Deccan Exploration Services Private Limited

Ganajur Gold Project Feasibility Study

Project Number AU9734

June 2017

ALL CHAPTERS

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Table of Contents

Figures

Tables

1 EXECUTIVE SUMMARY

1.1 Introduction

Deccan Exploration Services Private Limited (DESPL) is a wholly-owned subsidiary of Deccan Gold Mines Limited (DGML). DGML is a public limited company listed on the Bombay Stock Exchange (BSE Scrip Code: 512068). DESPL is involved in gold exploration activities in the state of Karnataka since 2003 and is working towards its long-standing vision of developing a producing gold mine in India.

DESPL proposes to establish 0.30 million tonne (Mt) gold project near Ganajur Village, Haveri Taluk and District in the state of Karnataka. The proposed gold project comprises of gold ore production from the mine and processing of gold ore in the processing plant to produce the gold. The gold processing plant will be located at a distance of 1.5 km south of the Ganajur Gold Ore Mine. The Ganajur Main Gold deposit was a discovery of DESPL as a result of strategic exploration carried out under reconnaissance permit and prospecting licence stages. DESPL's mining lease application over an area of 0.29 km^2 covering the Ganajur Main Gold Deposit. The Ganajur mining lease application in Ganajur village, Haveri Taluk and District in Karnataka has been approved by the Ministry of Mines, Government of India vide letter no. 4/113/2010-MIV dated 24 July 2015. The approval is per Section 10(A)(2)(B) of the New MMDR Act 2015. Prior to this, the mining lease application for the Ganajur Main Project was recommended by the Government of Karnataka. DESPL is awaiting the final grant order/Letter of Intent from the State Government of Karnataka.

1.1.1 Location

The Ganajur Gold Mining Project area is situated near the Ganajur village, 14°49'54.08" – 14°50'16.84" N latitude; 75°24'16.57" – 75°24'48.39" E longitude, forms a part of Survey of India topographic sheet no. 48 N/5 and falls in the jurisdiction of the Haveri Taluk and District in the State of Karnataka. The corner points of Ganajur mining lease area with UTM coordinates shown in Table 1.1.

Table 1.1 Coordinates of Ganajur mining lease block

Corner	UTM zone	Easting (M)	Northing (M)
A	43	543676.6	1640318.2
B	43	544476.4	1639908.1
C	43	544310.3	1639619.9
D	43	543525.4	1640027.6

**UTM Projection Everest Datum, Zone 43N*

Figure 1.1 and Figure 1.2 show the location map. Haveri Town, located on National Highway No. 4, is 335 km by road north of Bengaluru and 100 km south of Dharwar. The centre of the Ganajur mining lease block is located 4.53 km northeast of Haveri Town, and 0.76 km southeast of Ganajur Village. The Ganajur Project is well connected by an all-weather metalled road from Haveri and Ganajur.

Figure 1.1 Location map of the Ganajur Gold Project

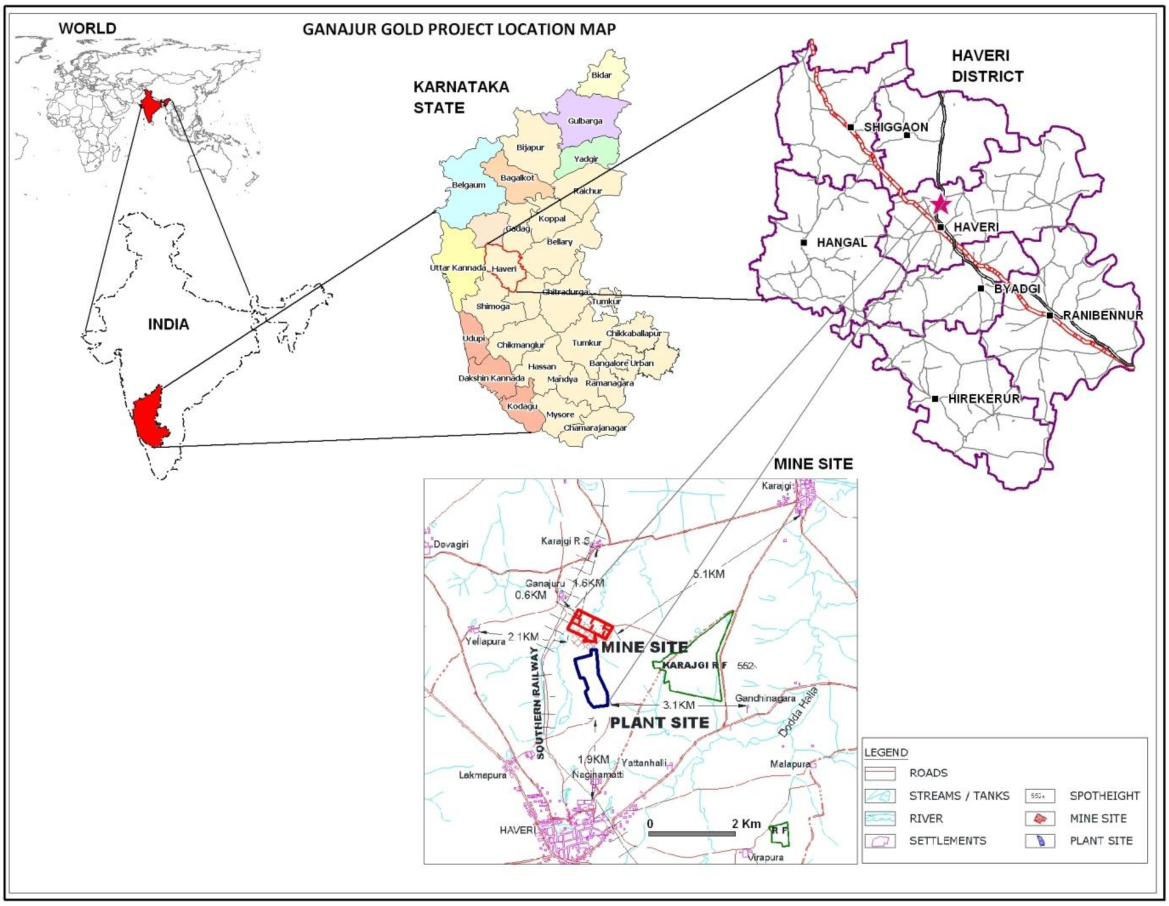
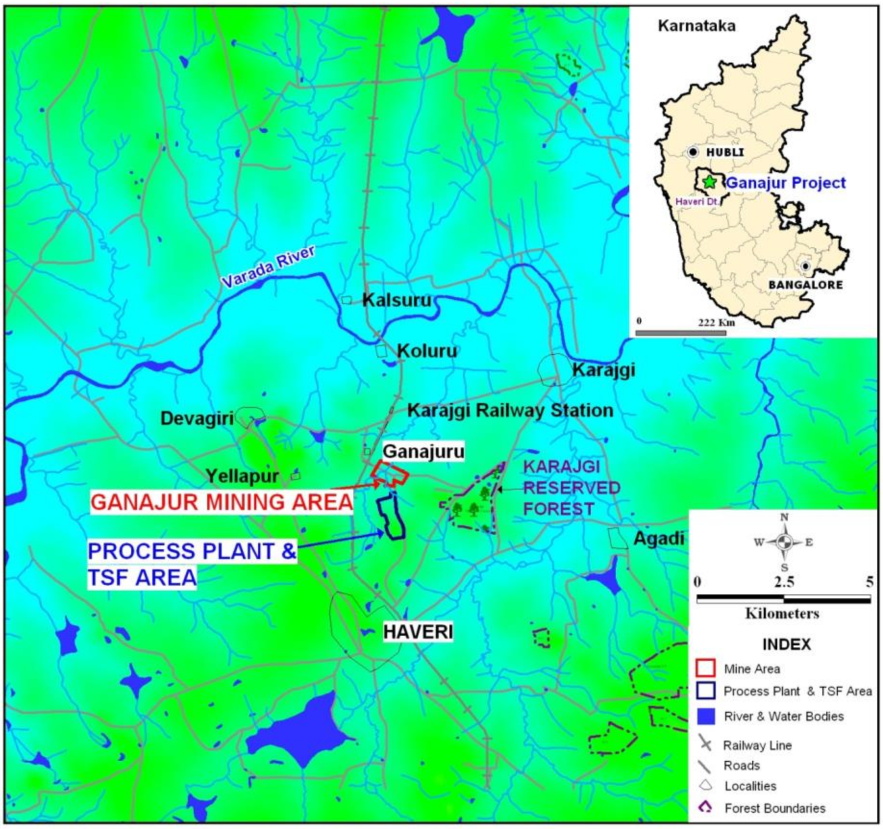


Figure 1.2 Location map of the Ganajur Gold Project



1.1.2 Climate

The Project area falls within a semi-arid sub-tropical region with moderate temperatures and rainfall. The normal annual rainfall for the Haveri District is recorded as 792 mm.

1.1.3 Physiography and infrastructure

The Project area and its surrounding area are generally flat to gently undulating terrain and a few northwest-southeast trending small ridges. The elevation of the ground surface varies from 525 metres above mean sea level (mamsl) to about 610 mamsl. Most of the Ganajur Main Gold Deposit is flat except for Ganajur Hill which is 551 mamsl (Figure 1.3).

Figure 1.3 Ganajur Main Gold Deposit

1.1.4 Drainage

The Ganajur Project and its surrounding area is drained by the Varada River. The Varada River rises in the Varada Moola in Sagara Taluk, in the hill ranges of the Western Ghats in Shimoga district of Karnataka.

The main crops grown in the district are jowar, maize, cotton, chilly, paddy, ragi, pulses, groundnut, horse gram, sugarcane and sunflower.

1.1.5 Transportation

The Project is well connected by road and rail and has well-established infrastructure in place. Ganajur Gold Project is located 4.53 km northeast of Haveri Town (Figure 1.1 and Figure 1.2) in Haveri District of Karnataka state on National Highway No. 4, connecting Bengaluru and Pune and 0.76 km southeast of Ganajur Village. Haveri is around 335 km from Bengaluru and 100 km south of Dharwar city. The Ganajur Gold Project is connected to Haveri and nearby villages by an all-weather metalled road.

1.1.6 Power

The HT 110 KVA powerline is located just 300 m south of the proposed processing plant (Figure 1.2). The nearest power substations are located at Haveri (3.37 km), Agadi (6.42 km), Gandhipura (3.6 km) and Basavanakatti (7.0 km). DESPL's application for obtaining 5 MW power from the 110 KV line has been sanctioned by Karnataka Power Transmission Corporation Limited (KPTCL), Bengaluru. KPTCL has approved for tapping the required power from Basavanakatti substation at a distance of around 7.0 km.

1.1.7 Water

The Varada River flows at a distance of 6.5 km north of the gold ore process plant area. It is proposed to pump water from the Varada River for the water requirement of the Project. Karnataka State High Level Clearance Committee (KSHLCC) has approved for drawing 3,000 kilolitres per day (kl/d) of water from Varada River for the project from the Kolur-Kalasur barrage.

1.1.8 Land

There are various facilities required for the project and a total of 255 acres of land will be acquired. The State Government, through a Government Order (GO) has also approved acquisition of 200 acres of land from farm-holders for the mines and processing plant. The GO has also facilitated land acquisition process through Karnataka Industrial Areas Development Board (KIADB), for which DESPL has submitted application to the KIADB in March 2013. DESPL will also be submitting application for an additional 55 acres of land after obtaining approval from Karnataka Udyog Mitra and KSHLCC. Most of the land is private agricultural land, with the remainder being government land. DESPL has obtained consent of more than 85% of the landowners, which complies with the land acquisition procedure. DESPL's long term lease agreement with landowners of the proposed gold mine will be an added advantage in the land acquisition process. Recently, KIADB processed the application for 200 acres of land and then issued a demand letter asking DESPL to remit 40% of the land cost. After payment of this deposit, KIADB will issue primary notification under Sections 3(1), 1(3) and 28(1) of the KIADB Act.

1.1.9 History

The area around the Ganajur Gold Mining Project is known for ancient artisanal gold mining activity. Chinmulgund, located southeast of Haveri, shows evidence ancient mining activities (i.e. old workings, ancient shafts, adits, waste dumps and pounding marks). However, Ganajur Main Deposit is a discovery of DESPL.

1.1.10 Tenement details

DESPL, under the Hanagal reconnaissance permit approved by the Ministry of Mines, Government of India in 2002, and granted by the State Government of Karnataka for 1,542 km^2 covering the Ganajur Main prospect, carried out initial exploration. This reconnaissance campaign included surface geological mapping, regional geochemical study of stream sediments and rock chips, channel sampling, limited reverse circulation (RC) and down-the-hole hammer (DTH) drilling. This exploration effort helped to identify and discover several gold-bearing prospects designated as Ganajur Main, South, South East, Central, Karajgi Main, Karajgi East and Hut.

DESPL recognised the mining potential of the Ganajur Main Gold Deposit and submitted a mining lease application over an area of 29.14 hectares (ha) covering the Ganajur Main Gold Deposit. The Ganajur mining lease application in Ganajur Village, Haveri Taluk and District in Karnataka was approved by the Ministry of Mines, Government of India vide letter no. 4/113/2010-MIV dated 24 July 2015.The approval is per Section 10(A)(2)(B) of the New MMDR Act 2015 and Section 5(1). Prior to this, the mining lease application for the Ganajur Main Gold Deposit was recommended by the Government of Karnataka.

DESPL signed a memorandum of understanding with the Government of Karnataka during the Global Investors Meet at Bengaluru on 4 June 2010 for commencement of a 2,000 tonnes per day (t/d) gold mine and setting up of a 2,000 t/d processing plant at Ganajur village. Prior to this, the KSHLCC approved the project on 24 May 2011.

1.1.11 Legal aspects

The Ganajur Gold Project is subject to the regulations of the Mines and Minerals (Regulation and Development) Act 1957 as amended in March 2015, the Mineral Concession Rules 1960, and the Mineral Conservation and Development Rules 1988 of India (as amended in March 2017).

1.2 Geology and Mineral Resource estimate

Snowden Mining Industry Consultants (Snowden) carried out the Mineral Resource estimate for the Ganajur Main Gold Deposit project during August 2016 on behalf of DESPL.

