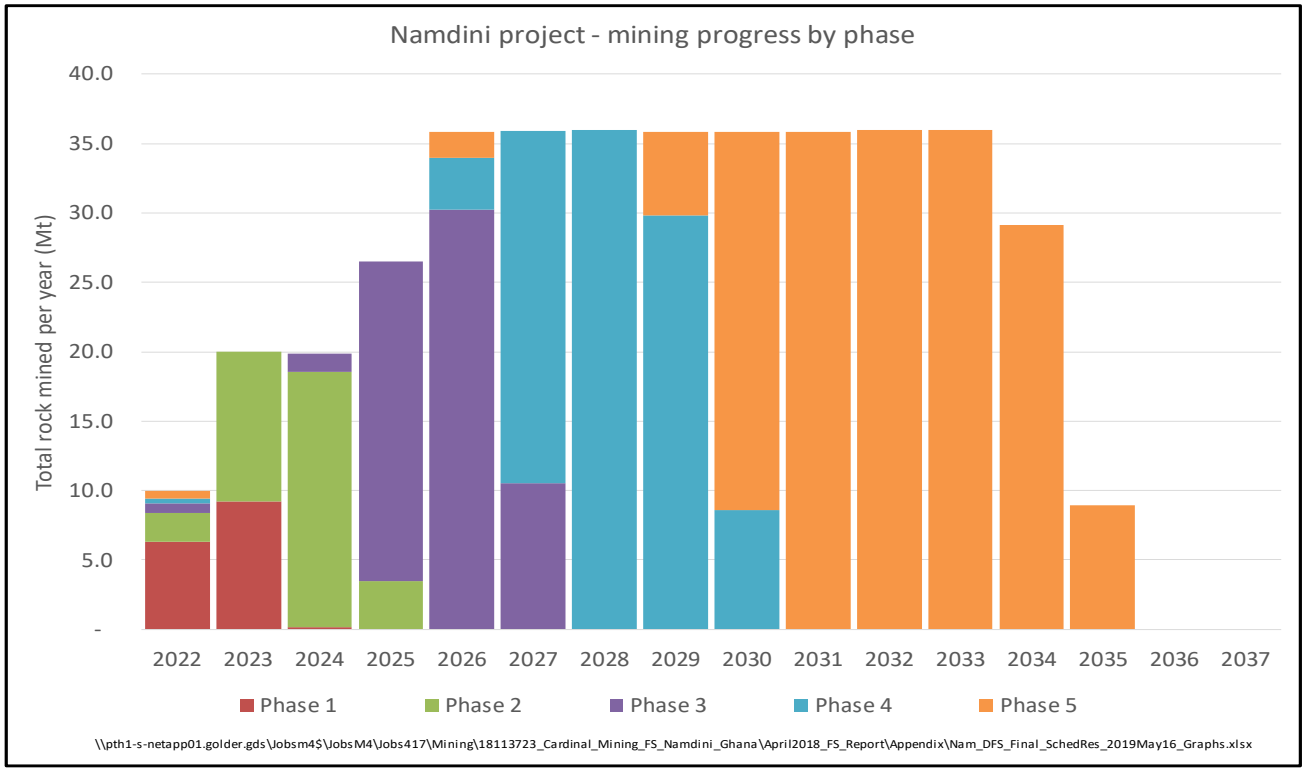


Figure 125: Mining Phase progression by year

Period progress plots for the scheduled mine plan have been produced showing the general progression of the mining Phases. The initial stage (Phase 1 & 2) will commence in the region of highest grade near-surface ore. The stage will require the removal and stockpiling of the Oxide material to allow access to the Transition and Fresh ore for mill feed. The first two mining Phases will last approximately four years, with commencement of Phase 3 two years after the start of mining operations. Figure 126 shows the first and second Phase mining sequence for the periods Qtr 1 to Qtr 7.

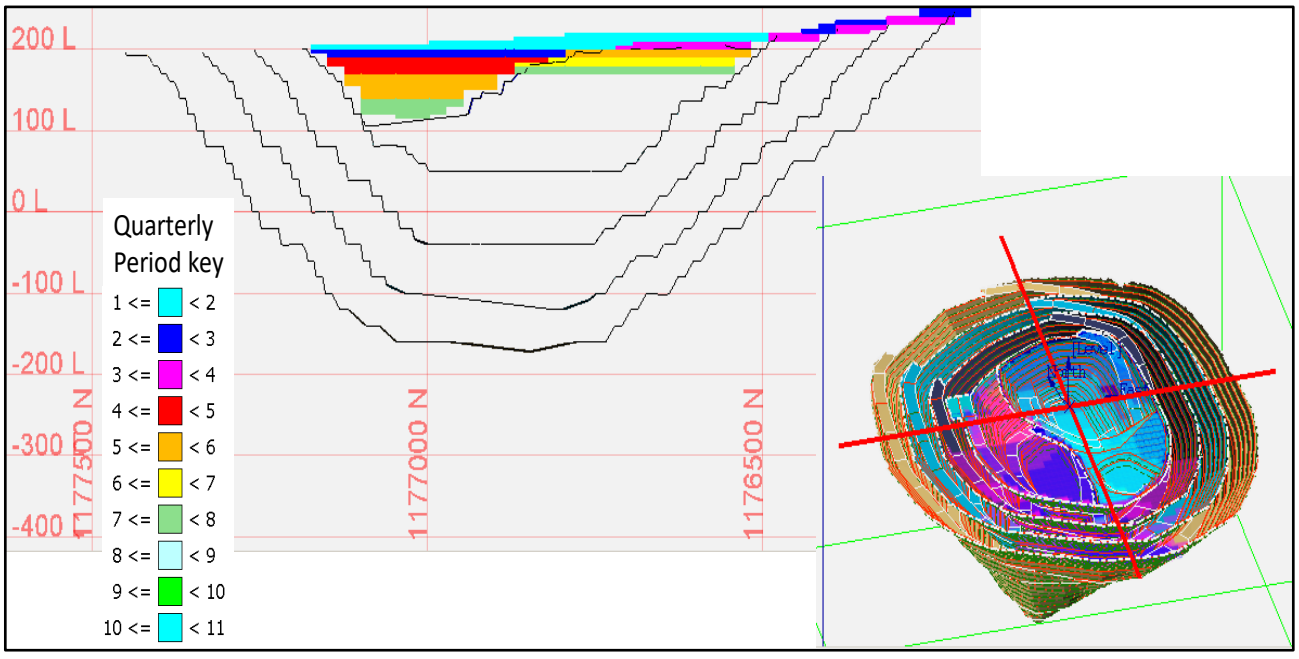


Figure 126: Period progress in Phase 1 Qtr1 to Qtr7 – May 2019

Mining of Phase 3 will commence some 3 years after start of the mining operations. Figure 127 shows the mining progress for the periods Qtr 8 to Qtr 20. The ore from Phase 2 forms the majority of the mill feed during this period while the waste removal of Phase 3 is progressing.

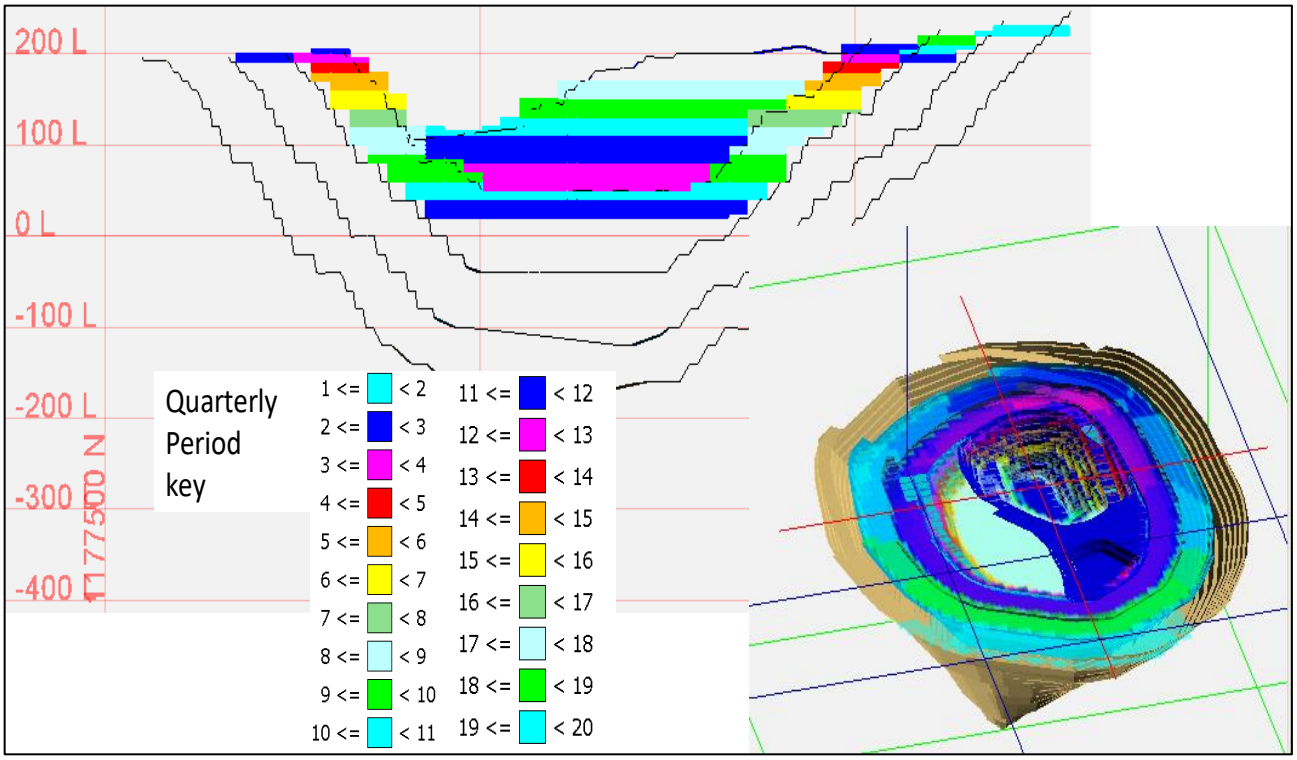


Figure 127: Period progress in Phase 2 and Phase 3 periods Qtr8 to Qtr20 – May 2019

Mining of Phase 4 will commence some 5 years after the start of the mining operations, with Phase 4 lasting five years. Figure 128 shows the mining progress plots for the period Qtr 20 to Qtr 30.

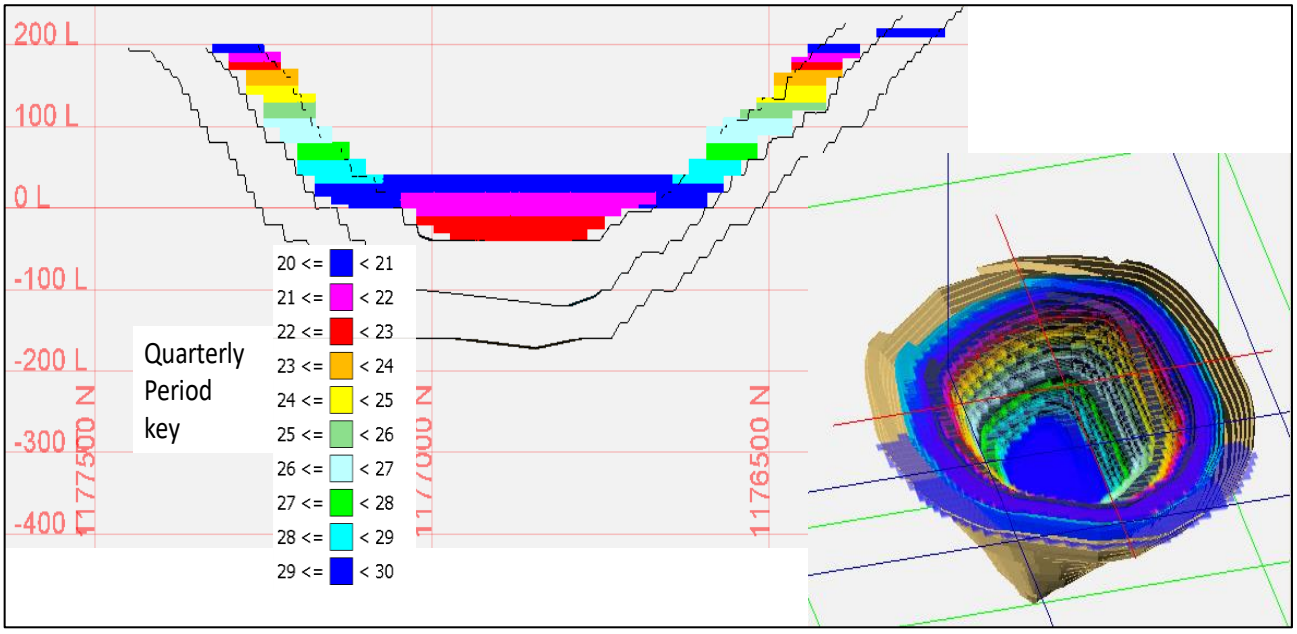


Figure 128: Period progress in Phase 3 Qtr 20 to Qtr 30 – May 2019

Mining of Phase 4 will commence in 2026 as Phase 3 is being completed. Phase 4 lasts some four years requiring a total rock movement of some 104 Mt. The initial stage of the Phase 4 mining cut is shown in Figure 129, with the remaining parts of the Phase 4 mining cut being shown in Figure 130.

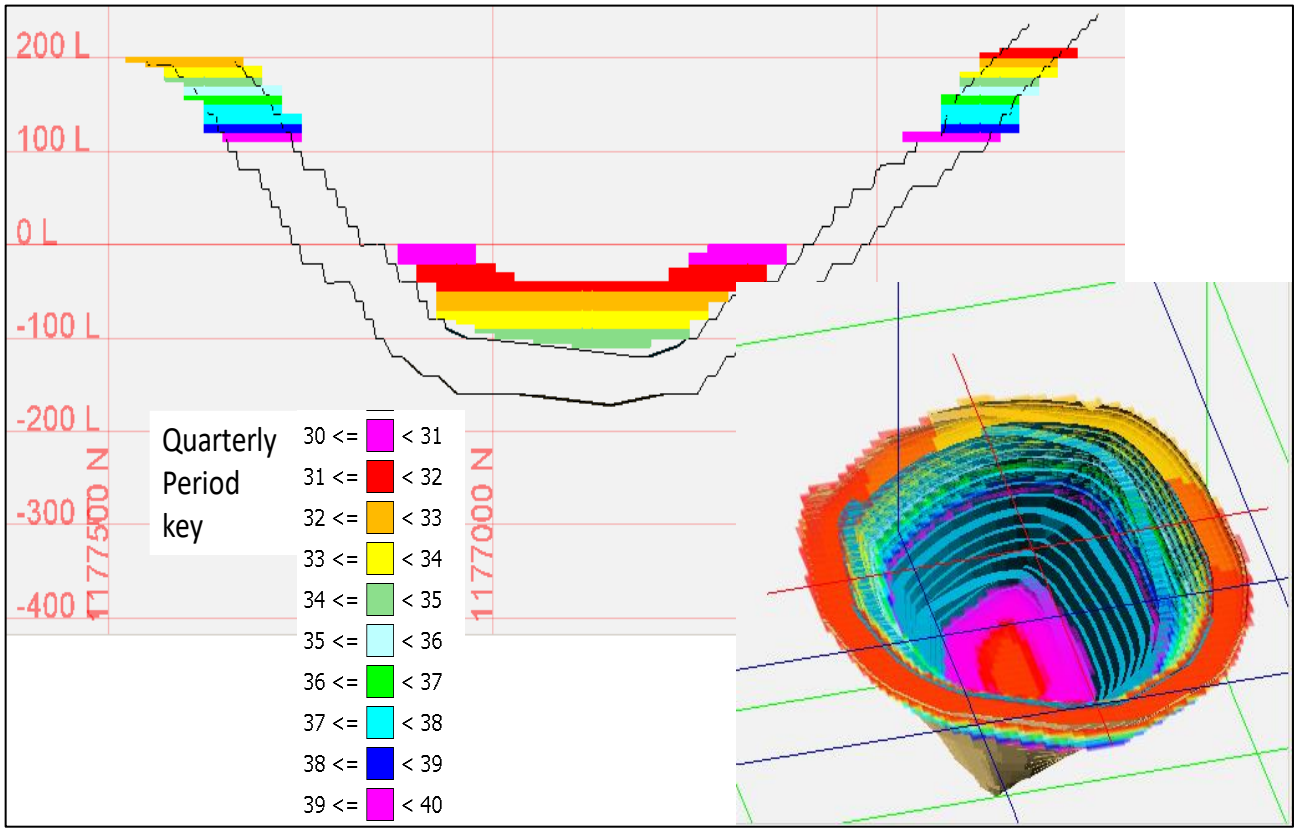


Figure 129: Start of the Phase 5 mining operation and period progress Qtr 30 to Qtr 40 – May 2019

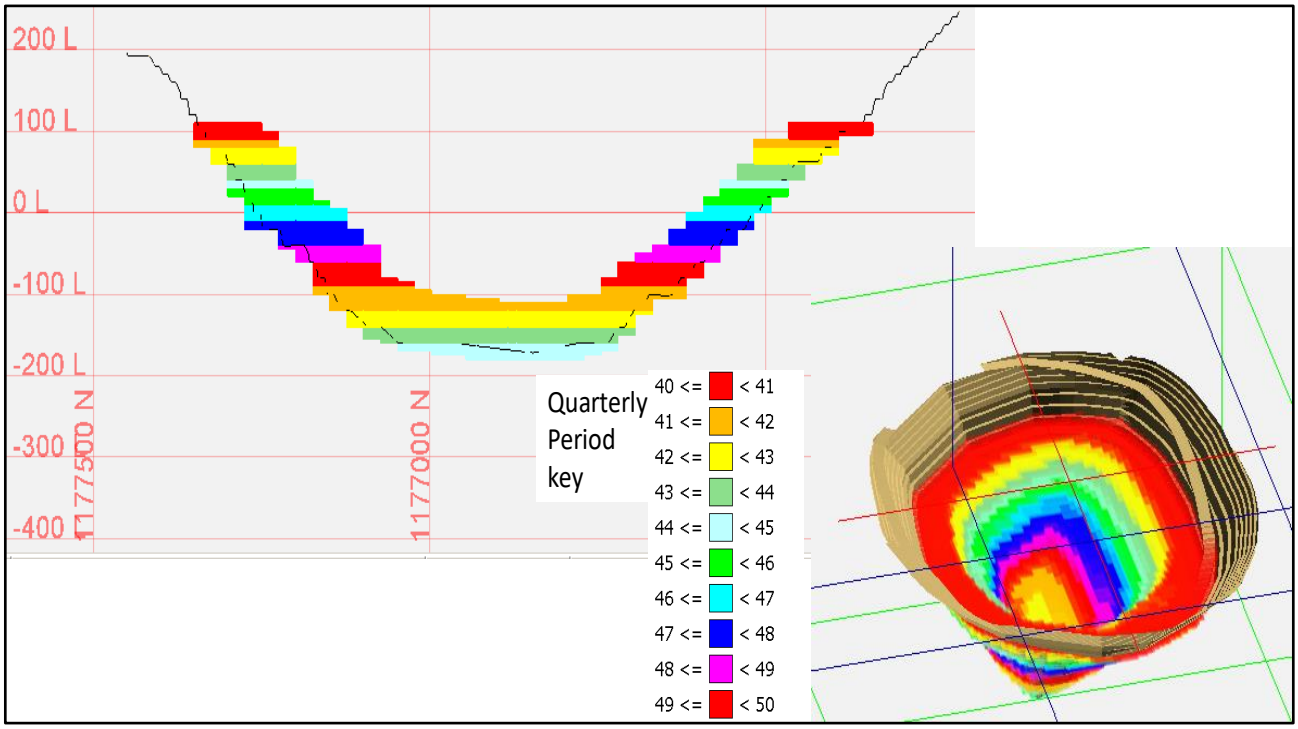


Figure 130: Latter stages of the Phase 5 mining cut – May 2019

16.12 Mining equipment

The Minemax Scheduler provided the period progress mined blocks by quarter from commencement of mining through to completion of the Phase 5 mining cut-back. The mined blocks were then imported into the Alastri Tactical Scheduler to identify fleet requirements based on detailed haul route analysis. The ATS allows rapid analysis of haul route paths and identifies potential congestion areas that are not visible at the strategic scheduler level.

The adoption of a tactical level scheduling tool for the FS of the Namdini Project is a key driver in identifying operational cost benefit areas and minimising mining spend to maximise project value.

An estimate of the truck profile for ore and waste by destination is shown in Figure 131. The maximum fleet requirement is for 27 of the 134 t rigid haul truck class. Mining will be carried out on 10 m benches, with three flitches allowed for in the ore and single bench bulk mining of waste where possible.

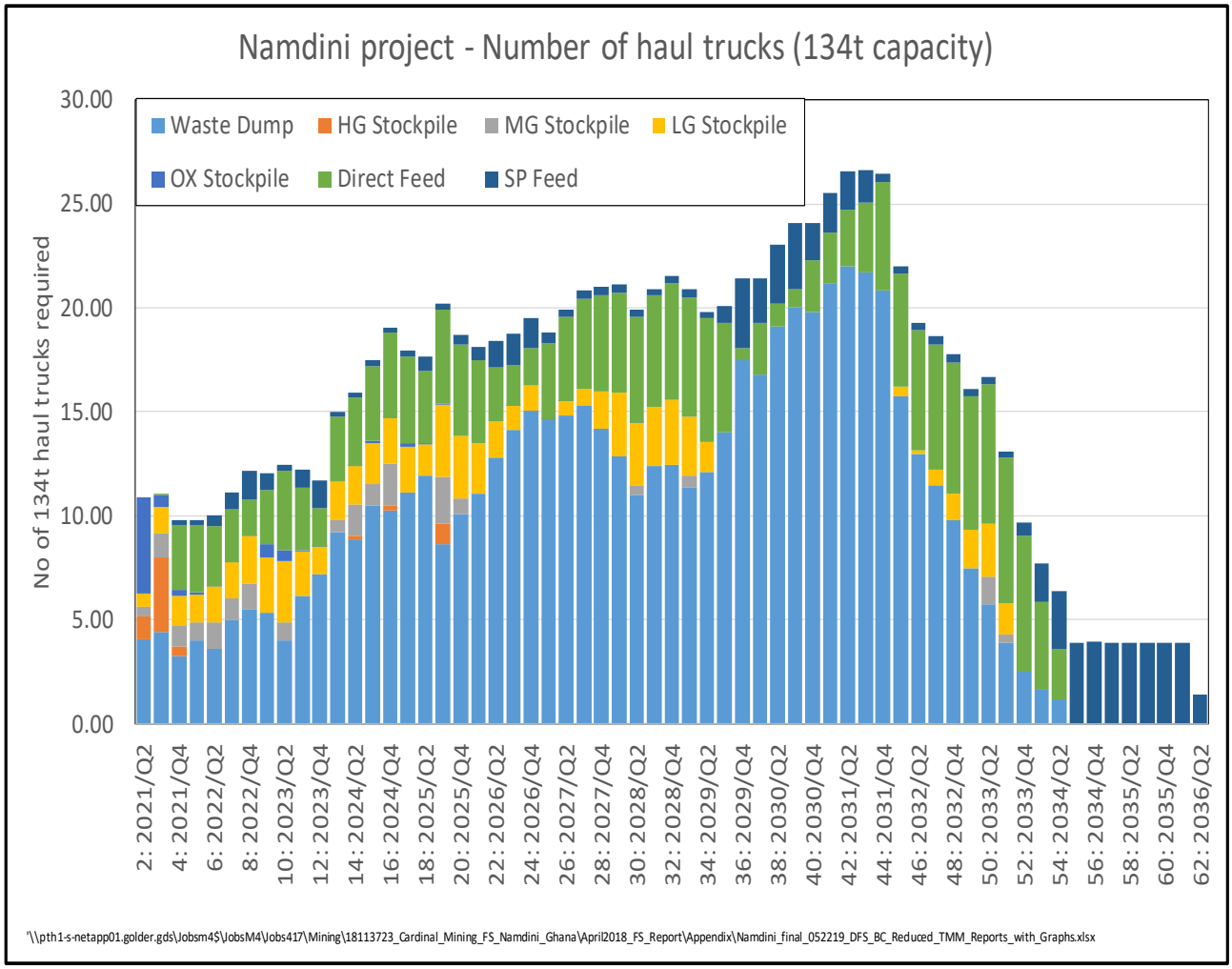


Figure 131: Haul truck requirement by period

16.13 Mining capital cost estimates

Cardinal has planned that the Namdini Project will be mined on an outsourced contract basis with capital equipment costs for the mining infrastructure and mining fleet amortised and charged back to Cardinal as part of the cost per tonne mining costs. Initial mobilisation and infrastructure costs to establish site facilities have been estimated for the mining contractor.

The ATS scheduling tool has been used to estimate the total required digging hours and hauling hours for the LOM based upon dig space and material destination. The equipment requirements were then used in a zero-based cost model to allow for comparison of owner versus contract mining costs for tender purposes. The detailed cost model approach will assist with adjudication of the mining tender process that will follow the Board decision to take the Namdini Project to the next stage. One added advantage of the detailed cost model is the ability to compare an owner mining alternative approach to the contractor mining option.

The maximum peak fleet requirements for the Namdini Project are shown in Table 146 with an expected maximum 27 (134 t capacity) rigid haul trucks and four shovels, three in face shovel configuration and 1 in back-hoe configuration.

Table 146: Peak equipment requirements for the mining fleet

Item (typical size/class)	Number
Excavator (Cat 6040 face-shovel)	3
Excavator (Cat 6020bh)	1
Rigid haul truck (Cat 785)	27
Large FEL (Cat 993)	2
Track dozer (Cat D10)	6
Pre-split drill (DML D65)	4
Medium FEL (Cat 992)	1
Production drill (Epiroc PV235)	6
Rubber tyred dozer (Cat 834)	2
Motor grader (Cat 16M)	3
Fuel tanker and water tankers (Cat 777)	4
Lighting plants	10
Light vehicles	20
Sump pumps	10
Explosive truck (PLTS Contractor)	2
Tyre handler (Cat 988) (Tyre Contractor)	1
Maintenance truck	2
Shift change bus	2
Compactor	1
Lowbed & prime mover	1
Crane	1
Tyre handler/forklift	1
Bobcat	1
Ambulance	1
Fire truck	1

16.13.1 Power supply

Electrical power requirements for the mining operation will be minimal, with most mining activities relying on diesel powered engines.

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

No specific heavy-duty road access or heavy-duty bridge requirements have been identified separate to the project infrastructure requirements assessed by Lycopodium and Knight Piésold.

16.13.3 Mining methodology

The Namdini Project has been evaluated by Golder based upon a conventional hard rock mining equipment fleet. There are no indicated requirements for untested or novel mining techniques.

16.13.4 Pit dewatering

Pit dewatering requirements have been identified as part of the hydrological study for the project. All mining has assumed a dry (de-watered) pit configuration. The requirement to maintain a de-watered operation is a critical aspect with respect to both safety and operational success. Provision for pit de-watering within the Oxide and Transition layers with horizontal drain holes has been allowed for in the mining plan (Section 16.3.2).

The pit will be ringed by a surface drain to capture rain and stormwater and lined to minimise inflow back to the pit. Sump pumping will be catered for by the provision of sump pumps and highwall piping as is typical in conventional open pit mining operations. Pit water will be decanted into silt ponds with excess water being supplied to the process plant.

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, with the latter generally being outsourced to a specialised explosives supplier. There are several reputable international explosives suppliers currently active in Ghana and supply is not a risk. Explosives contracts are typically awarded on a Prime Load Tie and Shoot ("**PLTS**") basis with the explosives contractor providing both the explosives magazine and manufacturing facility for the explosives emulsion mixing on site. The down-the-hole explosives trucks and all supporting infrastructure will be supplied by the successful PLTS contractor employed under the mining agreement.

16.13.6 Summary of mining 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, with the exception of the mobilisation and essential infrastructure components. The essential infrastructure components include the truck wash-down facilities, the fleet workshops, site offices and mining administration facilities for the mining contractor.

The contractor capital cost estimates include an allowance for the following items:

- Establishment to site
- Diesel power generators (if identified in other studies)
- Diesel tank farm
- Miscellaneous site buildings
- Mining equipment
- Haulage 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.

16.14 Mining operating cost estimates

Golder approached several leading mining contractor service providers in Ghana and has received budget estimates for the provision and supply of mining operations at the Namdini Project. The budget estimates have been used to provide project cost estimates for the mining operation over the life of the operation. Ghana currently has a very active mining contractor supply base. Specialist skills and core competencies in the mining contractor companies will be key considerations in the final selection criteria.

A detailed mining movement schedule was supplied to 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 five mining Phases. Detailed budget estimates have been provided covering the entire mining operation for the project. The estimates have provided an initial all-in contract mining cost similar to that used in this FS. Further discussions and negotiations will continue with suitable mining contractors as part of an official tender request prior to any award of the mining contract.

The budget estimate source data has been incorporated by Cardinal in the overall project economic model.

16.14.1 Load and haul

The budget cost estimates provided to date have included a detailed cost, by material type by mining bench, for each mining Phase. The mining costs for the load and haul have been compared to cost estimates calculated from the in-house cost model and ATS detailed schedule. The load and haul component costs supplied within the budget estimates compare favourably with the in-house cost model estimate. The incremental cost with depth, by 10 m bench, is as expected for this type of operation. The cost per bench for the waste loading and hauling operation component is shown in Figure 132. These costs compared favourably to those calculated with the TALPAC truck and loader simulation package and used for the PFS of the Namdini Project.

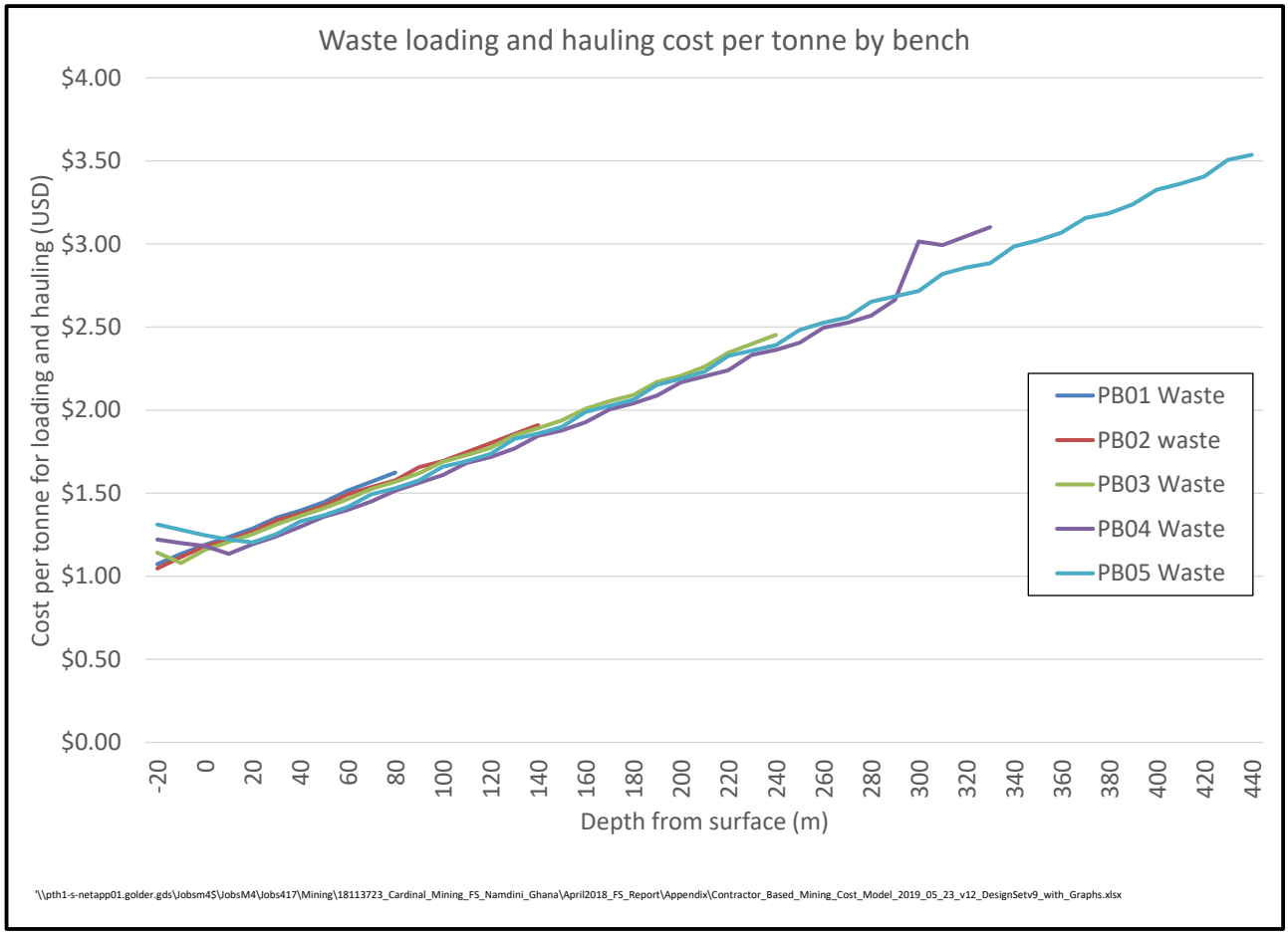


Figure 132: Incremental cost analysis by bench

16.14.2 Drill and blast

Drill and blast cost estimates have been supplied as part of the mining contractor budget estimates and have been compared to those derived from the mining cost model. It is expected that the blasting requirements for the waste rock will be relatively modest given the softer nature of the Metasediments in Northern Ghana compared to traditional hard rock environments in the Southern Ghana goldfields. The blasting requirements for the ore have been retained at a relatively high powder factor to ensure maximum throughput of the blasted ore at the primary crusher. Drill and blast costs are split into production, presplit for wall control, blast monitor movement balls and drain holes.

The drill and blasting configuration for the Namdini Project is shown in Table 147. The costs will form part of the overall mining contract services, with the blasting operations typically outsourced to the explosives supplier under a PLTS sub-contract.

Table 147: Namdini Project – drill and blast estimate

Standard pattern – staggered	Waste	Ore	Unit
Spacing	5.5	5	m
Burden	5	4.5	m
Depth	10	10	m
Rock density	2.75	2.75	t/m^3
Hole depth	10.5	10.5	m
Subgrade	0.5	0.5	m
Hole diameter	0.2	0.17	m
Explosive charge per metre	36.1	26.1	kg
Stemming length	6.2	4.5	m
Charge length	4.3	6.0	m
Powder factor (by rock tonnage)	0.20	0.25	kg/t
Emulsion density	1.15	1.15	t/m^3
Rock mass per hole	756	619	t
Powder factor (by rock volume)	0.56	0.7	kg/m^3

16.14.3 Operational mining costs

The contract mining cost for the Namdini operation is inclusive of drill and blast, load and haul and all associated mining operations with the exception of the mine planning, mine scheduling and grade control components which have been retained by Cardinal. The mining support functions, including mine planning, grade control, geology mapping and mine survey fall under the technical services structure of Cardinal and have been budgeted for as part of the Owner's costs within the overall Project cost model.

A comparison of the budget estimates over the LOM is shown in Figure 133 for ore and waste movement. A similar graph showing the split by main mining function is shown in Figure 134. This shows that the load and haul component accounts for some 50% of the total mining budget, as expected and in line with similar operations of this type and scale.

A breakdown of the mining operating costs is provided in Table 148.

Table 148: Breakdown of mining operating costs/t ore processed for LOM of 138.6 Mt

Item	Cost/t milled (US$)
Variable mining cost – Waste	5.08
Variable mining cost – Ore	3.53
Fees	1.68
Services	0.30
Total	**10.59**

The variable mining cost for ore and waste includes drill and blast (i.e. production drill and blast, presplits, blast movement monitors and drains holes), load and haul, road and other ancillary works. The variable mining cost for ore also includes rehandle of ore from stockpiles to the ROM and crusher.

Management fees include contractor fixed monthly management fees, supervision, explosives management, training, dewatering, health and safety, freight, communication, vehicles, transport and other general costs.

Services includes the mine dispatch system, contractor mobilisation and demobilisation, survey, dewatering, technical services running costs and other costs (e.g. consultants).