The August 2016 Ganajur Main Gold Deposit Mineral Resource estimate was classified and reported in accordance with the 2012 JORC Code.

The Mineral Resource has been classified as a combination of Measured, Indicated and Inferred Resources using the following criteria:

1) Measured Resources – Restricted to within the mineralised wireframe where drilling is approximately 20 mN x 20 mE or better, geological and grade continuity is confirmed and the mineralised body is at its thickest, typically 20 m to 50 m thick.
2) Indicated Resource – Restricted to within the mineralised wireframe where drilling is approximately 20 mN x 20 mE or better, geological and grade continuity is assumed. This has been restricted to areas where the mineralised body is typically less than 20 m thick.
3) Inferred Resource – Mineralisation with poor geological and grade continuity or which is defined by drilling on a grid greater than 20 mE x 20 mN.

Reporting of the Mineral Resource has been restricted to within the lease boundary provided by Deccan. Any mineralisation that has been interpreted as being outside of the lease is "unclassified" and excluded from the Mineral Resource. The classification is based on the confidence in the gold grade estimate. Given the lesser amount of data for the other variables, particularly sulphide sulphur, these should be considered of a lower confidence. The Measured classification assumes that mining will be conducted at around a 0.8 g/t Au cut-off and hence will mine the majority of the mineralisation, non-selectively.

The total Measured and Indicated Mineral Resource for the Ganajur Main Gold Deposit, reported above a 0.8 g/t Au cut-off grade, is estimated to be 2,700 kt grading at 3.40 g/t Au as detailed below in Table 1.2. The cut-off is based on preliminary results from the Feasibility Study (FS).

Table 1.2 Ganajur Main Gold Deposit Mineral Resource as at August 2016, reported above 0.8 g/t Au cut-off

Classification	Deposit	Tonnes (kt)	Au (g/t)
Measured	Oxide	580	2.82
	Sulphide	1,690	3.96
	Total Measured	**2,300**	**3.67**
Indicated	Oxide	130	1.85
	Sulphide	330	2.13
	Total Indicated	**450**	**2.05**
Measured + Indicated	**Total Measured and Indicated**	**2,700**	**3.40**
Inferred	Oxide	110	2.30
	Sulphide	110	2.29
	Total Inferred	**210**	**2.30**

Note: Small discrepancies may occur due to rounding.

The gold mineralisation in the Ganajur Main Gold Deposit is associated with a deformed iron formation hosted in a polydeformed greywacke sequence. The gold mineralisation is characterised by strong sulphide mineralisation, silica breccia and minor quartz veining developed within a sulphidic chert unit.

The gold mineralisation is epigenetic in nature but strata-bound because it is confined to the cherty iron formation. The main gold zones form a moderately to steeply dipping tabular body trending northwest to north-northwest and dipping northeast. Deccan carried out the geological interpretation using the geological logging of the chert domain and the Au assays at a nominal 0.3 g/t cut-off to define the mineralised envelopes. The mineralised domain is typically restricted to the chert with 1 m to 2 m of halo mineralisation in places and occasional small areas of unmineralised chert.

A block model was constructed using a parent block size of 10 mE x 10 mN x 5 mRL based on half the nominal drillhole spacing along with an assessment of the grade continuity. The search ellipse orientation and radius was based on the results of the grade continuity analysis, with the same search neighbourhood parameters used for all elements to maintain the metal balance and correlations between elements.

Estimation of gold, arsenic, copper, lead, sulphide sulphur (SS) and zinc was completed using ordinary block kriging with hard domain boundaries. Top cuts were not applied to the gold grades due to the low CV of 1.05 and 1.08 for the oxide and sulphide mineralised domains respectively, and lack of outliers. Top cuts were applied to SS in the oxide mineralised domain and arsenic in the sulphide mineralised domain. Grade estimation was completed using Datamine Studio 3 (Datamine) software.

Grade estimates were validated against the input drillhole composites (globally and using grade trend plots) and show a good comparison.

Extensive bulk density measurements were taken from diamond core with 264 taken in the oxide mineralised domain and 749 taken in the sulphide mineralised domain. Measurements were taken using the water immersion method. Bulk density was estimated into the model blocks by ordinary kriging in the oxide and sulphide mineralised domains. Where estimates were not possible, a default of 2.75 t/m^3 and 3.08 t/m^3 was applied to the oxide and sulphide mineralised domains, respectively, based on the average of the density measurements.

1.3 Metallurgical testing and recovery

The FS metallurgical testwork program focused on developing a gold recovery route on the predominant sulphide resource via a process flowsheet that involved flotation followed by the ultrafine grinding (UFG) and carbon in leach (CIL) on the sulphide concentrates. This flowsheet was assessed as the most likely process route that would provide the maximum net present value (NPV) for the Ganajur Project.

The sulphide variability samples that were selected for testwork are representative of the major tonnage area (“Belly” portion) of the Ganajur Main sulphide resource. These samples contained organic carbon (0.3%) which proved to be mildly preg-robbing and this effect could be negated by CIL. The x-ray diffraction (XRD) analyses confirmed that the major sulphide present is pyrite (9% to 12%) with minor arsenopyrite (1%), with silica levels at 50% and the carbonate minerals, siderite (20% to 25%), dolomite/ankerite (7% to 12%) representing the major gangue components.

The oxide variability samples and bulk oxide composite (BOC) are similar in mineralogy to the sulphide samples except that the sulphide content has been oxidised to goethite and hematite.

The comminution characteristics for the sulphide and oxide samples are similar and are considered to have a moderate to high competency with a moderate resistance to wear (abrasion).

The flotation response on the sulphide ore samples is positive with 95% gold and 97% sulphide sulphur recovery achieved into a low weight rougher concentrate stream (10% to 20%). These recoveries were attained via the optimum liberation grind size at P_{80}75 microns and by targeting a 20% to 22% sulphide sulphur concentrate grade. The flotation tailings stream at approximately 0.27 g/t Au and 0.13% SS reports directly to the tailings storage facility (TSF).

The flotation response on the oxide resource samples and oxide blends with the sulphide ore is negative compared to the gold recoveries achieved on the sulphide resource samples. Due to this poor response, conventional CIL was selected as the optimum recovery route for the oxide resource.

The optimum UFG grind liberation size and CIL conditions to attain high gold recovery from the sulphide concentrates were identified at:

- a P_{80} of 10 microns, which requires an energy consumption of 90 kWh/t
- a leach retention time of 48 hours, 0.20% initial cyanide dosage, 1 kg/t of lead nitrate, pH at 10.5 and 50 gpl of activated carbon addition.
- The reagent consumptions are 6 kg/t of NaCN, 1 kg/t of lead nitrate and 2.5 kg/t of hydrated lime.

A comprehensive gold deportment analyses on the sulphide CIL residues confirmed that the predominant gold losses were due to:

- Sub-microscopic gold with a gold content ranging between 3 g/t and 4 g/t. This residual gold cannot be recovered by CIL and sets the minimum gold residue grades.
- The remainder of the gold losses occur as unliberated fine gold particles locked in sulphides and equates to approximately 7% to 12% or 0.3 g/t to 0.8 g/t of the gold content.
- Based on this assessment an average of 4.5 g/t gold has been selected as the average sulphide residue gold grade that can be achieved from the CIL of the sulphide concentrates recovered via flotation.

Comparing the gold recoveries achieved from direct CIL vs. the flotation/UFG/CIL route on the sulphide samples is presented in Table 1.3. As shown the flotation/UFG/CIL route provides an incremental gold recovery increase ranging from 7.4% to 22.3% compared to direct CIL on the sulphide samples. Due to this significant gold recovery differential, it is recommended that the Ganajur Main Project incorporate a flotation/UFG/CIL circuit for the processing of the sulphide resource.

A gold recovery model for the sulphide resource was developed via the combination of a multivariate regression analyses (to predict the sulphide sulphur content of the Ganajur Main resource) and the interpreted results achieved from the flotation and UFG/CIL testwork program. The predicted vs. the actual gold recoveries achieved from testwork are in close agreement. Based on the average sulphide ore reserve gold grade at 3.7 g/t, the regression model estimated the average sulphide sulphur grade at 2.8%. At this average gold grade and estimated sulphide sulphur, an average gold recovery of 79% is estimated for the sulphide ore.

A gold recovery of 90% for the oxide resource can be achieved via a P_{80} grind liberation size of 75 microns and 24-hour CIL retention. The estimated key reagent consumptions are 0.5 kg/t of cyanide and 0.80 kg/t of lime.

The WAD cyanide content after CIL on the sulphide concentrates can be successfully decreased to below 10 ppm via two stages of the SO_2/O_2method (Inco) for cyanide destruction. The SMBS consumption is moderate at 5.2 kg/t of concentrate. Due to the high iron content in solution after CIL, the copper sulphate consumption is high at 3.2 kg/t of concentrate.

Removal of soluble Arsenic prior to discharge into the TSF has been achieved via the addition of ferric sulphate which is the Best Demonstrated Available Technology (BDAT) as recommended by the US EPA. A moderate consumption of ferric sulphate at 6 kg/t is estimated for this stage of the overall Ganajur flowsheet.

The rheology and settling properties for the Ganajur sulphide concentrate and final tailings stream are benign which should result in minimal slurry pumping issues, inter-tank screen head losses in CIL or thickener area requirements in order to achieve high thickener underflow slurry densities and clear overflow streams.

Table 1.3 Direct CIL vs. flotation/UFG/CIL

Sample	Feed		UFG/CIL Tail		Direct CIL Tail		Recovery Variance %
	Au	SS	Au	% recovery	Au	% recovery	
GM2	11.0	4.6	0.73	94.6			
GM3	5.2	3.8	0.78	87.4	1.85	65.1	22.3
GM4	3.7	2.4	0.86	79.5			
GM5	7.1	5.3	1.15	87.0	1.78	75.7	11.3
GM6	3.3	2.5	0.73	80.5	0.93	70.0	10.5
BSC	6.2	3.9	1.13	84.7	1.39	77.2	7.4

1.4 Mining engineering and Ore Reserve estimates

1.4.1 Geotechnical investigations

The Ganajur Main Gold deposit geotechnical engineering report was prepared by Sarathy Geotech and Engineering Services Pvt Ltd (SGES) in India for the FS, with technical oversight and review provided by Snowden.

Based on the geotechnical studies, pit slope design recommendations were developed with an acceptable level of risk of small-scale failures developing on batter faces. The recommended slope design angles in the pit design sectors are summarised in Table 1.4

Table 1.4 Pit slope design recommendations

Sector	Wall	Batter angles (°)		Maximum inter-ramp angle (°)	
		Weathered	Fresh	0 to 50 m	50 m to 100 m
Northwest	Footwall	45	60	46	51
	Hangingwall	45	80	48	56
	End wall	45	75	46	56
Southeast	Footwall	45	55	46	51
	Hangingwall	45	80	48	56
	End wall	45	75	46	56

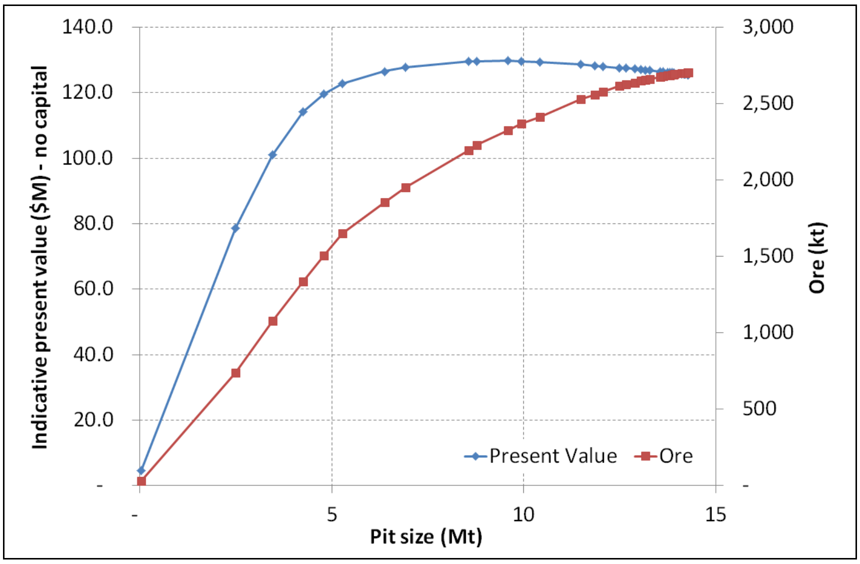
Notes to design recommendations:

- “Weathered” applies to top two 10 m benches.
- “Fresh” applies to all benches more than 20 m below surface.
- All batters to be maximum 10 m height.
- All berms to be minimum 5.0 m width.

1.4.2 Economic pit identification

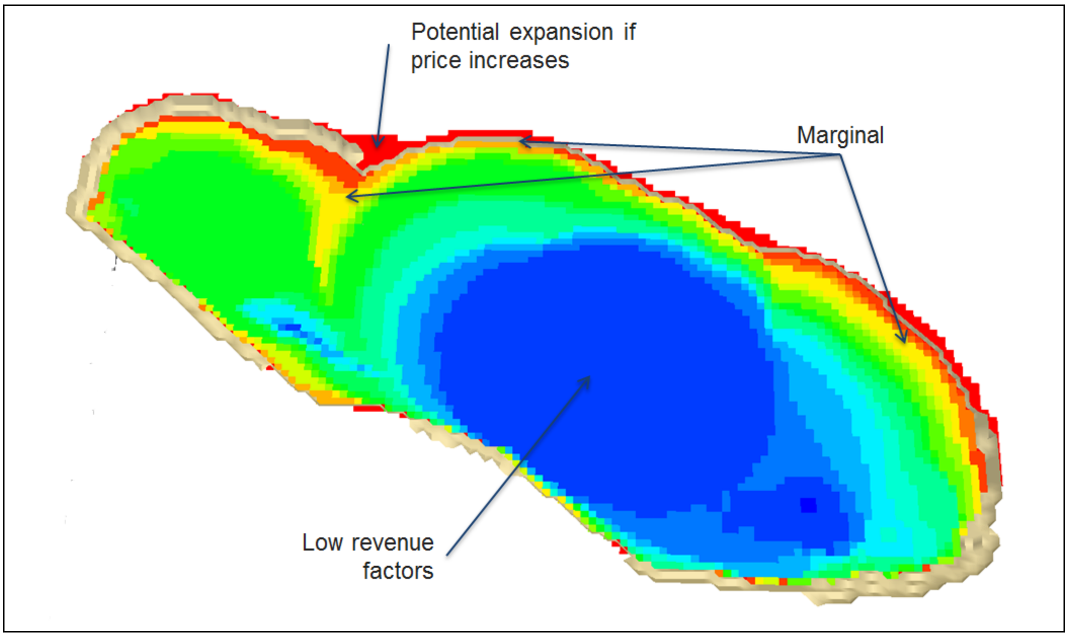
A Whittle pit optimisation confirmed an economic shell using the Measured and Indicated Mineral Resources. A graph of the pit size and present value is provided in Figure 1.4.