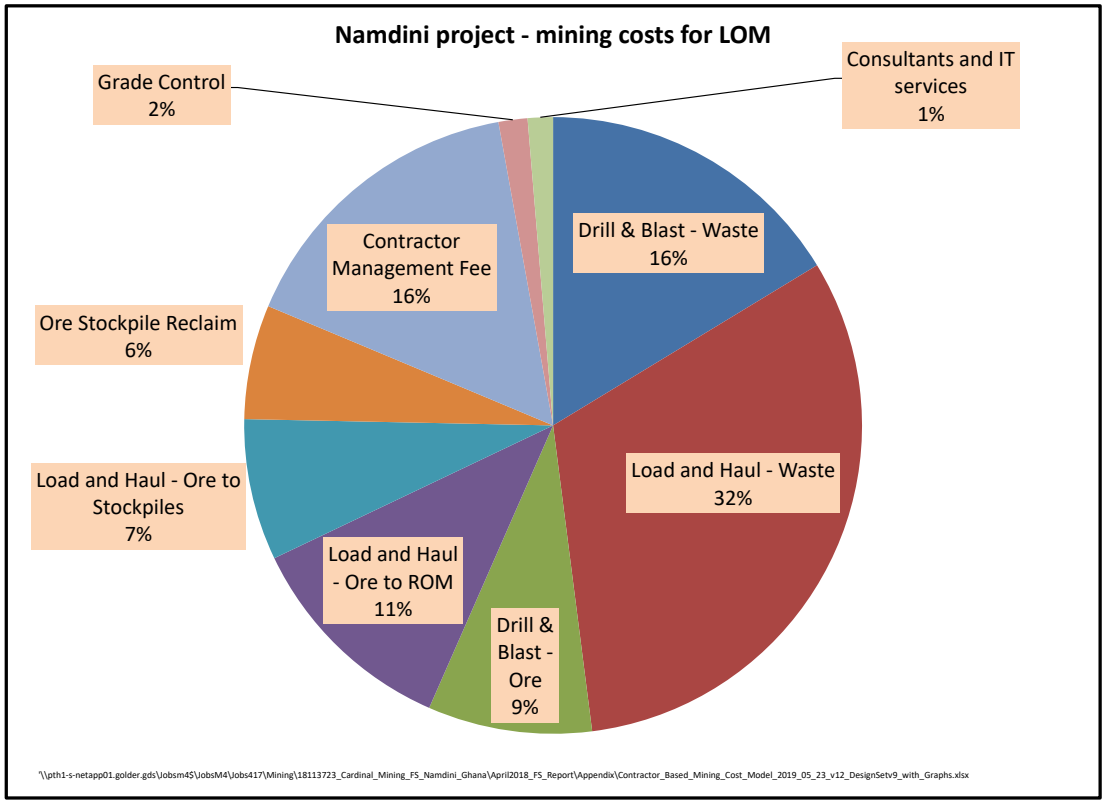


Figure 133: LOM breakdown of mining costs for the Namdini Project

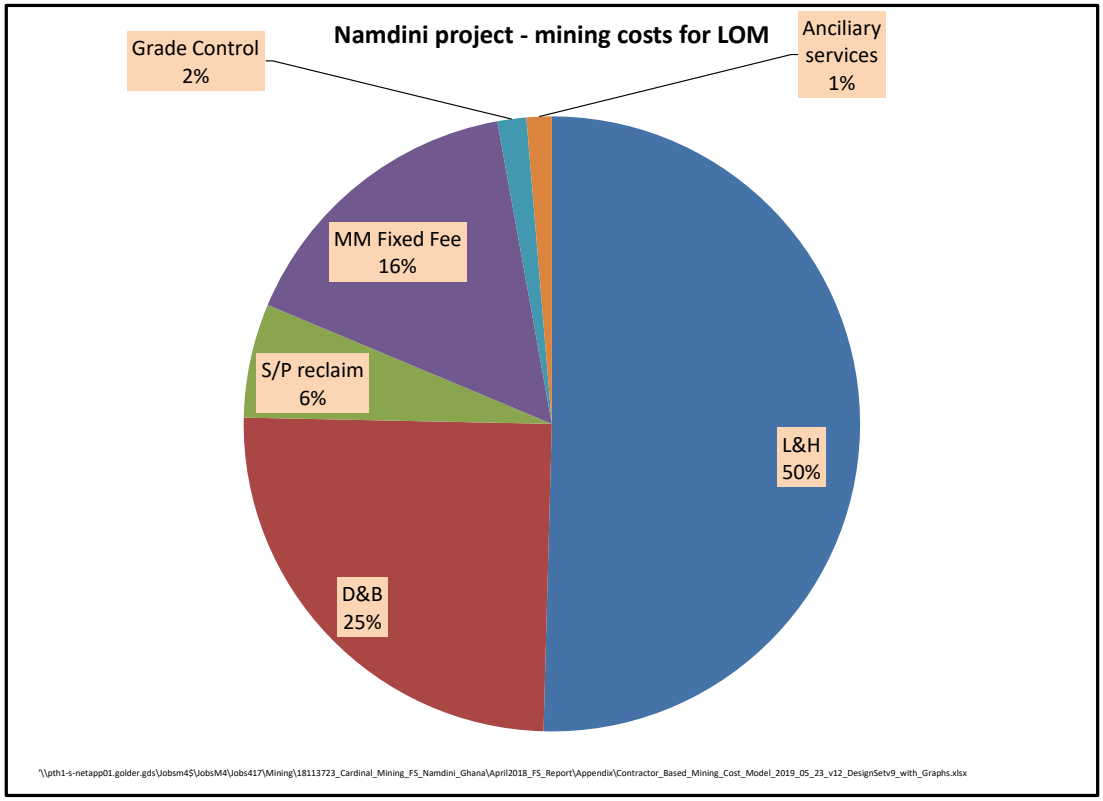


Figure 134: LOM breakdown of mining costs by function for the Namdini Project

16.14.4 Topsoil management

Topsoil will be removed from areas requiring disturbance and, when possible, immediately used for rehabilitation of final landforms. Where immediate use is not possible the soils will be stored separately in dedicated stockpiles of maximum 10 m height for future rehabilitation requirements. Final stockpile locations will be determined during operations based on quantities recovered and the ultimate site layout.

16.14.5 Waste rock dump

Waste rock from the pit will be disposed of at a designated waste rock dump located adjacent to the pit, and will also be used to construct the ROM pad and external walls of the TSF. The waste dump has been sited as close to the pit as safely possible taking into account the necessary width for infrastructure around the open pit. Approximately 100 M lcm of waste rock will be disposed of at the waste rock dump. The dump will be monitored on an ongoing basis in accordance with relevant environmental requirements.

The waste rock dump associated with mining operations will be constructed to meet the requirements of the Environmental Protection Agency of Ghana (the Akoben initiative and guidelines). The dump will initially be constructed with the natural rill angle of approximately 37 degrees (Figure 135), which is the angle of repose of the dumped waste rock material. This is then to be contoured progressively to an overall slope angle of 17 degrees (1:3.5) to allow for slope stability and re-vegetation (Figure 136). The waste dump will be progressed by tipping from a higher level against a windrow and progressively pushing the waste out with a dozer.

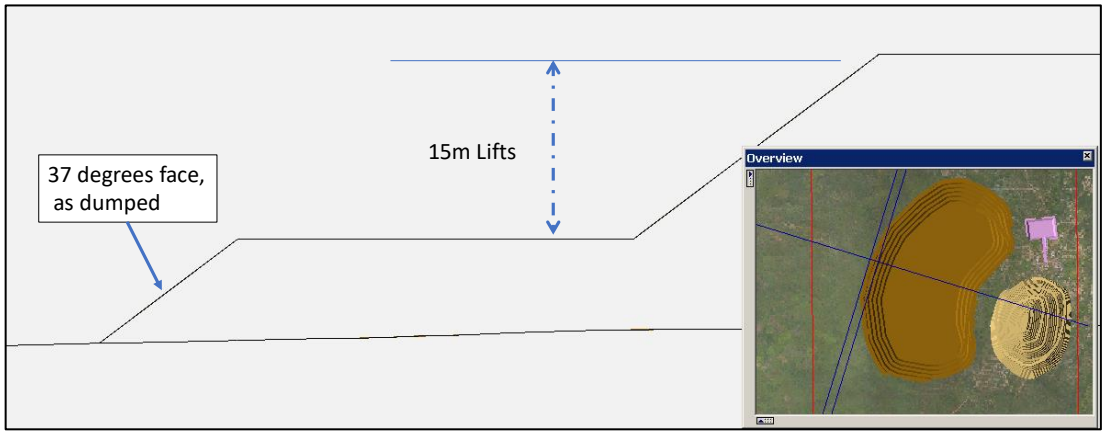


Figure 135: Waste dump section showing 15 m lift height

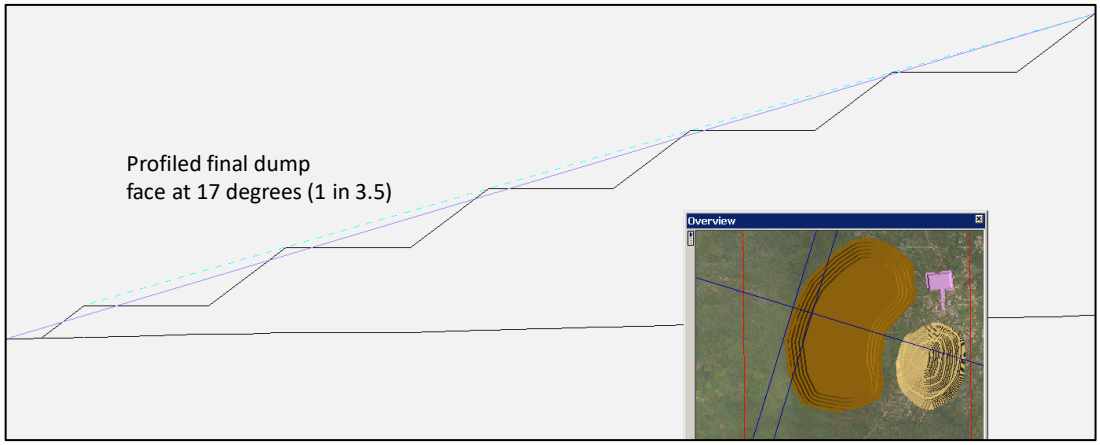


Figure 136: Profiled waste dump section with final slope angle of 17 degrees (1 in 3.5)

The waste rock dump will be progressively rehabilitated with topsoil, where possible. The surfaces of the waste rock dump will be contoured to minimise batter scour and ripped at 1.5 m centres to a depth of 400 mm where practicable. All such rehabilitation work will be carried out progressively. Seepage and shallow groundwater flow along the perimeter of the waste rock dump will be controlled with suitable toe drains.

Selected waste rock will also be used for the construction of the ROM pad, TSF walls and other infrastructure items during the site construction phase and for further TSF wall lifts during the life of mine.

Good mining practice dictates that mine sites be rehabilitated to a sustainable state after the mining operation reaches completion. Hence, all areas impacted by the project will be stripped of topsoil before commencement of construction. This topsoil will be stockpiled for progressive rehabilitation and future rehabilitation work at the end of the Project life.

There are a number of factors that need to be considered in evaluating the best option for a waste storage area. They include:

- Storage capacity
- Stored material
- Visual impact
- Haulage distance costs
- Site drainage
- Site access and preparation.

As result of the relatively gentle slope away from the pit on the western side of the deposit, the ability to fill this area in close proximity to the pit with minimum overall disturbance was considered a key factor that addressed the issues listed above. The planned maximum dump height of the waste rock dump at Namdini is 55 m above topographic elevation.

The overall position and final profile of the waste rock dump is shown in Figure 137.

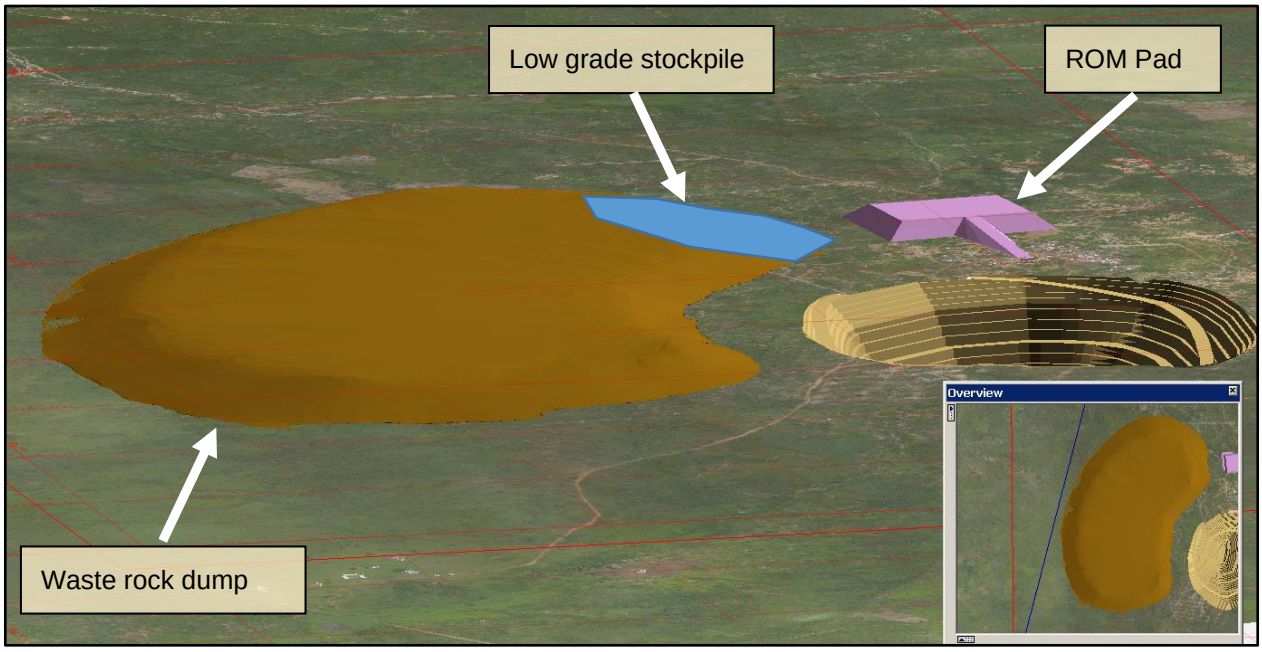


Figure 137: Final waste dump profile and footprint

Table 149: Waste rock dump and low-grade ore stockpile combined capacity for the Namdini Project

Waste Rock Dump Lift Name	Dump Lift Elevation from – to (m)	Note	Volume as lcm (M m^3)	Capacity at 2.0 t/lcm (Mt)
WD_250	250-265	final lift	25.9	51.8
WD_235	235-250	main lift	30.4	60.7
WD_220	220-235	pit crest level	34.8	69.5
WD_205	205-220	below pit crest level	36.3	72.7
WD_190	190-205	below pit crest level	31.7	63.5
WD_175	175-190	valley fill	11.2	22.4
WD_160	160-175	valley fill	0.0	0.0
Total capacity (including low-grade ore stockpile)			170.3	340.7

An allowance within the waste dump footprint includes some 80 Mt of low-grade ore which will be stockpiled over the life of the mine and then reclaimed and processed towards the end of the mine life. The total required waste rock capacity is 265 Mt (130 M lcm). An additional 40 M lcm temporarily required for stockpiling the low-grade ore.

16.14.6 Run of Mine pad

The ROM pad will be constructed using waste rock sourced from the open pit. The pad will be built up in stages over the mine development period to support the crusher chamber construction. Selected low-grade and marginal material may be tipped off the southeast side of the ROM for additional storage.

The final ROM pad will have a footprint of 44 Ha and will be constructed to a height of approximately 33 m. Approximately 3 M lcm of material will be required to construct the pad.

16.14.7 Mine operations and management

Mining activities will be conducted by a mining contractor with technical and managerial direction provided by Cardinal. The proposed mine operations model has the following advantages:

- Minimisation of upfront capital expenditure requirements by Cardinal
- Application of contractors' specialised open pit mining knowledge, systems and experience to lower operational risk.

The general mining method is summarised as follows:

- Clearing and grubbing of suitable material from all disturbed areas into discrete stockpiles
- Drilling and blasting of ore and associated internal waste on two flitches within the 10 m benches, while bulk waste which is outside the ore envelope is mined on 10 m benches. Trim blasts and pre-splits will be used to provide wall control in Fresh rock as required. The explosives usage is bulk emulsion, with all explosives supply provided by a subcontractor
- Loading and hauling utilising 400 t excavators and 134 t capacity haul trucks mining on 3 m high flitches in ore zones and bulk mining of the waste zones
- Grade control sampling will be provided by drilling on a 10 m by 15 m pattern of RC drill holes, using a subcontractor, and will be campaigned during the mine life (see Section 16.16)
- Ore blocks will be marked out by paint or tapes on the ground, supported by dedicated ore spotters as required
- Ore will be direct fed to the crusher or placed on stockpiles for future rehandle as required
- The waste dumps will be developed in 15 m lifts and progressively rehabilitated

- The TSF embankment will be constructed and raised with waste material from the mine as required
- Ancillary plant support for floor control, haul road construction and maintenance, rehabilitation, drill support, waste dump battering and the like will be provided by a fleet of dozers, graders and water carts
- Pit dewatering is expected to be minimal and will be managed by collection of water by in-pit sumps for use within the mining operation
- Crusher feed is provided by a combination of direct tip from the mine (90% of crusher feed) and rehandle from ROM or long-term stockpiles using either a front end loader ("**FEL**") only or a FEL and 134 t capacity haul trucks (depending on haul distance).

16.14.8 Technical services

A site based Cardinal technical services team will provide short-term and long-term technical direction and support to the operations in the following disciplines:

- Mine management
- Survey
- Geological control and grade control
- Mine planning
- Mine despatch
- Geotechnical monitoring
- Information management.

Manpower requirements to sustain the mining operation were estimated by Lycopodium in the Namdini Project cost model. The estimated manning level for the Cardinal mining department is 45 during the main mining period. Contractor manning will ramp-up from an initial 68 to around 240 with peak contractor numbers of 260.

16.15 Ore selectivity

16.15.1 Mining selectivity assumptions

The 2019 MIK resource estimate uses an assumed SMU size of 5 m by 10 m by 2.5 m and an information effect component based on assumed RC grade control drilling on slightly larger grid. This was produced by applying a local change of support to the MIK estimates using the Direct Lognormal Correction method for change of support.

The 2019 resource model assumes a relatively high level of mining selectivity considering the proposed production rate of 9.5 Mtpa of ore feed from expected total annual rock movements in excess of 25 Mtpa.

To test the impact of a change to a 10 m bench height, the block model was regularised to a 10 m panel size vertically.

Figure 138 compares the current (MIK SMU) grade-tonnage curve (5 m by 10 m by 2.5 m) to those curves for a 5 m panel (12.5 m by 25 m by 5 m) which is the e-type estimate, and a 10 m panel (12.5 m by 25 m by 10 m) which is the bench estimate. At a 0.5 g/t Au cut-off, grade there is about a 10% difference in both tonnes and grade between the MIK SMU estimate and the 10 m panel case. As expected, extra dilution is included causing an increase in the tonnage and reduction in the grade.

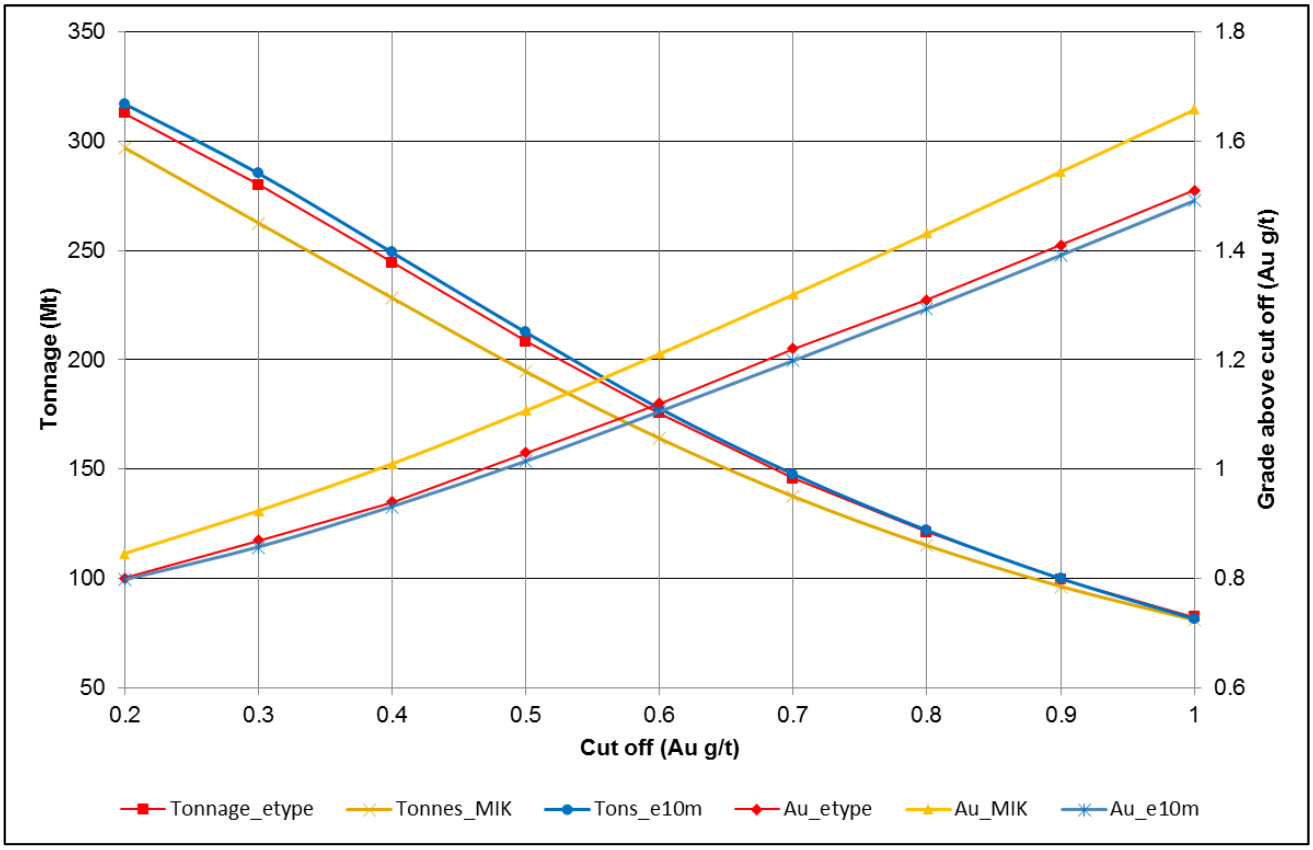


Figure 138: Grade-tonnage curves for the MIK (SMU), 5 m panel (e-type) 10 m panel (bench) estimates

Given the proposed blasting on 10 m benches, the planned selectivity may not be achieved. In Golder's view 10 m full-bench bulk mining would not be supported by the current SMU resource model.

Flitch mining in three flitches on the 10 m bench appears necessary particularly near grade boundary changes or waste/ore boundary changes. Bulk mining of portions of the waste is considered supportable.

16.15.2 Mining recovery

The Namdini Recoverable Mineral Resource estimate is expected to provide reasonable 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. Dilution relates to:

- SMU size
- sample spacing
- small scale continuity
- cut-off grade.

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 5 m by 10 m by 2.5 m, which has been applied to the low-grade large-tonnage disseminated Namdini deposit. The chosen mining equipment is of medium scale (134 t trucks and 400 t face-shovels) and so less amenable to selective mining.

At an RC grade control pattern of 10 m by 15 m with 1.5 m sampling, the short-term model confirmed the MIK Recoverable Resource estimate based on the Resource drilling. The 10 m by 15 m grade control pattern is a suitable grid for defining the ore zones at the mining scale anticipated.

The Namdini deposit presents as a largely diffuse ore zone with limited high-grade selectivity options. The area of highest grade potential has been chosen to commence the Starter Pit, given the positive results at very low gold prices. The degree of mining selectivity is however markedly less than one would normally see in similar massive gold deposits.

The grade-tonnage curve in Figure 115 shows the ore within the Final Pit region. This indicates that the Namdini Project will be less suited to a selective mining approach and more suited to a bulk mining option.

16.16 Grade control considerations

Recommendations of requirements for controlling the mined ore tonnes and grade during mining ('grade control' or more comprehensively 'ore control') were provided to Cardinal by Golder (2019c).

The feasibility study is assuming open pit mining by conventional load and haul methods feeding a process plant at a rate of 9.5 Mt per annum. Proposed mining cut-off grades for the future open pit do not conform to any observable geological feature and are determined by economic assessment. Future grade control will therefore require sound data collection, modelling, and tracking procedures to ensure mineral processing requirements and financial objectives are met.

Open pit gold mining operations like Namdini need accuracy at all levels of mine planning to ensure mill feed remains within specified tolerances and long-term value of the deposit is realised.

To prevent the operation from having a reactive operating mode, which would lead to decreases in mining efficiency, the Project will need to acquire good data for short-term mine planning. This is best achieved through the collection of grade control samples by RC drilling, collecting sample information three to six months in advance of production.

Mine planners will need access to a hybrid model that incorporates the latest grade control models with the long-term resource model. The potential value to the operation by having this data and information should not be underestimated.

Procedures to be considered should include:

- Golder (2019c) recommended that sampling should be by RC drilling on a 10 m by 15 m pattern with 1.5 m down-hole sampling. Holes should be orientated -60° towards the east and vertically cover four 10 m benches (48 m down-hole). Extending the depth of some drill sampling Figure 139 has some advantages in reducing the number of poor-quality collar samples, provides data below the benches of immediate interest, reduces modelling edge effects and progressively in-fills the exploration data and improves the accuracy of short- and medium-term mining models.
- In the resource estimation process (Section 14.7) it was assumed that the SMU of 5 m by 10 m by 2.5 m would be defined by an RC drilling program of 8 m by 12 m by 1.25 m. The final recommended drill spacing of 10 m by 15 m by 1.5 m (down-hole) is considered a practical implementation that better fits the resource model, SMU and short-term modelling requirements.
- Assaying all samples for Au by the same standard industry fire assay methods used for exploration and also assayed for S and As. Similar QAQC protocols should be adopted as are used for the Mineral Resource definition (see Sections 10.6 and 11.4).

- Grade modelling and ore block design using a method such as conditional simulation to model geology and grade of Au, S and As, increasing the reliability and repeatability of ore block design and facilitating the tracking of dilution and ore loss. This is important given the considerations in Section 16.15.1 regarding mining selectivity.
- A data system that captures all production data, including Ore Reserve depletion, grade control, mining and milling production. The system should have automated to semi-automated reporting functions for reconciliation, mine plan compliance, and production tracking.
- Defining and tracking ore blocks through the mining process requires 3D definition of the ore blocks and of lateral and vertical movement during blasting. Tracking blast movement, adjusting ore block boundaries for lateral movement, and raising mining floor levels to account for blast heave will reduce misdirection of ore and waste.
- Sufficient space should be allocated in the ROM pad design to facilitate blending of ore and to manage appropriate building and depleting of stockpiles.

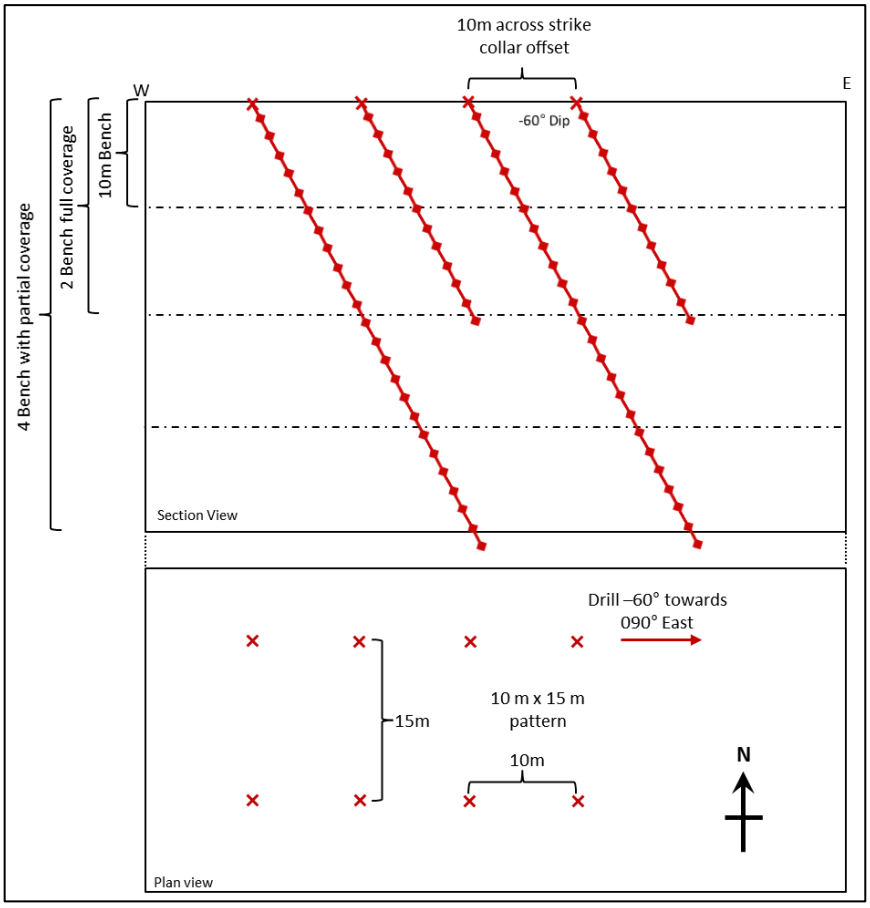


Figure 139: Extending the depth of some grade control drilling for better coverage

Estimated grade control costs (excluding supervision labour costs) were estimated by Golder (2018b) to be US$0.30/t milled (US$0.21/t mined). These costs contain allowances for drilling, sampling, assaying, QAQC, specialist software, and consumables. To advance the initial drilling well ahead of mining, higher costs will be incurred during the construction phase.

Cardinal will develop a comprehensive set of procedures for grade control tasks and functions. Specific recommendations have been made by Golder covering the RC drill sample spacing, sampling, assaying, ore blocking, blast movement, stockpile and reconciliation controls that are appropriate for such an operation.

16.17 Waste rock geochemical characterisation

Stripping requirements have been detailed for all the mining Phases in Section 16.9.1 and Table 149. Waste dumps will all be built to stable profiles and topsoil will be stockpiled appropriately. Under current Ghanaian law there is no requirement for backfilling and progressive backfilling of the open pit is not applicable with the current mine design.

To define the Acid and Metalliferous Drainage ("**AMD**") characteristics of the Namdini Project based on the preliminary mine planning, Cardinal requested Golder undertake a geochemical characterisation of the waste rock, low-grade ore and mineralized waste to be produced by mining of the deposit. The main findings (Golder, 2019g) were:

- The bulk of the waste rock materials are Non Acid Forming (NAF), especially Metasediments (MSE), Dacite (DAC) and Pyroclastics and tuffs (LTF).
- The Granite (GRA), Diorite (DIO) and Mafic Volcaniclastics (MVO) are the key lithologies that are consistently characterised as being Potential Acid Forming (PAF) or of uncertain classification (PAF-UC), for 20% of the samples of these lithologies analysed to date. These materials have elevated sulphur mainly associated with reactive sulphides, and insufficient Acid Neutralising Capacity ("**ANC**") to buffer the acidity produced by sulphide oxidation, although results suggest that even samples with elevated sulphidic sulphur may have significant lag times before producing acid leachate.
- Water extractable element testing suggested that there is likely to be minimal immediate mobilisation of trace elements in NAF and PAF materials represented by these samples, other than arsenic.
- MVO samples showed substantial mobilisation of arsenic, with deionised water leachate concentrations above 0.1 mg/L (World Bank Group International Finance Corporation ("**IFC**") mining guidelines and Ghana EPA Akoben mining effluent guidelines) in 90% of samples tested.
- GRA and DIO samples showed some mobilisation of arsenic, with deionised water leachate concentrations above 0.1 mg/L in over 20% of the samples tested, and concentrations above 0.01 mg/L in 67% and 100% of the samples tested respectively.

Based upon the above findings, considering the availability of beneficial oxide zone saprolite (which is non-acid forming and has characteristically low infiltration rates) and laterite (which has the ability to adsorb dissolved arsenic from leachate) it is planned that:

- The waste rock dump and the low-grade ore stockpiles should both be constructed with a basal layer of oxide zone saprolite
- Waste rock materials that are higher risk (PAF, PAF-UC and arsenic-bearing) should be contained within a central cell of the waste rock dump
- Low-risk waste rock materials should be dumped around the outside of the high-risk cell, they could be end-tipped in a doughnut shape, with the high-risk cell at the centre
- The high-risk cell and the low-grade ore stockpile should be constructed with a layer of laterite to manage arsenic in the seepage (above the basal layer of oxide zone saprolite)
- The Tailings Storage Facility (TSF) embankments, which are to be earthfill, should be constructed with oxide zone saprolite, with low-risk waste rock cladding where required (with suitable geotechnical properties).

16.18 Mining recommendations

Detailed further assessment of the drilling and blasting characteristics of the waste and ore material will be required as part of optimizing the mining costs and for contract tenders.

The mine schedule can still be refined and optimized for ore blending strategies to enhance the comminution characteristics of the ROM ore and possibly reduce costs.

Ongoing hydrological, hydrogeological and geotechnical analysis will be required for the area encompassed by the Final Pit design.

Additionally, the following requirements are also noted:

- The proposed plant sites, tailings disposal facilities and waste disposal sites may require some further geotechnical stability testing but the programme of sterilisation drilling is essentially complete to confirm the locations for the permanent infrastructure.
- Suitability and management of the current access roads to transport heavy mining equipment or wide loads to site prior to commencement of operations will need to be considered.
- The area around the proposed starter pit will require some minor pre-mining to rectify the legacy from the artisanal mining activities.
- The provision of adequate fencing and security are necessary prior to commencement of any mining on site.

17.0 RECOVERY METHODS AND PROCESS PLANT

The information presented in this Section is derived from Lycopodium (2019a) and Knight Piésold (2019a). Additional detail is contained in Appendices which are listed at the end of this Section.

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 minimise costs.

The main ore lithologies identified by Cardinal are classified as the Metavolcanics (MVO), Granite (GRA) and Diorite (DIO). Cardinal have advised 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 pit.

Metallurgical testwork was scoped and managed by Cardinal and reported in Section 13.0. The predominant association of gold with pyrite allows for conventional flotation technology to be applied, thus it was recommended to incorporate a rougher flotation stage followed by re-grind of the concentrate to liberate fine gold particles locked in the pyritic ore.

The key criteria for the plant design are as follows:

- The plant is designed for an initial throughput of 9.5 Mtpa.
- Crushing plant mechanical availability of 75%.
- Remainder of plant mechanical availability of 91.3%.
- An appropriate level of automation to provide control of the key process parameters.

A process design criteria document was prepared, for which the detail is available separately to Cardinal, incorporating the engineering and key metallurgical design criteria derived from the results of metallurgical testwork and comminution circuit modelling which are still ongoing.

17.1.2 Process flowsheet

The process plant design incorporates the following unit process operations:

- Single stage primary crushing with a gyratory crusher to produce a P_{80} 150 mm.
- Crushed ore feeding a coarse ore stockpile (12 hours live).
- Stockpile 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, including recycle and crush of pebbles from the SAG mill (SABC grinding).
- Gravity recovery includes scalping screens, centrifugal concentrators and an intensive leach reactor and dedicated electrowinning cell to recover gold.
- Rougher flotation to produce a gold-rich sulphide concentrate.
- Thickening of the flotation tails for water recovery prior to disposal in a separate non cyanide tailings storage facility.

- Thickening of the flotation concentrate for water recovery.
- Maelgwyn's Aachen high shear oxidation process on the regrind product before CIL.
- High intensity regrind of the flotation concentrate to a P_{90} grind size of 9 µm.
- A concentrate CIL circuit incorporating one pre-aeration tank and seven CIL tanks for gold and silver adsorption. Aachen technology will be utilised prior to the CIL.
- A 4.5 tonne split Anglo American Research Laboratory (AARL) elution circuit, electrowinning and smelting to recover gold and silver and produce doré.
- CIL tailings treatment incorporating cyanide destruction by sulphur dioxide and oxygen and arsenic precipitation.
- 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 140.

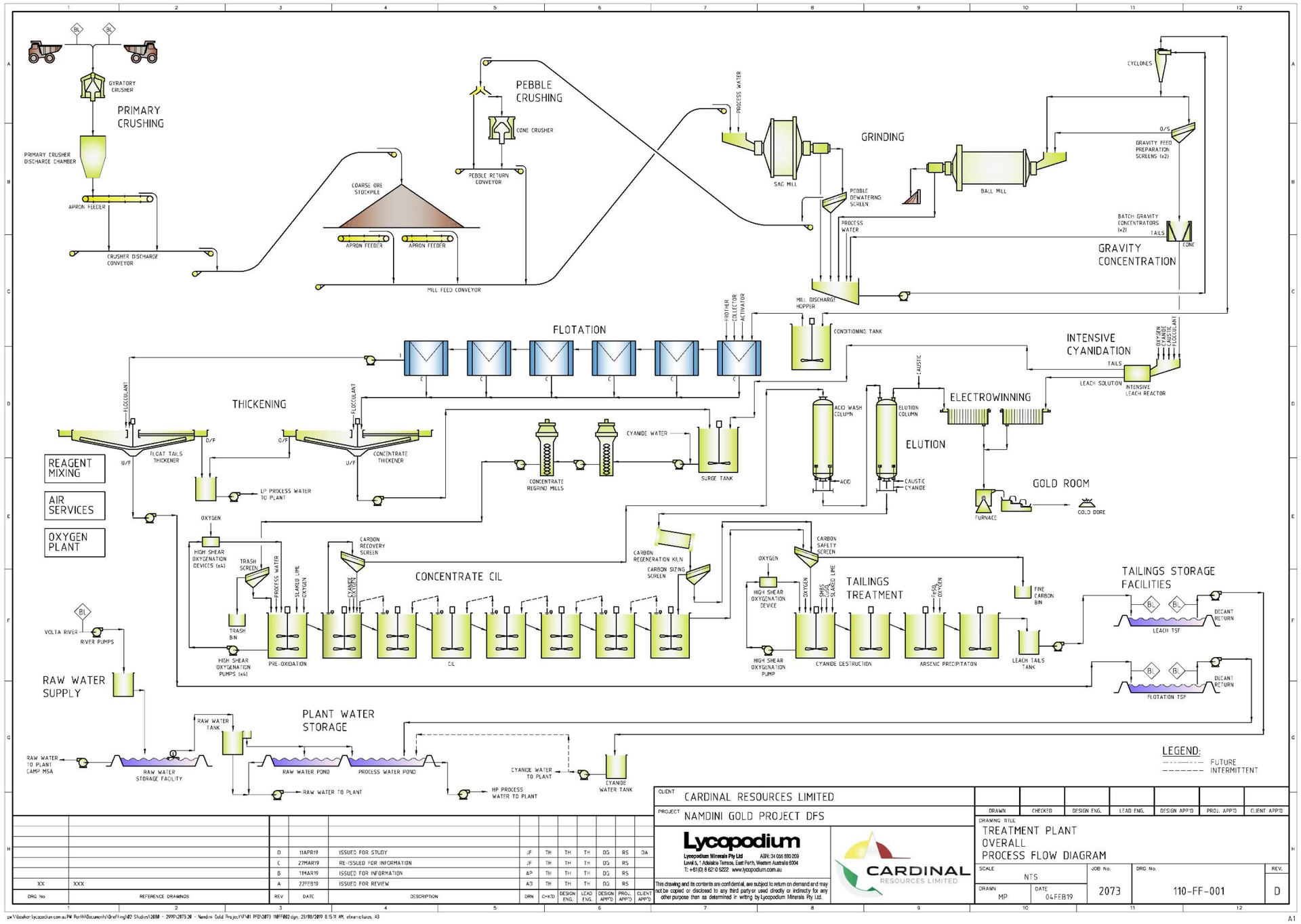


Figure 140: Process flowsheet showing the unit process operations

17.1.3 Plant design basis

Process plant

The plant design was based on a nominal capacity of 9.5 Mtpa. The feed will consist of blended ore from the pit consisting of an average of approximately 56% MVO, 29% GRA and 15% 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 for some blending of feed stock to help produce a consistent feed type and rate to the plant. A fixed hydraulic rock breaker will be used to break any oversized rocks in the crushing area.

Comminution selection

The historic comminution testwork started in 2016. This work focused on the main feed lithologies according to a preliminary mining schedule. This only comprises SPI and BBWi data.

Additional testwork was completed in two campaigns during early and late 2017 and further testwork was completed in June 2018.

As part of optimizing the feed methodology of the Namdini project, it was identified by Cardinal geologists that the testwork undertaken during early 2017 consisted of material close to the contact zone and is only expected to contribute to the mining schedule during the latter part of the production schedule.

It was noted that the November 2017 and 2018 testwork is more representative of the plant start-up blend (effectively operating Year 1 to 5). In terms of identifying the longer-term LOM feed blend, all of the testwork data was considered, including the 2016 and early 2017 data.

The comminution circuit design and throughput predictions reflect this methodology.

A future circuit upgrade is possible, transitioning from a primary crush only fed circuit to a secondary crush fed circuit.

The findings are summarized in the 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 29.9 to 40 putting these ores samples in the hard category.
- A gyratory crusher is recommended for primary crushing. A fixed rock breaker is also recommended.
- The SAG mill power efficiency (f_{SAG}) for this ore is low. The f_{SAG} is poor at 1.36 but could be improved by partial crushing.
- Assessment of the comminution circuit identified that 9.5 Mtpa is the largest throughput that can be achieved with dual pinion mill drives.
- Pebble crushing is included to manage build-up of critical size material in the SAG mill.
- The 85th percentile design abrasion index range of 0.090 for Metavolcanics to 0.290 for Granite shows relatively low ore abrasiveness and therefore translates into low and favourable wear liner and steel media consumption.

Selected comminution circuit

The comminution circuit option selected using the design criteria (Table 150) 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 minimises operating cost while still producing the relatively fine product size required.

Table 150: Summary of selected comminution design criteria

Parameters	Units	Start-Up (Year 1 to 5)	LOM
CWi	kWh/t	19.8	19.0
BBWi	kWh/t	17.3	18.9
A*b	-	31.3	28.8
Feed rate	tph	1,188	
Milling feed F_{80}	mm	137	112
Milling product P_{80}	µm	106	
Milling specific energy (total) P_{80}	kWh/t	23.7	24.7

A summary of the equipment selection is provided in Table 151 and the expected consumables are summarised in Table 152.

Table 151: Summary of selected mill criteria

	Units	Start-Up (Year 1 to 5)	LOM
SAG Mill			
Mill diameter	m	11.6	
Mill EGL	m	5.7	
Specific energy	kWh/t	12.0	11.7
Ball charge: ■ Duty ■ Max	 % %	 14 16	 12 16
Total load: ■ Duty ■ Max	 % %	 25 35	 25 35
Mill speed	% Nc	75	75
Drive type	-	Variable Speed	
Mill pinion power: ■ Nominal ■ Max	 kW kW	 14,312 17,396	 13,896 17,396
Installed power	kW	2 × 9,000	
Discharge arrangement	-	Grate	
Open area	%	8	
Grinding media size	mm	125	
Ball Mill			
Mill diameter	m	8.2	
Mill EGL	m	12.7	
Specific energy	kWh/t	11.5	12.8
Ball charge: ■ Duty ■ Max	 % %	 25.0 34.0	 28.0 34.0
Mill speed	% Nc	75.0	
Drive type	-	Variable Speed	
Mill pinion power: ■ Nominal ■ Max	 kW kW	 13,608 17,582	 15,208 17,582
Installed power	kW	2 × 9,000	
Discharge arrangement	-	Overflow	
Grinding media	mm	65	

	Units	Start-Up (Year 1 to 5)	LOM
Pebble Crusher			
% Feed	%	14.7	14.6
Feed rate	tph	175	174
Specific energy	kWh/t	0.2	0.2

Table 152: Estimate of primary grinding consumables

Parameters	Units	Start-Up (Year 1 to 5)	LOM
Abrasion Index	g	0.135	0.139
Primary Crusher			
Mantle	Sets /year	0.7	0.7
Concave	Sets /year	0.7	0.7
Gross power	kWh/t	0.3	0.3
Secondary Crusher			
Mantle / Concave	Sets /year	-	3.3
Gross power	kWh/t	-	0.2
SAG Mill			
Ball consumption	kg/t milled	0.333	0.259
Steel liner consumption	kg/t milled	0.067	0.067
Gross power	kWh/t	11.3	10.6
Ball Mill			
Ball consumption	kg/t milled	0.453	0.444
Steel liner consumption	kg/t milled	0.061	0.060
Gross power	kWh/t	10.2	9.9
Pebble Crusher			
Mantle/Concave	Sets /year	2.6	2.6
Gross power	kWh/t	0.2	0.2

Note: At circuit P_{80} of 130 µm and using LOM average comminution characteristics.

Crushing and ore storage

The primary crusher was sized on the basis of achieving 75% utilisation. The mine design is based on 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 150 t trucks 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 range of mill speed with a nominal design of 75% of critical speed. The ball mill will also be variable speed. The variability in ore competence can be addressed by varying the speed of the mills.

A gravity circuit was included as some of the ore types tested, in the Starter Pit samples in particular, appear to have a reasonably high gravity recoverable gold component.

Flotation concentrate thickening

Inclusion of a flotation concentrate thickener before the regrind facility will achieve a higher regrind feed slurry solids. Thickening also recycles water that is not cyanide bearing for re-use in the grinding and flotation sections of process plant.

The higher regrind feed slurry solids allows CIL tailings decant to be used as process water dilution to meet the regrind mill vendor's requirements for limiting the mill feed slurry solids content without compromising the plant water balance. The thickener was sized to allow for potential froth management.

A surge tank is installed to store concentrate thickener underflow slurry prior to delivery to the regrind circuit. This allows the primary grinding and flotation circuit to remain operational in the event of a short-term maintenance stoppage or breakdown in the regrind area.

Flotation

Testwork on samples of the ore has indicated that high gold recovery can be achieved by flotation, with a design point of 7.5% mass recovery from the feed selected by Cardinal for design.

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 each with individual level control. All flotation concentrate will report to a single concentrate hopper and be pumped to the concentrate thickener before transfer to the regrind circuit.

Concentrate regrind

Cardinal tested P_{90} regrind sizes from 5 to 45 µm. Leach testwork indicated that regrinding of flotation concentrate would be necessary to improve leach extraction. An optimization study indicated that a regrind to a P_{90} grind size of 9 µm should be targeted.

A high intensity grinding mill (the HIGmill) was selected for the regrind duty. This equipment minimises footprint, provides internal classification and has minimal overgrind issues, while providing suitable access for maintenance. The mill was sized based on testwork carried out by the Vendor (Outotec) on behalf of Cardinal.

The reground concentrate will be pumped to the trash screen in the CIL circuit.

Leach and adsorption circuit

Metallurgical testwork indicated that pre-oxidation improved both the gold leaching kinetics and overall dissolution. As a result, a pre-aeration tank was included in the design. Cardinal nominated a leach residence time of 48 hours for design.

A configuration of one pre-aeration tank and seven CIL tanks was selected. The pre-leach and CIL tanks will be the same size. As well as the pre-aeration tank oxygen addition facility, extra oxygen may be added to all CIL tanks to oxidize any cyanicides and maintain an adequate dissolved oxygen level. Cardinal have conducted metallurgical testwork and based on positive results have selected the Aachen technology as the method to add oxygen into the pre-CIL slurry.

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 4.5 t capacity split AARL elution circuit was selected requiring just under seven strips per week. The AARL circuit was selected as it offers the flexibility to run more than one elution per day if needed.

Flotation tails thickening

Tails from the flotation circuit will be dewatered in a high rate thickener to recover cyanide-free process water for re-use in the grinding and flotation circuits.

Cyanide destruction

Concentrate leach generally requires a higher cyanide concentration to achieve efficient extraction. As a result, tailings from the CIL circuit may contain high weak acid dissociable cyanide (CN_{WAD}) levels. A cyanide destruction facility incorporating sodium metabisulphite and oxygen was included to reduce the levels of CN_{WAD} that would be compliant with the International Cyanide Management Code limits (ICMI, 2016).

Arsenic precipitation

Testwork has indicated that high levels of arsenic can occur in leach solutions. The arsenic levels in CIL tailings solution can be reduced by the addition of iron sulphate. This process has been included in the design.

The design of the arsenic precipitation circuit has two agitated tanks, the same size as the cyanide destruction tanks. This provides additional residence time. One tank is designed for use as either a cyanide destruction or arsenic precipitation tank to provide additional cyanide destruction capacity if required.

Tailings disposal

Two separate tailings streams will be produced. CIL 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 a plastic lining. This will allow the two decant water streams to be recycled separately.

Water and air services

Three water systems will be used:

- Process water recovered from the flotation tails thickener and tailings storage facility will be recycled in the grinding and flotation areas.
- Cyanide water recovered from the cyanide tailings storage facility will be used in the CIL.
- Raw water sourced from the White Volta River via a raw water storage dam will be used for reagent mixing, cooling water make-up, pump gland seal water and will feed a water treatment plant for elution water and potable water plant feed.

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 and arsenic precipitation areas; Compressed air will be used for instruments and plant services.

17.1.4 Key process design criteria

Table 153 presents the key process and mechanical design criteria. The ‘gold head grade design’ was derived from the PFS Starter Pit head grade (1.31 g/t Au) with an optimistic allowance of 20% to ensure sufficient gold room capacity.

Table 153: Summary of key process design criteria

Parameter	Units	Primary	Source
Plant capacity	Mtpa	9.5	Cardinal
Gold head grade design	g/t Au	1.57	Cardinal
Crushing plant availability	%	75	Lycopodium
Rest of Plant availability	%	91.3	Lycopodium
ROM ore top size	mm	1200	OMC
Crushing Work Index (“**CWi**”)	kWh/t	19.8	Testwork – Start-Up
Bond Rod Mill Work Index (“**RWi**”)	kWh/t	22.1	Testwork – Start-Up
Bond Ball Mill Work Index (BBWi)	kWh/t	17.3	Testwork – Start-Up
Abrasion Index	g	0.283	Testwork – Start-Up
Drop Weight (SMC, A*b)		31.3	Testwork – Start-Up
SG		2.78	Testwork – Start-Up

Parameter	Units	Primary	Source
Comminution circuit type		SABC	OMC
Primary crush size (P_{80})	Mm	137	OMC
Primary grind size (P_{80})	µm	106	Testwork
Flotation mass recovery	%	7.5	Testwork
Concentrate thickener solids loading	t/m².h	0.25	Vendor
Concentrate regrind size (P_{90})	µm	9	Testwork
Regrind specific energy	kWh/t	120	Testwork
HIGmill grinding media consumption	g/kWh	8.5	Vendor
Leach circuit residence time	hours	48	Cardinal
Leach slurry solids	% w/w	35	Lycopodium
Number of pre-aeration tanks		1	Lycopodium
Number of CIL tanks		7	Lycopodium
CIL cyanide consumption	kg/t conc.	5.50	Testwork
CIL Lime consumption (100% CaO)	kg/t conc.	4.16	Testwork
Elution circuit type		Split AARL	Lycopodium
Elution circuit size	t	4.5	Lycopodium
Frequency of elution	strips/week	6.9	Lycopodium
Cyanide destruction process		SO2/O2 process	Lycopodium

17.2 Process plant description

17.2.1 Primary crushing

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.

A fixed rock breaker will be used to break up any oversize material. The crusher selected is a 60" by 89" gyratory crusher with a 600 kW motor.

The gyratory crusher product will discharge into a chamber with a live volume of 2 trucks or 300 tonnes. From the discharge chamber, the crusher discharge apron feeder will transfer crushed ore onto the crusher discharge conveyor in turn feeding the stockpile feed conveyor delivering onto the coarse ore stockpile.

A weightometer on the stockpile feed conveyor will measure crusher production rate.

Dust suppression sprays will be installed at the ROM feed pocket and stockpile feed conveyor. A wet scrubber type dust collector will draw dust laden air from key transfer points and effluent will be pumped to a silt pond.

A sump pump will be provided for fine spillage handling in this area. It will pump run-off to the silt pond.

Maintenance cranes will be provided for the crusher and the crusher drive. A hoist will be installed for apron feeder maintenance.

Space will be reserved in the plant layout in this area for the future installation of a secondary crushing circuit.

17.2.2 Ore storage and reclaim

The stockpile feed conveyor will discharge onto a 14,300 tonne live capacity stockpile providing a nominal 12 hours' worth of mill feed ore to allow for maintenance in the crusher area. The total stockpile capacity will be approximately 71,500 tonnes or 60 hours of mill feed. This extra material can 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 sump pump will be provided for fine spillage handling in this area. It will pump run-off to a silt pond.

17.2.3 Grinding and classification

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. An 11.6 m diameter and 5.7 m effective grinding length ("**EGL**") SAG mill will grind crushed ore from a F80 of 150 mm to produce the target transfer P_{80} size of 1.0 to 2.5 mm. The SAG mill will be equipped with two 9 MW variable speed drives to provide a total of 18,000 kW installed power. Variable speed drives will allow the mill speed to be adjusted to address variable ore competence.

SAG mill grinding media, typically 125 mm diameter steel balls, will be loaded by FEL into a SAG mill ball hopper for loading on to the mill feed conveyor to maintain the SAG mill media charge.

Product from the SAG mill will discharge to a 3.0 m wide by 5.5 m long vibrating single deck pebble dewatering screen fitted with a nominal 12 mm slotted aperture screen deck and high-pressure water sprays. Screen undersize will flow to the cyclone feed hopper.

Screen oversize, consisting of pebbles and worn steel grinding media will report to the pebble discharge conveyor. If required, for example on mill start-up, screen oversize can 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 by this should be detected by a downstream metal detector which will activate a diverter gate to prevent tramp steel entering the crusher chamber. This diverter gate will also allow manually initiated bypass of pebbles for recycle to the SAG mill 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 and record the mass of pebbles being recycled.

Crushed pebbles will discharge to the crushed pebble transfer conveyor which delivers them back to the SAG mill feed conveyor.

The SAG and ball mill discharge slurry will feed a combined cyclone feed hopper and will be diluted with water and pumped using duty / stand-by pumps to the classifying cyclones. The cyclone cluster will comprise 18 operating 650 mm diameter cyclones operating at 107 kPa. The cluster will have 15 cyclones in duty and three in stand-by for maintenance purposes. Cyclone overflow will report to flotation and cyclone underflow will feed the ball mill with a portion reporting to the gravity circuit.

The ball mill will be an 8.2 m diameter and 12.7 m EGL overflow mill with a target grind P_{80} size of 106 µm. The ball mill will be equipped with two 9 MW fixed speed drives giving a total installed power of 18,000 kW. The design mill normal rotation speed will be 75% of critical. A VWF controller will be used for starting the ball mill.

Ball mill grinding media consisting of 65 mm diameter balls will be charged from a kibble using a dedicated ball charging hoist.

A drive-in sump will be provided to recover coarse spillage from the circuit located near the discharge of the SAG mill. Three sump pumps are provided for fine spillage handling in this area.

Cyclone overflow at a nominal 35% w/w solids will flow by gravity via the two-stage flotation feed sampler, into the rougher conditioning tank ahead of the flotation circuit. The sampler consists of a timer controlled cross cutting primary sampler that that sends regular sample increments from a cut across the whole stream to a secondary Vezin type sampler. A shift composite sample is collected and submitted for analysis in the plant assay facility.

The grinding area will include an overhead gantry crane for cyclone maintenance.

Mill motors and drive trains are equipped with cooling systems. Cooling water from the heat exchangers will be recycled via a closed-circuit cooling system with a heat exchanger equipped with a closed-circuit cooling tower.

17.2.4 Gravity gold recovery

A bleed from the classification cyclone underflow will feed two 3.05 m wide by 4.8 m long scalping screens with 2 mm slotted apertures. The screen oversize will flow back to the ball mill feed chute while the screen underflows will feed the gravity concentrators. The concentrators will be 70 inch diameter centrifugal units with variable speed drives and dedicated fluidising water systems.

The units will operate on a 60 minute (nominal) cycle time. After 60 minutes the control system will bypass screen undersize slurry back to the cyclone feed hopper and start a flush cycle. Concentrate will gravitate to an intensive leach reactor. Once the flush cycle completes the unit will start up again and the bypass will again send screen underflow slurry to the concentrator.

The intensive leach system will consist of a feed cone, a rotating drum reactor and a solution cone. Concentrate will be leached at elevated pH using sodium hydroxide, cyanide and an oxidant. After completion of the daily batch leach cycle the pregnant solution will be pumped to a tank for electrowinning. The solids tails from the leach will be pumped to the feed stream of the regrind circuit.

The grinding area overhead gantry crane will be used for concentrator maintenance.

A sump pump will be provided for fine spillage handling in this area.

The gravity concentrators will be installed in fenced-off secure areas with access control limited by swipe card to selected personnel. The intensive cyanidation system will be installed in the gold room for security reasons.

17.2.5 Flotation

In the flotation conditioning tank, frother and collector will be added. A nominal five minutes residence time will be provided by the agitated tank. A pump will extract a slurry sample from the tank overflow stream, and this will be sent to the on-line elemental analyser.

The wavelength-dispersive X-ray fluorescence device provides monitoring of the iron and arsenic grades in the slurry solids going to flotation, for control purposes. A shift composite sample can be collected from this stream. A portion of the slurry sample will be sent to an on-line particle size analyser. The laser diffraction particle analyser monitors the primary grinding circuit product size distribution in the slurry solids going to flotation, for control purposes.

The conditioned slurry will overflow from the tank to feed a bank of six 300 m³ capacity forced air flotation cells arranged in a straight configuration with total residence time of at least 30 minutes.

Flotation concentrate slurry will gravitate via launders to a concentrate hopper and will be pumped by duty / standby pumps to the concentrate thickener. An in-line pressure pipe sample extracts a slurry sample from the concentrate stream, and this is sent to the on-line elemental analyser. This provides monitoring of the iron and arsenic grades in the concentrate, for control purposes. A shift composite sample can be collected from this stream.

A portion of the concentrate sample will also be sent to the on-line particle size analyser which will monitor the size distribution in the concentrate slurry solids going to regrind, for control purposes.

Flotation tails slurry will discharge from the last cell to the flotation tails hopper via the two stage flotation tails sampler. The sampler consists of a timer controlled cross cutting primary sampler that that sends regular sample increments from a cut across the whole stream to a secondary Vezin type sampler. A shift composite sample is collected and submitted for analysis in the plant assay facility.

A pump will extract a slurry sample from the tails hopper, and this will be sent to the on-line elemental analyser to provide monitoring of the iron and arsenic grades in the tails slurry solids from flotation, for control purposes. A shift composite sample can be collected from this stream.

Flotation tails slurry will be pumped by duty/standby pumps to the float tails thickener.

Two sump pumps will be provided for spillage handling in the flotation area. Spillage can be pumped to the conditioning tank or the tails thickener.

17.2.6 Flotation concentrate thickening

Concentrate slurry will be dewatered in a 23 m diameter high rate thickener ahead of the regrind circuit. Flocculant will be diluted with process water and added to the feed slurry. The underflow slurry, at typically 55% w/w solids will be pumped to the concentrate storage tank.

The tank provides some surge capacity, so the primary grinding circuit need not be shut down for a short or unplanned stoppage of the regrind circuit. The regrind circuit can also operate for a period during a mill shutdown for relining, etc.

The concentrate storage tank is designed to match the CIL tanks and has a nominal 13 hours residence time. Solid tails from the intensive cyanidation reactor are pumped to the concentrate storage tank.

Slurry from the storage tank will be pumped by duty / standby pumps to the regrind circuit. Cyanide process water will be injected into the pump suction to control the regrind mill feed density at a nominal 38% w/w solids.

Overflow from the thickener will report to the overflow tank for re-use in grinding and flotation as process water.

A sump pump will be installed at the thickener and another will be installed at the concentrate surge tank.

17.2.7 Regrind

The concentrate thickener underflow slurry will be diluted with cyanide process water to optimal grinding density and pumped into the regrind mill at base level.

Slurry enters a grinding chamber in the regrind mill containing media and rotating discs which provide momentum to stir the media charge against a series of static discs. As the slurry flows upwards, it passes through the free space between the rotating and static discs lining the wall.

Due to the vertical arrangement of the mill, classification is conducted simultaneously throughout the grinding process with larger particles remaining in the grinding chamber, while smaller particles move upwards. The process is a single pass with no external classification.

The first stage regrind mill product slurry overflows to the transfer pump hopper and will be pumped by duty / standby pumps through a heat exchanger to cool the slurry as it gets heated by the energy input in the regrind mill.

The cooled first stage regrind mill product slurry will be pumped into the second stage regrind mill. The second stage mill product overflows to the CIL feed hopper and will be pumped by duty / standby pumps through a second heat exchanger.

From the heat exchanger outlet the slurry will pass through the two stage CIL feed sampler and on to the trash screen ahead of the CIL circuit. The sampler consists of a timer controlled cross cutting primary sampler that sends regular sample increments from a cut across the whole stream to a secondary Vezin type sampler. A shift composite sample will be collected and submitted for analysis in the plant assay facility.

Cooling water from the slurry heat exchangers will be recycled via closed circuit cooling systems with heat exchangers equipped with closed circuit cooling towers.

A pump extracts a slurry sample from the transfer and CIL feed hoppers and these streams are sent to an on-line particle size analyser. The laser diffraction particle analyser monitors the primary and secondary regrind mills product size distribution, for control purposes.

The first and second stage regrind mill will be charged with nominal 4 and 2 mm ceramic media respectively. Media will be charged from bulk bags into a feed hopper located at the topmost level of each mill installation. A davit crane will be provided for the lifting the media bags to the hopper. Kibbles will be provided to collect the media charge when the mill is dismantled and the rotors removed for maintenance. The kibbles will be used to refill the media charge before restart using the davit hoist.

Either mill stage can be bypassed and the circuit operated as a single stage regrind if needed though capacity and product particle size will be limited by the reduced available regrind mill power.

A single sump pump will be provided for the regrind area.

17.2.8 Flotation tailings disposal

Flotation tailings will be thickened in a 36 m diameter high rate thickener ahead of the tailings disposal tank to maximise recovery of water. Flocculant will be added to the feed and the underflow slurry at 55% w/w solids then gravitates from the thickener cone into the tailings tank and gets pumped by duty / standby two stage pumps to the flotation TSF.

Overflow from the flotation tails thickener and the concentrate thickener will report to a single overflow tank and the duty / standby low-pressure water pumps will return process water for re-use in flotation and grinding.

A sump pump will be installed for the thickener.

Tailings will be deposited into the TSF using a peripheral discharge and decant method. Cyclic spigot deposition at various locations will be used to allow consolidation and drying of deposited material into beaches to direct supernatant water to a pond around the decant tower. Supernatant water will be recovered from the TSF and returned as process water for the plant.

Seepage and underdrainage pumps are installed at the flotation TSF and deliver solution back into the pond. A single decant pump is installed in the active TSF cell with an uninstalled standby pump available if required.

17.2.9 Leach and carbon adsorption

Reground concentrate slurry will be pumped to the 1.5 m wide by 3.0 m long vibrating trash screen with a nominal 0.63 mm aperture. Trash screen oversize will drop into a collection bin. Screen undersize will report to CIL feed distribution box.

The leach process will include one pre-aeration tank and seven CIL tanks. The pre-aeration tank and the CIL tanks will have a volume of 1,340 m³ each. 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 48 hours at 89 t/h solids at 35% w/w slurry solids.

The slurry pH will be adjusted with slaked lime slurry added to the pre-aeration tank.

Oxygen will be added to the pre-leach using Aachen technology and sparged into the CIL tanks through the agitator shafts to maintain a high dissolved oxygen profile. Oxygen will be supplied from the on-site oxygen generating plant.

The pre-aeration tank will be also equipped with proprietary high shear aeration devices that were developed to assist in dispersing oxygen in leach feed slurry. The technology nominated by Cardinal was sourced from Maelgwyn Mineral Services Africa (MMSA) who market the Aachen aerator device.

This concept for mineral slurry reactors originated in Germany and was developed in the 1970s for flotation. The aerator is designed to improve gas-liquid mass transfer using energy provided in pipeline flow. The fundamental principle is the use of slot aerators in high velocity fluid streams. Additionally, a secondary chamber provides regeneration of bubble surfaces using various hydrodynamic effects. The objective is to increase utilisation of the gas phase, thus reducing overall energy and reagent costs. The reactor contains no moving parts. It is designed to withstand erosion by mineral slurries.

A total of four reactors will be installed with each unit fed by a duty slurry pump drawing from the pre aeration. Each unit is designed to accept a slurry feed rate of about 700 m^3/h. Three units are expected to be in operation at a time with the fourth unit available as a standby. The capacity of installation will allow for the equivalent of ten times the new slurry feed rate to the pre-aeration tank to be passed through the aerators.

Sodium cyanide solution will be metered into the first four CIL tanks as required to maintain the desired cyanide concentration in the slurry.

Fresh and regenerated carbon will be added to the circuit in the last stage CIL tank and advanced by recessed impeller pumps counter-current to the slurry flow. Inter-tank screens in each CIL tank will retain carbon. Loaded carbon from the first stage CIL tank will be pumped to the loaded carbon recovery screen. The carbon will be rinsed with high pressure cyanide process water sprays on the screen and report to the acid wash column while the slurry draining through the screen will return to the CIL tank.

Tails slurry from the final stage CIL will flow to the carbon safety screen to capture any carbon leaking from worn screens or overflowing tanks. The safety 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 pre-aeration and CIL area will include an overhead gantry crane for operations and maintenance use, in particular removal and replacement of the interstage screens in CIL tanks.

Four sump pumps for spillage handling will be installed in the bunded area.

The loaded carbon screen will be installed in a fenced off secure area with access control limited by swipe card to selected personnel.

Cyanide gas monitoring will be installed in the CIL area. The monitoring is linked to a beacon light and audible siren which activate if the measurement triggers an alarm.

17.2.10 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 system
- Electrowinning of gold from pregnant solution
- Smelting of electrowinning products.

The spilt AARL stripping plant will be automated and contain separate acid wash and elution columns.

The total carbon movement around the elution plant on a daily basis will be approximately 4.5 tonnes with a solution flow rate in the elution column of two bed-volumes/h (19.2 m^3/h). There will be a nominal 6.8 strips/week. The circuit was designed at 4.5 tonne to accommodate fluctuations in feed grade and consequently carbon movement.

Acid wash

Loaded carbon will be received into the acid wash column. During acid washing, a dilute (3% v/v) hydrochloric acid solution will be pumped into the bed of carbon in the column and left to soak. This will remove contaminants such as calcium and magnesium from the carbon which can interfere with elution and carbon activity.

After the soak period, nominally 30 minutes, the loaded carbon will be rinsed with a nominal four bed volumes of fresh water. Dilute acid and rinse water effluent will be pumped to either the arsenic precipitation tanks or leach tailings hopper for disposal. Washed carbon from the acid column will be pressure transferred to the elution column and the water drained out.

Strainers are installed on the acid wash column outlet.

A sump pump will be installed in the acid wash area.

Cold cyanide wash

Leach testwork by Cardinal showed high levels of copper in solution. This solubilised copper will load onto the carbon in the CIL and overtime reduce its effectiveness to adsorb gold. Some copper may be removed in the acid wash step but a cold cyanide pre-wash step before elution has been allowed for in the design.

When high copper grades in solution are detected to be present in the CIL, a cold cyanide wash will be performed in the elution column prior to the main elution step. The cold cyanide wash solution will be a nominal 2% sodium hydroxide (NaOH) and 2% sodium cyanide (NaCN) concentration. A bed-volume ("**BV**") of solution will be made up and stored in the cold cyanide wash feed tank and as required pumped into the column and allowed to soak.

After the soak period, the column will be flushed with fresh water through the column to the cold cyanide wash product tank. The effluent in the product tank will be pumped out slowly to the cyanide destruction circuit to prevent copper overload to cyanide destruction feed.

Pre-soak and elution

The split AARL elution process will be used to recover gold adsorbed onto the carbon recovered from the CIL circuit

Prior to elution of gold from a batch of acid wash carbon, the lean eluate from the previous batch in 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% NaOH and 2% NaCN solution will be pumped into the elution column by the strip solution pump to pre-soak the loaded carbon for 30 minutes.

The pregnant solution will then be rinsed from the carbon by up to 10 BVs of solution heated to approximately 135°C. The first five BVs 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 BVs of the elution will be drawn from the stripping water tank and will be directed to the lean eluate tank for re-use during the next elution cycle. Two pregnant solution tanks will be installed to allow an elution batch to be initiated if the prior batch electrowinning is still underway.

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.

Strainers are installed on the elution column outlet.

A sump pump will be installed in the elution area.

Cyanide gas monitoring is installed in the elution area. The monitoring is linked to a beacon light and audible siren which activate if the measurement triggers an alarm.

The acid wash and elution columns are installed in a fenced off secure area with access control limited by swipe card to selected personnel.

Electrowinning and gold room

Gold will be extracted by electrowinning from the pregnant solution within the gold room. As the pregnant solution has a high gold tenor, it will be circulated through two parallel cells of 20 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 sludge from the cathodes was selected to simplify the cathode handling process.

Barren electrolyte will be pumped to the CIL feed to recover residual gold values and utilise the residual cyanide.

Rectifiers, one per cell, will be located in a position accessible from outside the gold room high security area to enable maintenance access to them 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.

Cathode sludge will be filtered using a vacuum pan filter and dried before mixing with fluxes and smelting.

A sump pump will be installed at the pregnant solution tank bunded area.

A sump pump will be installed in the gold room.

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 800°C and held at this temperature for at least 20 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.

17.2.11 Cyanide destruction and arsenic precipitation

CIL tailings from the carbon safety screen will gravitate to the cyanide destruction distribution box.

The SO_2 / O_2 process was selected to reduce weak acid dissociable cyanide (CN_{WAD}) 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 metabisulphite $Na_2S_2O_5$ ("**SMBS**"). will be supplied from the on-site oxygen generating plant. The detoxification process requires the presence of a soluble copper catalyst which will be supplied by metering copper sulphate solution directly from a mixing tank.

Slaked lime slurry 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 CN_{WAD} cyanide respectively and can be used to adjust the flow of SMBS and copper sulphate $CuSO_4$ to control CN_{WAD} cyanide discharge. Probes measuring Eh and pH will be will also be included.

The cyanide destruction tanks are also equipped with an Aachen high-shear aeration device. The unit has a nominal capacity of 700 m³/h and is fed by a single duty pump. This was selected by Cardinal with the objective of improving oxygen mass transfer in the cyanide destruction system.

Some ores contain soluble arsenic which will be treated and stabilised before discharge to the TSF. The arsenic precipitation and stabilisation reaction involves the use of iron sulphate. The reaction is rapid. Two tanks in series with one hour residence time in each, will be provided. Discharge from the cyanide destruction tanks will overflow to the arsenic precipitation tanks. Iron sulphate will be added in the presence of oxygen. The arsenic in solution will precipitate as ferric arsenate in the tails slurry.

Tailings slurry will overflow from the arsenic precipitation tank to the final tails sampler.

A sump pump will be installed in the cyanide destruction and arsenic precipitation area.

Cyanide gas monitoring is installed in the cyanide destruction and arsenic precipitation area. The monitoring is linked to a beacon light and audible siren which activate if the measurement triggers an alarm.

17.2.12 CIL tailings disposal

Tailings from the cyanide destruction process will flow through the two stage final tails sampler and in to the CIL tails hopper. The sampler consists of a timer controlled cross cutting primary sampler that that sends regular sample increments from a cut across the whole stream to a secondary Vezin type sampler. A shift composite sample will be collected and submitted for analysis in the plant assay facility.

The slurry in the CIL tails hopper will be pumped to the cyanide tailings storage facility (TSF) by two stage duty / standby pumps.

Tailings will be deposited into the CIL TSF using a peripheral discharge and decant method. Cyclic spigot deposition at various locations will be used to allow consolidation and drying of deposited material into beaches to direct supernatant water to a pond around the decant tower.

Seepage and underdrainage pumps are installed at the CIL TSF and deliver solution back onto the TSF. A single decant pump is installed with an uninstalled standby pump available if required.

Decant return water from the cyanide TSF will be pumped to the process water tank for reuse in the CIL circuit. Depending on the availability of decant water in the two tailings facilities, excess cyanide process water may need to be transferred to the flotation process water system. Any residual cyanide in the cyanide process water will be destroyed by the addition of an oxidant (e.g. hydrogen peroxide) dosed from an intermediate bulk container ("**IBC**") on an as required basis.

Flotation tails from the float tails thickener will be transferred by pinch valve control to a tails storage tank. From here, multi-stage pumps will transfer the tailings to the non-cyanide TSF. Decant return water from the flotation tails TSF will be pumped to the flotation water pond for reuse.

17.2.13 Reagents and consumables

Flotation collector

Potassium amyl xanthate ("**PAX**") will be delivered in bulk bags. PAX bags will be added to the mixing tanks by electric hoist via a bag breaker. The PAX solution will be transferred to the storage tank for use in the flotation process. PAX solution will be metered to the flotation circuit by dosing pumps.

A ventilation fan is installed at the mixing and storage tank to disperse fumes.

Flotation frother

Frother will be delivered to site in 1,000 L isotainers. The frother will be pumped into a storage tank. Frother will be metered to the flotation circuit by dosing pumps as required.

Quicklime

Quicklime will be delivered to site in bulk tankers from a local supplier, who maintains a depot and bulk storage nearby at Bolgatanga in the Northern Region of Ghana. This will allow 'Just-in-Time' deliveries to the mine site. Lime will be pneumatically transferred from the delivery tank into a silo.

Quicklime will be slaked on site and the slurry stored in an agitated tank. Lime slurry will be delivered via a ring main to the CIL cyanide destruction and arsenic precipitation systems.

A sump pump is installed at the lime storage tank.

Cyanide

Cyanide will be delivered to site in one tonne boxes containing bulk bags of cyanide briquettes. The briquettes will be added to the cyanide mixing tank using an electric hoist and enclosed bag breaker and dissolved in process water to achieve the required solution strength.

The cyanide solution will be transferred to the storage tank for use in the process. Cyanide will be reticulated to the CIL tanks and dosed into the intensive leach reactor, by a pump and ring main as required. A dedicated pump will meter cyanide solution to the elution circuit as needed.

A common sump pump is installed at the cyanide and caustic areas.

Cyanide gas monitoring is installed in the cyanide make-up area.

Caustic soda

Caustic soda (sodium hydroxide) will be delivered to site in bulk bags of 'pearl' pellets. Caustic bags will be added to the mixing tank by electric hoist via a bag breaker and screw feeder on the receiving hopper and dissolved in raw water to the required solution strength.

The caustic solution will be metered to the intensive leach reactor, elution, pregnant solution tanks as required, with separate dedicated pumps for the cyanide destruction and arsenic precipitation facilities.

Hydrochloric acid

Concentrated hydrochloric acid (32% w/w) will be delivered to site in 1,000 L isotainers. The concentrated hydrochloric acid will be pumped into the acid mixing / storage tank where it will be diluted with the correct quantity of raw water to achieve the required acid wash concentration.

The dilute acid solution will be pumped to the elution circuit as required.

Flocculant

Flocculant will be delivered to site in 750 kg bulk bags. Flocculant will be added to the flocculant mixing plant storage hopper using an electric hoist and bag breaker. The vendor supplied package flocculant mixing plant will automatically mix batches of flocculant with filtered water and transfer the mixed flocculant to a storage tank after each mixing cycle is complete.

The flocculant solution will be pumped to the two thickeners with separate dedicated pumps. A small amount of flocculant is required by the intensive leach reactor and will be delivered from the pump delivering to the flotation tails thickener as required.

A sump pump is installed at the flocculant mixing plant.

Sodium metabisulphite

SMBS powder will be delivered in bulk bags. SMBS bags will be added to the mixing tanks by electric hoist via a bag breaker. The SMBS solution will be transferred to the storage tank for use in the process. SMBS solution will be metered to the cyanide destruction circuit by dosing pumps.