Figure 1.4 Graph of present value without capital and pit size



The pit shell development is shown in Figure 1.5, with the low revenue factor shells robust for a low gold price and the outer shells for a higher gold price.

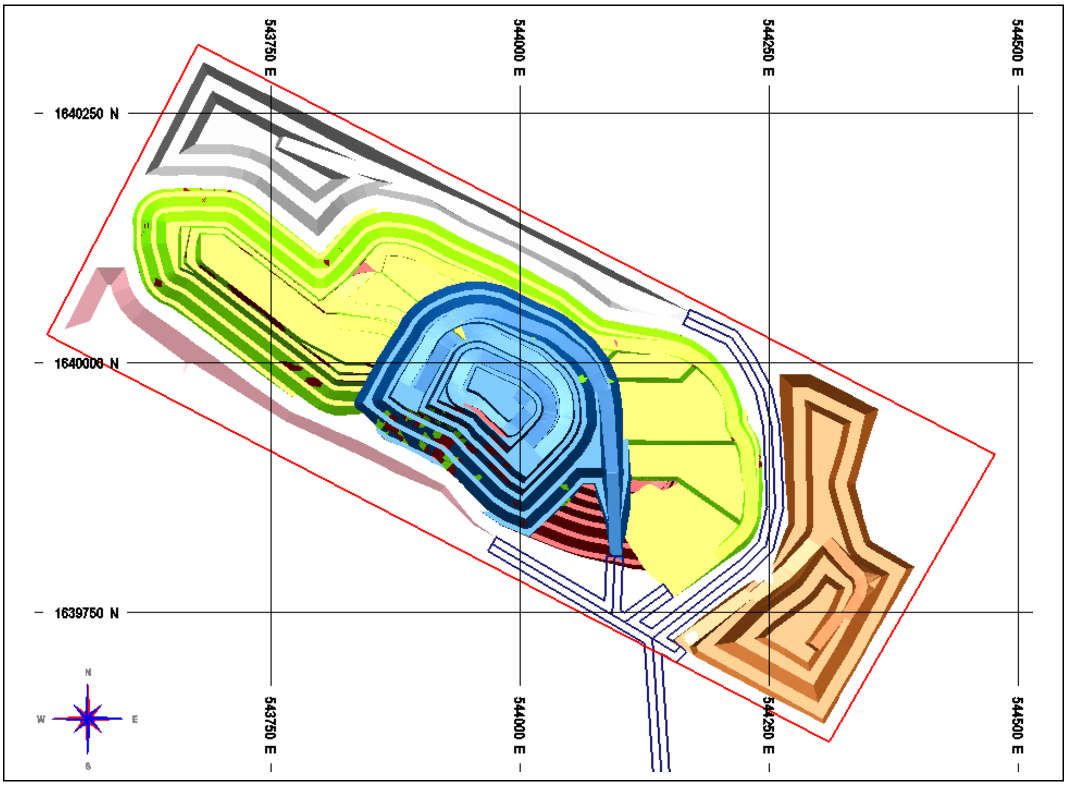
Figure 1.5 Pit shell development



1.4.3 Pit design

Pit designs were completed in Minesight software. Figure 1.6 shows the development of the staged pit design, the waste export pile and stockpiles and the proposed ex-pit road network inside the mining lease. Most of the waste is proposed to be exported to market as civil building materials.

Figure 1.6 Intra-pit road network

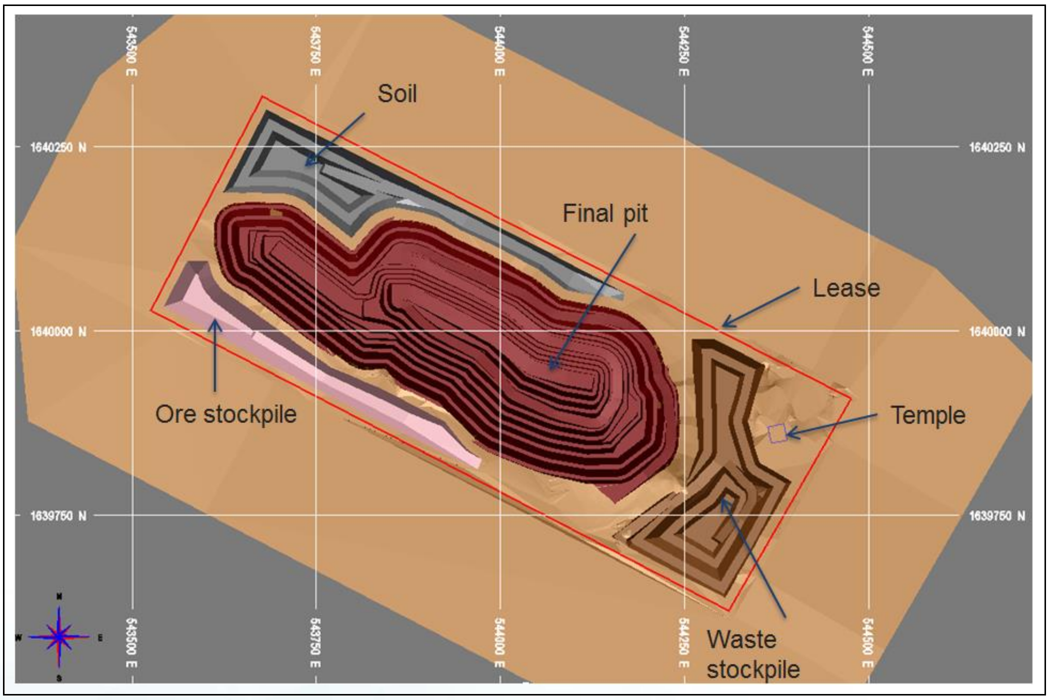


1.4.4 Site layout

There are no treatment facilities in the mining lease and ore is proposed to be processed at the Ganajur processing plant located 1.0 km south-southeast of the mine site.

The site layout with location of ultimate pit, existing temple, mining lease and stockpiles is shown below in Figure 1.7.

Figure 1.7 Ganajur mining lease layout



1.4.5 Mine production schedule

The total ex-pit movement is shown by stage in Figure 1.8.The mining rate is constrained by the maximum allowed during mine production quarters 4 to 6 and 9 to 11. The ore and waste movement is provided in Figure 1.9. The total life of mining is approximately 7.6 years.

Figure 1.8 Total ex-pit movement schedule by pit stage

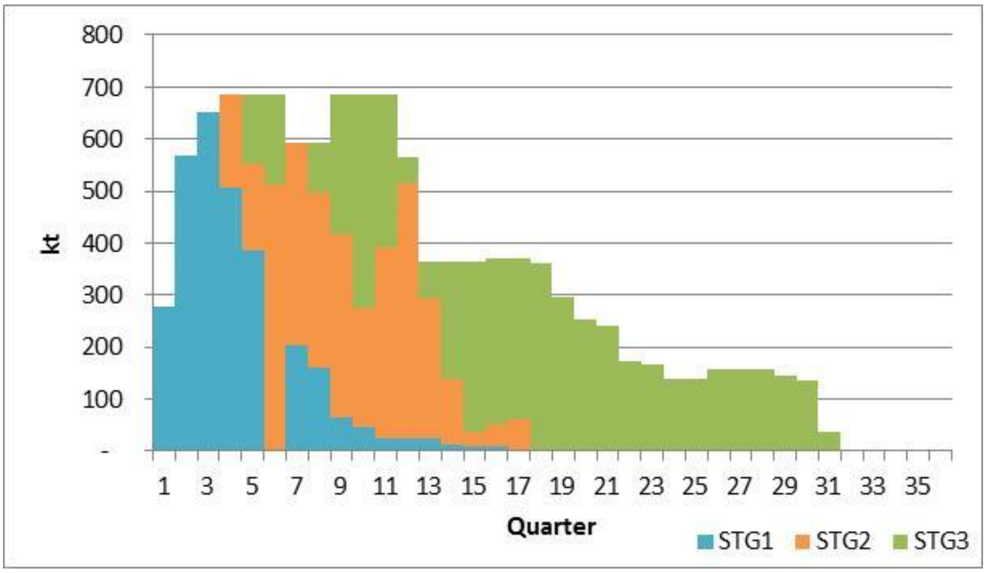
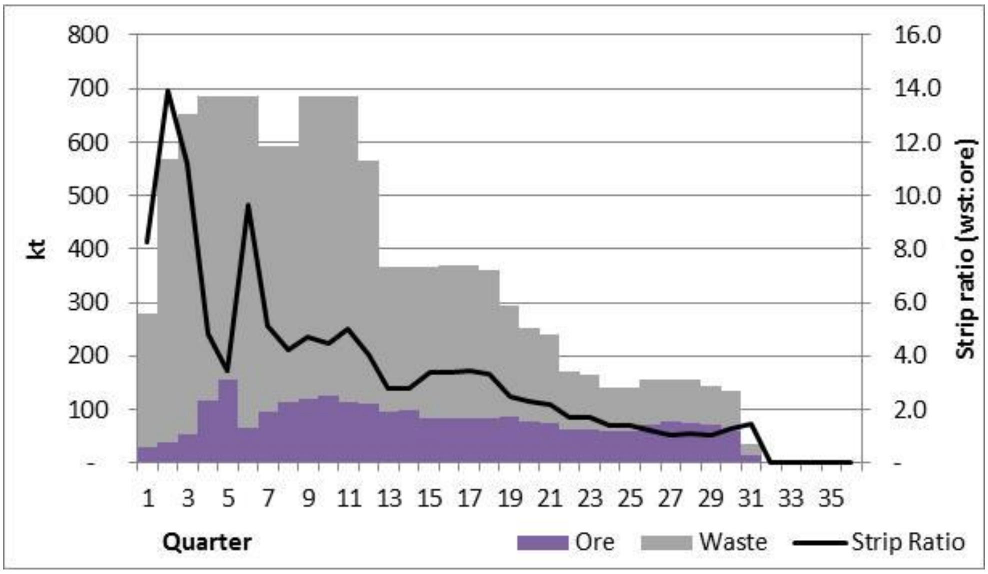
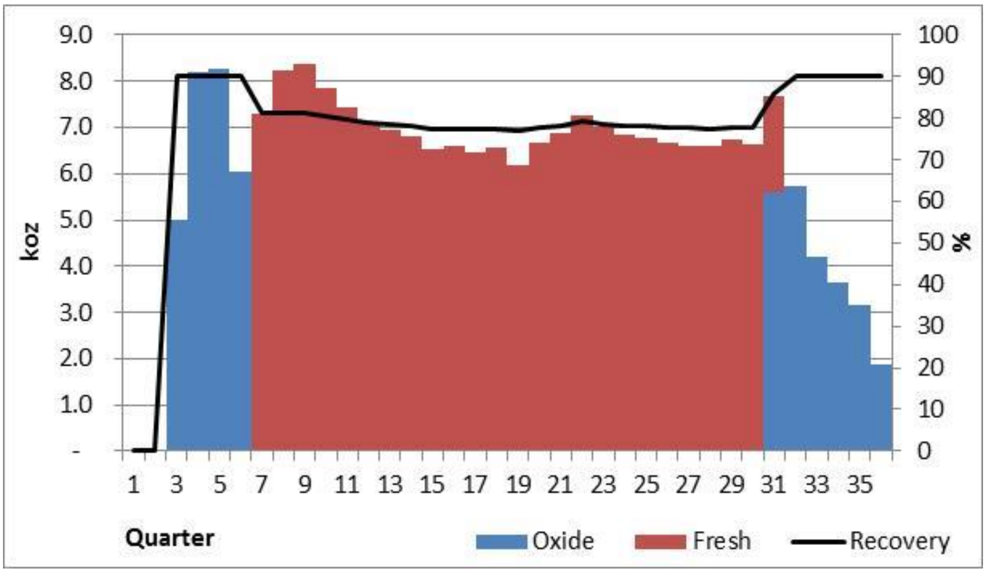


Figure 1.9 Total ex-pit movement schedule by material type



The gold production schedule is quite consistent at about 7 koz per quarter. When sulphide processing is complete, the production rate for the remaining oxide stockpile rapidly drops off as the gold grade decreases. Recoveries for the oxide material are constant 90% while fresh material varies around an average 79% recovery.

Figure 1.10 Recovered gold schedule



1.4.6 Mining methods and mine requirements

The mining method is a conventional open pit mining with load, haul and drill blast activities performed by an experienced mining contractor. It is planned that the mining contractor will buy back the waste for use in their civil operations elsewhere, subject to an offtake agreement with DESPL. Explosives and diesel fuel will be sourced locally in the city of Haveri and the costs covered by the mining contractor.

Semi selective mining is required to extract the main lode of ore that is nearly 30 m wide. A fleet of drill rigs (150 mm diameter), an excavator with a 2.0 m^3 bucket capacity and a fleet of 25 t capacity trucks will undertake the primary mining activities, supported by bulldozers, graders and loaders. Water will be encountered approximately 30 m below the surface and pumped from the pit.

The maximum manning for the Project is 115 persons and 10 technical staff will be required to control the mining operations. Accommodation for DESPL personnel and contractors will not be provided by DESPL but is available in the township of Haveri.

1.4.7 Ganajur Ore Reserves

Proved and Probable Ore Reserves were reported using the 2012 edition of the “Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves” (JORC Code 2012).

The Ore Reserve estimate for the Ganajur Main Gold Deposit as at the end of April 2017 is provided in Table 1.5. Note that tonnes and ounces have been rounded and this may have resulted in minor discrepancies.

Table 1.5 Ganajur in-situ Ore Reserve estimate as at April 2017

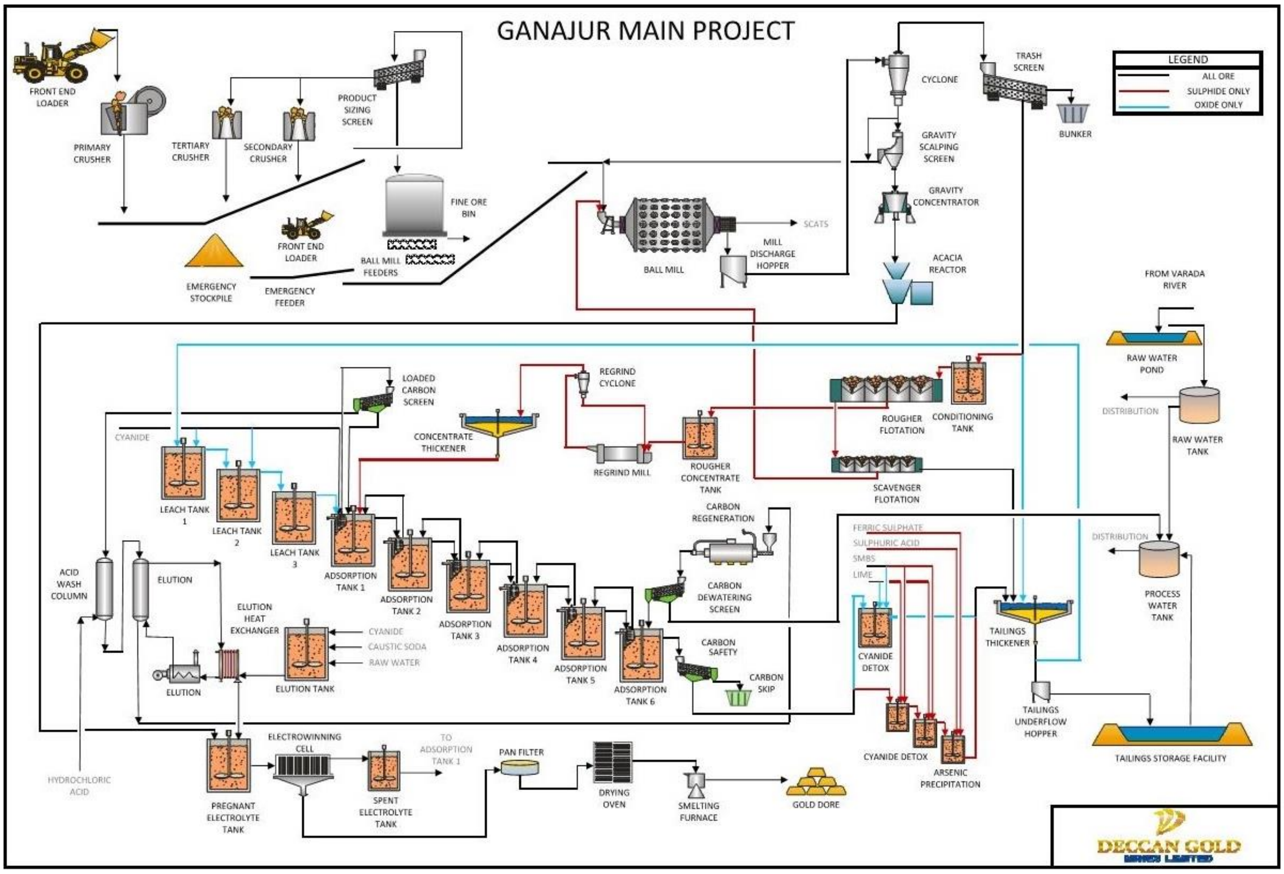
Classification	Weathering	Tonnes (kt)	Au (g/t)
Proved	Oxide	568	2.76
	Sulphide	1,567	3.94
Subtotal – Proved		**2,135**	**3.63**
Probable	Oxide	122	1.78
	Sulphide	250	2.08
Subtotal – Probable		**372**	**1.98**
TOTAL		**2,506**	**3.38**

1.5 Process plant and description

The process plant will treat a nominal 300,000 tonnes per annum (t/a) of gold-bearing ore with a crushing availability of 70% on a single shift and an overall plant availability of 91.3%. Ore will be processed via campaigns based on lithology as either sulphide or oxide material.

Both sulphide and oxide ores will be processed through the same crushing, milling, gravity and elution circuits, however, there are some processing differences around flotation, CIL and tailings.

Figure 1.11 Simplified process flowsheet



1.5.1 Sulphide ore processing

The ore will be delivered to the run of mine (ROM) pad using mine haulage trucks and stockpiled into fingers. A front-end loader (FEL) will reclaim ore from the various fingers as required. The jaw crusher will be fed from the FEL via a 30 m^3 ROM bin.

The jaw crusher will crush the ore to a P_{80} of 100 mm. A final crushed ore size of P_{80} 11 mm will be accomplished using two cone crushers and a product screen. The crushing circuit selected is a modular design to simplify the installation process and reduce construction costs.

The crushed ore will feed the grinding circuit which is a conventional ball milling circuit in closed circuit with cyclones. The cyclone overflow will have a P_{80} of 75 µm, suitable for the flotation of the sulphides and CIL of the oxide ore. Cyclone underflow will recycle back to the ball mill with a portion split off to feed the gravity concentration circuit.

The gravity concentration circuit consisting of a centrifugal concentrator and intensive leach reactor will treat a percentage of the cyclone underflow. The pregnant solution from the intensive leach reactor will be transferred to the gold room for electrowinning.

The cyclone overflow, via a trash screen will be fed to the rougher flotation circuit where the concentrate will feed the UFG circuit. The rougher flotation tailings will flow through a bank of scavenger flotation cells with the scavenger flotation concentrate recycled back to the ball mill feed and the tailings stream discharged into the tailings thickener.

The flotation concentrate will be further reduced in size via an UFG mill to a P_{80} of 10 µm which will operate in closed circuit with cyclones. Lead nitrate will be added to the mill feed to aid in gold dissolution in the leach circuit. The cyclone overflow will feed the flotation concentrate thickener while the cyclone underflow will be returned to the UFG mill.

The reground flotation concentrate will be thickened to 50% solids and agitated in a tank prior to six stages of leaching/adsorption in the CIL circuit. Loaded carbon will be removed periodically and replaced with regenerated and/or fresh carbon. The loaded carbon will be transferred to the elution circuit for gold recovery and doré production.

The tailings from the CIL circuit will feed the cyanide detox circuit which consists of two agitated tanks. Sodium metabisulphite (SMBS) and oxygen will be added in the first tank to convert the cyanide (CN^-) species to cyanate (CNO^-) which is relatively stable and with time will hydrolyse to ammonium and carbonate. Hydrated lime slurry will be added for pH control and the addition of copper sulphate will provide a catalyst for the reaction.

Discharge from the cyanide detox tank will flow into a single agitated tank for arsenic precipitation. Sulphuric acid (H_2SO_4) will be added to decrease the pH to approximately pH 6 and ferric sulphate ($Fe_2(SO_4)_3$) will be added to aid in the precipitation of arsenic from solution.

The arsenic precipitation circuit discharge will report to the tailings thickener where it will combine with the rougher flotation tailings and be thickened to 55% solids before being pumped to the TSF. Water will be decanted from the TSF for reuse in the process via the process water tank.

1.5.2 Oxide ore processing

Oxide ore will be processed through the same crushing, grinding and gravity circuit as the sulphide ore.

Cyclone overflow from the milling circuit will bypass the flotation circuit and instead will feed into the tailings thickener which will be used as a pre-leach thickener when processing oxide ore.

Thickened oxide ore will be leached in three large tanks prior to flowing into the six carbon adsorption tanks which are also used for processing the sulphide ore. The three larger leach tanks are required to maintain the residence time in the leach/adsorption circuit for the higher flowrate of the oxide ore stream.

Loaded carbon from the adsorption tanks are processed through the same elution circuit and gold room as the sulphide ore.

The leached tailings from the last adsorption tank flow through the carbon safety screen and report to a single stage CN destruct circuit, prior to being pumped to the TSF.

1.6 Surface geotechnical and tailings disposal

The Ganajur Gold Project infrastructure will include a TSF, a return water dam, stormwater dam, and other surface water management measures.

A geochemical assessment of the tailings material was undertaken and has been classified as potentially acid forming (PAF), with residual traces of arsenic released into solution despite the stabilisation step in the process plant. Due to the PAF geochemical classification, the selection of a suitable deposition and construction methodology required an analysis of the design criteria of both an upstream and downstream facility.

A risk-based approach was used in selecting a preferred construction method, taking into account aspects such as oxidation rates and other mitigation measures, drainage and long term environmental impacts. The key criteria for the selection of the TSF construction and operation methodologies included a cost-effective solution, environmentally acceptable practice, with maximum water conservation and minimum land use. A final recommendation and design option was made in favour of an upstream facility based on a smaller footprint, suitable pollution mitigation solutions and maximising water recovery.

Geomechanical laboratory testing of the tailings material was undertaken to determine its behaviour under conditions expected during the life of the TSF. These include density, permeability, shear strength and consolidation and were used in the design of the TSF.

A geotechnical investigation was undertaken to determine the suitability of the in-situ soils for its intended use as low a permeability liner and the construction of containment embankments. The results show that the in-situ soils have suited mechanical properties for embankment construction, but require additional compaction and permeability testing.

The TSF has been positioned adjacent to and downstream of the processing plant. The site selection was influenced by the required storage capacity and footprint, construction and development methods, local structural geology and topography, land ownership, rehabilitation requirements and existing significant surface infrastructures and features.

The TSF is designed to store a total dry tailings tonnage of 2,589,681 t over the 8.4-year life of mine requiring a volume capacity of 1.63 Mm^3 and footprint of 15.7 ha. The facility reaches maximum height of 19 m at a final rate of rise of 1.95 m per year and overall slope of 1V:3H.

The basin of the TSF will be lined with a low permeability clay layer, overlain with a geotextile and an HDPE geomembrane. Tailings will be deposited via a ring main distribution system. Drainage of supernatant water and rainfall will be via vertical penstock intake structures. Seepage water will be drained thought toe and blanket drains positioned along the basin of the TSF.

Decant and seepage water will be collected in a return water dam for reuse as process water. Stormwater runoff from the side slopes and the top surface of the TSF will collect in the perimeter catchment paddocks from where it will drain into the stormwater or event dam.

The Project site is located in a low risk seismic zone. A slope stability analysis, which incorporated tailings and soils mechanical properties and the TSF geometric profile, indicated a suitable factor of safety (FoS) against major slope failure.

A monthly deterministic water balance model has been compiled for the TSF, return water and stormwater dams. The input parameters to the water balance include the tailings characteristics, TSF layout and climate data. The rainfall data has been statistically analysed to incorporate the seasonal variability and to compile a conservative water balance. The annual return water volume was estimated at 130,476 m^3, approximately 53% of the slurry water deposited into the TSF per annum, which will return to the process plant.

The side slopes of the TSF will be progressively rehabilitated with a clay and topsoil layer and vegetation. Once the TSF has been decommissioned, the top surface of the TSF will also be capped with a rock, clay and topsoil layer, and vegetated.

1.7 Project infrastructure

The key infrastructure that has been incorporated into the overall Project design include:

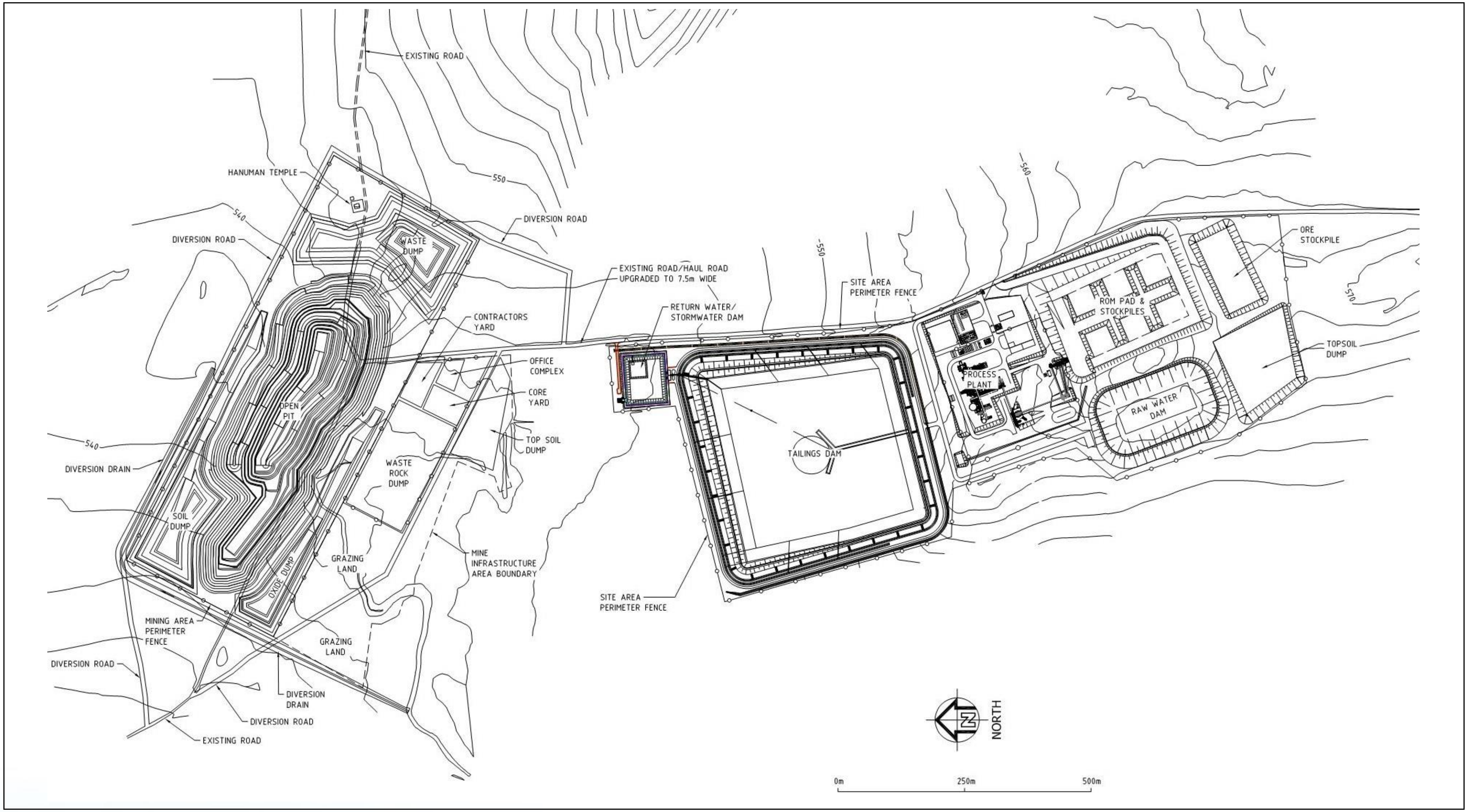
- Roads both within the Project area and access to the Project site
- Office buildings
- Gatehouses
- Laboratory and sampling areas

- Workshops and maintenance facilities
- Power supply and distribution
- Communications and computer network
- Water supply and storage infrastructure
- ROM pad and ore stockpile
- Stream diversion channel.