A ventilation fan is installed at the mixing and storage tank to disperse fumes.

A sump pump is installed at the SMBS mixing and storage tanks.

Copper sulphate

Copper sulphate will be delivered in bulk bags. Copper sulphate bags will be added to the mixing tanks by electric hoist via a bag breaker. The copper sulphate solution will be transferred to the storage tank for use in the process. Copper sulphate solution will be metered to the flotation and cyanide destruction circuits by dosing pumps.

Ferrous sulphate

Ferrous sulphate will be delivered in will be delivered in bulk bags. Ferrous sulphate bags will be added to the mixing tanks by electric hoist via a bag breaker. The solution will be metered from the mixing tank to the arsenic precipitation circuit by dosing pumps as required. A small buffer tank is connected to the mixing and storage tank for use while a new batch is being made-up.

Smelting fluxes

Sodium borate (borax), silica flour, sodium nitrate (nitre) and sodium carbonate (soda ash) will be used as fluxes for gold smelting. The fluxes will be delivered in 25 kg bags and mixed with the gold sludge prior to smelting.

Diesel

Diesel will be delivered to the plant from the mine site strange facilities. It will be stored in a plant diesel tank and pumped to feed head tanks that supply the gold room smelting furnace, carbon regeneration kiln and elution heater. Diesel is also supplied to the incinerator used to burn reagent packaging such as cyanide boxes and empty IBCs.

Activated carbon

Activated carbon will be delivered to site in 500 kg bulk bags. Carbon will be added to the last CIL tank, as required or via the regeneration kiln to allow fines removal on the sizing screen.

Grinding media

SAG mill grinding balls will be delivered to site in bulk. Grinding balls will be charged to the SAG mill via the SAG mill feed conveyor from a charging bin fed with a front end loader.

Grinding balls for the ball mill will be delivered in bulk bags and charged to the ball mill by loading a kibble from bags using a forklift into ball charging kibbles and lifting the kibble via a hoist into the ball mill feed chute.

Anti-scalant and de-scalant

Anti-scalant chemicals will be delivered to site in 1,000 L isotainers. They will be dosed into the raw and cooling water systems and the elution circuit using peristaltic dosing pumps as required.

Sulphamic acid H_3NSO_3 is used to de-scale the elution and regrind heat exchangers. A solution will be made up on site and a clean-in-place system used to periodically clean and flush the heat exchangers.

Reagent storage

Sufficient stocks will be maintained on site to ensure that supply interruptions due to transport or weather do not impact production. Plant reagents, excluding gold room consumables will be maintained at a minimum stock level of 60 days estimated consumption. Gold room flux inventories will be maintained at a minimum stock level of 30 days estimated consumption. Lime will be an exception, as it will be delivered by road tanker from a supplier's bulk storage depot nearby.

17.2.14 Services

Raw water supply and distribution

Raw water for the project will be extracted by pumps from the White Volta River and duty / standby transfer pumps used to deliver water to the raw water storage dam located east of the plant site. The impoundment will have a nominal capacity equivalent to ninety (90) days' worth of site raw water demand.

Raw water will be pumped by a single duty pump with an uninstalled standby from the dam to the plant 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 and cooling water make-up.

The raw water tank overflows to the raw water pond which has a capacity equivalent to a nominal 12 hours of plant raw water supply. The pond and tank are both connected to a suction manifold which supplies duty standby plant raw water pumps and low- and high-pressure gland water pumps. The high-pressure gland water systems are used for the multistage tailings pumps.

The tank overflow is also used for make-up to the process water pond.

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 using two hydrants at 36 m^3/h.

The fire water system will comprise a pump with an electric motor to supply fire water at the required pressure and flowrate and backup diesel driven pump that will automatically start in the event that electric power supply fails. An electric jockey pump will maintain firefighting ring main pressure when the system is not in use.

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.

Filtered and potable water

Raw water will supply the water treatment plant in the process plant. Water will be filtered and sent to the filtered water tank and a potable water treatment plant for sterilisation.

Additional ultra-violet sterilisation 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.

Filtered water will be used in the plant for cooling system make up, cyanide wash and eluant solution make-up, high pressure cathode wash sprays, mill feed chute seal water, carbon transfer and elution heat exchanger de-scalant make up.

A sump pump is installed at the plant potable water plant areas.

Process water

Process water will be separated into two systems. The two circuits are designed to operate separately to avoid cyanide and high pH water being returned from the CIL TSF to the flotation circuit. The presence of cyanide and/or a high pH in the flotation circuit will have a detrimental effect on flotation.

The grinding and flotation circuits use water recovered in the tailings and concentrate thickeners and decant water from the flotation tailings TSF with make-up from raw water.

Water recovered from the CIL TSF decant with make-up from raw water will be used in the regrind and the CIL circuits only. CIL TSF Decant water and raw water make-up are delivered to the plant cyanide water tank. Duty and standby pumps will deliver water to the regrind feed pump suction for dilution and the CIL area for washdown and service water. Cyanide process water will also be used for cyanide make-up in the reagent area.

The Process Water pond is filled from the flotation TSF decant and any excess thickener overflow water plus raw water make-up that overflow from the raw water tank. Duty and standby high pressure pumps will deliver water to the SAG mill pebble dewatering screen, ball mill trommel and gravity screen and flotation concentrate launder sprays plus make up to the thickener overflow water tank that supplies the low pressure process water pumps delivering water to the SAG and ball mill feed chutes and cyclone feed hopper dilution.

A duty fluidising water pump is provided for each gravity concentrator.

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 compressors. The instrument air will be dried and filtered before distribution. Separate air receivers supplying plant and instrument air are installed at key locations around the plant.

Duty and standby low-pressure air blowers will be installed for the flotation circuit.

A vacuum pressure swing adsorption type oxygen plant producing at least 90% purity oxygen gas with a capacity on 60 tonnes (100%) of oxygen per day will be installed to supply the requirements at the pre-aeration and CIL as well as the cyanide destruction and arsenic precipitation circuits.

Site spillage and run-off collection

An event pond will be constructed to contain spillage and run-off from the plant site. It will be located down slope from the plant area and be sized to contain the contents of the largest vessel on site if it were to rupture in an emergency.

The supernatant content of the pond can be reclaimed by a submersible pump to the cyanide process water circuit if it becomes full. High pressure water monitor guns can be used to reclaim settled solids in the pond after an event.

17.3 Plant design

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 (ICMI, 2016) 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 containment or treatment and bunding of relevant areas.

17.3.1 Plant layout and design considerations

The civil, mechanical and electrical design of the plant facilities will be based on standards deemed appropriate for the local climatic conditions and reflective of the local topography. The plant layout and equipment selection consider the requirements necessary for project implementation.

17.3.2 Site selection

The process plant location is shown in Figure 141. The plant will be to be located on relatively flat terrain to minimise earthworks and is to the north west of the open pit and directly north east of the waste dump. The tailings storage facility will be to the south and south west of the open pit.

The proposed plant layout is shown in Figure 142.

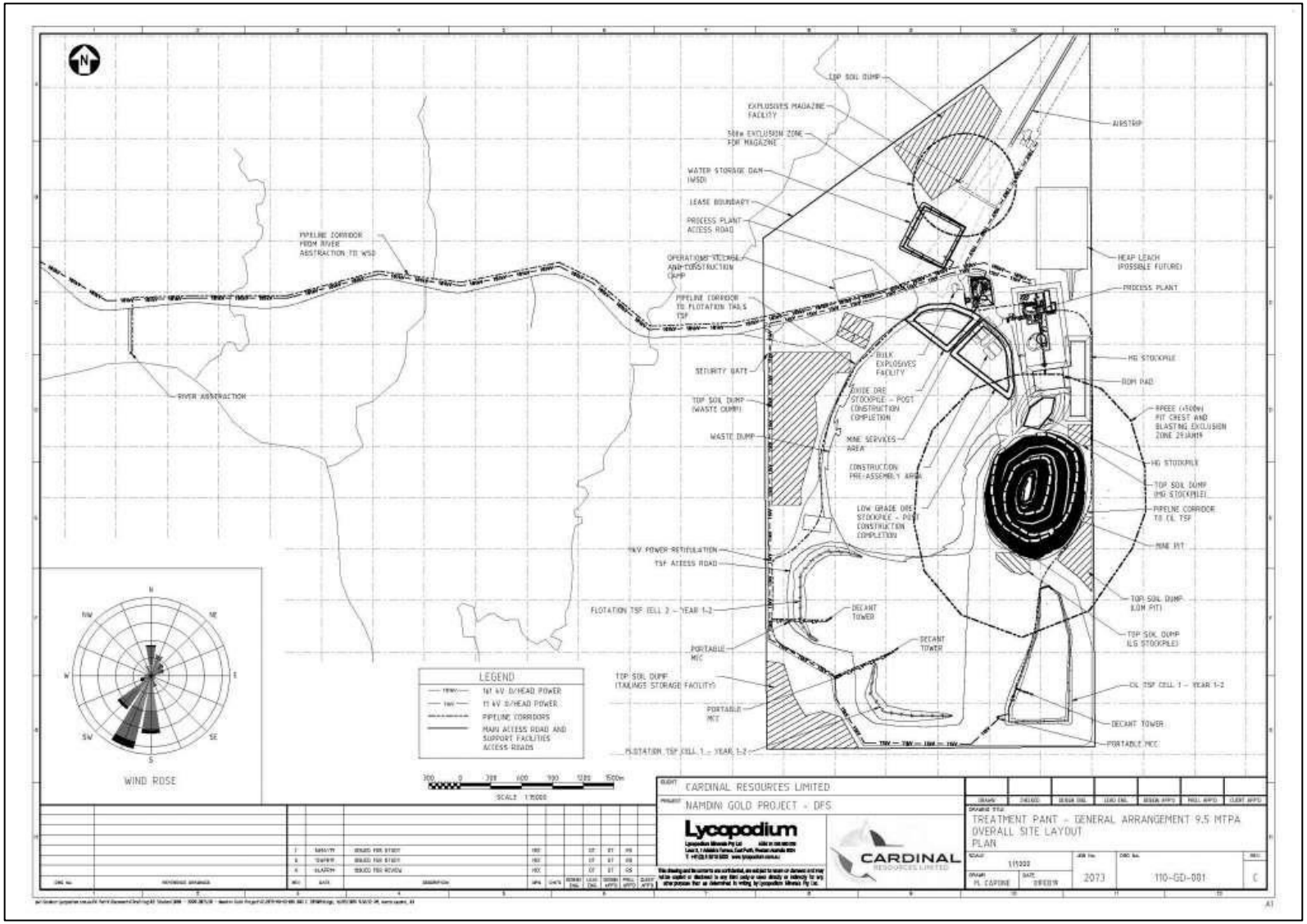


Figure 141: Proposed site layout

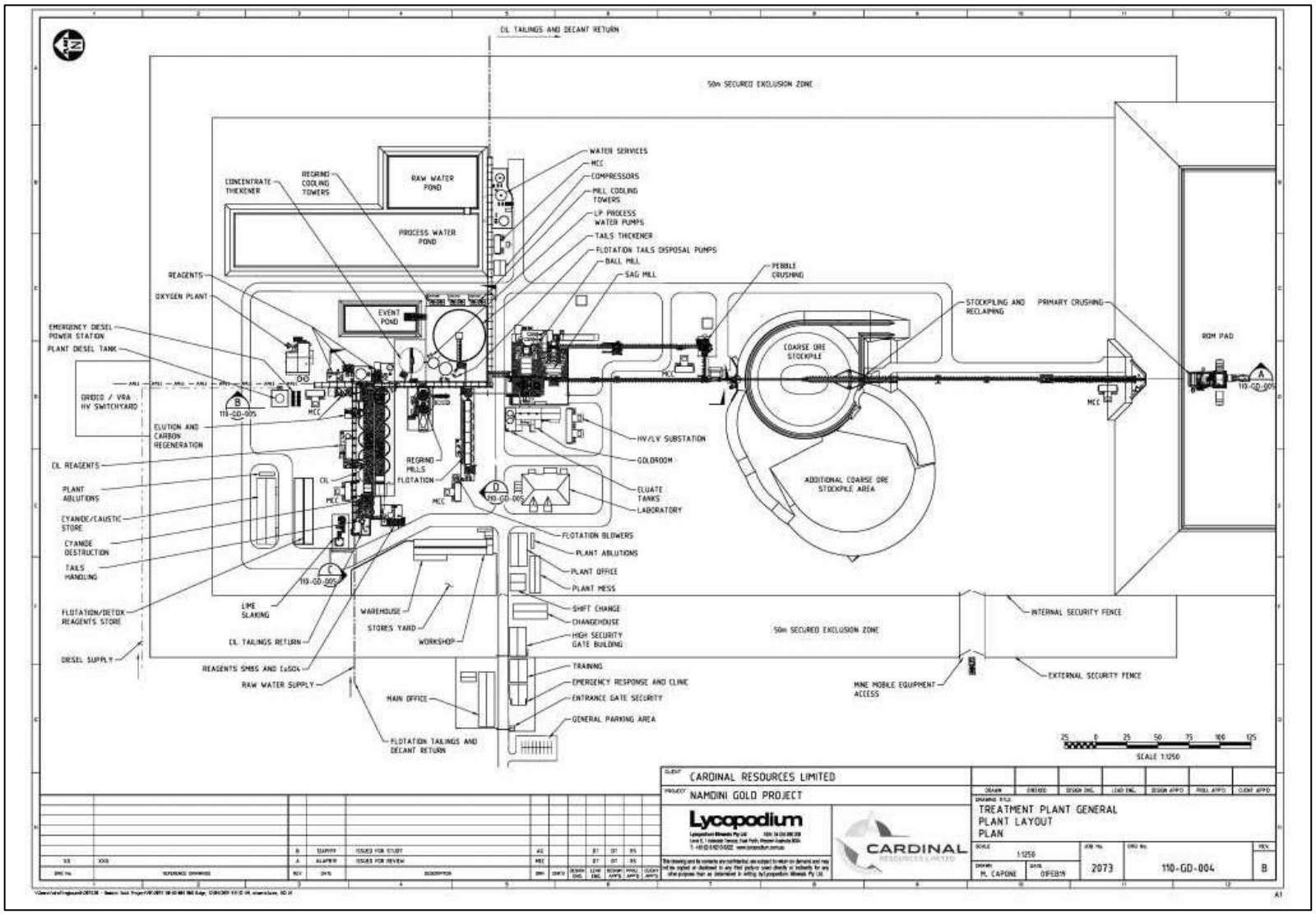


Figure 142: Proposed plant layout

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 flow where possible, 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 minimise 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 leaking vessel or run-off during a storm event and is sized 110% of the largest process tank.

Spillage within the reagents area will be directed to the locations nominated in Table 154.

Table 154: Spillage plan within reagents area

Reagent	Spillage directed to
Hydrated lime	Concentrate Thickener
Sodium cyanide and caustic	CIL Feed Box
Flocculant, flotation collector	Flotation Tails Thickener
Copper sulphate	Rougher Conditioning Tank
Sodium metabisulphite, 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.

Pipe off-take from these elevated structures enables piping to be distributed to the mechanical equipment. The distance between levels will be cognisant of maintenance requirements. Sufficient space allowance for light vehicle and maintenance access will be provided.

Low level pipe supports for water and other services will be also included in the layout.

Plant fencing and security

This is addressed in Section 18.16.

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 gravity gold recovery, loaded carbon screens, acid wash and elution and at the ROM tip at the primary crusher.

17.4 Instrumentation and control

The level of instrumentation and control will be selected to provide basic regulatory control to maintain steady operation with minimal process excursions. Following industry practice for similar size plants in similar locations, a supervisory control and data acquisition ("**SCADA**") and programmable logic controller ("**PLC**") architecture will be used for the plant wide process control system.

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 process control systems are not available, logic will be developed at the SCADA level for process control and monitoring of the vendor packages.

The control room will house multiple personal desktop computer-based operator terminals which function as the SCADA (control system) file servers as well as engineering or operator stations. The system will include a historian capability for data analysis and reporting.

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 mode selection will allow each drive to be started and stopped by the operator in the field by pushbutton. However, plant drives will generally be started remotely from the control room in Remote mode.

Status indication of process interlocks as well as the selected mode of operation will be displayed on the control room operator stations.

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 indicate and totalise the primary crushed ore tonnage. A second weightometer on the SAG mill feed conveyor will indicate and totalise mill feed tonnes. A weightometer on the pebble return conveyor will indicate and totalise 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 primary and regrind performance and flotation circuit metallurgical efficiency.

Density and flow meters on key streams (e.g. flotation tailings, regrind and CIL feed, CIL tailings, etc.) 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 a metallurgical balance for gold recovered to be calculated.

Regular 'in circuit' surveys will allow reconciliation of precious metals in feed and tails and inventory 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 flotation, CIL and elution and detoxification reagents will be measured and totalised using flow meters.

17.6 Additional information

Appendices provided in Lycopodium (2019a) include:

Appendix 17.1 Process Design Criteria

Appendix 17.2 Process Flow Diagrams.

18.0 PROJECT INFRASTRUCTURE

The information presented in this Section is derived from Lycopodium (2019b) and Knight Piésold (2019a).

Cardinal notes that it has sufficient area on its leases to cater for its planned infrastructure requirements.

Lycopodium and Knight Piésold completed FS level analyses covering all related aspects of the infrastructure requirement including power, water, road access and waste management.

The site is located approximately 50 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 to and from site as required.

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 Pwalugu 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 Pwalugu.

Infrastructure will include the following dedicated elements:

- Unsealed road
- High voltage power line
- Water supply pipeline from the White Volta River
- Raw water storage facility
- Airstrip
- Accommodation village
- Mine service area including heavy vehicle workshops
- Fuel depot.

18.1 Site facilities and layout

The site layout was developed based on the following information:

- Total available ore tonnage for the Life of Mine is 138.6 Mt
- Process throughput is 9.5 Mtpa.
- Flotation tail/concentrate split is 92.5% : 7.5%
- Tailings to Flotation TSF is 128.2 Mt
- Tailings to CIL TSF is 10.4 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.

The site infrastructure layout is shown in Figure 143.

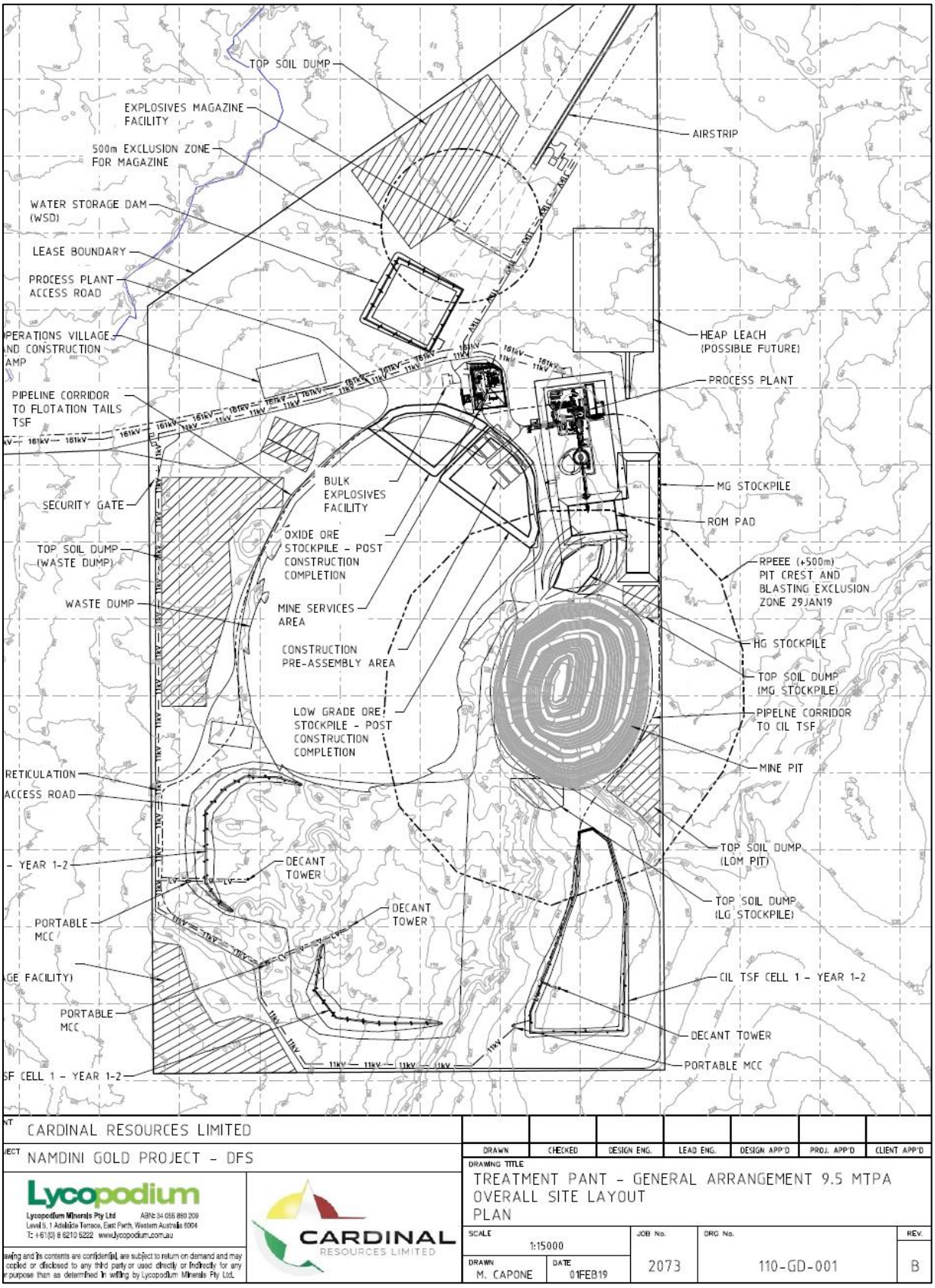


Figure 143: Site infrastructure layout

18.2 Waste dumps

Approximately 260 Mt (95 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 northwest of the open pit, and is bounded to the east by the Mining Lease. The waste dump design was carried out by Golder.

18.3 Process plant and mine services area

The process plant (and mine services area) will be located on relatively flat terrain to the north-northwest of the open pit and directly to the northeast of the waste dump.

18.4 Tailings Storage Facilities

The lease boundary, pit outline and waste dump footprint were provided by Knight Piésold. In addition, the consultant was advised of a number of medium- to high-grade anomalies to the north and northwest of the pit which were not yet sterilised. On this basis, the available area for siting of tailings storage facilities was limited to south and southwest of the open pit.

For the Pre-feasibility 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 FS, the TSF designs have been further optimized using these locations and incorporating more recent topography for the site area together with the updated design parameters.

18.5 Hydrogeology and hydrology

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. The implications of the work reported previously in the PFS (Golder, 2018b) and updated in Golder (2019f) are addressed in a discussion of the mine groundwater in Section 16.3.

The broader hydrogeological aspects of the region and the impacts on community use of groundwater are discussed in Section 20.3.3.

The hydrology is discussed in Section 20.3.4 and the results of water quality sampling are discussed in Section 20.3.5.

18.6 Geotechnical investigations

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. A geotechnical investigation of the site access road, water storage facility, airstrip and potential borrow sources was carried out as part of the FS. The scope of the site investigations comprised the following:

- 4 diamond cored drill holes within the Flotation TSF footprint
- 73 test pits within the Flotation TSF footprint
- 2 diamond cored drill holes within the CIL TSF footprint
- 24 test pits within the CIL TSF footprint
- 4 diamond cored drill holes within the process plant footprint
- 18 test pits within the process plant footprint
- 2 diamond cored drill holes within the water storage facility footprint

- 9 test pits within the water storage dam footprint
- 25 test pits within the airstrip footprint
- 15 test pits within the open pit footprint
- 50 test pits along the main site access road
- 25 test pits at other potential borrow materials sources
- 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 the TSF and process plant locations generally comprises 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.
- The sub-surface profiles at the other infrastructure locations comprises a variable thickness of silty clay/laterite gravel and residual soils overlying bedrock. A layer of topsoil, silty clay or sandy silt covers some of the proposed sites. The topsoil or clay/silt comprises low to medium plasticity clay or silt with sand, of firm to very stiff consistency. Laterite gravel covers the majority of the locations investigated. The sandy/clayey gravel is typically yellow brown to red brown in colour with low to medium plasticity fines, and of loose to medium dense consistency. The laterite gravel is underlain by residual soil or extremely weathered rock, which was typically excavated as clayey silt. The clayey silt is typically yellow-brown, grey or green in colour, low to medium or high plasticity, of firm to stiff consistency. Cobbles were excavated in some locations.
- Groundwater was not encountered during the investigations. It is understood that groundwater resides at an average depth of 42 m within the open pit footprint.
- 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. Details relating to specific borrow materials sources are available.

18.7 Roads

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-southwest to east-northeast 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.

Site roads will be 'fit for purpose' and will comprise haul roads for mining use, full width gravel roads for frequent traffic by site vehicles (light and heavy) and basic access tracks for infrequent access by light vehicles to site infrastructure.

18.8 Power

18.8.1 Power supply

The design is based on establishing a grid power supply to the process plant. This will be achieved by constructing a new GRIDCo switchyard near the process plant to step-down the incoming 161 kV supply to 11 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 kV, 45/50 MVA, switchyard, near the process plant.

No allowance has been made for Reactive Compensators to stabilise the grid power supply during start-up of mill drives, as it was assumed that the grid supply is steady and healthy, based on the previous observations and measurements carried out by Cardinal.

18.8.2 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 switchroom for distribution of 11 kV power supply to various sections of the process plant.

For the process plant utilisation, the 11 kV supply will be stepped down to 415 V at the switchrooms using 11 kV/415 V distribution transformers. These transformers will be fed from the main 11 kV switchroom/switchboard. These low voltage ("**LV**") switchrooms will house 415 V motor control centres ("**MCC**"s). These LV MCC/switchrooms will be located at various load centres of the process plant for LV power distribution.

18.8.3 Emergency power supply

Emergency power for process plant essential drives and facilities during grid power supply outages will be supplied from two 2.5 MVA, High Speed diesel generator units operating on diesel fuel oil. These emergency power generators will be connected to the process plant main 11 kV distribution switchboard to be operated during grid power outages. The emergency power is not meant to be used for sustaining the operations of the plant. The purpose of the emergency power supply is to provide power during grid power outage to the following critical areas of the process plant:

- Plant administration building.
- Guard house.
- 30% of area lighting.
- Control room power.
- Thickener rake system.

- CIL tank agitators.
- Thickener under flow pumps (50%).
- Fire detection and alarm systems.
- Essential facilities in the mine services area.

Based on the power supply reliability study done by Cardinal, the grid supply is assumed to be very reliable and the expected number of power outages per annum could be low, hence the utilisation of the emergency power station could be relatively low.

The changeover from the grid supply to the emergency power will be done manually. No automatic changeover has been allowed in the study.

18.8.4 Construction power

The construction power demand is estimated to be about 1,500 kVA during peak construction activities. One 2,500 kVA diesel 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.

18.9 Water management

18.9.1 Modelling

A detailed site-wide water balance model and stormwater management plan was developed for the PFS (Golder, 2018b) and is briefly discussed in Sections 20.3.5.4 and 20.3.5.5.

A water management model was also developed to understand the 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 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 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 will only require 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 will require 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 from the river abstraction system.

18.9.2 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 sterilisation 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.

18.10 Sewage and waste management

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 has been 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.

18.10.1 Solid wastes

Wastes 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 burnt and the ashes buried, under supervision, on site beneath mine waste to prevent unauthorised use.

18.10.2 Hydrocarbon wastes

Waste lubricating oils will be returned to the supplier for recycling. Hydrocarbon contaminated materials will be spread on volatilisation pads for decontamination before disposal in landfill sites.

18.11 Tailings Storage Facilities

18.11.1 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 in terms of percent water release, 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 pH values. Therefore, the Diorite and Metavolcanics flotation tailings are classified as acid consuming (AC) and the Granite rougher tailings as non-acid forming (NAF) using the AMIRA (2002) classification standard (Table 155).
- The CIL tailings sample recorded a positive NAPP and a low and a low net acid generation pH, resulting in a classification of potentially acid forming (PAF).
- On the basis of the multi-element results, both the Flotation and CIL TSFs are 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 ongoing 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 a compacted soil liner with overlying HDPE liner and underdrainage system.

Table 155: Acid formation classification (AMIRA, 2002)

Classification	NAPP ($kg/H_2SO_4/t$)	NAG pH
Potentially acid forming (PAF)	> 0	< 4.5
Non-acid forming (NAF)	< 0	≥ 4.5
Acid consuming (AC)	< -100	≥ 4.5
Uncertain (UC)	> 0	≥ 4.5
	< 0	< 4.5

18.11.2 Design of the 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 474 Ha and is designed to accommodate 139 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 (Knight Piésold, 2019b).

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 will drain 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 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.

18.11.3 Design of the 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 earth fill 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 16 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 (Knight Piésold, 2019b) to a final elevation of 266.0 m RL.

The TSF basin area will be cleared, grubbed and topsoil stripped, and a 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 smooth HDPE geomembrane liner.

The TSF design incorporates an underdrainage system comprising a network of branch and collector drains. The underdrainage system will drain 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 mounted in a decant tower located along the western embankment of the facility.

In order to mitigate seepage losses through the basin area, minimise 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
- Leachate collection and recovery 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 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.

18.11.4 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, construction management ("**EPCM**") contractor staff and senior contractor personnel subject to availability.

18.12 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 centres 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 and 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.

18.13 Communication system infrastructure

Site communications will consist of the following:

- Internal communications and IT services will be via 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.

18.14 Fuel and lubricant supply

Diesel fuel will be stored on site. Allowance has been 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.

18.15 Explosive storage and handling

It is anticipated that a contract will be entered into with a recognised supplier of mining explosives to establish their own facilities on site and supply emulsion explosives, initiators, detonators and other blasting consumables as needed.

18.16 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 decentralised infrastructure are located will be patrolled by the security team.

The process plant, mine services area and general administration area 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 via 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 '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 unauthorised access.

The tailings storage facility 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.

18.17 Operational facilities

Site facilities, buildings and the airstrip have all been allowed for in the design and costings. An Administration Office, site warehouse and clinic combined with the emergency response facility will be located together.

Workshops, warehouses and the like will be of structural steel frame and metal cladding construction on concrete slabs. Office and amenity area structures will generally be of transportable or prefabricated style construction although concrete blockwork construction will be considered to provide additional local content if the schedule allows.

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 and mining technical team who will share the contractor's facilities such as change-rooms to avoid duplication.

The Mine Services Area facilities based on a quotation are summarised below:

- Vehicle workshop, tyre store and warehouse
- Mine vehicle wash-down bay
- Mine contractor's offices

- Explosives contractor facilities
- Lube storage
- Crib/Training rooms
- Ablutions.

18.18 Airstrip

An airstrip will be constructed to service the site with freight shipments both inwards and outwards, and gold shipments outwards. The airstrip will be located approximately 2.5 km to the west-southwest of the accommodation camp and directly to the south of the site access road.

18.19 Additional information

Appendices provided in Lycopodium (2019b) include:

Appendix 18.1 TSF Drawings etc.

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$1,350/oz for the purposes of the financial model at the date of this FS.

The gold price assumptions used for the purposes of this FS and the project gold price for the financial analysis are presented in Table 156.

Table 156: Gold price assumptions

Selection case	Gold price
Trial open pit optimizations were run in Whittle 4X at a US$1,300/oz gold price to define the base of potentially economic material. Five cut-back pits (Phases) were then selected and full mine designs applied.	US$1,300/oz
Mine scheduling was used to maximise 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 shell of approximately $1,235/oz using estimated LOM input prices and costs. Pit shells were converted into engineering designs.	US$1,235/oz
The financial model input gold price was based on a long-term average of US$1,350/oz.	US1,350/oz

The FS has been evaluated at a gold price of US$1,350 per ounce. The average US$ gold price per ounce for the period from October 2017 to October 2019 is US$1,387 per ounce. During this period the US dollar gold price has been in steady uptrend from a low of US$1,177 in August 2018 to a high of US$1,550 per ounce traded during September 2019.

At this time there are no contracts or arrangements material to the issuer that are required for development of the Project, including for mining, concentrating, smelting, refining, transportation, handling, sales and hedging, or forward sales.

20.0 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

The information in this Section is derived from Golder (2019h) which is based on NEMAS (2017; 2018), and on Geosystems (2019). It is presented in a similar format to that in the PFS (Golder, 2018b) which is a comprehensive report provided on Sedar (https://www.sedar.com). Additional detail is contained in Appendices which are listed at the end of this Section.

20.1 Introduction

Cardinal Resources Limited (Cardinal) engaged NEMAS Consult Ltd (NEMAS) as an independent consultant to undertake the Environmental Impact Assessment (EIA) study for the Namdini Gold Project (the Project) in Ghana, Africa. The 19.5 km^2 concession/lease (the Project area) is located near the town of Datoko within the Talensi District in the Upper East Region of Ghana, 50 km south east of Bolgatanga, the regional capital, about 20 km from Tongo, the District capital, and approximately 776 km north of the National Capital, Accra.

The Project falls under Schedule 2 of the *Environmental Protection Agency Act 1994* (EPA Act 490) (mining and processing of minerals in areas where the mining lease covers a total area in excess of 10 ha), and the Environmental Assessment Regulations 1999 (LI 1652) whereby the Project is subject to a mandatory Environmental Impact Assessment (EIA).

The Project's EIA process involved a scoping phase leading to preparation of an EIA Scoping Report in 2017 and an EIA study phase culminating in preparation and submission of an Environmental Impact Statement (EIS) report to the Ghanaian Environmental Protection Agency (EPA) in 2018. NEMAS completed and submitted the Draft EIS (NEMAS, 2018) for the Project. The EIS Report is still in Draft form pending approval from the EPA.

In accordance with the EPA's Environmental Assessments Regulations (LI 1652) 1999, Cardinal has also registered the Tailings Storage Facility (TSF) portion of the Project with the EPA as a standalone project. This enables a TSF specific EIA to be conducted to ultimately obtain an environmental permit for the construction, operation and closure of the TSF. Therefore, Cardinal engaged Geosystems Consulting Limited (Geosystems) to prepare a Draft TSF Scoping Report (Geosystems, 2019), which precedes the EIA specifically for the TSF. The Draft TSF Scoping Report sets out the scope and extent of the EIAs to be carried out and includes the draft terms of reference to be addressed in the EIS during the EIA process.

Information provided in the following key documents has been summarised in this Section:

- Namdini Gold Project Environmental Impact Assessment Scoping Report (NEMAS, 2017)
- Namdini Gold Project Pre-feasibility Study (Golder, 2018b)
- Namdini Gold Project Draft Environmental Impact Statement (NEMAS, 2018)
- Namdini Gold Project Tailings Storage Facility Draft Scoping Report (Geosystems, 2019).

This Feasibility Study report provides a high-level summary of the environmental and social baseline conditions with a focus on key findings from the baseline studies conducted to date for the Project. Numerous desktop and field studies and assessments were undertaken to support the EIA Study and EIS, including examination of stakeholder engagement, flora, fauna, aquatic biology, hydrology, soils, environmental quality, archaeology and culture.

The various acts and regulations that are applicable to obtaining a 'social licence to mine' in Ghana, and with which Cardinal is complying, are:

- Environmental Assessment Regulations, 1999 (LI 1652)
- Environmental Protection Agency (EPA) Act, 1994, Act 490

- Forestry Commission Act, 1999 (Act 571)
- Minerals & Mining (Compensation & resettlements) Regulation, 2012 (LI 2175)
- Minerals & Mining (Health, Safety and Technical) Regulation, 2012 (LI 2182)
- Minerals and Mining (General) Regulations, 2012, LI 2173
- Minerals and Mining (Support Services) Regulations, 2012 (LI 2174)
- Minerals and Mining Act, 2006 (Act 703)
- Mining and Environmental Guidelines, May 1994
- Water Use Regulations, 2001 (LI 1692).

20.2 Permitting and approvals

Under the EP Act 490 and in accordance with LI 1652 (Regulations 1 and 2), 'undertakings' (i.e. the proposed mining project) require registration and issuance of an environmental permit. Cardinal registered the Project with the EPA for environmental permitting, under the title 'Savannah Gold Mining Project'. In 2016, the EPA indicated that the Project required a full-scale EIA Study and submission of an EIS to the EPA.

In accordance with Regulation 11 of LI 1652 and an EPA letter dated 23 November 2016, an EIA Scoping Report was prepared by NEMAS (2017) and submitted to the EPA on 22 June 2017. The submission of the EIA Scoping Report initiated the process of the subsequent EIS in accordance with Regulations 15(1b) and (1c) of LI 1652 and Ghana's EIA Procedures. The EIA Scoping Report (NEMAS, 2017) provided the Terms of Reference (ToR) for the subsequent EIA.

Upon review of the EIA Scoping Report and draft ToR, in letter dated 13 July 2018 the EPA requested that the Company prepares an EIS for the Project.

NEMAS conducted the EIA Study to support the preparation and submission of the draft EIS Report (NEMAS, 2018) in October 2018 for the Project in accordance with the EPA Act 490. The EIS Report is still in Draft form pending approval from the EPA.

To advance the development of the Project, Cardinal also applied to the EPA for an environmental permit to construct and operate the proposed TSF portion of the Project within its concession area. In a letter dated 15 March 2019 (reference Number: CM6954/18) the EPA advised Cardinal to conduct an independent standalone EIA for the TSF portion of the Project as part of the process for the granting of an environmental permit for the proposed TSF. The environmental permit for the TSF must be obtained independent of the environmental permit for the main Project.