1.7.1 Layout

A proposed layout of the Project site and associated infrastructure is shown in Figure 1.12. The layout shows the location of the raw water dam, ROM pad and ore stockpiles, processing facilities, roads and buildings.

Figure 1.12 Overall plant site layout



1.7.2 Roads

Access to the site will be via an existing public road to the east of site area. The existing paved road, accessed from Ganajur-Nagendrana Matti-Haveri road is only 3.5 m wide and will be upgraded as it will be used as a haul road from the mining pit to the ROM pad.

The existing road currently runs through the proposed mining pit. The road will be diverted to the north and south around the pit and reconnected to the eastern site access road. Access to the Hanuman Temple will be provided from the northern diversion road.

1.7.3 Power supply and distribution

The electrical power for the Ganajur Gold Project will be supplied from a take-off bay at a nearby 110 kV KPTCL substation located near Basavanakatti village which is approximately 7.5 km from the processing plant site. The exact location has not been finalised, however there are ongoing discussions to locate the take-off closer to the project site.

An 8 MVA substation located in close proximity to the plant site will be constructed to transform the 110 kV voltage supply to 11 kV. The substation will be constructed to meet the specifications and requirements of KPTCL.

The exact route of the 7.5 km 110 kV transmission line will need consultation with KPTCL and local farmers to an agreed right of way.

Refer to drawing, 7056-492-ED-001 in Appendix C of Chapter 8, for the overall plant site high voltage single line diagram which illustrates the main power distribution around the plant.

1.7.4 Raw water supply and storage

Raw water will be supplied from the Varada River, with supply limited to the monsoon months of June through September. During this period, the flood gates will be opened and the water flows through the weir located at the Kolur-Kalasur barrage.

An intake well will be constructed at Kolur-Kalasur barrage that will allow for two submersible pumps to draw water from the river on a duty/standby basis.

A 6.5 km buried pipeline will transfer raw water from the Kolur-Kalasur barrage to a 300,000 m^3 raw water storage dam constructed on the plant site. The pipeline will be buried and will utilise easements next to existing roads. This design requires minimal government involvement and ensures that once the water is pumped it remains within DGML control.

1.7.5 Buildings

Table 1.6 lists the proposed site buildings, size and use.

Table 1.6 List of site buildings

Building	Size
Administration building	480 m^2
Process plant office	110 m^2
Security office	110 m^2
Maintenance office	110 m^2
Workshop/warehouse shed	540 m^2
Medical facility	135 m^2
Mining office	110 m^2
Site area gatehouse	12 m^2
Mining area gatehouse	12 m^2
Plant ablutions	36 m^2
Crushing area switch-room	55 m^2
Main (HV/LV) switch-room	108 m^2
Lunch room	36 m^2
Control room	12 m^2
Titration room	12 m^2
Laboratory/sample preparation building	325 m^2 / 66 m^2

1.7.6 Control system

Communications around the plant will be via optical fibre cable configured in a ring topology to enable a robust communications media with redundancy. Twelve (12) core fibre cables have been specified enabling the fibre network to carry a variety of applications on dedicated fibres. They include:

- Business IT
- VoIP phones
- SCADA
- Security
- CCTV.

1.7.7 Potable water treatment plant

A brackish water reverse osmosis water treatment plant will process raw water from the raw water dam for potable water use. The treated water will be stored in a tank with a 20-hour capacity to provide potable water to the office buildings, mine site and process area.

The potable water treatment plant has a design capacity of 5 m^3/h.

1.7.8 Fire protection

The plant site facilities will be protected with a pressurized fire protection system that comprises a fire water reserve, an electric driven jockey pump, an electric driven fire pump, and an emergency diesel driven fire pump. The firewater distribution system will consist of a dedicated buried firewater loop and hydrant system for the process, ancillary buildings and warehouse/workshop.

1.7.9 Sewage

Two septic tanks will be provided; one for the plant site area and the other for the mining area.

1.7.10 Security

Security fencing around the plant/TSF and the open pit will restrict non DGML personnel, farming animals from access to these operational areas. Additional fencing will be provided for the process plant to enhance the level of security.

The gold room will be a heavily secured building, with three CCTV cameras for monitoring. Limited access will be provided into the gold room to authorised personnel via the access card reader system.

The gold room will be a sheeted building with security meshing on the inside for added security. The gold bullion will be secured in a safe which will be located in a concrete vault room within the gold room.

1.7.11 Run of mine pad and ore stockpile

The ROM pad is designed to stockpile 130,000 t of ore configured in six finger stockpiles, each 6 m high. The six finger stockpiles will be used to blend the feed ore to the process plant.

The ROM pad will incorporate a skyway to allow trucks to directly tip ore to the dedicated stockpiles.

An area south of the ROM pad has been allocated to stockpile an additional 120,000 t of ore.

Both the ROM pad and ore stockpile will be 300 mm clay lined to prevent any seepage into the ground from potential acid mine drainage (AMD) solutions.

1.7.12 Stream diversion channel

The open pit is situated in a low-lying area. The stream flows through the pit during the monsoon season. To minimise water ingress into the pit, a diversion channel will be constructed upstream to direct the water around the western side of the pit and the same will be connected downstream.

The channel will be sized for a 1:50-year rain event and be constructed with a 1:400 slope. The channel will be an unlined excavated structure.

The channel width at the two diversion road crossings is approximately 25 m. Two concrete bridges have been allowed for these road crossings.

1.7.13 Accommodation camp

No allowance for an accommodation camp has been made. The construction workforce and the mining staff will be housed in nearby towns and villages such as Haveri, Ganajur and Karajgi.

1.8 Marketing information

India is a mineral rich country with wide availability of minerals in the form of abundant rich reserves and favourable eco-geological conditions. The Indian mining industry is characterised by a large number of small operational mines.

Given the traditional significance given to gold possession in India, it generates one of the highest quantum of demand in the world. In 2015, India accounted for nearly 26% of the global fabrications demand (jewellery, bar and coin and technology). The demand for gold in India is not only the highest in the world but also the fastest growing. With the Indian economy projected to grow at 8% during 12th plan, the demand for gold can only increase further.

Although India has a long history of gold mining, current production levels are very low; during 2014/2015, India's gold production was a negligible 1.43 t (less than 2 t). Over the coming years, mine production is expected to grow modestly as new mines enter the production phase. But the industry faces significant challenges. For mining to develop in India, regulations need to be reviewed and the industry needs investment.

India is a traditional and stable market for gold consumption imports of gold in significant quantities will continue. The present and future production of gold will not be sufficient to meet the ever-increasing demand. Therefore, efforts will be required to reduce the gap between production and demand.

The analysis, using annual data from 1990 to 2015, reveals two significant factors affecting gold consumer demand over the long term. All else being equal, gold demand is driven by:

- Income – gold demand rises with income levels; for a 1% increase in income per capita, gold demand rises by 1%
- Gold price level – higher prices deter gold purchases; for a 1% increase in prices, gold demand falls by 0.5%.

Going forward, the International Monetary Fund (IMG) has forecast per capital GDP to grow by 35% for 2015 to 2020 and the National Council of Applied Economic Research expects India's middle class to double, exceeding 500 million by 2025.

By 2020, it is expected that the Indian gold demand would average 850 t/a to 950 t/a. India's relationship with gold goes beyond income growth; gold is intertwined with India's way of life. And as we look ahead, India's gold market will evolve.

1.9 Geochemistry

1.9.1 Run of mine ore

Due to the ROM stockpile being unsaturated and air moving freely through the pile, oxidation of exposed surfaces of material commences shortly after placement of the material. Rainwater percolates through the stockpile materials and the release of drainage from the ROM stockpile will be periodic. The chemistry of the released water will be affected by the length of time the material has been on surface and the degree of weathering, the time between rainfall events (when last the material was rinsed), and the intensity of the rainfall. The material is classed as potentially acid forming and the SPLP and distilled water leach tests indicated that a fraction of arsenic (As) was present in readily soluble phases. Therefore, the ROM stockpile would require appropriate clay lining to prevent seepage into groundwater resources and percolating rainwater should be diverted away from any water resources.

1.9.2 Tailings material

Given the high As abundance in the ore samples, it is likely that there is sufficient As in the tailings to pose a long term leaching risk at the site. Although the As stabilisation step was successful, As is still released from tailings material during batch leaches at concentrations close to or exceeding General Effluent Standards. The leaching profiles of acidity, As and sulphate (SO_4) from tailings material that has undergone As stabilisation are currently being assessed during long term column leaching experiments. These results are expected after June 2017.

The TSF requires an appropriately lined facility (HDPE and clay) to prevent groundwater contamination. During construction or post closure the facility should be capped to minimise oxygen ingress and oxidation of tailings. Surface water decant should be contained and treated if necessary. Rainwater should be diverted where possible.

1.9.3 Waste rock material

The waste rock is intended to be sold as aggregate. For a period of time prior to sale, waste rock might be stored on site as a waste rock pile. Geochemical tests were therefore undertaken to assess the release of metals and acidity from the waste rock.

The waste rock is classified as non-hazardous according to Schedule II of the Indian Hazardous and Other Wastes (Management and Transboundary Movement) Rules, 2016. If the waste rock will be sold as aggregate it is the responsibility of the seller (according to IS 383:1970) to determine the physical characteristics and provide further information regarding presence of reactive minerals as requested by the purchaser.

1.9.4 Outcome of the geochemical studies

Acidic to near neutral saline drainage is expected to be released following oxidation of the ROM stockpile and tailings material. The waste rock material is less sulphidic and does not have long term potential to generate acid, therefore neutral or saline drainage is expected to decant from any waste rock piles on site.

The abundance of As in the ore, tailings and to extent in waste rock material is of concern. Liberated As species are mobile across the pH scale. In the material, As is present as both readily soluble and insoluble phases. Potential soluble phases include adsorption onto or inclusion into iron hydroxide or association with carbonate minerals. Insoluble phases include arsenopyrite which would require oxidation for the As to become mobile.

1.9.5 Hydrology and hydrogeology

Total make-up water requirement for the Ganajur Gold Project is 3,000 kl/d or 0.035 cubic metres per second (m^3/s). The main make-up water supply for the ore processing will be pumped from the nearby Varada River, over a four-month period during the monsoon season, for the months from July to October and will be stored on site in a HDPE-lined water storage dam with an approximate volume of 300,000 m^3. A pump station at the Varada River and a buried HDPE pipeline will transfer the river water to the plant site raw water dam for further distribution.

From the hydrogeological modelling, there is very limited groundwater ingress up to 20 m below ground level. Once the mining operation progress below the water table, groundwater ingress will occur; preliminary modelling estimates have this inflow as increasing to a maximum of around 2,000 m^3/day based on the numerical flow model predictions toward the end of Phase 3 at 85 m pit depth. Groundwater drilling and future modelling may need to be planned at the start of the mining.

The groundwater ingress predictions are built into the seasonal water budget for the mine – to effectively optimise groundwater use, minimise water abstraction from the Varada River and to minimise water disposal as part of the water management plans. Up to 1,000 m^3 per day (m^3/d) of water from the mine can be recycled to the Raw Water Reservoir to replace water pumped from the Varada River pumping system. Any additional water produced from the mine will need to be treated to comply with Indian discharge standards and released to the stream and/or provided to local farmers.

1.10 Environmental studies, permitting and social or community impact

The semi-arid sub-tropical climate of the region comprises hot and humid summers, moderate monsoon seasons and mild winter seasons. May is the hottest month in the year. The months of December, January and February are considered to have pleasant climate.

The study area includes a 10 km radius around the proposed Ganajur Gold Project that includes the open pit mine, gold processing plant and related infrastructure near Ganajur Village, Haveri Taluk and District of Karnataka State. The baseline environmental conditions determined represent the background environmental conditions in the study area and buffer zone of the Project. Baseline environmental monitoring was carried out during the 2016 summer season (March, April and May 2016) per Indian legal requirements.

Potential impacts to the physical, ecological and socio-economic environments which may arise as a result of the proposed mining and related activities were assessed for the following aspects:

- Air emissions
- Transportation and roads
- Noise generation
- Waste water generation
- Solid waste disposal.

1.10.1 Air emissions

The following risk areas were identified as the main sources for controlling fugitive dust emissions on site:

- Drilling
- Blasting
- Excavation
- Loading operation
- Transportation of ore and overburden.

The proposed environmental management measures for controlling air pollution are as follows:

- Utilisation of drilling equipment with built-in water injection systems.
- Regular wet suppression (spraying) on blasted heaps, dumps and haul roads. Water sprayers controlling conveyor-borne dust with an efficacy of 90% or greater. Additives should be added to the sprayer arrangements of stockpiles of the crushed material.
- Wet suppression should also be implemented on stockpiles of the ore at the processing plant. Crushed fine ore must be stored in closed bins. Dry dust collectors (using dust socks which are cleaned via high pressure reverse air jets) and water sprays should further be utilised at the crushers and transfer points of all the conveyors for ore handling.
- Implementation and maintenance of a green belt around the mining area (afforestation).
- Best-practice measures for drilling and blasting (sharp drill bits, using optimum blast charges and time delay detonators).
- Avoiding blasting during high windy periods, night times and temperature inversion periods.
- Regular grading of haul roads and service roads.
- Managing vehicle loads to avoid overfilling and spillages.
- Ongoing maintenance and servicing of vehicles and machinery.
- Progressive revegetation of denuded areas to stabilise surfaces.