20.3 Environmental and social context

The following Sections provide a high-level summary of the environmental and social baseline conditions with a focus on key findings from the baseline studies conducted to date for the Project. Numerous desktop and field studies were undertaken to support the EIA Study and draft EIS (NEMAS, 2018), including stakeholder engagement, flora and fauna study, hydrological assessment, aquatic biological study, soils study, environmental media quality assessment, archaeological, and cultural profiling.

20.3.1 Climate

The Project is located in a dry part of Ghana, within the Guinea savannah climatic zone. The region's climate is influenced by the Southwest Monsoon Winds and Northeast Trade Winds. Warm, moist southwest monsoon winds from the Atlantic Ocean and the warm, dry Tropical Continental air mass, which also induces the dusty Harmattan winds from the Sahara Desert move from the opposite sides of the Equator and merge into a low-pressure belt, known as the Inter Tropical Convergence Zone (ITCZ). The effects of the Harmattan winds (i.e. poor air quality, low visibility) occur between November to February and are most severe in the northern regions of Ghana, where the Project is located. These months also coincide with lower rainfall and relative humidity, higher temperatures and strong winds.

The region experiences an annual wet season between May to October with intermittent monthly rainfall rise, which typically peaks in August, and a dry season between November to April. Typical annual rainfall ranges from 786 mm to 1464 mm, and rainfall events are often intense and torrential, and associated with strong winds and thunderstorms. Mean day time temperatures range from 30.6°C in August to 40°C in March. Mean relative humidity ranges from 13% in January to 70% in August.

20.3.2 Topography, geology and soils

The topography of the Project area is relatively flat with the lowest elevation at 170 m AMSL and the highest elevation at 240 m AMSL. Sections of the Project area have been impacted by illegal mining activities, with open pits remaining across the site.

The Project lies in a Paleo-Proterozoic granite-greenstone terrane in the north-eastern district of Ghana and lies along the northwest trend of the Bole Greenstone Belt. It is separated from the Nangodi Belt by the Bole-Bolgatanga fault. Much of the area to the south of the tenements is covered by later Voltaian Basin sediments.

Soil in the area is slightly acid to neutral with moderate to high organic matter, and low to moderate ion exchange capacity. The overall fertility assessment from chemical analysis indicated poor soil, partly due to continuous cropping, prolong drought and ground disturbing activities by artisanal mining.

20.3.3 Hydrogeology

Groundwater resources are an important source of fresh water for local communities within and around the Project area. Mine development may pose a risk to these resources due to mine dewatering activities, particularly given the nature of local aquifer conditions.

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. Characterisation of groundwater quality by sampling and laboratory analysis, groundwater monitoring and hydraulic testing was completed.

The water table level is summarised 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.

Golder Associates Pty Ltd has developed models to estimate pit inflow rates and groundwater level drawdowns in areas away from the immediate mining area. The average predicted pit inflow rates are between 22 and 26 litres per second (L/s) at the end of the mining period. Maximum pit inflows are simulated for the fourth year of mining when pit inflows will increase to between 27 and 33 L/s.

The presence of faults may generate higher pit inflows but are not expected to significantly change the total inflow. One or several sumps at the base of the Pit are likely to be sufficient to dewater the Pit.

The Accra community wells are likely to be impacted by mine dewatering due to groundwater level drawdowns, with a simulated drawdown of 28.6 m at groundwater bore NMBH02 and 52.2 m at the Accra site after 14 years of mining. The Bingo community wells will also be impacted with drawdowns of up to 8.6 m. Ongoing monitoring of groundwater levels will be important during mine operations to confirm that model assumptions and predictions are valid and provide early-warning of a need to implement measures (if indeed) to prevent these community wells from being negatively impacted.

20.3.4 Hydrology

A comprehensive hydrology field program was completed and reported in the PFS (Golder, 2018b) based on work done by Golder Africa (2018a) and updated by Golder Africa (2018c).

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 (Figure 144).

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. The Project area drains to the west towards the White Volta River 6 km west of the site, which eventually forms part of the Volta River. The Project area drains to a tributary within the White Volta sub-basin. The Red Volta is located to the east of the Project area.

A small ephemeral stream, the Zoan Buliga is the only visible source of surface water that traverses through the Project area. It passes through the northern boundary of the Project area before joining the White Volta. The Zoan Buliga stream flows from a few days to weeks during the months of heavy rainfall, typically between May to October. The White Volta River is the proposed source of raw water supply for the Project. A few stagnant ponds were also observed in the Project area, which were as a result of rainfall collecting in pits left by illegal miners.

Figure 144 shows the Project area in relation to nearby rivers.

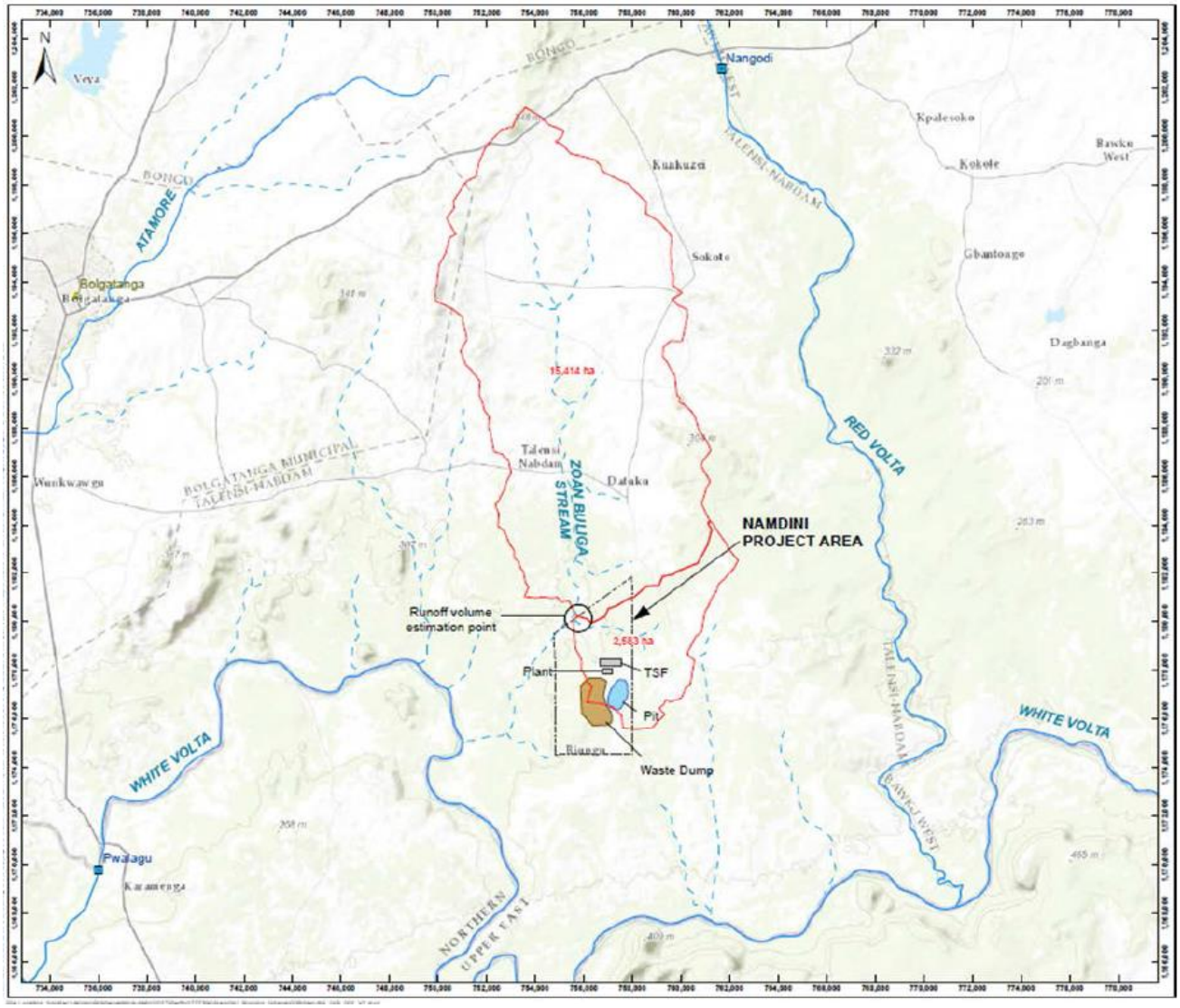


Figure 144: Location of the Project area in relation to nearby rivers

20.3.5 Water quality

20.3.5.1 Surface water

Golder's search of the available hydrological databases showed that there is limited site specific hydrological data. As part of the hydrological assessment, Golder Africa (2018c) collected additional baseline data by:

- Installation of on-site rainfall gauges
- Installation of a flow gauging station which will assist in defining the flow regime of the surrounding water courses and the availability of water for the plant if needed. The long-term measurement of the flow within the White Volta tributary will inform the pumping decisions for water required at the Plant
- The creation of a water monitoring network and subsequent quality measurements of the major streams around the lease area.

Golder set up a surface water baseline monitoring programme to supplement the current data sets, provide data for later stages of the project development and provide baseline data against which impacts of the proposed project on the receiving environment can be measured.

Three site visits were undertaken to set up the monitoring programme. The first visit was undertaken during the rainy season from 29 September 2017 to 4 October 2017 and the second site visit during the dry season was from 30 November 2017 to 2 December 2017. The third site visit was undertaken during the rainy season from 4 September 2018 to 12 September 2018.

Baseline surface water quality was monitored from four locations within the White Volta and one at Zoan Buliga Rivers, with sampling following standard UNESCO/WHO/UNEP protocols. NEMAS (2018) identified that there were exceedances of the World Health Organisation ("**WHO**") Maximum Permissible Levels ("**MPLs**") in electrical conductivity, total suspended solids and turbidity at three sampling location and in iron at all White Volta sampling locations.

20.3.5.2 Groundwater

Groundwater quality was analysed from ground water sources (boreholes, hand-dug wells and dug out) of host communities within and around the Project. NEMAS (2018) identified that six (EC, alkalinity, turbidity, sodium, iron and manganese) out of the 28 water quality parameters analysed, registered concentration levels higher than the WHO MPLs from some of the sources investigated.

20.3.5.3 Training site personnel

Site personnel based at the Namdini Mine were trained during the third site visit in September 2018, to ensure the continuation of the surface water monitoring programme by Cardinal Resources. The training was run concurrently with surface water monitoring undertaken during that visit. The training included step-by-step walkthroughs to take surface water flow measurements, water quality samples and to download and maintain the installed rain gauges.

20.3.5.4 Site water balance model

As part of the PFS (Golder, 2018b) the hydrology scope included development of a site-wide water management model. The model integrated the Plant, Pit, TSF, Waste Rock Dump and groundwater aspects of the mine water system into a model. The model was applied to determine mine make-up water requirements and to size pollution control dams to meet the spill criteria. Specific aspects of this plan relating to the TSFs are addressed in Section 18.9.1.

20.3.5.5 Stormwater management plan

As part of the PFS the hydrology scope also included development of a stormwater management plan for the site to prevent flooding of the pit and separate the runoff from clean and polluted catchment areas. The system of channels, berms and dams was sized to meet the design criteria and costed. This work was fully developed in reports by Golder Africa (2018a; 2018b; 2018c).

20.3.5.6 Water quality summary

The hydrology studies have shown:

- The monitoring of the stream-flow, rain gauges, and water quality should be continued.
- More instantaneous velocity, flow depth and in-stream flow monitoring is required, with the emphasis on the planned abstraction source, the White Volta River.
- The water quality of local runoff generally falls within WHO drinking water standards. Elevated levels of total dissolved solids, manganese and aluminium should be tracked by future monitoring.

20.3.6 Air quality and noise

Total Suspended Particulates (TSP) and Particulate Matter (PM) less than 10 microns in diameter (PM_{10}); gases comprising sulphur dioxide (SO_2) and nitrogen dioxide (NO_2) and noise dispersion were measured at six different sites within over three sampling rounds (December 2017, March 2018, June 2018). The six environmental media sampling sites include the communities of Bingo, Accra, Tarkwa and Wankala (Figure 145). The Bingo and Accra communities are located within the Project area.

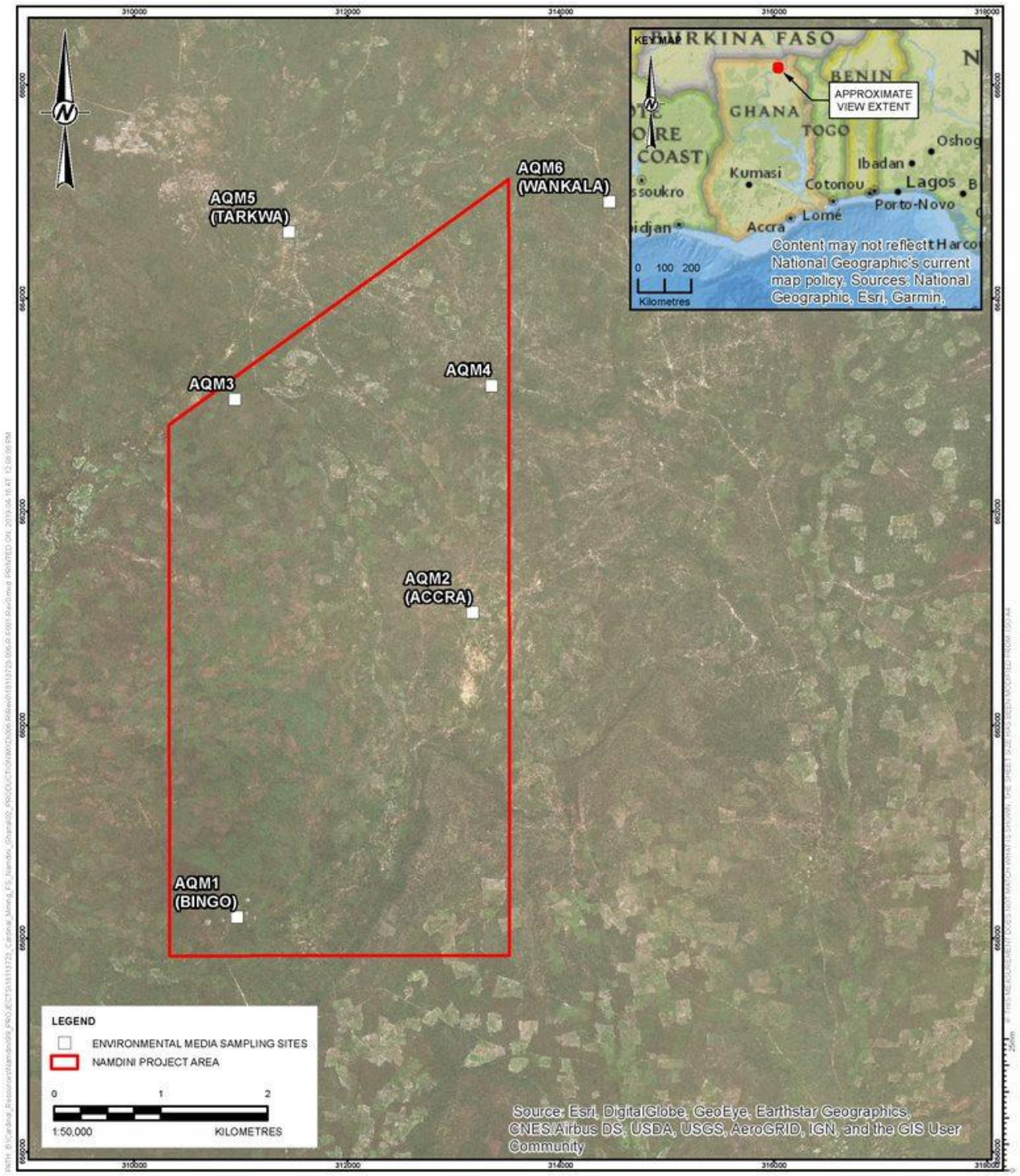


Figure 145: Project area environmental media sampling

20.3.6.1 Dust concentration

Maximum recorded TSP concentration was 81.9 µg/m^3 from the Tarkwa site and maximum PM_{10} of 4.5 µg/m^3 from the Accra site. These levels were all lower than the Ghana EPA recommended values of 230 µg/m^3 TSP and 70 µg/m^3 PM_{10}. Primary sources of dust were cleared or bare land, old small-scale illegal mining (galamsey) as areas, and traffic.

20.3.6.2 Gas emissions

Ambient monitoring was completed to ascertain the current concentrations of SO_2 and NO_2 pollutants, December 2017, March 2018 and June 2018. The highest 24-hour average ambient gases sampling results recorded were 109.2 µg/m^3 at the Tarkwa site in December 2017 and 36.9 µg/m^3 in March 2018 at the Bingo site for SO_2 and NO_2 respectively.

SO_2 and NO_2 concentrations were all minimal and within the Ghana EPA limit for industrial sector. Concentrations recorded could be attributed to motorcycle emissions since it is the main source of shuttle transportation between the community's diesel-powered generators used by the illegal small scale mine operators.

20.3.6.3 Noise dispersion

Noise levels from all sites recorded lower values with respect to the 70 dB(A) MPL set by the Ghana EPA. The highest noise level of 50.1 dB(A) was recorded at the Tarkwa site during the second monitoring round in March 2018. Baseline noise levels were mainly generated by domestic noise sources within the communities, crushing noise of small-scale miners' machines and vehicular movement.

20.3.7 Flora and vegetation

The Project area lies in the northern Guinea Savannah vegetation zone of Ghana, and vegetation is characterised by a continuous grassy ground layer with an open canopy tree stratum. The structure of the vegetation of the Project area has uniform species composition as a result of regular annual grass burning. The trees and shrubs that occur in the Project area are fire-tolerant and are mostly left standing during cultivation, however the trees are stunted and show visible damage caused by wildfires.

The vegetation of the Project Area was recorded to be highly disturbed, and areas have been extensively cultivated. Weeds occur in the same proportion as native Savannah species which could be attributed to cultivation and fires.

A total of 39 species belonging to 35 genera in 13 families were identified in the Project 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 composition.

Most of the flora species recorded within the Project area occur widely in the Savannah and were stated by NEMAS (2018) to have no immediate conservation concern. However, African oak (*Afzelia Africana*) is listed as a Red Star Species in the Forest Reserves of Ghana Geographic Information Exhibitor (FROGGIE) and the shea butter tree (*Vitellaria paradoxa*) is listed as Vulnerable on the International Union for Conservation of Nature's (IUCN) Red List of Threatened Species. Some of the tree species recorded in the Project area are of economic and medicinal importance, such as the shea butter tree (*Vitellaria paradoxa*), African oak (*Afzelia africana*), African copaiba balsam (*Daniellia oliveri*), Baobob tree (*Adansonia digitate*) and African locust bean (*Parkia biglobosa*).

Annual bush burning and illegal mining activities are existing threats to the flora and vegetation of the Project area.

20.3.8 Fauna

Several species of mammals and birds occur in the Project area. The proximity of the Project area to the eastern wildlife corridor means that occasionally large mammals stray into the Project area. Some of the commonly observed wildlife in the area includes the Patas monkey (*Erythrocebus patas*), Green monkey (*Cercopithecus aethiops*) and Senegal galago (*Galago senegalensis*).

The African elephant (*Loxodonta africana*), Warthogs (*Phacochoerus aethiopicus*) and the West African Bushbuck (*Tragelaphus scriptus*) are occasionally observed in the Project area. The Togo hare (*Lepus zechi*), Giant rat (*Cricetomys gambianus*) and the Cane rat (*Thrynomys swinderianus*) are commonly found in the area. The avifauna includes species such as the Grey Heron (*Ardea cinerea*), Double-spurred francolin (*Francolinus bicalcaratus*), and Hammerkop (*Scopus umbretta*). These species have a status of 'least concern' under the IUCN Red List, except for the African elephant which has a status of 'vulnerable'.

Annual bush burning and illegal mining activities are existing threats to the fauna of the Project area.

20.3.9 Aquatic ecology

Typical Savannah vegetation for swamps and floodplains are found in the Project area. Plant diversity were dominated by herbs, legumes and grasses which are of least concern in the IUCN ratings. The most dominant grasses were *Typha sp*., *Sporobolus spp*., *Echinocloa spp*., and *Panicum sp*. found at the margins of the White Volta River.

Dominant aquatic and semiaquatic fauna are fish, frogs, monitor lizards, snakes, crocodiles and hippopotamus in decreasing abundance. As part of the fieldwork conducted for the aquatic biological study, experimental fishing was carried out at various locations along the White Volta, along the western corridors of the Project area. Species caught included *Synodontis clarias*, *Synodontis eupterus*, *S. ocellifer*, *Schilbe mystus*, *Alestes nurse*, *Mormyrus rume*, *Gnathonemus tamandua*, *M. macrophthalmus*, *Heterobranchus bidorsalis*, and *Tilapia zillii.*

The sediment was very fine and clayey at all areas visited along the river-banks, with visible presence of gastropod mollusc shells and polychaete fragments and feeding tubes of polychaetes in moderate abundance indicating moderate productive water body.

20.3.10 Social environment

A social baseline study was conducted as part of the EIA Study to outline the socio-economic and cultural settings of the host communities at the pre-project phase and to use it as a reference to measure impact (positive and negative) of the Project on the host communities (NEMAS, 2018).

The host communities affected by Cardinal's 19.5 km^2 Project area are the Biung (Bingo) and Datoko which also captures Wankala and Zangore. In addition, the Accra Site which is an illegal mining community established approximately 20 years ago, is centrally located within the Project area and will be subject to resettlement and relocation. Other peripheral communities are referred to as the Tarkwa and Obuasi sites.

The dominant tribe is the Talensi which forms about 90% of the local population with few migrants from Southern Ghana and neighbouring countries. The area is under the Nabdam Traditional Council and is male dominated, both in terms of demography and cultural practices. The people are mostly traditionalist and/or Christians with Muslims forming the minority.

Land tenure in the Talensi community is customary, whereby the land is either owned by individuals or by families in portions and held in trust for the dead, present and future by the Tindana, who is the priest and custodian of the land. In some communities, land ownership is vested in one person (the chief) who also serves as the Tindana and who holds the allodial title. The power to decide on the transfer of ownership is vested in the individuals or families. Where the chief holds the allodial title (in Pusu-Namogo, Tongo-Beo and Buing) he has the right to decide on land ownership. Though most often inimical to physical development, the religious adherence to these conventions provide an avenue to protect and conserve land and natural resources.

20.3.11 Stakeholder consultation and engagement

As part of the EIA Study, and to support the draft EIS, Cardinal identified approximately 29 key stakeholders for the Project, including government regulatory agencies, government institutions, traditional and political authorities, Project communities and leaders, and non-government authorities.

Extensive social consultations and stakeholder engagements have been held with key stakeholders, including one-on-one interviews with approximately 298 respondents from the host communities of Buing, Accra, Tarkwa, Datoko, Obuasi and boundary communities such as Bapella Tolla Degare, Zingoire and Kejetia.

All key stakeholders consulted expressed their support for the Project. Key concerns raised related to security, Social Corporate Responsibility, payment of royalties, and protection of juveniles from engaging in artisanal mining activities and advised Cardinal to operate an ‘open door’ policy based on mutual respect.

Other concerns raised during social consultations and focus group discussions included:

- Fear of loss of jobs (mainly expressed by illegal miners)
- Fear of relocation (expressed by illegal miners especially at the Accra Site)
- Fear of loss of gathering opportunities (mainly expressed by women)
- Fear of deprivation of support to continue education (mainly expressed by recurrent miners who come on holidays to engage in illegal mining to support their education)
- Fear of water pollution (expressed by all)
- Fear of increase in social vices especially prostitution (mainly expressed by the women)
- Expectation of socio-economic improvement and job creation for indigenes (expressed by all, especially the youth).

With regards to the proposed TSF portion of the Project, Cardinal have held initial consultations with leaders of host communities, traditional area and identified groups, governing District Assembly, Talensi District Assembly, regional and national regulatory agencies, and government institutions. These consultations identified social and environmental issues and concerns that will need to be addressed in the EIA for the TSF. The Draft Scoping Report for the TSF (Geosystems, 2019) provides the list of stakeholders and issues raised during initial consultations. Issues and concerns raised by stakeholders included the topics of:

- Permitting – Compliance, regulator involvement, etc.
- Land take and compensation – Adequate compensation, demarcation, resettlement, etc.
- Health and safety – site security, etc.
- Communication – Host community involvement, cooperation with traditional authorities, etc.
- Socioeconomic issues – Community involvement, adequate community liaison, provision of community assistance, etc.

- Employment – Local communities considered for employment, unskilled jobs, etc.
- Other issues such as royalty payments, Memorandum of Understanding, etc.

Detailed stakeholder mapping is proposed to be undertaken to identify relevant stakeholders and stakeholder engagement is also proposed to develop an 'issues trail' so Cardinal can address these in the EIS stage of the TSF portion of the Project.

20.3.12 Local economy

Most residents either have no formal education or are without any employable skills. Their economic mainstay is subsistence farming with maize and millet being the major crops, and livestock rearing, whilst communities within the fringes of the White Volta use fishing to supplement household incomes. Other major livelihood activities include harvesting and processing of local fruits to produce groundnut oil, sheabutter, dawadawa (food additive or ingredient), as well as processing of local paddy rice. These are primarily practiced by women.

Illegal artisanal gold mining has dominated economic activity particularly within the Project area over the past 20 years. Evidence of artisanal workings occur throughout the Project area. This has led to establishment of the Accra community as well as other adjoining towns. Concerns regarding loss of access to land and therefore loss of this economic activity due to project development is a key concern for these communities.

The potential negative impacts and proposed mitigation measures regarding the socio-economic environment are provided in Table 157.

20.3.13 Archaeology and cultural heritage

Fifteen heritage resources, two archaeological settlement sites and 13 shrines were identified and documented at Biung. Three shrines and 14 archaeological settlement sites were identified and documented at Datoko.

A total of 155 artefacts were collected from excavated units and the surface, comprising local and foreign ceramic sherds, glass fragments, metal objects, bones, querns, and one plastic bead.

The archaeological survey conducted at Biung and Datoko found archaeological sites and shrines within or adjacent to the Project area. These included Kpinkpalyen, Kpagsab Babuk, Sirkugri, Zietod, Voyen Wari, Zoozee, and Kpari Gayi.

One of the old settlements; Nanbang, Yin and Tobik Dabogu is within the Project area. Consultation with the relevant local communities must be undertaken prior to commencing mining activities to promote cooperation and mutual understanding regarding these resources.

In a situation of relocation of communities, the Company would require further collaboration with the traditional authorities regarding the removal of affected sacred sites (NEMAS, 2018).

20.3.14 Resettlement

All issues related to relocation and/or resettlement shall come under a Resettlement Action Plan (RAP) to be produced as a standalone report for the Project. The handling of relocation and/or settlement will consider the following principles:

- Fair and adequate compensation
- Compensation based on negotiation
- Payment before destruction
- Maintenance and/or betterment of facilities to pre-project condition
- Regards to traditional and spiritual needs of the people

- Grievance redress mechanism.

Cardinal will liaise with relevant stakeholders as provided in the RAP to establish the Relocation and Compensation Negotiation Committee (RCNC) within the Project approval phase prior to commencement of construction. The RCNC will approve or agree on compensation rates for affected properties/assets, livelihood restoration programmes for project affected persons and deprivation of use of land/surface rights issues as provided in the *Minerals and Mining Act 2006* and the Minerals and Mining (Compensation and Resettlement) Regulations, 2012 (LI 2175).

Potential compensation amounts cannot be predicted at this stage.

20.4 Potential environmental and social impacts, and mitigation

The Project shall interact both positively and negatively with the biophysical and socioeconomic resources and/or receptors within and beyond the Project area. The key potential positive impacts and opportunities, and key potential adverse impacts from Project construction, operation, and decommissioning/closure identified in the draft Project EIS (NEMAS, 2018), and draft Scoping Report for the TSF (Geosystems, 2019) are summarised in the following Sections.

20.4.1 Positive impacts and opportunities

The key potential positive impacts for the Project include:

- Employment creation and skills development
- Creation of business opportunities for locals/Ghanaians
- Improvement in the district/regional and national revenue base
- Improvement in the local Economy of beneficiary district
- General improvements in livelihoods and reduced poverty
- Community development and improvements in infrastructure and social amenities.

20.4.2 Negative impacts and proposed mitigation measures

Table 157 summarises the key potential negative impacts and mitigation measures for the Project, including the development of the TSF.

Table 157: Key potential negative impacts and proposed mitigation measures

Factor/Aspect	Key Potential Negative Impacts	Key Proposed Mitigation Measures
Physical Environment		
Ambient air, noise and vibration	Fugitive emissions potentially impacting air quality and subsequently impacting human health and flora/vegetation health/quality caused by: ■ Dust/exhaust emissions produced by vegetation clearing, grubbing, soil disturbing activities, stockpiling, vehicle movements and general construction and operational activities ■ Waste rock dump (WRD) and Run of Mine (ROM) pad ■ TSF construction and operation ■ Dust produced from drilling, blasting, loading, crushing, milling, and screening or ore Generation of noise potentially impacting communities and site workers caused by: ■ Operating generators, machinery, processing plant and blasting activities Generation of vibrations potentially impacting the safety of buildings, and subsequently human and animal lives: ■ Vibrating machines from breaking sub-base rocks ■ Blasting activities.	■ Use lowest emission machines and use qualified operators ■ Regular maintenance of vehicles, machinery and equipment ■ Use of noise minimising equipment (i.e. mufflers/silencers) ■ Use of good quality fuel/lubricants ■ Use of qualified operators ■ Implementation of speed limits and speed ramps (humps) ■ Use of dust suppression (i.e. water carts, water bowsers) ■ Strategic placement of material stockpiling to minimise dust lift-off ■ Limit clearing to only areas designated ■ Adherence to design drawing specification in vegetation clearance ■ Avoid working under aggressive weather conditions (i.e. strong winds) ■ Use of appropriate personal protection equipment ■ Regular dust and gaseous particles monitoring.
Soil and land degradation	Soil and land degradation (including soil erosion and loss of soil fertility) as a result of: ■ Vegetation clearing/grading/general earthworks ■ Alteration of landscape (i.e. construction of the TSFs, placement of soils on steeper slopes) ■ Silted water courses could lead to flooding and erosion of farmlands leading to loss in soil fertility ■ Improper disposal/storage/transportation, spills, and leaks of wastes, oils, lubricants, hydrocarbons, chemicals etc. ■ Poor contouring of the TSFs during development ■ Increased erosion ■ Soil compaction.	■ Vegetation clearing activities undertaken in accordance with approved design drawings ■ Bund wall installation around fuel storage areas and use of drip pans at workshops ■ Installation of sumps at workshops ■ Conservation of topsoil for future use in rehabilitation ■ Proper handling and preservation of topsoil during construction and operation phases of the Project for final reclamation works ■ Practicing of good landscape engineering (TSFs) ■ Re-contouring, pegging and terracing of areas prone to erosion (TSFs) ■ Implement progressive re-vegetation or re-graveling ■ Compacting of loose soil (TSFs).
Surface water and groundwater	Impact on surface water from: ■ Vegetation clearing and construction activities ■ Erosion of surfaces of the TSFs, ultimately leading to sedimentation of surface water features ■ Tailings spills from delivery lines at the plant, along the delivery lines and at the TSFs ■ Perennial flooding at the peak of the rainy seasons could overflow TSF infrastructure such as the tailings pipeline corridor and sediment control structures (SCS) that may lead to contamination of surface water within the Project area ■ Potential TSF dam failure ■ Improper disposal/storage transportation, spills, and leaks of wastes, oils, lubricants, hydrocarbons, chemicals, sewage etc. ■ Runoff of leachate from the WRD ■ Runoff from the ROM Pad ■ Pit water discharges, pit dewatering, construction activities, and raw water abstraction for both domestic and production usage Impact on groundwater from: ■ Improper disposal/storage transportation, spills, and leaks of wastes, oils, lubricants, hydrocarbons, chemicals etc. ■ Seepage of leachate from the WRD, leachate of metals from exposed waste rock ■ Seepage from the ROM Pad ■ Seepage through the basin and/or embankments of the TSFs, including supernatant water ■ Potential TSF dam failure ■ Tailings spills from delivery lines at the TSFs ■ ARD and chemical releases from the WRD and Pit dewatering.	General: ■ Construction of drains to channel sediment flows ■ Horizontal heaping of stockpile material ■ Use of terraces and other landscaping measures to reduce surface runoff ■ Vegetation clearing activities undertaken in accordance with approved design drawings ■ Avoid working during destructive weather conditions (i.e. strong winds) ■ Installation of a bund wall at fuel/oil storage areas and bunded hardstand ■ Capture stormwater from bunded areas into sumps ■ Ready accessibility of oil spread preventive materials and adherence to Safety Data Sheets when handling/storing chemicals ■ Installation of a sediment containment pond to contain runoff from the WRD and Pit dewatering ■ Stabilise WRD with local grasses/plants to reduce runoff ■ Reuse Pit water in processing. TSF specific: ■ Construction of sediment control structures ■ Lining of tailings delivery pipeline corridor ■ Appropriate collection and storage of spent oil ■ Use 300 mm compacted soil liner (“**CSL**”) for the Flotation TSF and a composite liner consisting of CSL and HDPE geomembrane for the CIL TSF ■ Implement underdrainage and leak collection systems in the CIL TSF and an underdrainage system in the flotation TSF to drain the tailings mass (above the CSL) and return all seepage and leakage into the dam ■ The proposed TSF will be operated with a freeboard sufficient to contain a 1:100-year, 24-hour rainfall/storm event and a well-designed spillway to allow flow of 1:5,000 ARI ■ Prepare and implement a hydrocarbon management plan ■ Equip vehicles with spill kits ■ Immediate clean-up of accidental spills ■ Installation of groundwater monitoring wells ■ Conduct regular groundwater and surface water quality monitoring ■ Where necessary treat supernatant water to meet baseline values or EPA guidelines where applicable before release into external environment ■ Embankments to be designed to prevent dam breaches

Factor/Aspect	Key Potential Negative Impacts	Key Proposed Mitigation Measures
		■ Conduct a detailed study during the final TSF design stage to assess the impact of TSF dam failure on the White Volta and its tributaries that drain the Project area ■ TSF seepage control and underdrainage collection systems to include: ▪ Cut-off trench excavated into the foundation soils and backfilled with low permeability fill ▪ Low permeability soil liner of either 300 mm thickness (Flotation TSF) or 200 mm thickness (CIL TSF) to be constructed in the basin area of each TSF ▪ Synthetic HDPE geomembrane ▪ Basin underdrainage collection system ▪ Underdrainage collection sump to be constructed against the upstream toe of the embankment in each facility ▪ Leakage collection system (CIL TSF) ▪ Upstream toe drain ▪ Decant return water system ▪ Emergency spillway.
Waste Management	Impact on soil (i.e. fertility), surface water (including increased breeding grounds for vectors such as mosquitoes), groundwater quality, and general biodiversity of the area due to improper management, disposal, storage and transportation of hazardous and non-hazardous wastes, such as construction, installation, mine and domestic wastes.	■ Usage of excavation materials for backfilling of galamsey sites ■ Disposal of waste to appropriately licenced landfill facilities/agents ■ Drainage system to contain and treat waste water ■ Adequate storage of spent oil on site and disposal of spent oil by accredited oil companies ■ Treatment of effluent on-site.
Biological Environment		
Terrestrial and aquatic biodiversity	Impact on terrestrial flora and fauna including: ■ Direct clearing/loss of vegetation (including economic/medicinal trees/plants) ■ Dust deposition impacts on flora/vegetation health ■ Damage/loss of fauna habitat/food ■ Displacement of fauna ■ Direct impact on sensitive arboreal, terrestrial and burrow receptors due to soil erosion and loss of soil fertility ■ Ambient air quality/noise/vibration impacts on fauna ■ Fauna injuries/fatalities (i.e. vegetation clearing activities, establishment of mine facilities etc.) ■ Increased breeding grounds for vectors such as mosquitoes due to poor waste management and/or disposal ■ Water in the supernatant pond will attract birds and animals to the TSF, especially in the long dry season when all ephemeral streams and water bodies except the White Volta River are dry ■ Improper disposal/storage/transportation, spills, and leaks of wastes, oils, lubricants, hydrocarbons, chemicals etc. impacting terrestrial flora and fauna and habitat. Impact on aquatic biodiversity: ■ Decrease in water quality due to increased sediment loads in surface waters, particularly during periods of high rainfall resulting from erosion of excavated sites ■ Effect of increased water pollution on aquatic life, water resources and usage (i.e. reduced fishing catch by fishers) ■ Effect of installation of abstraction pipe and abstraction pump in the White Volta on aquatic life ■ Effect of dumping of waste, runoff on contaminant soil from the WRD and Pit water, and deposition of silt on aquatic life ■ Improper disposal/storage/transportation, spills, and leaks of wastes, oils, lubricants, hydrocarbons, chemicals etc. impacting aquatic flora and fauna and habitat.	■ Limit vegetation clearing in areas not required for the Project ■ Limit excavation to protect soil composition/quality ■ Implement a buffer area and offsets ■ Adherence to forestry rules on tree felling within the right of way (RoW) of intended facilities ■ No unauthorized roads to perverse vegetation ■ Prohibition of game by workers and allowing safe passage by distressed animals ■ Design TSF to limit fauna access ■ Restricted and supervised clearance according to management plan to avoid/limit animal kills ■ Avoid bush burning ■ Stockpile topsoil to be used in rehabilitation activities ■ Restriction of gaming/hunting and safe passage for stray animals ■ Design and implement an appropriate cover system for the TSF that takes into account the specific climatic conditions and material characteristics of the overall Project Area ■ Implement progressive re-vegetation or re-graveling.
Socio-economic Environment		
Aesthetic view	Distortion of landscape/topographic views, natural aesthetic values and Visual intrusion as a result of: ■ Vegetation clearing ■ General construction (i.e. mine facilities, TSFs) and mining activities (i.e. increased presence of light and heavy vehicles and machinery in the Project area).	■ Backfill areas affected by illegal mining ■ Limit vegetation clearing in areas not required for the Project ■ Avoid using artificial floor lighting in areas not regarded as safety/security threat zones ■ Implement progressive re-vegetation or re-graveling ■ Implement good landscape engineering (TSFs) ■ Re-contouring, pegging and terracing of areas prone to erosion (TSFs).
Archaeological and Cultural Heritage	Impact on archaeological and cultural heritage as a result of: ■ Vegetation clearing and development of mine facilities (relocation of 12 shrines and 1 archaeological site of interest within the concession) ■ Destruction of archaeological/cultural sites and heritage ■ Pit excavation (chance archaeological finds).	■ Develop a roadmap with the Tindanis institution for the relocation of shrines and sacred sites ■ Fulfilment of all agreed protocols for relocation and completion of relocation before site are disturbed ■ Development of chance find procedure for any relics/antiquities found during clearing/construction works ■ Promptly notify the Ghana Museums and Monument Board office at Bolgatanga, of chance finds.
Conflict and disputes	Community/social conflicts and disputes due to: ■ Failure to establish true ownership of surface rights (the land) between the two claimants (i.e. compensation arrangements), and economic/medicinal trees/plants	■ Proper demarcation of areas of control for each chief ■ Compensation amounts dependant on land size.