1.10.2 Noise generation

The following noise abatement measurements are proposed for implementation during the operational phase:

- Ongoing maintenance of vehicles, machinery and equipment.
- Limiting blasting activities to daylight hours and employing optimum explosive charges, proper delay detonators and proper stemming to prevent blowout of holes.
- Limiting time exposure of personnel to excessive noise and adequate provision of personal protective equipment.
- Limiting vehicle speeds appropriate to the type of vehicle and the working area.
- Noise generating sources at the plant should be sufficiently away from residential dwellings. Crushers, grinding mills and diesel generating sets could be housed in closed buildings to help attenuate the noise level.
- In order to reduce noise generation/absorb noise from air compressors, pumps and diesel generators, the machinery will be placed on vibration isolators.
- The proposed 7.5 m wide green belt should encompass the plant area, office buildings, township and internal roads wherever possible to attenuate noise.

1.10.3 Terrestrial ecology (fauna and flora)

To manage potential impacts to flora and fauna of the study area, the following conservation measures should be undertaken in the study area:

- Plantation of suitable native species preferably indigenous species on degraded or waste-land and open degraded forest:
  - It is noted that a comprehensive list of suitable species is not provided in this regard, nor is an actual implementation plan.
- Planting of palatable grasses will be undertaken to support the herbivore population.
- Artificial waterholes will be created and natural water sources will be maintained on a spatial–temporal distribution basis.
- Development of the greenbelt around mining area (mentioned elsewhere).
- Planting of native fruit and fodder species within the buffer zone.

1.10.4 Blasting and ground vibration

The drilling and blasting technical parameters specified must be strictly adhered to in order to limit the potential for fly-rock and ground vibrations and to ensure that the PPV remains within the allowable limits at all times. The following measures are planned for controlling ground vibration and fly rock:

Ground vibrations should be limited to less than 6 mm/sec by implementing the following measures:

- Free face must be provided for each hole and the charge per delay must be kept within permissible limits.
- Sand-covered, delayed detonating fuses must be used during blasting.
- The burden of holes in the first row, as well as the effective burden of other blast holes, must be optimised. Blast holes must also be charged with the optimum quantity of explosives.
- A staggered pattern of blasting must be adopted.
- Benches must only be blasted one at a time.

1.10.5 Afforestation

The proposed green belt, which is recommended to mitigate various potential impacts arising due to mining and related activities at the Ganajur Gold Project, includes the following areas:

- 26% (36 ha) of the plant area
- Waste rock stockpile – 10.87 acres
- Temple Buffer Zone and Safety Zone (5.8 acres).

1.10.6 Social

The demographic profile of the socio-economic changesstudy area is shown in Table 1.7.

Table 1.7 Population demographics

Population, household size and sex ratio in the study area	Total (0 to 10 km)
No. of villages	20
Households	60,585
Population	292,696
Male population	150,671
Female population	142,025
Household size	4 to 5
Sex ratio	943 females : 1,000 males

The Ganajur Gold Mine and plant area does not involve any displacement of human settlements as the land is private agricultural land and will be purchased by DESPL. A farmhouse is present at the location of the proposed open-cast pit; however, it is assumed that this property will be dispensed with in terms of the land acquisition agreement arrived at.

1.11 Operating cost estimate

1.11.1 Process operating cost

The projected life of mine (LOM) average process operating cost for the 300,000 t/a Ganajur Gold Project is $23.53/t of sulphide ore processed and $18.36/t of oxide ore processed. This cost excludes all mining operating costs, taxes, permitting costs, non-process administrative costs and other government imposed costs unless otherwise noted.

Table 1.8 Summary of process operating costs

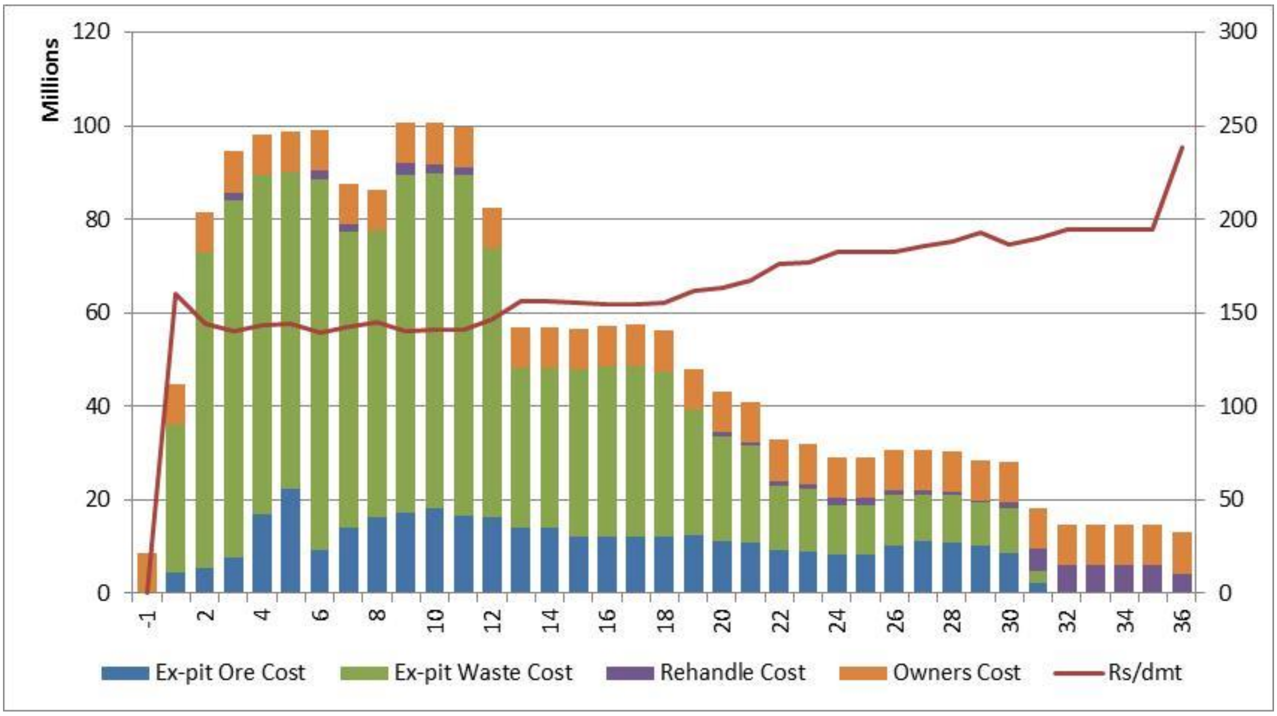
Category	Cost (US$)		
	$M/year	$/t	$/oz
300,000 t/a sulphide ore	7.06	23.53	249.31
300,000 t/a oxide ore	5.51	18.36	243.00

1.11.2 Mining operating cost

Mining costs were estimated in Indian rupee (IDR) and converted to US$ for financial modelling

Snowden estimated a LOM operating cost of ₹1,917 million (US$28.13 million). The operating cost schedule is shown in Figure 1.13.

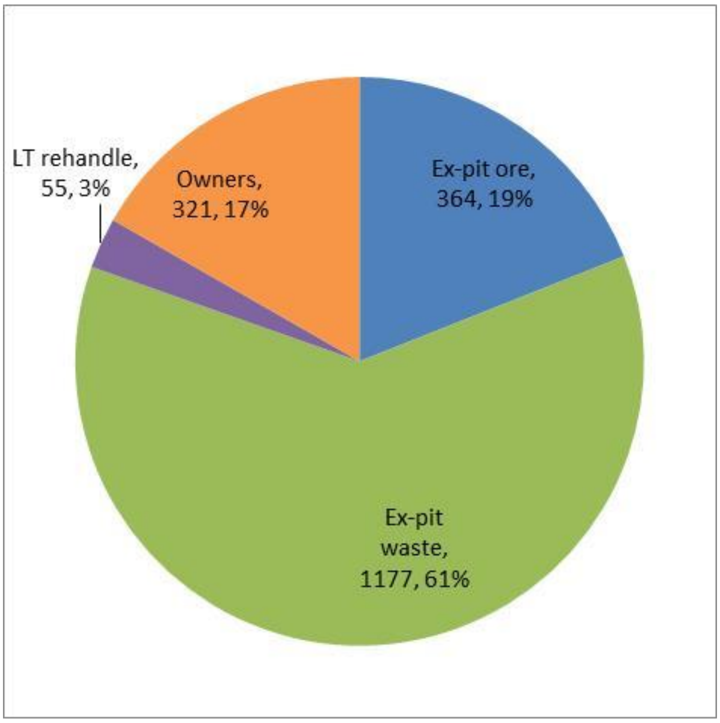
Figure 1.13 Operating cost summary (₹ million)



At the peak in Quarters 9 to 11, operating costs are estimated to be around ₹100 million (US$1.47 million) per quarter. The unit operating cost is relatively consistent at approximately ₹150/t (US$2.20/t) moved for the first half of the mine life before rising over the second half to around ₹200/t (US$2.93/t) moved.

Figure 1.14 summarises the total mining operating cost over the mine life. Nearly two-thirds of the total cost is attributable to mining waste.

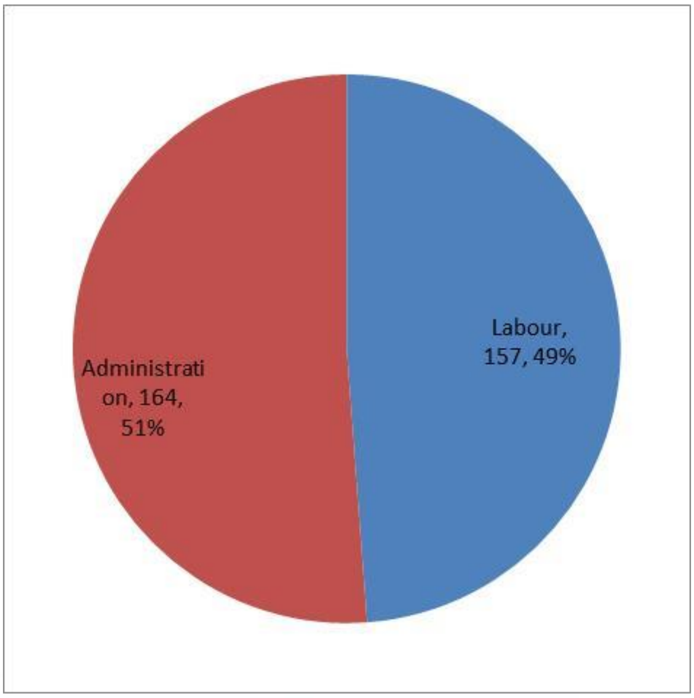
Figure 1.14 Total operating cost spilt (₹ million)



1.11.3 Owner's costs

Owner's costs are split almost evenly between labour and administration (Figure 1.15).

Figure 1.15 Owner's cost summary



1.12 Capital cost estimate

CPC Project Design Pty Ltd (CPC) has compiled the total capital cost estimate for the Ganajur Main Project, which is summarised in Table 1.9.

Table 1.9 Capital estimate summary – Ganajur Main Project (1Q17, ±15%)

WBS	Description	US$
1	Mining	874,470
3	Process plant	19,028,226
5	Process plant infrastructure	5,042,640
6	Infrastructure plant and equipment	4,283,556
8	Construction indirects	2,152,503
9	Indirect costs	14,961,410
Total		**46,342,805**

While CPC has prepared the majority of the estimate scope and pricing, Prime Resources (Pty) Ltd (Prime Resources) provided the engineering quantities for the TSF and DGML provided the owner's costs and assisted with obtaining indicative in country construction rates.

The work breakdown structure (WBS) is based on the standard CPC WBS for capital projects.

Note that the final pre-production capital number used in the Economic Analysis makes an allowance for progressive mine closure cost requirements and presents as $46.6 million.

1.13 Economic analysis

Snowden prepared an economic cashflow and financial analysis model based on inputs derived from mining and processing schedules, as well as capital and operating cost estimates, including royalties for the Project. The model was prepared from construction and mining schedules estimated on a quarterly basis for Project life. All inputs are consolidated annually in this report. The cash flow model was based on the following:

- 100% equity ownership
- Costing from January 2017
- 1.75-year production period for plant construction
- No cost escalation
- All costs reported in US$ and where costs were estimated in Indian Rupee (INR),the exchange rate used was INR66 to the US$.

Table 1.10 and Table 1.11 provide the Project headline results before and after taxation for a gold price of $1,250/oz of gold (base case).

Table 1.10 Economic model headline results before taxation

Item	Unit	Value at $1,250/oz Au
Net cash flow	$ M	133.0
NPV_5	$ M	91.6
IRR	%	39.1

Table 1.11 Economic model headline results after taxation

Item	Unit	Value at $1,250/oz Au
Net cash flow	$ M	93.1
NPV_5	$ M	61.4
IRR	%	29.6

Table 1.12 shows the inputs were used in the economic cash flow model.

Table 1.12 Economic model inputs

Item	Unit	Value
Pre-production	years	1.75
Life of process production	years	8.35
Project life	years	10.1
LOM ore mined	kt	2,506
LOM waste mined	kt	9,237
LOM total material mined	kt	11,743
Strip ratio w:o		3.68
LOM ore processed	kt	2,506
LOM average Au grade	%	3.38
LOM average Au recovery sulphide	%	79.0
LOM average Au recovery oxide	%	90.0
LOM average gold recovery	%	81.7
LOM contained ounces	koz	273
LOM recovered ounces	koz	221
Average annual gold produced	koz	27
Plant throughput (average)	Mt/a	0.30
LOM Au price	$/oz	1,250

A summary of total LOM costs is shown in Table 1.13 below. Note that no depreciation of capital was included in the taxation estimation.