Factor/Aspect	Key Potential Negative Impacts	Key Proposed Mitigation Measures
	■ Labour termination.	
Land take	Effects of land take, and termination, restriction and/or disruption in land use: ■ Loss of land/pasture for livestock/grazing leading to decline in production ■ Limitations on hunting leading to reduction in bush meat, and decline in protein for dietary requirements ■ Loss of gathering opportunities for economic/medicinal trees/plants (i.e. sheanut) by women leading to loss of income/revenue and deprivation of health needs for community members.	■ Allowing accessibility to areas not immediately affected by mining activities ■ Provide women with training to achieve alternative livelihood ventures ■ Improvement in local health care facilities to reduce the reliance on traditional bush medicine.
Physical displacement	Cessation of Buing and Accra sites located within the concession (Project boundary) as a dwelling place for human habitation, leading to displacement, relocation and resettlement: ■ Buing – Approximately 20 houses/100 people ■ Accra – Approximately 105 houses/500 people ■ The entire Biung (Bingo) community lies within the TSFs footprint and will require resettlement.	■ Resettlement Action Plan (RAP) for a comprehensive resettlement programme based on the following principles shall be established: ▪ Fair and adequate compensation ▪ Compensation based on negotiation ▪ Payment before destruction ▪ Maintenance and/or betterment of facilities to pre project condition ▪ Regards to traditional and spiritual needs of the people ▪ Grievance redress mechanism through acceptable Grievance Redress Committee ■ Formation of relevant committees to ensure acrimony free implementation of compensations and resettlement in accordance with provisions contained in a RAP Report (to be prepared) ■ Provision of 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 affected community members who may lose their livelihoods as a result of the establishment of the TSF.
Occupational and Public Health and Safety	Construction and operation of the Project leading to: ■ Increased potential for hazards, incidences, injuries and fatalities ■ Increased potential for spread of infectious diseases and infections ■ Increased potential for vector infestations such as through black fly bite, tsetse fly bite etc. ■ Increased fire outbreaks. Construction and operation of the TSFs leading to: ■ Increased potential for accidental spills ■ Potential for TSF seepage ■ Potential for TSF dam failure, which poses a threat to occupational and public safety.	■ Formulation and adherence of Occupational Health and Safety (OHS) policy ■ Mandatory OHS training for all workers ■ Enforcement of rules and regulations for operating in hazardous areas ■ Provision and compliance to the use of Personal Protective Equipment (PPE) ■ Use of quality electrical equipment and installation by certified electricians/engineers ■ Installation, regular servicing of firefighting equipment ■ Clearance of health conditions of workers before recruitment and routine checks during working period ■ Implement adequate lighting/visibility ■ Implement effective signage ■ Recruitment of qualified staff/operators ■ Observance of traffic rules and regulations both on and off sites ■ Observance of break/rest within working hours ■ Hoisting of caution flags and flashlights on operating vehicles and moving machines ■ Implement effective communication system ■ 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, covering of trailer buckets in use during construction periods ■ Develop and implement a policy that covers vaccination of all workers against cerebrospinal meningitis (CSM) diseases ■ Vaccinate all workers against CSM diseases ■ Develop Company's Code of Conduct to be signed by all workers engaged that compels them to respect community norms and traditions ■ Develop good public relations between the community and the mine. A Community Relation 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 ■ Collaborate with the District Security Council to provide police/military presence to ensure security and enforce law and order within the resettled communities ■ Educate the communities and workers on good hygienic practices ■ Implement Malaria Control Programmes ■ Undertake health screening for host communities ■ Non-involvement of mine workers in local politics and chieftaincy issues ■ Periodic community/mine engagement and sensitization, especially on HIV/AIDS awareness and other sexually transmitted infections ■ Maintain and train an effective fire department and call on the District and Regional fire offices for assistance when required

Factor/Aspect	Key Potential Negative Impacts	Key Proposed Mitigation Measures
		■ Fire hydrants to be located at fire prone locations ■ Fire belts to be constructed around the mine operational areas.
Socio-Economic, diseases and conflict	Construction and operation of the Project leading to: ■ Socio-economic impacts: ▪ Disturbances of social and cultural norms and values by mine workers ▪ Loss of land/livelihood/employment ▪ Displacement of small-scale miners (galamseys) ▪ Influx of people to the area in search of employment, along with an increase in crime and insecurity ▪ Increased potential for accidents on the public/community roads ▪ Annual occurrence of bush burning could impact equipment and other assets at the mine. ■ Impacts from diseases: ▪ Increased potential for spread of infectious diseases and infections, including HIV/AIDs and STIs ▪ Increased potential of bites/stings from disease-causing vectors causing spread of diseases ▪ Sediment control ponds and raw water ponds could provide as breeding grounds for mosquitoes ▪ Usage of contaminated water sources from mine runoffs could affect community health. ■ Community and social conflict: ▪ Aggrieved illegal miners could pose a threat to mine workers ▪ Layoff construction workers after construction period could destabilise community/company relations ▪ Increased potential of social problems such as prostitution, teen pregnancy, drugs, drunkenness, and crime.	■ Transparency in job recruitment and high job consideration for native people ■ Provision of shuttle buses to ease pressure on local transport ■ Open door policy with efficient grievance redress channels ■ Pledge of respect for local tradition as part of workers contract agreement ■ Strict enforcement of ‘No Interference Policy’ in local politics and chieftaincy issues ■ Sensitization on HIV/AIDS as part of worker-community educational programmes ■ An Influx Management Plan to be developed as part of the Namdini gold project development ■ Reinforcement of security by establishing a Police Post within the host community assisted by private security from a registered security company ■ The company commits to rely on locally made goods and services during the implementation of the TSF project ■ Clearing of agricultural land will be avoided, where feasible for the TSF project ■ Project components shall be sited as close together as possible to reduce land take ■ Continue to engage relevant stakeholders, such as the District Assembly, District Security Council, traditional authorities, the Regional Coordinating Council, and the Minerals Commission to ensure a smooth and peaceful relocation of illegal miners from the concession.

20.5 Environmental management and monitoring

20.5.1 Provisional Environmental Management Plan

A Provisional Environmental Management Plan (EMP) has been developed for the Project that captures Cardinal's corporate policies on environment, health and safety with clear policy objectives and targets as well as establishing a comprehensive Environmental Management Structure.

Cardinal is committed to finance the environmental aspects of the Project from its annual operating budget for the Mine and will make provisions for financial allocations during budget preparations for the implementation of all environmental management actions, including monitoring. The detailed cost estimates will be calculated prior to the construction and operational phases.

A detailed EMP will be prepared and submitted to the Ghana EPA within 18 months of commencement of operations as per Regulation 24 of LI 1652. Cardinal shall update the EMP every three years throughout the life of the Project.

20.5.2 Environmental monitoring

Environmental monitoring programmes will be implemented to confirm the extent and magnitude of predicted impacts and to ensure effectiveness of the proposed mitigation measures as well as to confirm any residual impacts, which are expected with the development of such Projects. The proposed monitoring programme will cover meteorology, ambient air quality and noise levels, blast vibration, water resources and the socio-economic environment.

20.6 Closure, decommissioning and reclamation

The Mining Lease is for 15 years, renewable for a further 30 years. The projected mine life is 15 years using a 9.5 Mtpa throughput processing plant.

20.6.1 Reclamation and decommissioning planning

Within two years of commencement of operation Cardinal is required to prepare and submit a detailed Reclamation and Decommissioning Plan for the Project to guide the reclamation security agreement to be signed between Cardinal and the Ghana EPA. That Plan shall address land stabilisation, erosion and sediment control among other issues. The Plan will serve as the basis for concurrent reclamation during the Project's life. A final Reclamation and Decommissioning Plan is required to be submitted to the Ghana EPA two years before closure of the Project (NEMAS, 2018).

20.6.2 Reclamation

Cardinal's aim of reclamation is to achieve a socially acceptable balance between mining and the biophysical and social environment and to ensure that internationally accepted standards of health, safety and environmental protection are observed during and after active mine life.

Project reclamation shall comply with the EPA criteria summarised by NEMAS (2018) in Table 158.

Table 158: Ghana EPA reclamation criteria

Project Facility	Ghana EPA Reclamation Criteria
Pit	■ Stable slopes free from rock fall and soil movement ■ Water discharge meets with Ghana's environmental effluent quality guidelines ■ Protected access to rock faces and all steep slopes with inert waste rock as perimeter bund at minimum distance of 50 m from the pit face ■ Establishment of protective zones around the pit to protect community members from water hazards.

Project Facility	Ghana EPA Reclamation Criteria
Waste Dumps	■ Stable slopes and surfaces free from rock and soil movement and erosion ■ Slopes battered between 20 – 30 degrees ■ Water run-off managed effectively ■ Gentle rolling landforms of similar aesthetics to existing area.
Rom Pad and Ore Stockpiles	■ Stable slopes and surfaces free from rock and soil movement and erosion ■ Slopes battered between 20 – 30 degrees ■ Water run-off managed effectively ■ Gentle rolling landforms of similar aesthetics to existing area.
Plant site and contractor facilities	■ Soils free from erosion ■ Water run-off managed effectively ■ Facilities not required by local community/government removed ■ Any contaminated soils (hydrocarbons) encapsulated and free from leachate ■ Free from all hazardous materials and waste.
Leach Pad and Ponds	■ Stable slopes and surfaces free from rock and soil movement and erosion ■ Slopes battered between 20 – 30 degrees ■ Water run-off managed effectively ■ Gentle rolling landforms of similar aesthetics to existing area ■ Water discharge meets with Ghana's environmental effluent quality guidelines.
TSFs	■ Embankments stable and free from erosion ■ No steep slopes or faces leading into storage facility ■ Protected access to unconsolidated tailings ■ Provision of 1 m embankment freeboard ■ Overflow spillway constructed.
Water Bodies	■ Retaining structures stable and free from erosion ■ Mine related facilities removed ■ Water levels reduced to minimum practical elevations ■ Quality conform to Ghana EPA guidelines.
Backfill and borrow pit (Recreation Area)	■ Soils stable and free from erosion ■ Water run-off managed effectively ■ All facilities related to construction removed ■ Prevent contamination of soils but any contaminated soils (hydrocarbons) shall be encapsulated and free from leachate.
Townsite	■ Soils stable and free from erosion ■ Water run-off managed effectively ■ All facilities related to construction removed ■ Prevent contamination of soils but any contaminated soils (hydrocarbons) shall be encapsulated and free from leachate.

The primary Project facilities that will require reclamation and decommissioning are provided in Table 159 using data from NEMAS (2018) with TSF requirements from Geosystems (2019).

Table 159: Project facilities to be decommissioned

Mine Infrastructure/Facilities	Area (ha)	Reclamation/Decommissioning Options	Growth Media Recovery
Open pit	78	Partial reclamation	Topsoil to be removed and stockpiled prior to development.
Wastes Dump	240	Reclamation	
Haul roads	95	To remain as access route	
Cell 1 Flotation TSF	245	Reclamation and subsequent closure of the TSF will be completed in accordance with the provisions of LI 2182 and with the conditions of the environmental permit and EPA guidelines so that water leaving the facility meets regulatory requirements	
Cell 2 Flotation TSF	102		
CIL TSF	127		
Plant site	15	Removal of plant facilities and reclamation	

Mine Infrastructure/Facilities	Area (ha)	Reclamation/Decommissioning Options	Growth Media Recovery
ROM Pad	40	Removal of ore and reclamation	Topsoil to be removed and stockpiled prior to development.
Raw water storage facility	8	To remain. Potential for aquaculture or as source of water for irrigation	-
Settlement Ponds	Included in Plant	Decommission and reclamation	Topsoil to be removed and stockpiled prior to development.
Mining Contractor Laydown area (includes workshop/fuel storage area, explosive magazine etc.)	4	Decommissioning of facilities and reclamation	-
Sediment control structures	28	Decommissioning of structures and reclamation	Topsoil to be removed and stockpiled prior to development.
Topsoil stockpile area	175	Reuse of topsoil for reclamation	-
Office/administration facility	Included in Plant Area	To remain	-
Camp/accommodation site	78	To remain	-
Site access roads	86	To remain	-
Airstrip	15	To remain	-
Other facilities (utilities and water lines, crushers, pumps, piping, process solution tanks, electrical cables, and related equipment.	-	To be removed and reused or relocated outside project area or disposed of properly.	-

20.6.2.1 General approach

The short-term reclamation approach will be to stabilise all areas that were heavily impacted during mining operation as well as its immediate areas not disturbed from any land degradation, either by erosion, sediment transport or deposition.

The long-term reclamation approach will be to ensure complete site stabilisation and re-establish a productive floristic heterogeneous community capable of supporting and sustaining a healthy faunal population. The post mining decommissioning is also to ensure public safety within the mine sites and its immediate areas of influence and establish and regenerate the sites to acceptable land use that conforms to the pre-mining land use or land use that enhances land utilization.

20.6.2.2 Tailings Storage Facilities

Reclamation and subsequent closure of the proposed TSFs will be completed in accordance with the provisions of LI 2182 and with the conditions of the environmental permit and EPA guidelines so that water leaving the Project site meets regulatory requirements.

The proposed TSFs will be rehabilitated to promote long term stability, minimise erosion, and provide a discharge water quality that meets the baseline quality of the receiving environment or the EPA guidelines for discharge into waterbodies. Rehabilitation of the TSF would begin when process plant operation ceases and tailings are no longer generated, with initial measures taken to stabilise the tailings surface. This is a key issue that will be appropriately addressed to ensure compliance with the legislation and result in long term environmental sustainability.

Decommissioning and rehabilitation of the TSF will be linked to the general rehabilitation of the mine site and the permitting conditions for the mining lease.

20.6.2.3 Post mining land use

The post mining land use objectives will be determined through stakeholder engagement during the operational phase, and the final land use will be compatible with community expectations and needs. Post mining land use options to be proposed to stakeholders include:

- Woodlot or afforestation for WRD, TSF sites and mining contractor areas and the plant sites
- Pit lake for the mined-out pit as a tourist site
- Agriculture for the topsoil stockpile site.

20.6.2.4 Closure and post-closure monitoring

A comprehensive monitoring programme is required to be prepared and shall include physical observation and assessment of reclaimed sites. Closure and post-closure monitoring shall aim to ensure that the mine area after closure does not suffer from residual mine effects and to satisfy statutory authorities on both environmental and public safety of areas decommissioned.

Short term monitoring shall occur during the reclamation period and will commence during and just after reclamation. Critical issues for monitoring consideration shall include water quality and air quality, erosion, and state of vegetation regrowth etc. Critical areas will be pit water, pit stability, WRD area, plant site, domestic and monitoring boreholes, rehabilitated areas for erosion checks and vegetation growth rate and return of biodiversity. Frequency of monitoring at the initial stage shall be monthly or as agreed with the EPA (NEMAS, 2018).

Long term monitoring shall remain for three years after mine life and monitoring shall be conducted quarterly. Monitoring results shall be compared with baseline data gathered during the EIA baseline studies. Community feedback, physical observations and monitoring results shall indicate areas of concern or arising challenges that call for modification of reclamation strategies.

20.6.3 Provisional cost estimate of decommissioning and reclamation

Cardinal proposes to finance the closure, decommissioning and reclamation of the Project from the annual operations budget. The total cost of decommissioning and reclamation of the Project is discussed in Section 21.0.

Cardinal is not currently required under Ghanaian law, regulations or agreements to post a reclamation bond or other assurance for the financial obligations of mine closure.

20.7 Recommendations

Environmental, Health, Safety and other issues will be mitigated by standard industry measures as required.

Measures will be adopted to ensure that the welfare of the public and host communities is not undermined by the operations of the mine.

20.8 Additional information

Appendices provided in NEMAS (2018) include:

Appendix 1	Team of Experts
Appendix 2	Deeds of Assignment
Appendix 3	Letter of Notification to EPA on Deeds of Assignment
Appendix 4	Company Registration Documents
Appendix 5	EPA Correspondences
Appendix 6	Evidence of Stakeholders Consultation
Appendix 7	Ecological Data
Appendix 8	Meteorological Data
Appendix 9	Soil Baseline Study Report
Appendix 10	Archaeological and Cultural Heritage Report.

21.0 CAPITAL AND OPERATING COSTS

The information presented in this Section is derived from Lycopodium (2019c), Golder (2019b) and Cardinal.

21.1 Capital cost estimate work breakdown structure

The capital cost (Capex) estimate for the Namdini Gold Project has been compiled by Lycopodium with input from Knight Piésold on water infrastructure and tailings storage.

The FS capital cost estimate was completed to an accuracy of +15% to -5% for both Plant and Infrastructure Capital Costs, based on open pit mining of the Ore Reserve.

Cardinal has elected to compile project specific portions of Mining and Owner's costs external to the capital cost estimate presented below. The mining equipment is identified in Section 16.13.

21.2 Capital estimate summary

The capital cost estimate is summarised in Table 160 for which the detail is available separately to Cardinal.

Table 160: Capital estimate summary (US$, 1Q19, +15/-5%)

Main area	US$'000
Construction distributables	41,699
Treatment plant costs	149,727
Reagents and Plant services	28,562
Infrastructure	60,138
Mining Services	6,253
Management costs	36,313
Owner's Project costs	25,207
Subtotal – Process Facility & Infrastructure	**347,911**
Contingency	42,196
Taxes & duties	*
Escalation	Excl.
TOTAL PLANT DEVELOPMENT	**390,107**

Note: * Taxes of $37M are excluded and are subject to negotiation and/or deferment under a Development Agreement.

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 including 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 requests 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 infrastructure items that were subsequently costed by Lycopodium.

The Capex estimate includes:

- Direct costs of the Project development including infrastructure
- 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
- Operating spares and 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
- HV Powerline to the process facility and switchyard
- Taxes and duties (subject to Development Agreement).

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

21.3 Estimate currency and base date

The estimate is expressed in US dollars based on prices and market conditions current at first quarter 2019 (1Q19).

The following exchange rates have been used in the compilation of the estimate:

- US$1.00 = A$1.38 (Australian Dollar).
- US$1.00 = €0.87 EUR (Euro).
- US$1.00 = £0.77 GBP (British Pound).
- US$1.00 = R14.29 ZAR (South African Rand).

Foreign currency exposure is shown in Table 161 below.

Table 161: Foreign currency exposure

Currency	Percentage of capital estimate
USD	~61%
AUD	~18%
EUR	~15%
GBP	~5%
ZAR	<1%

The estimate capital cost accuracy is as follows:

- +15/-5% Plant Capital Costs.
- +15/-5% Infrastructure Capital Costs.

21.4 General estimating methodology

General arrangement drawings and a 3D model have been produced with sufficient detail to permit the assessment of the engineering quantities for earthworks, concrete, steelwork, mechanical and electrical for the crushing plant, processing plant, conveying systems and infrastructure. Unit rates that reflect the current market conditions have been established for bulk materials, capital equipment and labour from project specific budget quotation requests and from similar projects currently under construction.

Labour rates from the market have been benchmarked against in-house labour gang rates and indirect cost modelling to ensure consistency with the current projects market.

Budget pricing for equipment and infrastructure facilities were obtained from suitable suppliers and contractors. The rates used in the estimate have been reviewed and deemed to reflect the current market conditions.

21.5 Engineering status

The design status varies depending on the basis of design, from recently completed facility designs, modified construction and as built drawings of current and past project facilities, to initial concept drawings.

The key process and engineering design criteria used for equipment selection in the development of the capital cost estimate are described in Section 17.0.

21.6 Quantity development

The Project works were quantified to represent the defined scope of work and to enable the application of rates to determine costs. Allowances for compaction, waste, rolling margin and the like are included in the build-up of unit costs.

Quantity information was derived from a combination of sources and categorised to reflect the maturity of design information as follows:

- Project engineering that includes quantities derived from detailed engineering or similar projects that have previously been constructed or were under construction at the time of the estimate.
- Study engineering that includes quantities derived from concept or preliminary engineering for the purpose of the study. Includes equipment lists and redline mark-ups of previous project drawings / data by engineering.
- Estimates that include quantities derived from sketches or redline mark-ups of previous project drawings / data by estimating.
- Factored quantities derived from percentages applied as a factor from previous estimates or projects.

The derivation of quantities within these categories by percentage is provided in Table 162 weighted by value of the direct permanent works (i.e. excluding temporary works and construction services).

Table 162: Derivation of quantities

Classification	Quantity	Unit	Project engineering %	Study engineering %	Estimated %	Factored %
Plant earthworks	3,397,936	m³	-	91%	9%	-
Infrastructure earthworks	3,948,056	m³	-	97%	3%	-
Plant concrete	17,312	m³	19%	63%	18%	-
Infrastructure concrete	3,705	m³	-	51%	49%	-
Structural steel	2,369	t	13%	82%	5%	-
Platework	624	t	39%	28%	33%	-
Field erected tanks	886	t	-	81%	19%	-
Mechanical equipment	554	each	-	100%	-	-
Conveyors	650	m	-	100%	-	-
Piping – Plant	1	lot	-	-	-	100%
Piping – overland	36.9	km	-	100%	-	-
E & I Plant & infrastructure	1	lot	-	95%	5%	-
Steel buildings	4,891	m²	-	100%	-	-
Blockwork / modular buildings	1	lot	-	100%	-	-

21.7 Pricing basis

Estimate pricing was derived from a combination of the following sources:

- Market Pricing – Budget pricing solicited specifically for the study or project estimate or actual costs from similar projects that have recently been constructed or were under construction at the time of the estimate
- Database – Historical database pricing that is less than six months old
- Estimated – Historical database pricing older than six months, escalated to the current estimate base date
- Factored – Factored from costs with a basis.

Table 163 summarises the source of pricing by major commodity, weighted by value of the direct permanent works (excluding temporary works, construction services, commissioning assistance, EPCM costs and contingency), including supply and installation.

Table 163: Source of pricing

Classification	Market pricing %	Database %	Estimated %	Factored %
Earthworks	99%	-	<1%	-
Concrete	95%	3%	2%	-
Structural steel	100%	-	-	-
Platework	89%	10%	<1%	-
Mechanical equipment	85%	13%	2%	-
Piping – Plant	-	-	-	100%
Piping – overland	96%	4%	-	-
SMP	95%	5%	-	
E&I equipment	75%	20%	5%	-
Prefab / modular buildings	92%	3%	5%	-

Pricing is categorised by the following cost elements, as applicable, for the development of each estimate item.

Bulk Materials

This component covers all materials normally purchased in bulk form, for installation on the project. Costs include the purchase price ex-works, off-site fabrication (if applicable), transport to site (unless otherwise stated), and over-supply for anticipated wastage.

Plant Equipment

This component represents prefabricated, pre-assembled, off-the-shelf type of mechanical or electrical equipment items. Pricing is inclusive of all costs necessary to purchase the goods ex works, generally excluding delivery to site (unless otherwise stated) although does include operating and maintenance manuals. Vendor representation and commissioning spares have been allowed for separately in the estimate.

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 trade labour, equipment and contractors' distributables.

The labour component reflects the cost of the trade labour workforce (excluding management, supervision and other on-site support staff) 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 includes small tools, consumables, personal protective equipment ("**PPE**") and the applicable contractor's margin.

Contractors' distributable costs encompass the remaining cost of installation and include items such as off-site management, on-site staff and supervision above trade level, site facilities, cranes and crane drivers, mobile equipment, scaffold, mobilisation and demobilisation, rostered breaks off-site ("**R&R**"), meals and accommodation costs, durations costs general and the applicable contractors' margin.

The package average labour gang rates and equipment estimated for each major trade commodity are shown in Table 164.

Table 164: Direct labour gang rates

Item	Trade Labour $	Equipment $	Total Hourly Rate $
Concrete installation	3.14	3.12	6.26
SMP	17.43	11.10	28.53
Field erected tanks	14.00	9.02	23.02
Architectural	2.40	0.95	3.35
Electrical & instrumentation	12.00	10.56	22.56

21.8 Construction infrastructure

Project construction offices and establishment, construction services, power, water, PPE, communications, computers, IT services, servers and telephones are all included in the capital estimate.

Construction fuel facilities will be provided by the individual construction contractors.

Construction equipment and project cranes are included in the capital cost estimate.

21.9 EPCM services

The Project will be implemented using an EPCM approach, whereby the EPCM Engineer will provide design, procurement and construction management services on behalf of the Owner based on the Project Schedule.

The EPCM services cost estimate includes head office support and site staffing, subconsultants, office consumables, equipment and associated project travel. The cost of a fully equipped home design office and all project computing requirements are included under Management costs.

The estimate for EPCM services costs has been based on a preliminary manning schedule for the anticipated project deliverables and Project Schedule. The resulting EPCM cost estimate is consistent with other projects of this nature in terms of the percentage of the plant capital cost.

The engineering design component of the EPCM estimate for the home office is based on a calculation of required manning levels to complete the Project and benchmarked against Lycopodium experience on similar projects.

Engineering, design and procurement is assumed to commence shortly after the Cardinal Board approval.

21.10 Owner's costs

As part of the project development Cardinal will provide the Project management team.

The Owner's construction team will interact closely with operations management personnel recruited during the construction phase of the Project.

The costs for the Owner's Management team and operational management costs have not been included in the capital estimate and have been dealt with directly by Cardinal in the financial model.

In addition to the above, the following items have been dealt with directly by Cardinal in the financial model:

- Owner's project team and expenses
- Mining mobilisation and establishment
- Mine pre-stripping costs
- Bulk fuel and lubricant storage
- HV power supply / grid connection
- Plant pre-production labour
- Admin pre-production labour
- Working capital.

The following allowances have been included in the capital cost estimate:

- First fills (grinding media, lubricants, fuel, and reagents)
- Opening stocks
- Plant mobile equipment
- Project spares
- Vendor representative and training costs for the process plant
- Fit out of buildings
- Information technology and communications.

21.11 Spares

A value for Project spares has been estimated and included in the estimate.

21.12 Duties, taxes and insurances

Government duties and taxes

The capital estimate excludes government taxes and duties.

Import duties and taxes

The capital estimate excludes import duty for material supply, land tax, withholding tax, and other taxes and duties.

These items have been dealt with directly by Cardinal in the financial model.

21.13 Contingency

An amount of contingency has been provided in the estimate to cover anticipated variances between the specific items allowed in the estimate and the final total installed project cost. The contingency does not cover scope changes, design growth, etc., or the listed qualifications and exclusions.

Contingency has been applied to the estimate as a deterministic assessment by assessing the level of confidence in each of the defining inputs to the item cost being engineering, estimate basis and vendor or contractor information. It should be noted that contingency is not a function of the specified estimate accuracy and should be measured against the project total that includes contingency.

A contingency analysis has been applied to the estimate that considers scope definition, materials / equipment pricing and installation costs.

The resultant contingency for the project is 11.2%.

21.14 Escalation

There is no allowance for project escalation in the capital estimate. Escalation shall be calculated from the base of 1Q19.

21.15 Qualifications and assumptions

The capital estimate is qualified by the following assumptions:

- The estimate base date in regard to project pricing is the first quarter 2019 (1Q19).
- Prices of materials and equipment with an imported content have been converted to US$ at the rates of exchange stated previously in this document. All pricing received has been entered into the estimate using native currency.
- Contractor rates and distributables include mobilisation / demobilisation, recurring costs, direct and indirect labour, construction equipment, construction cranes up to 100 t, materials, materials handling and offloading, temporary storage, construction facilities, off site costs, insurances, flights, construction fuel (US$1.20/L), tools, consumables, meals and PPE.
- Site construction offices will be containerised units only, with the intention that the permanent administration building construction schedule will be accelerated.
- The bulk commodities for earthworks that include for imported material are based on the assumption that suitable construction / fill materials will generally be available from borrow pits within 2 km of the work fronts, other than roads that will likely have longer haulage distances. Concrete imported materials have been included in the concrete installation rates by the contractor.
- ROM pad earthworks construction will be performed by a commercial contractor.
- Engineering quantities for the Tailings Storage Facility (TSF), Water Storage Pond (“**WSP**”), river abstraction, and sediment control structures (“**SCS**”) have been provided by Knight Piésold and appropriate rates applied to complete the capital cost of these items. Knight Piésold subconsultant costs for design and construction supervision are included.
- The bulk fill for the TSF and WSP embankments (Zone C) will be won from borrow and construction by a commercial contractor.
- Engineering quantities for Site Drainage Management have been provided by Knight Piésold and appropriate rates applied to complete the capital cost of these items.
- The estimate allows for aggregate and sand for concrete batching to be provided by the concrete contractor and are assumed available locally to the project site.
- The estimate allows for all reinforced bar and mesh for construction to be provided by the concrete contractor. Free issue of materials would be a project capital opportunity.

- There is no allowance for blasting in the bulk earthworks.
- Clearing, grubbing and topsoil removal for the treatment plant has been allowed in the mining cost estimate.
- The estimate allows for supply of structural steel and minor platework from South East Asia.
- No allowance for landscaping in the permanent village has been included in the capital estimate.
- Owner's mobile equipment to be purchased early and made available for construction and operations.
- The capital estimate includes an allowance for mill installation supervision by the vendor.

21.16 Exclusions

The following items are specifically excluded from the capital cost estimate:

- Various items discussed previously that have been dealt with directly by Cardinal in the financial model
- Permits and licences
- Project sunk costs
- Exchange rate variations
- Escalation.

21.17 Processing operating cost estimate

21.17.1 Summary

The purpose of this operating cost (Opex) estimate is to provide substantiated costs which can be utilised for an assessment of the viability of the Namdini Gold Project.

The operating costs have been compiled by Lycopodium.

Operating costs have been determined for a throughput rate of 9.5 Mtpa operating 24 hours per day, 365 days per year at a primary grind size of P_{80} 130 µm and a flotation concentrate regrind size of P_{90} 9 µm.

The estimate is considered to have an accuracy of +15% to -5%, is presented in United States dollars (US$) and is based on prices obtained during the first quarter of 2019 (1Q19).

21.17.2 Operating cost summary

The processing operating cost estimate is summarised in Table 165 and operating cost detail is available separately to Cardinal. The operating costs have been compiled from a variety of sources, including the following:

- OMC modelling for crushing and grinding energy and consumables, based on comminution testwork
- The LOM expected mass recovery of gold to flotation concentrate is 7.5% based on testwork managed by Cardinal
- Flotation reagent consumption based on testwork managed by Cardinal
- CIL reagent consumption based on based on testwork managed by Cardinal
- Calculated reagent usage for cyanide destruction and arsenic precipitation prior to testwork
- Typical industry data from equipment vendors

- Budget pricing from vendors or Lycopodium's database of unit prices for consumables
- Lycopodium's database of costs for similar sized operations
- Mining costs were solicited from five 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 increases in the depth of mining in line with standard mining cost principles.

Table 165: Processing operating cost estimate summary (±15%,1Q19)

Opex – Process costs	US$ M/y	US$/t	% of cost
Power[1]	49.5	5.21	45%
Operating consumables	48.0	5.06	44%
Maintenance	8.0	0.84	7%
Laboratory	1.9	0.20	2%
Process & maintenance labour	2.7	0.28	2%
Total Processing	**110.1**	**11.59**	**100%**
Administration labour	7.7	0.82	50%
General & administration costs	7.2	0.76	47%
Camp power[1]	0.4	0.05	3%
Total G&A	**15.4**	**1.62**	**100%**
TOTAL – Process	**125.4**	**13.21**	

Notes: [1]The average power tariff for the LOM is US$0.12/kWh. Cardinal's financial model uses a third party power offer at a variable tariff as detailed in Table 168 and Table 169

Costs and totals are rounded.

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 consumables
- 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.

21.17.3 Estimate basis

Labour

The processing labour cost includes all labour costs associated with plant operations and maintenance personnel.

The site laboratory is operated on a contract basis with the personnel included in the process labour head 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.

This includes the Owner's team responsible for management of mining operation but excludes the mining contractor's personnel. All labour costs for mining personnel are excluded (included in the mining costs).

The camp is operated on a contract basis with the personnel included in the administration labour head count but the labour costs are included in the camp contract cost.

The labour costs and manning numbers for plant operations and maintenance and site G&A are summarised in Table 166 and Table 167.

A list of positions, number of personnel, labour rates and costs detail is available separately to Cardinal.

The manning levels and rosters used to determine the labour operating cost estimate were based on similar operations. The estimate of the labour contingent has been 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 has been made for four weeks leave and two weeks sick leave per year per person.

Unit rates for labour have been based on information advised by Cardinal. An overhead cost allowance has been 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 166: Plant and administration personnel

Personnel	Number of personnel
Foreign expat	14
Regional expat	120
Local	361
Total	**495**

Table 167: Plant and administration labour cost

Area	Annual US$ M	US$/t
Plant operations	1,505,825	0.16
Plant maintenance	1,183,400	0.12
Total Plant	2,689,225	0.28
Administration	7,747,935	0.82
Total	10,437,160	1.10

Power

The power consumption for the SAG and ball mills has been 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 estimated specific grinding energy of 120 kWh/t to achieve a 10 µm P_{90} in the regrind product from testwork completed by the mill vendor (Outotec) and managed by Cardinal.

The power consumption for the remainder of the Project has been estimated from calculated power required for the process plant equipment and infrastructure. Typical drive efficiency and utilisation factors were applied to the installed power to estimate the plant average continuous power draw.

The average continuous power draw and power cost by plant area are summarised in Table 168. The power requirements and costs for the accommodation camp are also included.

The power cost estimate has been based on grid power at an average LOM unit cost of US$0.12/kWh. Cardinal's financial model uses a third party power offer at a variable tariff. This was provided as a build own operate ("**BOO**") option, being a unit cost of US$0.125/kWh for the first three years of operation (Table 168) and US$0.115 thereafter (Table 169).

Table 168: Power summary Years 1 to 3, based on BOO option

Area	Installed power kW	Peak cont. draw kW	Average cont. draw kW	Power cost US$/y	Power cost US$/t
Primary crushing	1,764	1,394	933	1,021,532	0.11
Stockpile & reclaim	380	318	274	300,059	0.03
Milling power	36,000	25,531	23,316	25,531,250	2.69
Grinding (excluding mills) & pebble crushing	6,665	3,438	2,749	3,010,029	0.32
Gravity	486	216	194	212,452	0.02
Flotation	2,032	1,528	1,377	1,507,629	0.16
Concentrate thickening & regrind	11,725	11,481	10,455	11,448,570	1.21
Concentrate CIL	1,618	1,183	1,010	1,105,659	0.12
Elution & gold room	307	239	108	118,056	0.01
Cyanide destruction & CIL tails disposal	965	550	484	529,577	0.06
Flotation tails disposal	3,485	1,707	1,550	1,696,982	0.18
Reagents & fuels	326	237	86	94,161	0.01
Water services	1,727	1,030	835	913,969	0.10
Plant sewage	74	59	43	46,954	0.00
Air services	2,425	1,317	1,203	1,317,000	0.14
Raw water supply	746	364	256	280,538	0.03
Tailings storage facility	221	124	113	123,675	0.01
Plant buildings	321	257	242	264,771	0.03
Total Plant	**71,266**	**50,972**	**45,226**	**49,522,862**	**5.21**
Camp	898	716	391	428,047	0.05
Total	**72,163**	**51,688**	**45,617**	**49,950,909**	**5.26**

Table 169: Power summary Post-Year 3, based on BOO option

Area	Installed power kW	Peak cont. draw kW	Average cont. draw kW	Power cost US$/y	Power cost US$/t
Primary crushing	1,764	1,394	933	939,810	0.10
Stockpile & reclaim	380	318	274	276,054	0.03
Milling power	36,000	25,531	23,316	23,488,750	2.47
Grinding (excluding mills) & pebble crushing	6,665	3,438	2,749	2,769,227	0.29

Area	Installed power kW	Peak cont. draw kW	Average cont. draw kW	Power cost US$/y	Power cost US$/t
Gravity	486	216	194	195,456	0.02
Flotation	2,032	1,528	1,377	1,387,018	0.15
Concentrate thickening & regrind	11,725	11,481	10,455	10,532,684	1.11
Concentrate CIL	1,618	1,183	1,010	1,017,207	0.11
Elution & gold room	307	239	108	108,611	0.01
Cyanide destruction & CIL tails disposal	965	550	484	487,210	0.05
Flotation tails disposal	3,485	1,707	1,550	1,561,223	0.16
Reagents & fuels	326	237	86	86,628	0.01
Water services	1,727	1,030	835	840,851	0.09
Plant sewage	74	59	43	43,197	0.00
Air services	2,425	1,317	1,203	1,211,640	0.13
Raw water supply	746	364	256	258,095	0.03
Tailings storage facility	221	124	113	113,781	0.01
Plant buildings	321	257	242	243,589	0.03
Total Plant	**71,266**	**50,972**	**45,226**	**45,561,033**	**4.80**
Camp	898	716	391	393,803	0.04
Total	**72,163**	**51,688**	**45,617**	**45,954,836**	**4.84**

Consumables

Costs for processing operating consumables, including reagents, liners, fuels and process supplies have been estimated and are summarised by plant area in summarised in Table 170.

The consumables cost for mining is included in the mining operating cost.

Table 170: Processing consumables summary

	Annual cost US$ M	Unit cost US$/t
Crushing	244,415	0.03
Milling	11,150,994	1.17
Flotation	3,443,750	0.36
Regrind	9,842,363	1.04
Leaching	10,655,952	1.12
Thickening	281,483	0.03
Refining	448,789	0.05
Cyanide destruction	10,540,863	1.11
Water treatment	7,487	0.001
Fuel	662,429	0.07
Smelting	38,983	0.004
Other	707,000	0.07
Total	**48,024,459**	**5.06**

The consumption of reagents and other consumables has been calculated from laboratory testwork and comminution circuit modelling or has been estimated based on experience with other operations. No additional allowance for process upset conditions and wastage of reagents has been allowed. Consumption rates detail is available separately to Cardinal.

Reagent costs have been sourced from budget quotations and in-house data relating to similar projects in the region. Transport and freight cost to site have been added. Import duties and taxes are excluded.

A diesel price, delivered to site, of US$0.90/L has been advised by Cardinal. 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 utilisation. The diesel usage for elution heating, carbon regeneration and the gold room furnace has been calculated from first principles.

Allowances have been made for water treatment reagents and operator supplies. Lubricants are excluded.

Maintenance materials

The plant maintenance cost allowance has been nominated by Cardinal as summarised in Table 171, for which the detail is available separately to Cardinal.

Table 171: Maintenance summary

Area	Plant maintenance cost	
	US$/y	US$/t
Plant maintenance	7,125,000	0.75
Mobile equipment	239,910	0.03
Maintenance general	177,000	0.02
Contract labour	446,052	0.10
Total Cost	**7,987,962**	**0.85**

The allowance covers mechanical spares and wear parts, but excludes crushing and grinding wear components, grinding media and general consumables which are allowed for in the consumables cost.

The maintenance cost excludes site maintenance labour which is included in the labour cost. Contract labour has been allowed for crusher and mill liner changes and plant shutdowns.

Allowances for plant mobile equipment, plant building maintenance and general maintenance expenses have been 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.

Laboratory

The laboratory cost is based on assaying and analyses of periodic plant samples and 260 mining grade control samples per day.

Table 172: Laboratory sample schedule

Item	Samples/month
Grade control (assay)	7,800
Plant solids (assay, sizing, etc.)	480
Plant solutions (assay)	960
Plant carbon (assay)	90
Bullion (Au)	60
Environmental solutions (CN_{TOTAL}, CN_{WAD}, etc.)	120
Reagent quality	120

Laboratory costs have been based on a budget price supplied by a potential assay laboratory service provider.

Table 173: Laboratory summary

Item	US$ M/y
Fixed fee	1,155,204
Variable fee	701,775
Total	**1,856,979**

General and administration

The general and administration costs were based on information from the Lycopodium database. The G&A costs are summarised in Table 174 for which the detail is available separately to Cardinal. The administration labour cost was calculated as described above.

Table 174: General and administration summary

Cost Centre	US$/y
Site Office	811,200
Insurances	1,103,000
Financial	184,000
Government Charges	0
HR Administration	508,184
Contracts	3,425,540
Community Relations	168,000
Other	991,371
Total G&A	**7,191,295**

21.18 Processing pre-production and working capital

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.

Pre-production costs associated with mining are excluded.

The pre-production cost estimate is summarised below, for which the detail is available separately to Cardinal:

- Administration labour and expenses US$6,169 million
- Process plant labour and expenses US$0.560 million.

21.18.1 Consumable first fills and opening stocks

Costs have been 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 until regular ordering of supplies can be established.

Quantities allowed have been based on either consumption over a minimum period or minimum shipping quantities, considering package size.

Costs, for which detail is available separately to Cardinal, are summarised as follows:

- First Fill US$2.899 million
- Opening Stocks US$7.074 million.

21.18.2 Laboratory pre-production cost

The lab facility will be operational about six months prior to plant start-up. The monthly fee for laboratory operations is approximately US$677 600.

21.18.3 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, for which the detail is available separately to Cardinal, are summarised as follows:

- Vendor representatives US$0.646 million
- Training US$0.295 million.

21.18.4 Working capital

Six weeks of processing and administration operating costs has been allowed to cover the cost of operating the plant before the first revenue is received from bullion sales.

The cost for this is estimated to be US$12.320 million.

21.19 Other operating costs

Contract mining costs per tonne of ore processed (138.6 Mt of ore) at a throughput of 9.5 Mtpa are shown in Table 175.

Table 175: Contract mining costs

Item	Cost/t milled (US$)
Variable mining cost – Waste	5.08
Variable mining cost – Ore	3.53
Management fees	1.68
Services	0.30
Total	**10.59**

The variable mining cost for ore and waste includes drill and blast (i.e. production drill and blast, presplits, blast movement monitors and drains holes), load and haul, road and other ancillary works. The variable mining cost for ore also includes rehandle of ore from stockpiles to the ROM and crusher.

Management fees include contractor fixed monthly management fees, supervision, explosives management, training, dewatering, health and safety, freight, communication, vehicles, transport and other general costs.

Services includes the mine dispatch system, contractor mobilisation and demobilisation, survey, dewatering, technical services running costs and other costs (e.g. consultants).

Owner's costs per tonne of ore processed (138.6 Mt of ore) at a throughput of 9.5 Mtpa are shown in Table 176.

Table 176: Owner's costs

Opex – Owner's costs	US$/t
Grade control	0.76
10% ROM rehandle	0.07
Owner's G & A	0.60
TOTAL	**1.43**

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. These costs include rehabilitation and mine closure.

Sustaining capital costs per tonne of ore processed (138.6 Mt of ore) at a throughput of 9.5 Mtpa are shown in Table 177.

Table 177: Sustaining costs

Sustaining capital cost	US$ '000	US$/t
Flotation TSF sustaining capital	102,696	0.74
CIL TSF sustaining capital factor	20,910	0.15
Vehicles & equipment sustaining capital	6,678	0.05
TSF sustaining costs – mining waste to TSF	27,513	0.20
Mine closure cost	23,789	0.17
TOTAL	**181,586**	**1.31**

21.20 Additional information

Appendices provided in Lycopodium (2019c) include:

Appendix 21.1 Capital Cost Estimate

Appendix 21.2 Operating Cost Estimate.

22.0 ECONOMIC ANALYSIS

22.1 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 16.16, 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.

22.2 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, the estimation of the 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.

22.3 Methodology

The economic assessment was carried out Pre-Tax and has assumed a total allowance for royalties of 5% of the gross gold price.

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 is the total revenue generated, minus total costs. The calculation therefore is total recovered ounces (4.196 Moz), multiplied by the gold price selected (US$1,350/oz), minus the total cost per ounce (US$/oz). Pre-Tax cash flow is before tax is deducted, while Post-Tax is after tax is deducted.

22.4 Financial summary

The Project has been evaluated on a discounted cashflow basis. The results of the analysis show the Project to be economically very favourable. The net present value of the net cashflow based on a 5% discount rate (NPV_5) is US$914 million on a Pre-Tax project basis, using a base gold price of US$1,350/oz. The Post-Tax NPV_5 at a US$1,350 gold price for the Project is US$590 million on an all equity basis. Project internal rates of return (IRR) are 43% Pre-Tax and 33% Post-Tax.

Project payback period at a gold price of US$1,350/oz is expected to be 15 months on a Pre-Tax basis and 21 months on a Post-Tax, all equity basis. Payback period is defined as the time on commencement of gold production that is required to recover the initial expenditures incurred in developing the Project. This is the point in time when the Project cumulative undiscounted net cashflow is zero.

The total Life of Mine (LOM) of the Project is 15 years gold production following a 27-month construction period. The ore starts near surface, hence mining of the Ore Reserve commences immediately with minimal waste.

This economic analysis assumes that Cardinal will secure all development funding via loans to the Project, which will be paid back with interest from future gold sales.

The Government of Ghana will own a direct 10% free-carried interest in the Project. This economic analysis has been prepared on a whole of Project basis.

The key economic indicators of NPV and IRR are most sensitive to changes to revenue, principally the US dollar gold price. A US$100/oz reduction in the gold price would reduce the Post-Tax NPV for the Project by US$175 million and reduce the Post-Tax IRR by 7%. A US$100/oz increase in the gold price would increase the Project Post-Tax NPV by US$186 million and increase the Post-Tax IRR by 7%.

The Project Post-Tax NPV and Post-Tax IRR are equally sensitive to a US$100/oz increase or decrease in the gold price from the base price of US$1,350/oz.

The cashflow analysis has been prepared on a quarterly basis and incorporates an annual inflation rate of 2.5% pa on all operating costs. No inflation or escalation of gold price has been incorporated and the gold price is assumed to be constant for the period of the economic analysis.

22.5 Key economic results

The key economic results at a gold price of US$1,350/oz are provided in Table 178.

Table 178: Key economic results

Key Economic Results		Unit	Feasibility Study
Capital cost *(including 11% contingency)*		US$ M	390
All-in Sustaining Cost *(AISC)*[1]	*Starter Pit*	US$/oz	585
	Life of Mine		895
Total Project payback		Months	21
Pre-Tax NPV US$ *(5% discount rate)*		US$ M	914
Post-Tax NPV US$ *(5% discount rate)*		US$ M	590
Pre-Tax IRR		%	43
Post-Tax IRR		%	33

Notes: [1] Cash Costs + Royalties + Levies + Life of Mine Sustaining Capital Costs (World Gold Council Standard)
Royalties calculated at a rate of 5.5% and a corporate tax rate of 32.5%; both subject to negotiation and expected to be finalised over coming months.

22.5.1 Production summary – Starter Pit and Life of Mine

A summary of the expected production is provided in Table 179 for the Starter Pit and Table 180 for the Life of Mine (including the Starter Pit).

Table 179: Starter Pit proposed production summary

Key Estimated Production Results	Unit	Feasibility Study
Gold price	US$/oz	1,350
Gold produced *(average for full production years)*	koz/y	361
Gold head grade *(Starter Pit, Ore Reserve)*	g/t Au	1.41
Gold recovery *(Starter Pit)*	%	85
Strip ratio *(Starter Pit)*	W:O	0.9 : 1
Ore tonnes mined *(0.5 g/t cut-off grade)*	Mt	47
Waste tonnes mined	Mt	43
Starter Pit life *(including ramp-up)*	Months	27
Total Project payback	Months	21

Table 180: LOM proposed production summary, including the Starter Pit

Key Estimated Production Results	Unit	Feasibility Study
Gold price	US$/oz	1,350
Gold produced *(average for LOM full production years)*	koz/y	287
Gold produced *(LOM)*	Moz	4.2
Gold head grade *(LOM, Ore Reserve)*	g/t Au	1.13
Gold recovery *(LOM)*	%	83
Strip ratio *(LOM)*	W:O	1.9 : 1
Ore tonnes mined *(0.5 g/t cut-off grade)*	Mt	138.6
Waste tonnes mined	Mt	263
Mine life *(Including ramp-up and mine closure)*	years	15
Capital cost *(including 11% contingency)*	US$ M	390
Total Project payback	Months	21

22.6 Economic sensitivity studies

Based upon Life of Mine production and cost parameters, the Post-Tax NPV sensitivities are shown for the 9.5 Mtpa throughput in Table 181.

Table 181: Post-Tax NPV (in US$ M) and gold price sensitivity

Post Tax Real Discount Rate (%)	Gold Price (US$/oz)				
	US$1,150	US$1,250	US$1,350	US$1,450	US$1,550
0	441	720	**980**	1,257	1,527
5	228	411	**590**	769	950
10	105	233	**355**	486	614

Note: All NPVs are Post-Tax values in US$ M

Bar charts in Figure 146 to Figure 149 show the expected Pre-Tax and Post-Tax economic sensitivities for the 9.5 Mtpa throughput at a gold price of US$1,350/oz.

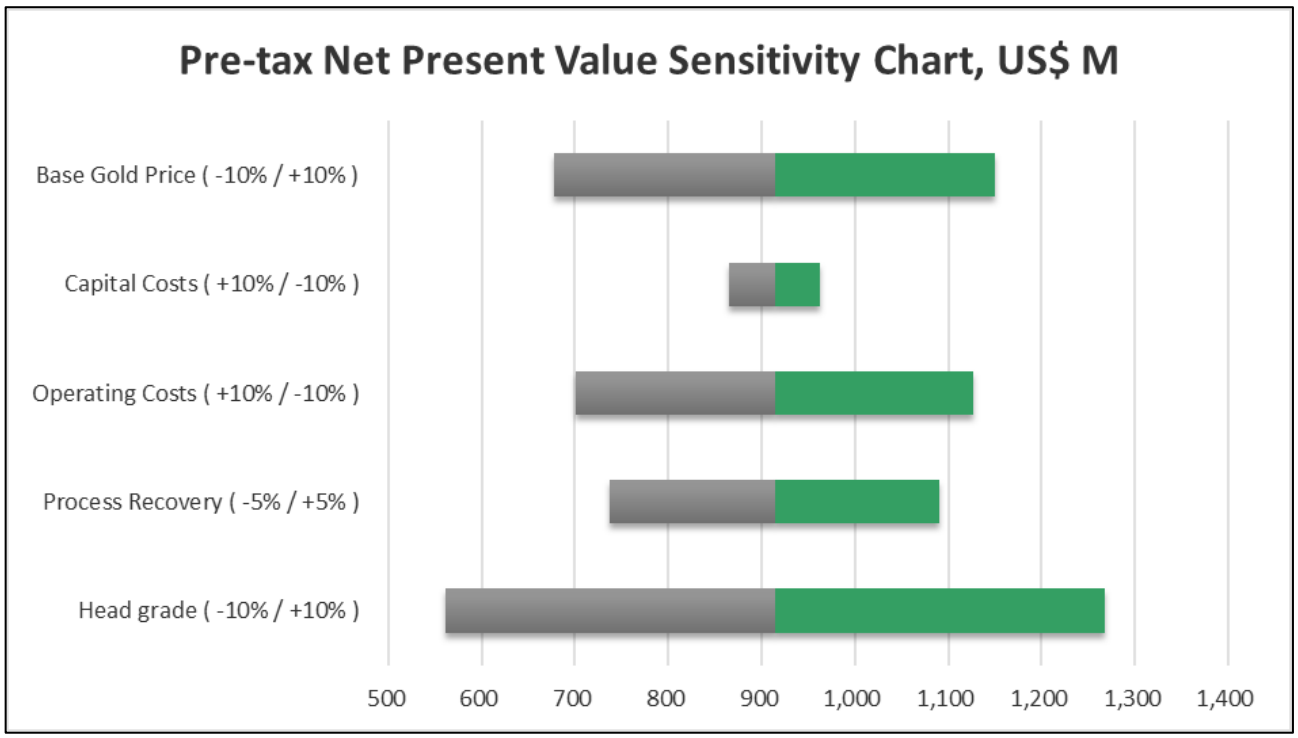


Figure 146: Pre-Tax NPV sensitivity at 5% discount (US$ M) for the 9.5 Mtpa throughput

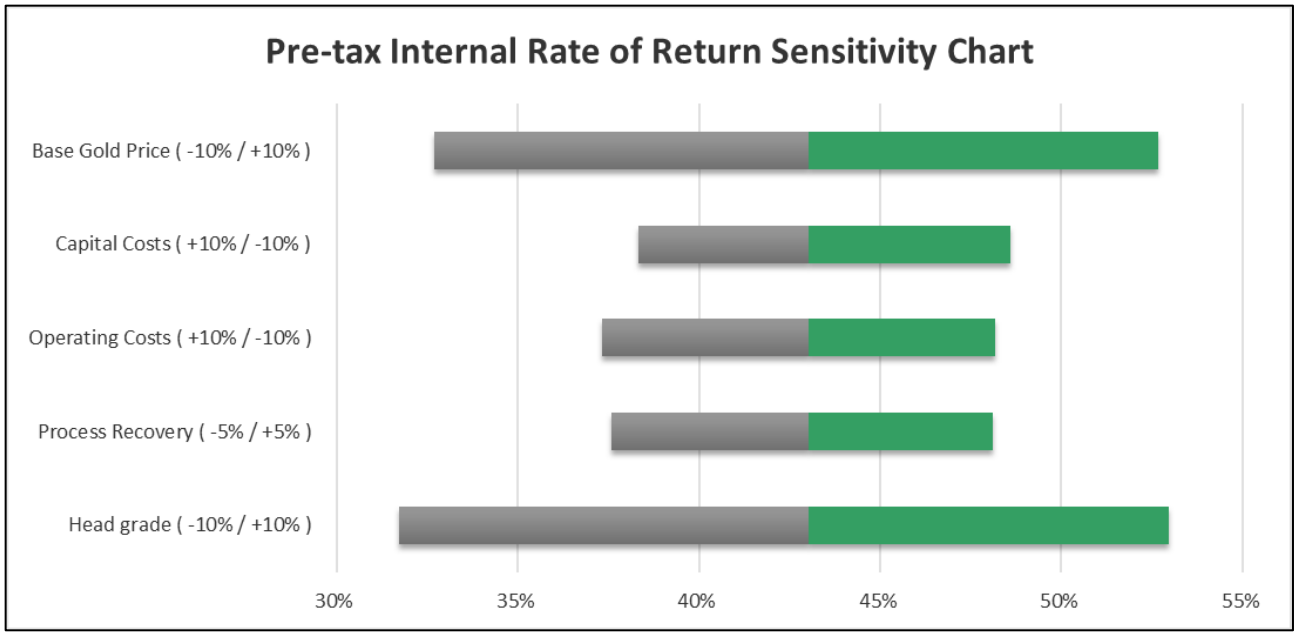


Figure 147: Pre-Tax IRR (%) sensitivity for the 9.5 Mtpa throughput

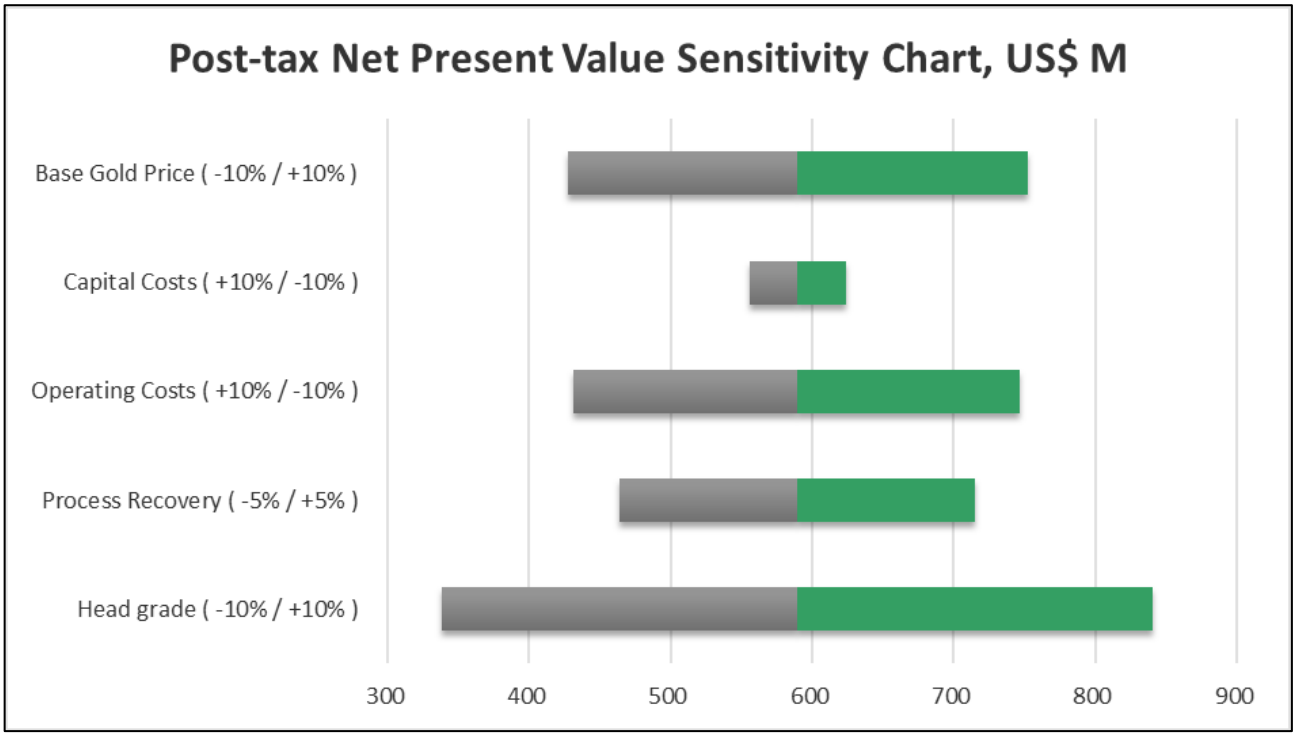


Figure 148: Post-Tax NPV sensitivity at 5% discount (US$ M) for the 9.5 Mtpa throughput

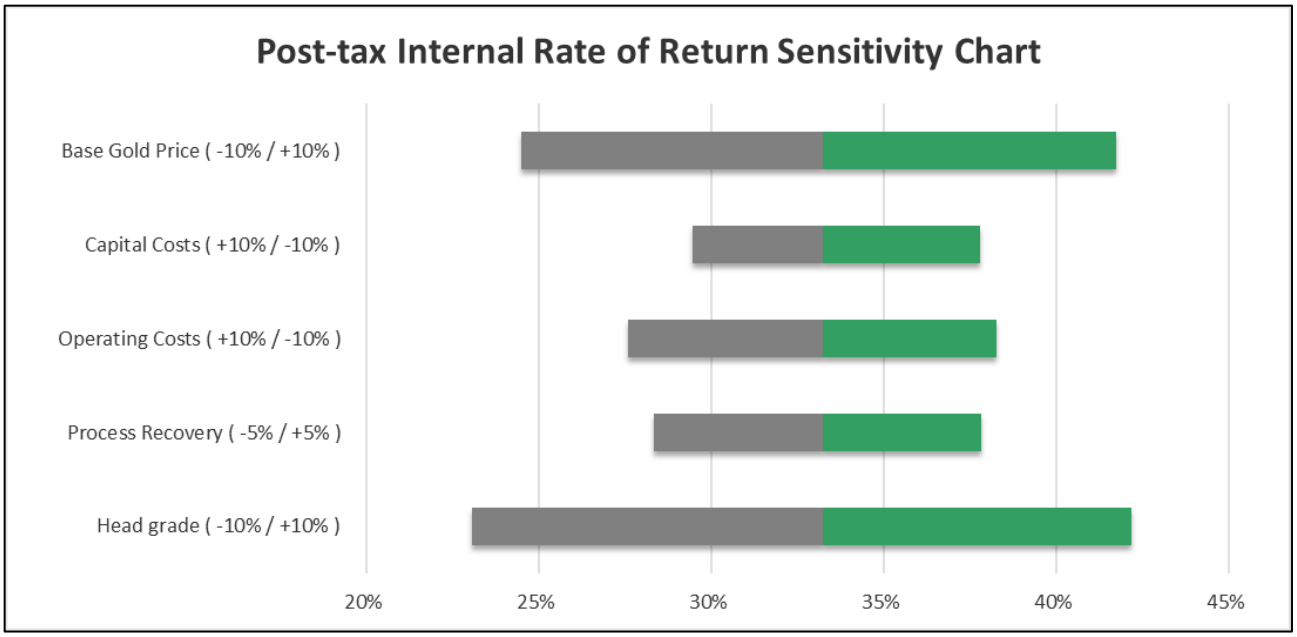


Figure 149: Post-Tax IRR (%) sensitivity for the 9.5 Mtpa throughput

Figure 150 shows the annual gold production against the accumulative post tax cash flow at a gold price of US$1,350/oz.

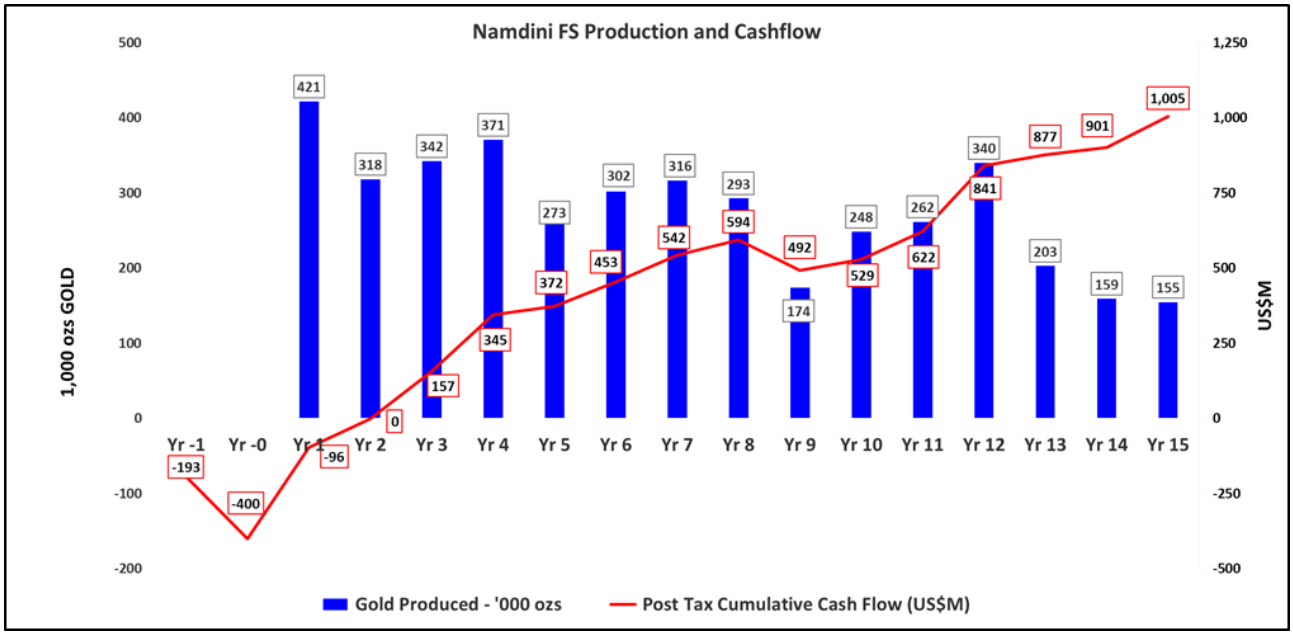


Figure 150: Namdini FS ounce production and net cash flow (100%)

Figure 151 shows the annual gold production against the All-in Sustaining Cost (AISC) in US$/oz.

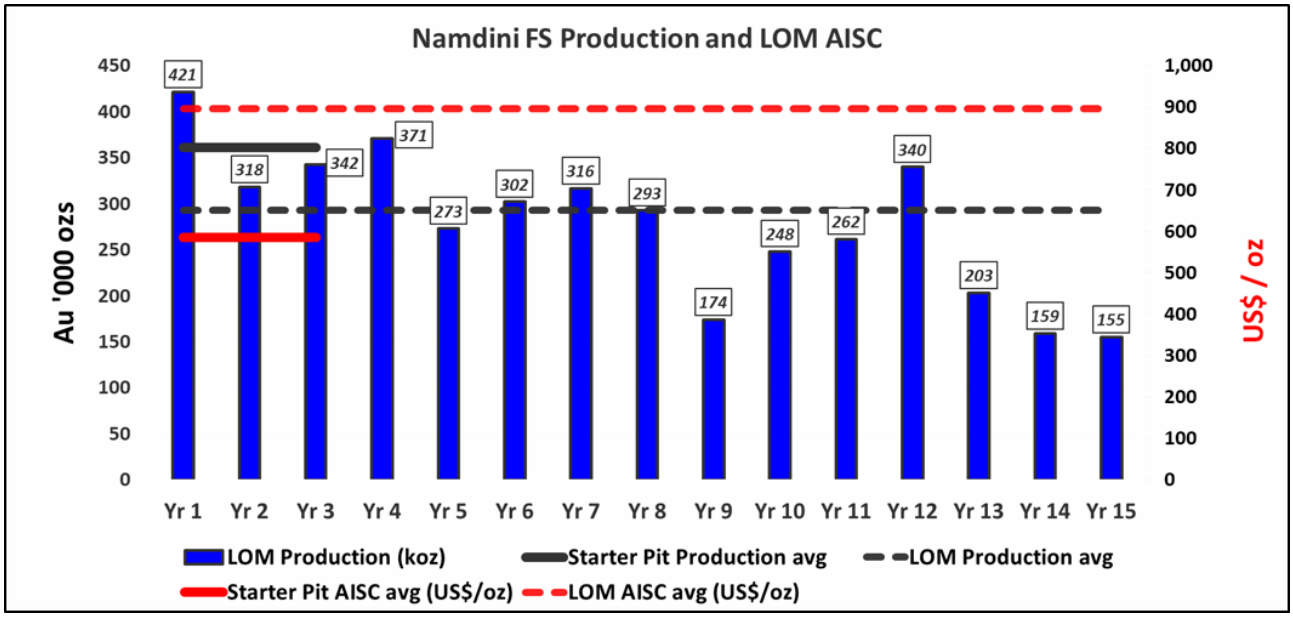


Figure 151: Namdini FS ounce production & All-in Sustaining Cost (100%)

22.7 Mine production and mill feed

The mine production schedule presented in Section 16.0 of this Report is summarised into an annual mine production and mill feed schedule shown in Table 183.

Total LOM mill feed is 138.6 million tonnes (Mt) at an average LOM grade of 1.13 g/t Au. The average mill feed grade during the first three years of operations is 1.41 g/t Au.

Open pit mining commences in the quarter immediately prior to the commencement of mill operations and continues for the 12 years and 6 months of processing plant operations.

The existence of ore from surface allows a short mining start-up, with minimal waste in the first two years of mining. Total LOM waste to ore ratio is 1.9 tonnes waste to 1 tonne of ore. In the first three years of operation the waste to ore ratio is 0.5 tonnes waste to 1 tonne of ore.

Ore will be stockpiled adjacent to the crusher and rehandled to the plant as required to meet the processing schedule. These stockpiles represent an inventory buffer between the mine and the processing plant.

Total ore is milled over a total period of 14 years and 6 months at an annual throughput rate of 9.5 Mtpa.

The ore processing rate is a function of the significant process and metallurgical testwork (Section 13.0) and the process flowsheet design (Section 17.0). A lower plant throughput rate is assumed during the initial quarter for commissioning and ramp-up.

Table 183 includes annual estimates of recovered gold, derived from the overall (i.e. gravity, leach plus cyanidation) process recovery curve described in Section 13.0. The average overall process recovery over the LOM is estimated to be 82.9%. Total recovered gold over the LOM is estimated to be 4.2 Moz, for an average of 287 koz per year over the fifteen-year ore processing period. The average production rate during the first three years of operations is estimated to be 361 koz per year with expected production of 421 koz during the first year of operation.

22.8 Gold price assumption

The Project economic assessment has been evaluated at a gold price of US$1,350/oz.

The average US dollar gold price per ounce for the period October 2017 to October 2019 was US$1,387/oz. During this period the US dollar gold price has been in steady uptrend from a low of US$1,177/oz to a high of US$1,550/oz traded during September 2019.

The gold price has been relatively stable over the past two years as shown in Figure 152. The impact of a range of higher and lower gold prices on project economic results is addressed as part of project sensitivity analysis.



Figure 152: Average US dollar gold price per ounce for the period October 2017 to October 2019

22.9 Cost estimates

22.9.1 Capital and operating costs

Capital cost estimates are presented in Section 21.0 of this Report. Initial capital expenditure is estimated at US$348 million plus an additional contingency allowance of US$42 million. LOM sustaining capital is estimated at US$182 million.

The initial capital cost estimate for mine development is US$4.5 million of pre-production waste mining.

Operating cost estimates are presented in Table 165. In addition to the process operating costs, an allowance of US$1.10/oz of recovered gold has been included for refining, freight and insurance costs. The capital and operating cost estimates are based on a diesel fuel price of US$0.90/L. Electricity is to be supplied from the main power grid of Ghana and processing operating costs presented are based on an average power price of US$ 0.12/kWh for the LOM. Cardinal's financial model uses a third party power offer at a variable tariff.

The Project All-in Sustaining Cost (AISC) averages US$585/oz of gold produced over the first three years of operations and US$895/oz over the LOM.

22.9.2 Closure and salvage value

Mine closure is discussed in Section 20.6.2.4. A net amount of US$24 million has been included within sustaining capital costs for mine closure. This amount is calculated as the difference between the mine closure costs and the salvage value of the equipment and facilities. This cost is incurred over the final two production years.

22.9.3 Working capital

Cardinal has assumed that approximately four weeks of gold production will be contained within mill circuits and in doré inventory on site or in transit to the refinery. These delays in the receipt of gold revenue contribute to project working capital requirements.

Working capital is also required to maintain an operating supplies inventory. Approximately six weeks of operating supplies will be purchased in advance of commencement of operations and stored on site.

Cardinal has assumed that accounts payable, equivalent to the average of one month of capital and operating costs, will partially off-set these working capital requirements.

22.10 Taxes and royalties

A number of taxes and royalties are included in the economic evaluation, as described below.

Cardinal is in the process of negotiating the applicability and the rates of taxes that will be levied on the Project. In addition, Cardinal is negotiating the timing of assessment of various taxes. Cardinal provided advice on Ghanaian mining legislation and taxation and assisted in the modelling of the tax treatment.

22.10.1 Surface taxes and Mining Licence fees

The Government of Ghana assesses annual surface rates on the Mining Licence which are included in the sustaining capital estimate.

22.10.2 Government royalty

The government of Ghana assesses a variable rate gross revenue royalty on gold projects. Cardinal is negotiating the rate of royalties that will apply to the Namdini Gold Project The royalty rates used in the economic assessment is based on royalty rates that presently apply to projects of the size of the Namdini Gold Project. The gold royalty rates used are:

- 3% of gross revenue if the gold price is less than US$1,300/oz;

- 3.5% of gross revenue if the gold price is greater than or equal to US$1,300/oz and less than or equal to US$1,450/oz;
- 4% of gross revenue if the gold price is greater than US$1,450 and less than or equal to US$2,300, and
- 5% of gross revenue if the gold price exceeds US$2,300/oz.

22.11 Corporate income tax

A corporate tax rate of 32.5% is has been adopted for the Namdini Gold Project. Cardinal is negotiating the rate of income tax applicable for this economic assessment based on current income tax rates as they apply to projects of the size of the Namdini Gold Project.

Deductions from income for the purpose of estimating income subject to tax include the items described below.

22.11.1 Depreciation

All capital construction costs and fixed facilities are depreciated based on concessional depreciation rates. Cardinal is negotiating the rate of income tax applicable for this economic assessment based on current income tax rates as they apply to projects of the size of the Namdini Gold Project.

The depreciation rate adopted in the economic assessment was 30% per annum until fully depreciated.

22.11.2 Carry forward income tax losses

Sunk exploration and other eligible project costs are presently carried forward and deducted in determining taxable income. Cardinal estimates that its eligible sunk project costs total US$15.4 million. Mine operating losses can also be carried forward and deducted from income in future years.

22.12 Duties and levies

The Government of Ghana assesses a customs duty of 5% and other levies totalling 2.5% on imported goods. Cardinal is negotiating the exemption from customs duty, but the other levies are applicable. After start up all imported goods (as published in the 10th edition of the Ghana Revenue Authority Mining List) are assessed at a total of 2.5% duties and levies.

A detailed estimate of duties and levies applicable to the Project has not been completed for this study. For the purposes of cashflow forecasting it is assumed that levies are applicable to 31% of the initial capital costs and duties and levies are applied to 11% of sustaining capital costs.

22.13 Value Added Tax

Ghana has a Value Added Tax ("**VAT**") rate of 12.5%. A detailed estimate of VAT for operating cost items (excluding labour) has not been completed for this study.

For the purposes of cashflow forecasting it is assumed that VAT is not applicable to 70% of the initial and sustaining capital costs and 74% of mining, processing and general and administrative costs and is refunded monthly after it is charged.

22.14 Withholding tax

The Government of Ghana assesses withholding taxes on dividends and interest payable. Cardinal is negotiating the rate of withholding tax applicable for dividends and interest payable and has adopted a withholding tax rate of 8% for this economic assessment based on the rate applicable to double tax treaty rates as they currently apply.

22.14.1 Government carried interest

The Minerals and Mining Act reserves for the Government a ten per cent (10%) free carried interest in the Mining Operations conducted on the Namdini Mining Licence and the Government shall receive dividends of a sum equal to 10 per cent of the total amount paid as dividends to the shareholders of the Mining Licence on each occasion when dividends are distributed.

This economic assessment is based on a Project basis and the Project results are before the distribution of dividends.

22.15 Economic results

The economic results for the Project, as summarised in Table 182, are very favourable.