Table 1.13 Total LOM costs

Item	Unit	Value
Pre-production capital	$ M	46.6
Production sustaining capital	$ M	3.1
Total Capital Costs	**$ M**	**49.7**
Total mining	$ M	21.6
Total processing	$ M	55.8
Onsite labour	$ M	1.2
Total Operating Costs	**$ M**	**78.5**
Royalties	**$ M**	**14.9**
Taxation	**$ M**	**39.8**
TOTAL ALL COSTS	**$ M**	**183.0**

The Project LOM key performance indicators (KPIs) after taxation are presented in Table 1.14 below.

Table 1.14 KPIs after taxation

Item	Unit	Value at $1,250/oz Au
Total value of product sold	$ M	276.1
Cash cost	$/oz	423
Total cost	$/oz	829
Production year payback	year	2.7
Brooke Hunt methodology C1 cost	$/oz	356
Brooke Hunt methodology C2 cost	$/oz	356
Brooke Hunt methodology C3 cost	$/oz	423

The cash costs include all direct operating costs plus royalties, the total costs include the cash costs plus capital costs and taxation. The Brooke Hunt methodology C1 costs include all direct operating expenses but do not include royalties, C2 is C1 plus depreciation and C3 is C2 plus royalties.

A breakeven analysis after taxation was undertaken on the gold price and gold grade for NPV_5. This analysis is conducted on the sensitivity analysis data and provides the gold price which will bring either the NPV_5 to \$0.0. The results of this analysis are presented in Table 1.15.

Table 1.15 Breakeven analysis after taxation

Item	Unit	Breakeven
Gold price	\$/oz Au	701
Gold grade	g/t Au	1.90

1.14 Project implementation

The recommended development methodology for the design and construction of the Ganajur Main project is engineering, procurement and construction management (EPCM). This approach allows DESPL to monitor and control the budget, schedule and quality through all stages of project development and execution.

It is intended that procurement of all equipment and bulk materials will be completed by the EPCM engineer and will be free-issued to the construction contractors for installation. This will ensure control over the critical procurement activities to achieve the desired completion schedule and ensure control of quality that meet DGML's requirements.

1.14.1 Project objectives

The strategic objectives for the project are to:

- Deliver the project with zero lost time and medical treatment injuries
- Zero major environmental incidents
- 100% compliance with all approvals
- Positive community relations
- Low impact on surrounding communities
- Implementation and delivery of an operational process plant which achieves the availability, reliability and metallurgical performance given in the process design criteria
- Low cost, fast track, high quality implementation of the process plant and associated infrastructure
- Utilise Indian manufactured equipment and materials where practically possible and cost effective.

1.14.2 Project implementation stages and schedule

This project implementation strategy provides the overall methods of managing the project from the detail design, procurement and construction through to commissioning. To meet the schedule proposed, the implementation schedule is structured into four stages:

- Feed engineering
- Detail design
- Construction
- Commissioning and handover.

1.14.3 Project schedule

The project schedule (Figure 1.16) and is based on the following:

- Off-site – 40-hour week, no work on public holidays, between Christmas and New Year and the first week of January
- On-site, the engineer and construction contractors will work 13 days per fortnight, 10 hours per day, with no site activities between Christmas and New Year.

Figure 1.16 Ganajur Gold Project summary project implementation schedule

The milestone dates for the development of the project are:

- May 2017 – Commence feed engineering
- July 2017 – DGML approval for the project
- September 2017 – Award of EPCM contract
- December 2017 – Mobilisation of mining contractor
- January 2018 – Site works earthworks
- May 2018 – Water storage dam and river extraction facility and pipeline completed
- October 2018 – Project completion and ore commissioning.

1.15 Risks and opportunities

A risk and opportunity assessment for the Ganajur Gold Project was performed via a one-day workshop held in March 2017 at the CPC office in Perth, Western Australia. The risk assessment was attended by representatives from Snowden, CPC and DESPL. Prime Resources' input was via a conference call link.

Risks were rated on the current state of the Project (Figure 1.17) and then re-rated after control measures were identified to manage the risks (Figure 1.18).

Figure 1.17 Comparison of current risks

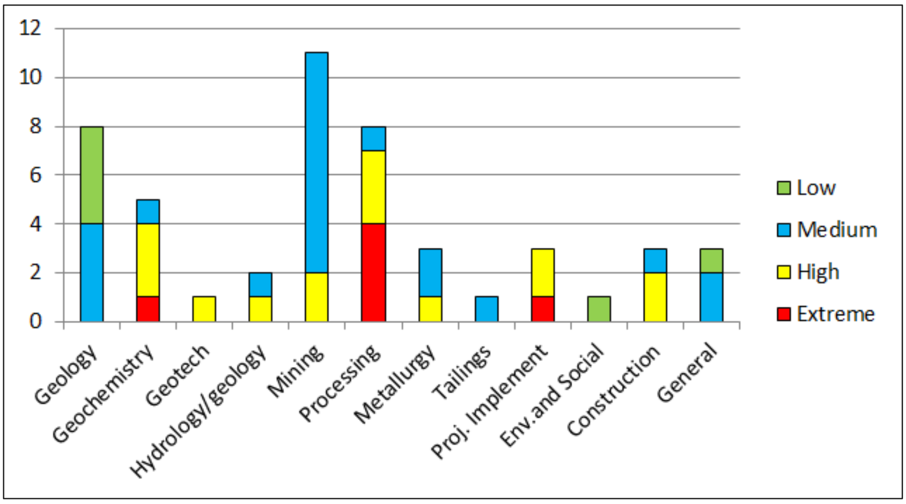
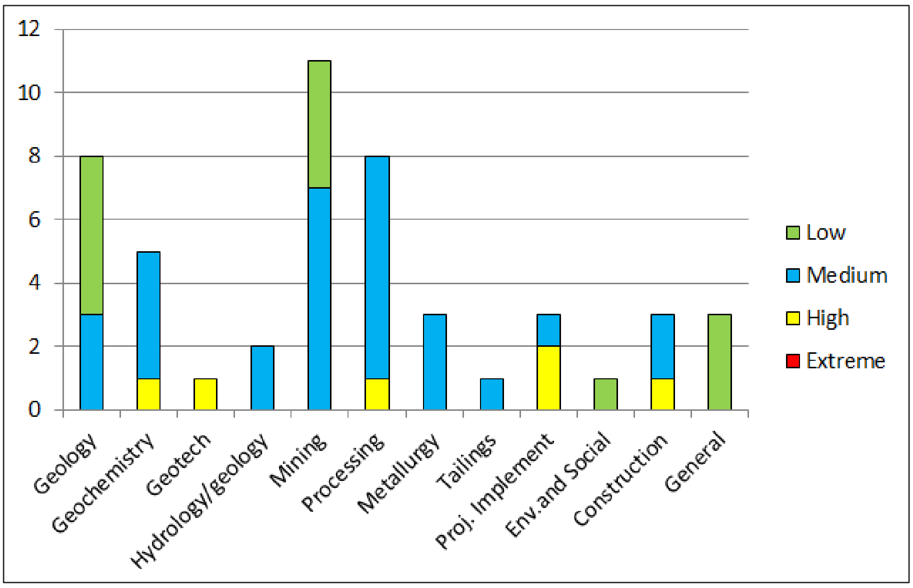


Figure 1.18 Risks after identified controls were applied



1.16 Recommendations

Recommendations for actions to occur both during project implementation and project operations have been made by the technical contributors for each of the following areas:

- Geology
- Geotechnical
- Geochemical
- Hydrogeology and hydrology
- Tailings management
- Mining operations
- Metallurgy and process plant
- Infrastructure, and
- Environmental, social and community.

The authors indicate that the best outcomes for the Project will be achieved if these recommendations are followed.

1.17 Interpretation and conclusions

Following review of the FS technical information, DESPL have advised their intention to progress with Project Implementation as per the schedule provided.

Snowden has reviewed the content of each technical chapter and interprets the mine as viable at the given assumptions, inputs and financial criteria. Snowden endorses the recommendations made by the technical contributors as required for the project implementation and future operations.

2 INTRODUCTION

Deccan Exploration Services Private Limited (DESPL) is a wholly-owned subsidiary of Deccan Gold Mines Limited (DGML). DGML is a public limited company listed on the Bombay Stock Exchange (BSE Scrip Code: 512068). DESPL is involved in gold exploration activities in the state of Karnataka since 2003 and is working towards its long-standing vision of developing a producing gold mine in India.

2.1 Property description

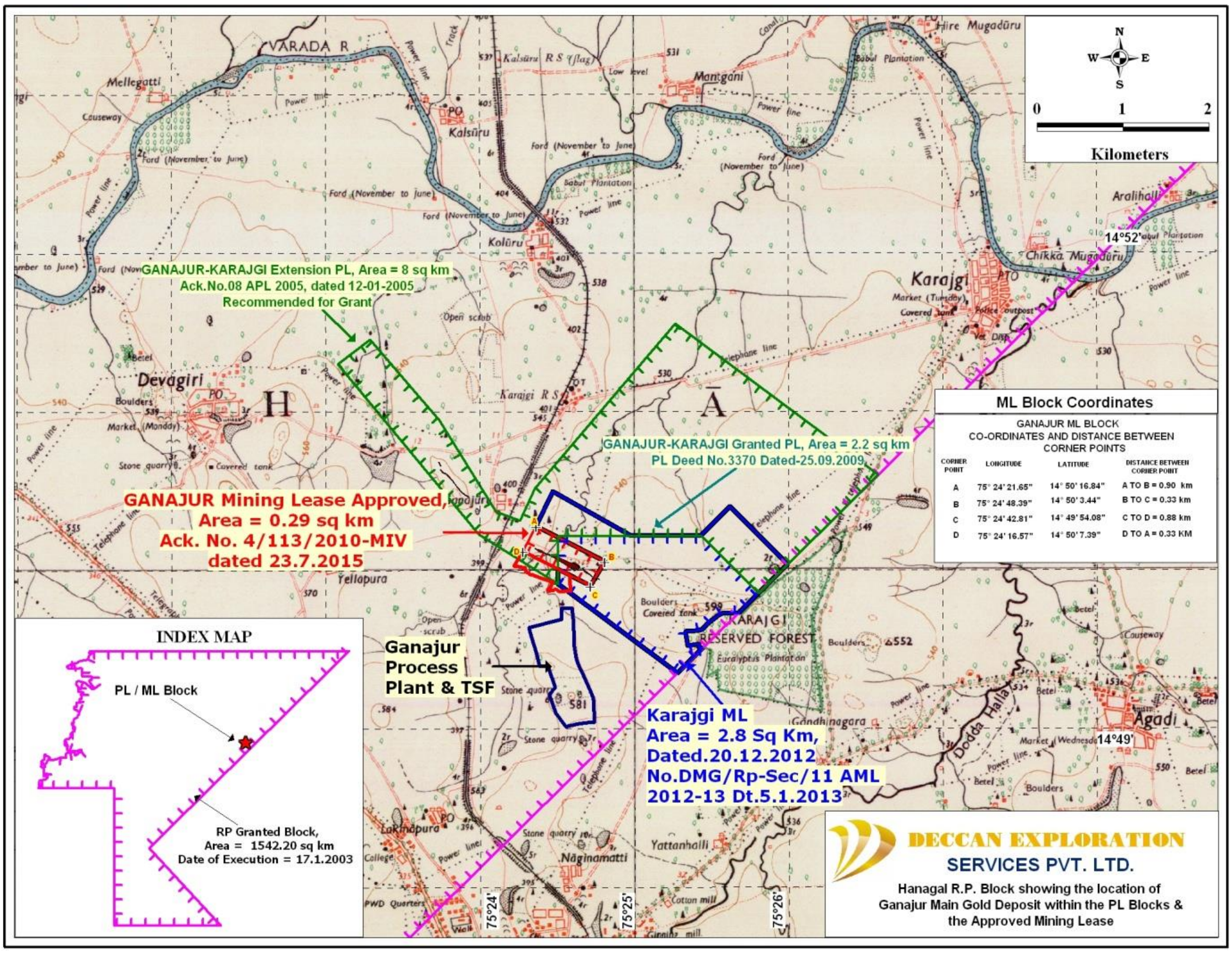
DESPL proposes to establish a 0.30 Mt gold project near Ganajur Village, Haveri Taluk and District in the state of Karnataka. The proposed gold project comprises of gold ore production from the mine and processing of gold ore in the processing plant to produce the gold. The gold processing plant will be located at a distance of 1.5 km south of the Ganajur Gold Ore Mine. The Ganajur Main Gold deposit was a discovery of DESPL as a result of systematic exploration carried out under reconnaissance permit and prospecting licence stages. DESPL's mining lease application over an area of 0.29 km^2 covering the Ganajur Main Gold Deposit. The Ganajur mining lease application in Ganajur Village, Haveri Taluk and District in Karnataka has been approved by the Ministry of Mines, Government of India vide letter no. 4/113/2010-MIV dated 24 July 2015. The approval is per Section 10(A)(2)(B) of the New MMDR Act 2015. Prior to this, the mining lease application for the Ganajur Main Project was recommended by the Government of Karnataka. DESPL is awaiting the final grant order/Letter of Intent from the State Government of Karnataka.

DESPL has been exploring the Ganajur-Karajgi block since the reconnaissance permit which was granted in 2003. Figure 2.1 shows the tenement details and Ganajur mining lease block. Exploration was carried out under different phases as per the international norms. The discovery of an open pitable mineable resource in Ganajur Main has been one of the significant achievements for the company. DESPL's sustained exploration efforts during the last seven years have resulted in overall value addition of the whole Ganajur-Karajgi block in general and Ganajur Main Gold Deposit in particular. Gold resources were also estimated in two other satellite prospects viz the Ganajur SE and Karajgi Main prospect. The exploration data generated so far revealed that this block is a major mineral corridor with substantial potential to host significant ore resource.

Ganajur-Karajgi block is part of Ranibennur group of the late Archaean-Dharwar-Shimogagreenstone belt in the western Dharwar Craton. Gold mineralisation in the area is hosted within sulphidic banded iron formation (BIFs).

The Dharwar-Shimoga Belt projects are presently considered as flagship projects with the Ganajur Main Gold Deposit being the lead project in respect of the planned commencement of mining and commercial gold production in 2018.