The net present value of the cashflow with a 5% discount rate (NPV_5) is US$914 million on a Pre-Tax, project basis, using a base gold price of US$1,350/oz. Project Post-Tax NPV at a US$1,350 gold price is US$590 million on an all equity basis. Project internal rates of return (IRR) are 43% Pre-Tax and 33% Post-Tax.

At a gold price of US$1,350/oz, the Project Post-Tax, all equity basis payback period is expected to be 21 months.

Detailed cashflow estimates by year are presented in Table 183.

Table 182: Namdini Gold Project economic results summary

	Units	LOM Total
Gold Revenue		
Gold Price	US$/oz	1,350
Gold Sales	oz	4,168,522
Gold Sales Revenue	**US$ M**	**5,628**
Pre-Production Costs		
Development Capital	US$ M	-390
Pre-production Open Pit Mining	US$ M	-4
Production Period Costs		
Mining costs	US$ M	-1,429
Processing	US$ M	-1,774
Refining, Freight and Insurance	US$ M	-5
Sustaining Capital and Closure costs	US$ M	-182
General and Administrative	US$ M	-64
Royalties	US$ M	-311
Project Net Cashflow, Pre-Tax	**US$ M**	**1,458**
NPV 5%	**US$ M**	**914**
IRR	**%**	**43%**
Payback Period	**Months**	**15**
Project Net Cashflow, Post-Tax, All Equity Basis		
Project Net Cashflow, Pre-Tax, from above	US$ M	1,458
Income Tax, all equity basis	US$ M	-433
Project Net Cashflow, Post-Tax, all equity basis	**US$ M**	**1,025**
NPV 5%	**US$ M**	**590**
IRR	**%**	**33%**
Payback Period	**Months**	**21**

Table 183: Production summary at a gold price of US$1,350/oz

	Unit	LOM Total	Yr -2	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15
PRODUCTION																			
Mine Plan – Waste	Mt	263		11.4	10.3	15.6	20.6	20.1	26.4	24.8	22.8	29.0	33.1	29.4	14.0	5.0	0.6		
Mine Plan – Ore	Mt	139		4.3	15.7	15.4	15.4	15.9	9.6	11.2	13.2	7.0	2.9	6.1	9.3	10.5	1.9		
Strip Ratio	W:O	1.9		2.6	0.7	1.0	1.3	1.3	2.8	2.2	1.7	4.1	11.6	4.8	1.5	0.5	0.3		
Production																			
Processed Tonnes	Mt	138.6			9.2	9.5	9.5	9.5	9.5	9.5	9.5	9.5	9.5	9.5	9.5	9.5	9.5	9.5	5.9
Grade	g/t Au	1.13			1.65	1.24	1.33	1.42	1.07	1.19	1.23	1.15	0.75	1.01	1.05	1.32	0.85	0.69	0.98
Recovery	%	83			86.1	84.2	84.2	85.4	83.7	83.0	83.9	83.6	75.7	80.4	81.7	84.2	78.1	75.0	86.2
Gold Produced	**koz**	**4,177**			**421**	**318**	**342**	**371**	**273**	**302**	**316**	**293**	**174**	**248**	**262**	**340**	**203**	**159**	**155**
Revenue																			
Gold Price	US$/oz	**1,350**			1,350	1,350	1,350	1,350	1,350	1,350	1,350	1,350	1,350	1,350	1,350	1,350	1,350	1,350	1,350
Revenue	**US$ M**	**5,628**			**568**	**429**	**461**	**500**	**368**	**407**	**426**	**394**	**234**	**334**	**353**	**458**	**273**	**214**	**209**
Operating Costs																			
Mining	US$ M	1,433		19	82	99	114	119	123	126	127	126	132	140	106	72	24	16	9
Processing	US$ M	1,774			122	125	125	122	122	122	122	122	122	122	122	122	122	122	64
Owner's Costs	US$ M	80	3	5	8	5	5	5	5	5	5	7	7	6	5	5	4	1	1
Royalties – 5%	US$ M	311			32	24	25	27	20	22	23	22	13	18	19	25	15	12	11
Total	**US$ M**	**3,598**	**3**	**24**	**244**	**253**	**269**	**273**	**270**	**275**	**277**	**276**	**273**	**285**	**252**	**224**	**164**	**150**	**86**
Capital Costs																			
Construction Capex	US$ M	**390**	**190**	**183**	**17**														
Sustaining Capex	US$ M	158			3	12	15	11	9	10	10	9	9	12	8	7	9	19	15
Mine Closure	US$ M	24														4	8	8	4
Taxes	US$ k	453				68	20	28	61	41	51	57	54	0	0	4	56	13	0
Cash Flow from Operations																			
Net Cash Flow – Pre Tax	**US$ M**	**1,458**	**-193**	**-207**	**303**	**164**	**177**	**216**	**89**	**122**	**140**	**109**	**-48**	**37**	**92**	**224**	**92**	**37**	**104**
Net Cash Flow – Post tax	**US$ M**	**1,005**	**-193**	**-207**	**303**	**96**	**157**	**188**	**27**	**81**	**89**	**52**	**-102**	**37**	**92**	**219**	**36**	**24**	**104**

Note: Differences between the total cashflow over Life of Mine and the sum of the cashflows in each year is due to rounding.

22.16 Funding

Cardinal's Board has approved this Feasibility Study and recommends progressing the Project to construction pending successful completion of financing activities.

- The gold price is currently trading at approximately US$1,500/oz, which compares favourably to the Project's financial assumption of US$1,350/oz.
- The Company has been actively working with its debt advisor Cutfield Freeman to secure Project finance on competitive terms and to ensure that Project finance can be secured in a timely manner following completion of the Feasibility Study. A number of leading financial institutions are involved in the process which is well advanced and is continuing. Cardinal looks forward to providing more details in the future.
- In addition to traditional financing solutions, the Company will also evaluate alternatives to bring the project into production.
- The strong production and economic outcomes delivered in the Namdini FS are considered by Cardinal's Board to be sufficiently robust to provide confidence in the Company's ability to secure project funding.
- A final financing decision will be made at a time assessed as the most appropriate and beneficial to the Company.
- In the meantime, the Company remains in a strong financial position with cash reserves of approximately A$27 million.

23.0 ADJACENT PROPERTIES

The Namdini Gold Project site is located approximately 6 km southeast of the operating Shaanxi underground which uses grid power.

24.0 OTHER RELEVANT DATA AND INFORMATION

In the opinion of the Authors, all relevant and material information is provided in this NI 43-101 Technical Report, based on the Feasibility Study documentation.

Further information relating to the Project Execution Plan ("**PEP**") and schedule is available in Lycopodium (2019d) which forms part of the complete Feasibility Study Report documentation. This document also outlines at a high level the Project Execution Strategy ("**PES**") that has been incorporated in the Project Master Schedule ("**PMS**") for the successful management of the Project. It also summarises the Value Engineering Assessment ("**VEA**") option studies that have been carried out to date.

25.0 CONCLUSIONS

25.1 Overall

- The Namdini Gold Project is a significant gold deposit with highly attractive economics, which enjoys both regional and national support, strongly reinforcing the decision to proceed with development of the project.
- The project is located in a socially and economically stable region of the world. The climate is moderate and local infrastructure is available and has the required capacity to support the project.
- Cardinal has a long standing and robust relationship with the region's Development Committee, Talensi Traditional Council, regional communities, Chiefs and Elders.
- There is strong support nationally and regionally for the project to be developed in a sustainable way to provide an uplift in an area that has not realized local investment and development opportunities. The social benefits are expected to be similar to those experienced in southern parts of Ghana where gold mines operate.
- The region is not densely populated and relocation requirements are not onerous. Community engagement with Cardinal's RAP consultants shows support for the Project.
- Vegetation is sparse and the tenement area is flat thereby allowing for low cost and relatively easy bulk earthwork and transportation access requirements and activities.
- The project site is approximately 25 km from the N10 highway which is the main transportation route from Tema, Accra, and into Burkina Faso in the north.
- The White Volta River is approximately 7 km away allowing for a sustainable raw water source for the Project. Testing by ALS laboratory in Perth indicates that this water is suitable for process operations including flotation.
- A stable 161 kV HV grid powerline is approximately 25 km due west of the Project. Testing at the Bolgatanga substation close to the Project has shown that the power supply is stable and authorities have assured Cardinal that sufficient capacity from the current grid exists for the Project. Ghana is currently in an HV oversupply situation currently, which further de-risks this aspect of the Project infrastructure.
- Cardinal has a 15 year renewable Mining Lease. The environmental permitting is well advanced with the Ghanaian EPA and other regulatory authorities.

25.2 Sampling and assay information

The quality control measures adopted for assaying of the Namdini resource drilling 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.

Cardinal's exploration and resource drilling activities have defined a substantial Measured and Indicated Mineral Resource. Inferred Mineral Resources, which represent a small proportion of model estimates within the RPEEE pit shell, are generally at depths below 250 m.

25.3 Mineral Resource estimation

Based on an external review, Golder concluded that the additional drilling has not materially changed the tonnes and grade in the Mineral Resource. The most significant change is to the reclassification of 7 Mt from Indicated to Measured Resource.

The 2019 resource estimate is supportable as an appropriate input to mining studies at the FS level and is a reasonable development from the previously reviewed 2017 and 2018 resource estimates.

25.4 Metallurgy

The Transition material at Namdini is only partially weathered (oxidized) and behaves in metallurgical test work similar to the Fresh material, with similar recoveries. The Transition material is to be processed with the Fresh material as part of the mill feed and thus these are combined in all the mine and mill schedules.

The metallurgical work carried out to date indicates that gold can be recovered from Namdini Transition and Fresh ore feed using conventional flotation, regrind and Carbon In Leach (CIL) cyanidation techniques, and from Oxide ore feed using conventional CIL cyanidation techniques. The work is considered sufficient to define a technically and economically viable gold mining project.

A substantial amount of drill core sample has been tested which adequately covers the entire deposit and rock types for the life of the operation. The sample representativity is considered by Cardinal to be appropriate.

No unproven technologies are planned for the Project. Many process plants of this size have been constructed on time and within budget.

25.5 Mine planning

The Namdini Project has been evaluated by Golder based upon a conventional hard rock mining equipment fleet. There are no indicated requirements for untested or novel mining techniques.

The Namdini Project is only moderately sensitive to both capital and mining costs, and relatively insensitive to the overall geotechnical slope angle of the pit as a result of the relatively low stripping ratio for a pit of this size.

The adoption of a tactical level scheduling tool for the FS of the Namdini Project is a key driver in identifying operational cost benefit areas and minimising mining spend to maximise project value.

The MIK adjustment assumed a moderately selective SMU of 5 m by 10 m by 2.5 m, which has been applied to the low-grade large-tonnage disseminated Namdini deposit. The chosen mining equipment is of medium scale (134 t trucks and 400 t face-shovels) and so less amenable to selective mining.

At an RC grade control pattern of 10 m by 15 m with 1.5 m sampling, the short-term model confirmed the MIK Recoverable Resource estimate based on the Resource drilling. The 10 m by 15 m grade control pattern is a suitable grid for defining the ore zones at the mining scale anticipated.

The Namdini deposit presents as a largely diffuse ore zone with limited high-grade selectivity options. The area of highest grade potential has been chosen to commence the Starter Pit, given the positive results at very low gold prices. The degree of mining selectivity is however markedly less than one would normally see in similar massive gold deposits.

Given the proposed blasting on 10 m benches, the planned selectivity may not be achieved. In Golder's view 10 m full-bench bulk mining would not be supported by the current SMU resource model.

25.6 Ore Reserve estimation

Based on the information presented in this FS, the Ore Reserve estimation process has converted approximately 80% of the combined Measured and Indicated Mineral Resource to Proved and Probable Ore Reserve.

The proposed mine plan is technically achievable. All technical proposals made for the operational phase involve the application of conventional technology that is widely used in the gold industry. Financial modelling completed as part of the FS 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.

25.7 Water balance

A site wide water balance was completed to ensure that adequate storage is available for both the mill process and other water requirements. Results indicate that the raw water demands for the mill process can be met for the proposed production rate.

25.8 Infrastructure

Approximately 260 Mt (95 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 northwest of the open pit.

The process plant and mine services area will be located on relatively flat terrain to the north-northwest of the open pit and directly to the northeast of the waste dump.

The TSF designs have been further optimized using locations defined in the PFS and incorporating more recent topography for the site area with the updated design parameters. The designs are based on downstream raise construction methods.

A hydrogeological fieldwork programme was undertaken comprising a hydro-census of surrounding properties to identify groundwater users. Characterisation of groundwater quality by sampling and laboratory analysis, groundwater monitoring and hydraulic testing were completed. Development of a conceptual model for assessment of pit inflows, and assessment of the potential impacts of mine dewatering on groundwater and surface water systems, have been completed in support of the mine design.

A hydrology programme including the development of a stormwater plan and overall site water balance has also been completed.

25.9 Environmental and other permitting requirements

As part of the EIA Study and to support the EIS, Cardinal identified approximately 29 key stakeholders for the Project, including Government regulatory agencies, Government institutions, traditional and political authorities, Project communities and leaders, and non-Government authorities.

Extensive social consultations and stakeholder engagement meetings have been held with key stakeholders.

All key stakeholders consulted expressed their support for the Project. Key concerns were raised and are being addressed related to security, Corporate Social Responsibility, payment of royalties, and protection of juveniles from engaging in artisanal mining activities. Stakeholders advised Cardinal to operate an ‘open door’ policy based on mutual respect.

The process plant is designed to comply with the International Cyanide Management Code (ICMI, 2016) for the manufacture, transport, and use of cyanide in the production of gold.

Construction, operational and post-closure monitoring is required to ensure regulatory and permitting compliance.

25.10 Overall outcome

Based on the positive FS outcome, the Cardinal Board has approved initiatives which will further de-risk the project. These include immediate advancement of the Namdini Gold Project to the FEED and Early Works Construction stages, plus advancement of the Relocation Action Plan subject to final project finance.

The positive results of this FS of the Namdini Gold Project have confirmed that progression of the Project to the development and construction stage is warranted.

26.0 RECOMMENDATIONS

26.1 Opportunities

During the course of the feasibility study, several potential opportunities were identified that will be evaluated in the next phase of work, along with the Namdini Gold Project FEED phase.

- *Analysis of current Gold Metal Prices* – The selected FS gold price is US$1,350/oz, the average over the last 12 months. At the completion of this study, gold was trading at US$1,500/oz, which improves the project economics substantially and may present other optimization opportunities.
- *Additional Mineral Resources* – Mineral Resources that are not part of the current Ore Reserve may represent potential lateral strike and depth extensions to the current LOM pit design at US$1,235/oz. Additional drilling beyond the current pit design may extend the life of the current proposed open pit mine or provide an opportunity to investigate an underground mine plan.
- *Regional Land Package Drilling* – Cardinal has the potential to define satellite pits with close proximity to Namdini. Recent results from drilling completed within the Ndongo Project, located approximately 24 km north of Namdini, are within hauling distance. Drill testing new zones of high-grade gold mineralization to define satellite pits will recommence shortly.
- *Flotation and Regrind* – Cardinal is considering an option of eliminating siliceous gangue associated with the rougher concentrate, which is likely causing significantly high specific grinding energy requirements in the regrind circuit. This could reduce operating costs.
- *CIL* – Cardinal will evaluate the benefits of reducing the 48 hour leach down to 36 hours.
- *Detoxication circuit upgrade using Aachen™* – The cost-effective Maelgwyn cyanide detoxification process has been developed to comply to the strict effluent limits of 50 ppm as prescribed by The International Cyanide Management Code. The Aachen process in combination with the use of oxygen, carbon, selected reagents and catalysts, efficiently and economically destroy all forms of cyanide, including WAD cyanide, to levels well below 50 ppm. This testwork could reduce operating costs.
- *TSF and Waste Dump Location* – Cardinal has secured an additional lease area to the west of Namdini which has allowed for TSF and waste dump optimization. This evaluation will likely result in improved sustaining and mining operating costs.
- *Mine Pit and Enterprise Optimization* – Cardinal are continuing optimization work with Whittle Consulting to further optimize mine plans and process configurations. This work is planned to be complete in the early 2020 and could lead to improved overall project economics.
- *Pit slope Angles* – There may be opportunities to increase the inter-ramp slope angels, especially in the hanging wall of the pit. The result of steeper slopes would be reduced waste (overburden) stripping and better potential economics.
- *Recruitment Strategy* – Cardinal will evaluate the recruitment strategy to determine the optimal time to bring new staff on for safety orientation and training to ensure successful startup.

26.2 Risks

- *Sterilization of Process Plant and Infrastructure area* – Some areas of the current site lay-out have not been sterilized to prove there is no gold mineralization at depth. Cardinal's sterilization process is continuing and will be complete shortly.

- *Owner Team Development* – Cardinal is dependent on consultants providing strong capabilities and teams to assist with the development of the project. A plan is in place to onboard key owner positions to enhance Cardinal capacity and capability.
- *Relocation Action Plan* – Access to site (and therefore the project execution schedule) is dependent on timeous and successful completion of the RAP. The RAP and social study is well advanced and on schedule.
- *Long Lead Equipment Supply Market* – The current lead times for large equipment being procured by Cardinal is favourable, however, demand may increase should competing projects come on line. Cardinal has identified long lead items and has commenced negotiations with suppliers for manufacturing and supply slots to mitigate this.
- *Permitting* – Cardinal has not yet been issued an EPA permit but considers that the process is progressing well with the permitting requirements and authorities.
- *Development Agreement* – The Development Agreement with the Ghanaian Government defining taxes and duties has not yet been formalized. Where appropriate the current position is noted in this Report (Table 10 and Table 160).

26.3 Further work

Cardinal has commenced with its FEED programme utilising available funds. A brief summary of the programme is as follows:

- Further evaluation of the opportunities (Section 26.1) and risks (Section 26.2).
- Further development of Cardinal policies and procedures for project execution and mine operation.
- Owner's team onboarding of key positions for effective project execution.
- Update of the geological model with further structural and lithological data.
- Further evaluation of owner mining *versus* contract mining *versus* lease mining scenarios
- Modification of the mining cost model for application of variable haulage optimization
- Compilation of contract tender documentation where required
- Evaluation of autonomous mining methods
- Update of the site design and mine schedule based on the outcomes of the TSF, waste dump and enterprise optimization studies
- Review of current Phase mining and cut-off grade strategies
- Blasting and fragmentation studies to finalize the Powder Factor assumptions for project execution
- Finalization of the early works and key contracting strategies (priority items have commenced for some Front End Engineering Design including early works design and tender package preparation for long lead items)
- Implementation of the already developed Project Execution Plan (PEP)
- Finalization of the RAP
- TSF EIS submission and EPA permitting.

The budget for the initial FEED and optimization programme is presented in Table 184.

Table 184: Namdini Gold Project further work budget

Activity	Budget cost (US$ k)
Metallurgy testwork programme	143
Mining programme	104
Initial FEED activities	829
Finalisation of EPA and other permitting	204
Sterilisation drilling	918
Contingency (10%)	220
TOTAL	**2,418**

Note: Costs are budgetary and will be finalised as the project progresses

Based on the positive FS outcome, the Cardinal Board has approved initiatives which will further de-risk the Project. These include immediate advancement of the Namdini Gold Project to the FEED and Early Works Construction stages, plus advancement of the Relocation Action Plan subject to final project finance.

The preliminary Project development schedule is shown in Table 185 and a Project Gantt Chart is provided as Figure 153. This is subject to available funding based on the positive outcome of this FS and to favourable timelines for permitting, contracting and procurement.

Table 185: Project schedule

Milestone	Target timeline
Commence Front End Engineering and Design (FEED)	Q4 2019
Advance Relocation Action Plan to Completion	H1 2020
Target production commencement	H2 2022

Note: The Company's proposed project development schedule assumes that the Company is fully funded to proceed throughout and beyond the FEED process, into early works and full construction. The actual development schedule will depend upon the manner and timing of the Company's financing plans. Dates are therefore indicative only.

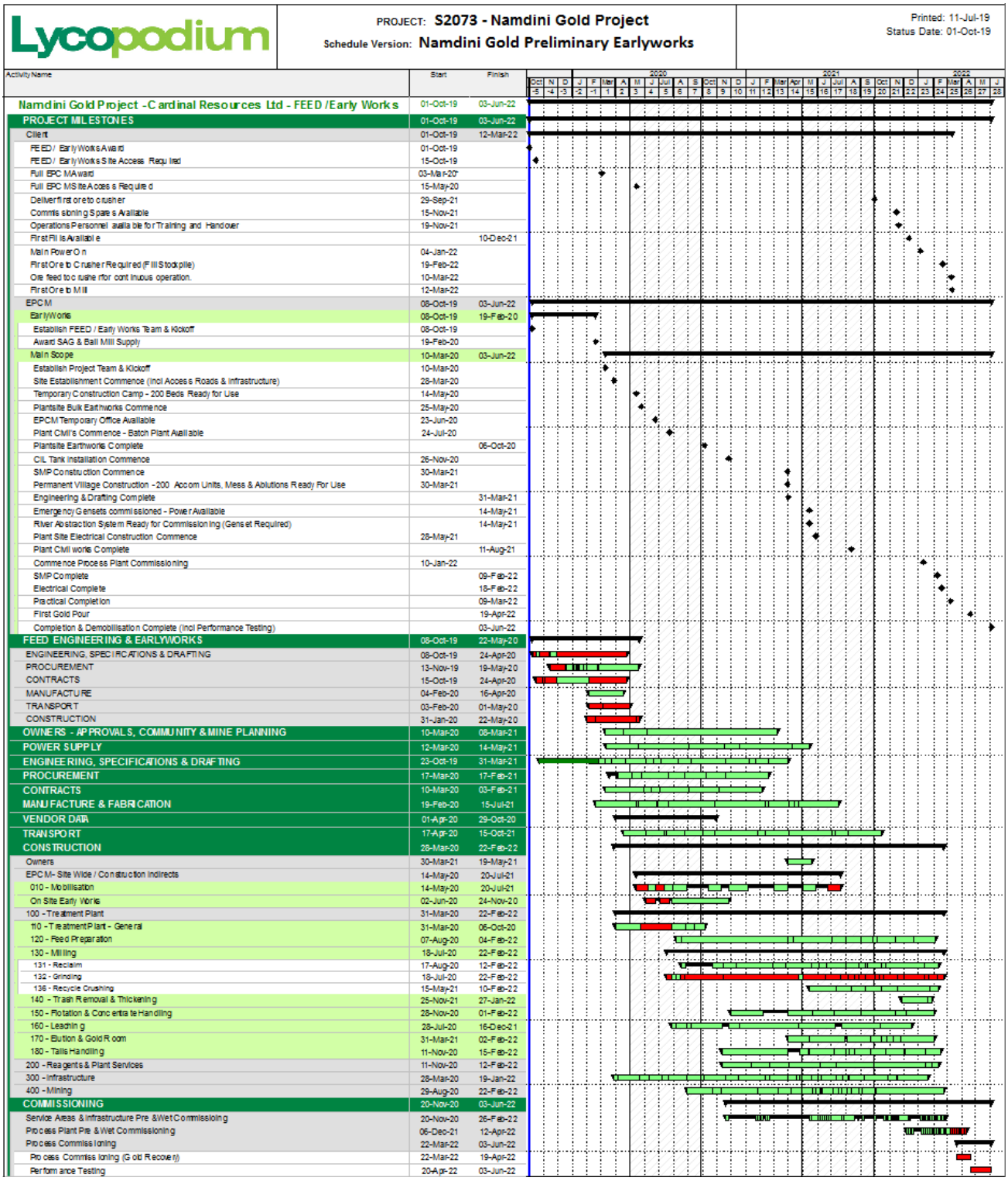


Figure 153: Project schedule Gantt timeline (source: Lycopodium 2019d)

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27.2 Abbreviations, acronyms, definitions

AARL	gold elution circuit (developed by Anglo American Research Laboratory)
AC	acid consuming
AEP	annual exceedance probability (for rainfall events)
AIF	Annual Information Form (company report to the TSX)
ATS	Alastri Tactical Scheduler
AMSL	above mean sea level
ANC	acid neutralising capacity
ARI	average recurrence interval (for rainfall events)
ASX	ASX Group Ltd (the Australian Securities Exchange)
Authors	Cardinal, Lycopodium, Golder, and other consultants who have contributed to this Report
BBWi	Bond ball mill work index
BFA	bench face angle (for open pit design)
BDO	BDO Advisory Pty Ltd
BH	bench height (for open pit design)
BOO	build own operate (by contractor)
BoS	basis of schedule
BQR	bulk quantity request
BV	bed-volume (for tanks)
BW	berm width (for open pit design)
C.Eng	Chartered Mining Engineer
Capex	Capital expenditure
Cardinal	Cardinal Resources Limited
Cardinal Namdini	Cardinal Namdini Mining Limited a wholly owned subsidiary of Cardinal
CCWA	Chemistry Centre of WA
CIL	carbon-in-leach
CN_{WAD}	weak acid dissociable cyanide (also CN_{FREE}, CN_{TOTAL}, etc)
COG	cut-off grade (in g/t Au)

CSL	compacted soil liner (for TSF)
CSV	comma separated value computer format
CV	coefficient of variation
CWi	Crushing Work index
Davg	discounted average NPV (used in Whittle 4X)
DGPS	differential GPS
E&I	electrical and instrumentation
EGL	effective grinding length (mill size specification)
EIA	Environmental (and Social) Impact Assessment
EIS	Environmental Impact Statement (a comprehensive public report)
EPA	Ghana Environmental Protection Agency
EPCM	Engineering, Procurement, Construction & Management (construction management)
Eur.Ing.FEANI	Registered European Mining Engineer
FEL	front-end loader
fSAG	SAG mill power efficiency
FS	The Namdini Gold Project Feasibility Study (the basis of this Report)
G&A	General and Administration (cost category)
Ga	Billion years (giga annum)
GCST	Gold Coast Selection Trust
Geosystems	Geosystems Consulting Limited
Golder Africa	Golder Associates Africa (Pty) Ltd
Golder Ghana	Golder Associates Ghana Limited
Golder	Golder Associates Pty Ltd
GPS	global positioning system
GPCP	Global Precipitation Climatology Project
HDPE	High-density Polyethylene
HIGmill	High Intensity Grinding mill
IBC	intermediate bulk container
IFC	World Bank Group International Finance Corporation
IGR	International Gold Resources
IMO	Independent Metallurgical Operations Pty Ltd
IRA	inter-ramp angle (for open pit design)

IRR	internal rate of return
Knight Piésold	Knight Piésold Consulting Pty Ltd
LOM	Life of Mine
LV	low voltage
Lycopodium	Lycopodium Minerals Pty Ltd (also “**Lyco**” or “**LMPL**”)
M+I	Measured plus Indicated Mineral Resource categories
MCC	motor control centre
MIK	Multiple Indicator Kriging
MMSA	Maelgwyn Mineral Services Africa Pty Ltd
MKM	MKM Social Limited
MPL	maximum permissible levels
MPR	MPR Geological Consultants Pty Ltd
MSA	mine services area
NAF	non-acid forming
NAF-ML	metal-leaching NAF
NAPP	net acid producing potential
Namdini	Namdini Gold Project (or the Project)
NAPP	net acid producing potential
NEMAS	Nemas Consult Ltd
NI 43-101	National Instrument 43-101
NPV_5	net present value (at a discount rate of 5%)
NSG	non-sulphide gangue
OK	Ordinary Kriging
OMC	Orway Mineral Consultants Pty Ltd (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
PAX	Potassium Amyl Xanthate
PB	pushback, stage or Phase of development of the open pit mine design
PDC	process design criteria

PEA	Preliminary Economic Assessment (first stage of study under NI 43-101), see Golder (2018a)
PEP	Project Execution Plan
PFS	Pre-feasibility Study (second stage of study under NI 43-101), see Golder (2018b)
PLC	programmable logic controller (process plant control system)
PLTS	prime, load, tie and shoot (basis for explosives supply contracts)
PM_{10}	particulate matter less than 10 microns
PMS	Project Master Schedule
PPE	personal protective equipment
PRI	preg-robbing index (for evaluation of metallurgical processing of the ore)
QP(s)	Qualified Person(s) under NI 43-101
Qtr	quarter (quarterly period)
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)
RWi	Bond rod mill work index
SABC	open circuit SAG mill with recycle pebble crushing followed by closed circuit ball mill/hydro-cyclones
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)
SCS	sediment control structures
SGI	Sebbag Group International Pty Ltd
Shaanxi	Shaanxi Mining Company Limited (nearby mine)
SMBS	sodium metabisulphite $Na_2S_2O_5$
SMC	SAG mill comminution (test)

SMP	structural, mechanical and piping
SPI	SAG mill power index
S-MC	sulphide master composite (metallurgical sample for LOM flotation optimization)
SP-MC	Starter Pit master composite (metallurgical sample)
TML	Townend Mineralogy Laboratory, Perth
TSF	Tailings Storage Facility
TSX	Toronto Stock Exchange
VAT	Value Added Tax
VEA	Value Engineering Assessment
WC	Whittle Consulting Pty Ltd
WBS	work breakdown schedule
WHO	World Health Organization
WRD	waste rock dump
WSP	Water Storage Pond

27.3 Units

Many of these units are combined in measurement, e.g. kWh is thousand-Watt hours

µ	millionth (as in µm for micron or micro-metre)
bcm	banked cubic metre (volume of *in situ* material)
cP	viscosity
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
K	Fleming kinetic constant (gold adsorption)
l or L	litre
l/sec	litres a second (flow rate)
lcm	loose cubic metre (volume of disturbed material)
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)
mg	milligram (thousandth of g)
mm	millimetre
Mtpa	million tonnes a year
pH	measure of acidity
S/cm	Siemens per centimetre (for electrical conductivity)
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*)

27.4 Trademarks

Aachen™ shear reactor technology by Maelgwyn Laboratories South Africa for ore processing

Alastri™ Tactical Scheduler software for mine planning

GS3M™ geostatistical estimation software from FSSI Consultants (Australia) Pty Ltd

HIGmill™ High Intensity Grinding mill for ore processing by Outotec

Logchief™ geological logging software from Maxwell Geoservices (Perth)

Minemax™ Scheduler software for mine planning

Rocscience™ 2D and 3D geotechnical software for civil engineering and mining

Slide™ pit slope modelling software for mine design

Vulcan™ software by Maptek for geological modelling and mine planning

Whittle 4X™ software for mine planning.

28.0 QUALIFIED PERSON CERTIFICATES

The Qualified Persons named in this NI 43-101 Technical Report are:

David Gordon (Lycopodium)

Daryl Evans (IMO)

Nicolas Johnson (MPR)

Glenn Turnbull (Golder)

All production targets for the Namdini Project referred to in this Report are underpinned by Mineral Resource and Ore Reserve estimates which were prepared by Competent Persons in accordance with the requirements of the JORC Code and who are also Qualified Persons in accordance with NI 43-101 respectively.

The information in this press release that relates to process plant, infrastructure, and capital costs is based on information compiled and reviewed by **Mr David Gordon**, who is a Fellow of the AusIMM and a full-time employee of Lycopodium Minerals Ltd. Mr Gordon has sufficient experience which is relevant to the activity which he has undertaken to be a Competent Person as defined in the JORC Code (2012) and a Qualified Person for the purposes of NI 43-101. He has no economic, financial or pecuniary interest in the Company and consents to the inclusion in this Report of the matters based on his information in the form and context in which it appears. Mr Gordon has not personally inspected the Namdini Gold Project site, as noted in Section 3.4 of this Report. He has relied on the information about site setting and infrastructure provided by his colleagues Michael Cicanese of Lycopodium and Simon Smith of Knight Piésold who have made site inspections and have been involved in developing the relevant parts of this Report.

Scientific and technical information contained in this press release that pertains to metallurgical results and interpretations has been reviewed by **Mr Daryl Evans**, who is a Fellow of the AusIMM and a full-time employee of Independent Metallurgical Operations Pty Ltd. Mr Evans has sufficient experience which is relevant to the activity which he has undertaken to be a Competent Person as defined in the JORC Code (2012) and a Qualified Person for the purposes of NI 43-101. He has no economic, financial or pecuniary interest in the Company and consents to the inclusion in this Report of the matters based on his information in the form and context in which it appears. Mr Evans has not personally inspected the Namdini Gold Project site, as noted in Section 3.4 of this Report, because that was not required for him to do the metallurgical testwork studies. He was involved in the selection of the samples to be tested and they were collected under his instruction by Cardinal geologists. He has relied on the information about site setting, drilling, sampling, sample preparation and QAQC provided by Mr Nicolas Johnson as documented in this Report.

The information in this press release that relates to Mineral Resources is based on information compiled and reviewed by **Mr Nicolas Johnson**, who is a Member of the Australian Institute of Geoscientists and a full-time employee of MPR Geological Consultants Pty Ltd. Mr Johnson has sufficient experience which is relevant to the style of mineralization and type of deposit under consideration and to the activity which he has undertaken to be a Competent Person as defined in the JORC Code (2012) and a Qualified Person for the purposes of NI 43-101. He has no economic, financial or pecuniary interest in the Company and consents to the inclusion in this Report of the matters based on his information in the form and context in which it appears. Mr Johnson has personally inspected the Namdini Gold Project site, on the dates noted in Section 3.4 of this Report.

The information in this press release that relates to Ore Reserves and mining studies is based on information compiled and reviewed by **Mr Glenn Turnbull**, who is a Chartered Engineer, a Fellow of the Institute of Materials, Minerals and Mining, a Member of the Australasian Institute of Mining and Metallurgy, and an employee of Golder Associates Pty Ltd. Mr Turnbull has sufficient experience which is relevant to the style of mineralization and type of deposit under consideration and to the activity which he has undertaken to be a

Lycopodium

CERTIFICATE OF QUALIFIED PERSON

This certificate applies to the technical report prepared for Cardinal Resources. ("Cardinal") entitled: "National Instrument 43-101 Namdini Gold Project Feasibility Study" signed on 28 October 2019 (the "Technical Report") and effective 28 October 2019.

1. My name is David Gordon and I am the Group Manager Process of the firm of Lycopodium Minerals Pty Ltd of level 5, 1 Adelaide Terrace East Perth, WA, 6004, Australia.
2. This Certificate relates to the Report.
3. I am a practicing process engineer and am a Fellow of the Australian Institute of Mining and Metallurgy. My membership number is 108413.
4. I am a graduate of the Western Australian Institute of Technology and hold a Bachelor of Applied Science in Engineering Metallurgy (1983).
5. I have worked as a metallurgist, operations manager, laboratory manager and process engineer/manager for over 30 years since my graduation. My relevant experience for the purpose of this Technical Report is:

   Managed and interpreted results from numerous leaching and mineral processing testwork programmes on gold and silver ores.

   Involved in the process design of treatment plants for over 15 years.
6. I am a "qualified person" as that term is defined in National Instrument 43-101 (Standards of Disclosure for Mineral Projects) (the "Instrument").
7. I have not visited the Namdini Gold Project area.
8. I have not been involved in any previous Technical Report on the Namdini Gold Project.
9. I am responsible for the preparation of only the following sections of the technical report, Sections 17, 18, 21 and 24, in the report "National Instrument 43-101 Namdini Gold Project Feasibility Study" and dated 28 October 2019.
10. I am not aware of any material fact or material change with respect to the subject matter of the Report which is not reflected in the Report, the omission of which would make the Report misleading.
11. I am independent of the Namdini Gold Project and the Issuer (Cardinal Resources Limited) pursuant to section 1.5 of the Instrument.
12. I have read the National Instrument and Form 43-lOlFl (the "Form") and the parts of the Report for which I am responsible have been prepared in compliance with the Instrument and the Form.
13. I do not have nor do I expect to receive a direct or indirect interest in the Namdini Gold Project of Cardinal Resources Limited and I do not beneficially own, directly or indirectly, any securities of Cardinal Resources Limited or any associate or affiliate of such company.
14. At the effective date of the Report, to the best of my knowledge, information and belief, the parts of the Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Report not misleading.

Dated at Perth, Western Australia, on date 28 October 2019.

David Gordon (FAusIMM - 108413)
Group Manager Process
Lycopodium Minerals Pty Ltd

Competent Person as defined in the JORC Code (2012) and a Qualified Person for the purposes of NI 43-101. He has no economic, financial or pecuniary interest in the Company and consents to the inclusion in this Report of the matters based on his information in the form and context in which it appears. Mr Turnbull has personally inspected the Namdini Gold Project site, on the dates noted in Section 3.4 of this Report.

Certificates for these Qualified Persons are provided in the following pages.

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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 Feasibility Study" signed on 28 October 2019 (the "Technical Report") and effective 28 October 2019.

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. In addition to direct operational experience, I have 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 the preparation of Section 13.0 of the technical report titled 'National Instrument 43-101 Namdini Gold Project Feasibility Study', dated 28 October 2019 (the "Technical Report") relating to Namdini Gold deposit, Ghana.
6. I have not visited the Namdini Gold Project, Ghana site.
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 all metallurgical aspects and methods and in preparation of the Preliminary Feasibility Study dated 25 October 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 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 25 October 2019



Daryl Evans FAusIMM (224540)

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 Feasibility Study" signed on 27th November 2019 (the "Technical Report") and effective 28th October, 2019.

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 28 October 2019.
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 Feasibility Study dated 25 October 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 27 November 2019



Nicolas James Johnson, MAIG

GOLDER

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 28 October 2019 (the "Technical Report") and effective 28 October 2019.

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 35 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 seven 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, 17, 18, 21 and 24, in the report "National Instrument 43-101 Namdini Gold Project Feasibility Study" and dated 28 October 2019.

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 Preliminary Feasibility Study dated 25 October 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 28 October 2019 at Perth, Australia.



Glenn Turnbull

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