Figure 2.1 Ganajur Gold Project and tenement details



2.2 Location, accessibility and infrastructure

The Ganajur Gold Mining Project area is situated near the Ganajur village, 14°49’54.08” – 14°50’16.84” N latitude; 75°24’16.57” – 75°24’48.39” E longitude, forms a part of Survey of India topographic sheet no. 48 N/5 and falls in the jurisdiction of the Haveri Taluk and District in the State of Karnataka. The corner points of the Ganajur mining lease area with UTM coordinates shown in Table 2.1.

Table 2.1 Coordinates of Ganajur mining lease block

Corner	UTM zone	Easting (M)	Northing (M)
A	43	543676.6	1640318.2
B	43	544476.4	1639908.1
C	43	544310.3	1639619.9
D	43	543525.4	1640027.6

**UTM Projection Everest Datum, Zone 43N*

2.2.1 Location of Ganajur Gold Project

The Ganajur Gold Project forms part of Survey of India topographic sheet no. 48 N/5 and falls in the jurisdiction of the Haveri District in the State of Karnataka. Figure 2.2 and Figure 2.3 show the location map. Haveri Town, located on National Highway No. 4, is 335 km by road north of Bengaluru and 100 km south of Dharwar. The centre of the Ganajur mining lease block is located 4.53 km northeast of Haveri Town, and 0.76 km southeast of Ganajur Village. The Ganajur Project is well connected by an all-weather metalled road from Haveri and Ganajur.

Figure 2.2 Location map of the Ganajur Gold Project

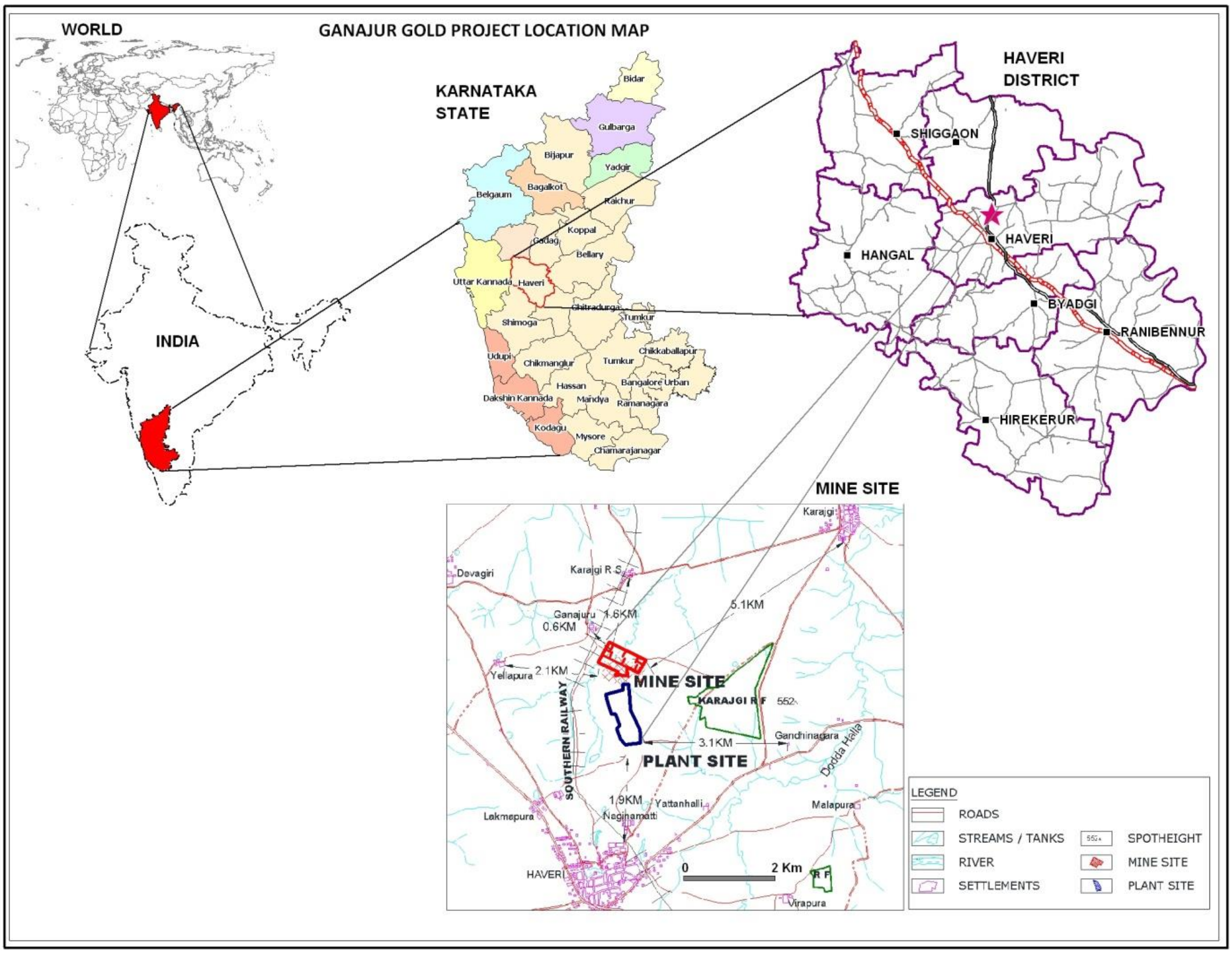
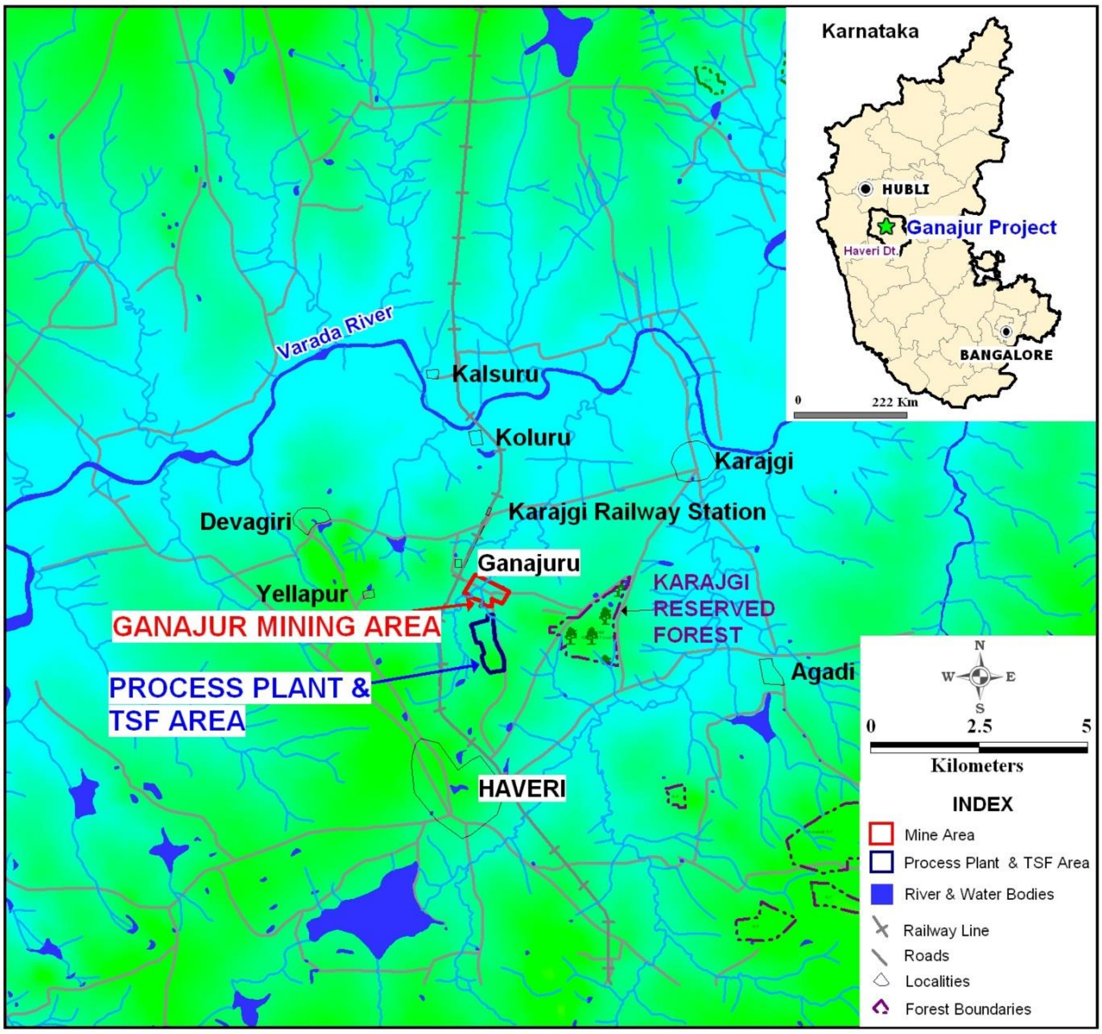


Figure 2.3 Location map of the Ganajur Gold Project



2.2.2 Climate

The Project area falls within a semi-arid sub-tropical region with moderate temperatures and rainfall. The normal annual rainfall for the Haveri district is recorded as 792 mm, as per the Indian Meteorological Department (IMD), Government of India. As per the document “Haveri District at a Glance 2014-2015”, of the district authority, the normal (1951 to 2000) annual rainfall of Haveri District is 793 mm.

The area falls in a semi-arid region with normal maximum temperatures varying from 29.1°C to 37.3°C, reaching the maximum in April. The normal maximum temperature of 30°C and above lasts for five months from February to June. The mean minimum temperatures range from 16.4°C to 22.5°C with temperatures of less than 20°C from November to February. The highest normal temperature observed, 39.6°C, occurred in May; the lowest normal temperature observed, 13.6°C, occurred in December.

2.2.3 Physiography and infrastructure

The Project area and its surrounding area are generally flat to gently undulating terrain and a few northwest-southeast trending small ridges. The elevation of the ground surface varies from 525 mamsl to about 610 mamsl. The highest elevated land lies to the north-northwest of Haveri with a peak point of 627 mamsl near the Project area, except the Karajgi Hill which is 596 mamsl. Most of the Ganajur Main prospect is flat except for Ganajur Hill which is 551 mamsl (Figure 2.4).

Figure 2.4 Ganajur Main Gold Prospect

2.2.4 Drainage

The Ganajur Project and its surrounding area is drained by the Varada River. The Varada River rises in the Varada Moola in Sagara Taluk, in the hill ranges of the Western Ghats in Shimoga district of Karnataka. The river, after flowing for about 25 km eastwards, joins the Tungabhadra River that flows 30 km east of the Project. Tungabhadra is a tributary to the Krishna River, which is one of the major east-flowing rivers in South India. A major stream, Dodda Halla flowing in a southwest-northeast direction, drains most of the southern part of the study area into the Varada River. The drainage is well developed over the study area. Due to the presence of strong orientation of strike joints/fractures in the compact rocks of the area, the drainage tends to follow these weak directions. It also exhibits annular and dendritic drainage patterns. The density of the drainage pattern is moderate. Fracture/lineament controlled drainage is noticed in most of the buffer area.

The main crops grown in the district are jowar, maize, cotton, chilly, paddy, ragi, pulses, groundnut, horse gram, sugarcane and sunflower.

2.3 Transportation

The Project is well connected by road and rail and has well-established infrastructure in place. Ganajur Main prospect is located 6.0 km north of Haveri Town (Figure 2.3) in Haveri District of Karnataka state on National Highway No. 4, connecting Bengaluru and Pune and 1.5 km southeast of Ganajur Village. Haveri is around 335 km from Bengaluru and 100 km south of Dharwar city. The Ganajur Gold Project is connected to Haveri and nearby villages by an all-weather metalled road.

Bengaluru-Pune Broad Gauge Railway on South Central Railway passes through Ganajur Village at a distance of 0.32 km from the northwest (A corner) corner. The Karajgi and Haveri railway stations are 1.85 km and 4.5 km respectively, from the centre of the mining lease block (Figure 2.5). The nearest domestic airport is at Hubli, which is 76 km by road in a north-northwest direction from the Haveri city. Bengaluru is the major international airport and Karwar is the nearest sea port located 190 km west of Haveri.

Figure 2.5 Karajgi Railway Station near the Ganajur Gold Project

2.3.1 Power

The HT 110 KVA powerline is located just 300 m south of the proposed processing plant (Figure 2.6). The nearest power substations are located at Haveri (3.37 km), Agadi (6.42 km), Gandhipura (3.6 km) and Basavanakatti (7.0 km). DESPL’s application for obtaining 5 MW power from the 110 KV line has been sanctioned by Karnataka Power Transmission Corporation Limited (KPTCL), Bengaluru. KPTCL has approved for tapping the required power from Basavanakatti substation at a distance of around 7.0 km.

Figure 2.6 HT110 KVA powerline near the proposed process plant area

2.3.2 Water

The Varada River flows at a distance of 6.5 km north of the gold ore process plant area. It is proposed to pump water from the Varada River for the water requirement of the Project. Karnataka State High Level Clearance Committee (KSHLCC) has approved for drawing 3,000 kl/d of water from Varada River for the project from the Kolur-Kalasur barrage (Figure 2.7). The existing abandoned quarry near the proposed process plant area will be part of the raw water storage facility (Figure 2.8).

Figure 2.7 Kolur-Kalsur barrage across Varada River north of the Project

Figure 2.8 Abandoned quarry near the proposed process plant area earmarked for raw water storage facility

2.3.3 Land

As per the land use pattern of the various facilities of the project a total of 255 acres of land will be required. The State Government through a Government Order (GO) has also approved acquisition of 200 acres of land for the mines and processing plant. The GO has also facilitated land acquisition process through Karnataka Industrial Areas Development Board (KIADB), for which DESPL has submitted application to the KIADB in March 2013. DESPL will also be submitting application for an additional 55 acres of land after obtaining approval from Karnataka Udyog Mitra and SHLCC. Most of the land is private agricultural land, with the remainder being government land. DESPL has obtained consent of more than 85% of the landowners, which complies with the land acquisition procedure. DESPL's long term lease agreement with landowners of the proposed gold mine will be an added advantage in the land acquisition process. Recently, KIADB processed the application for 200 acres of land and then issued a demand letter asking DESPL to remit 40% of the land cost. After payment of this deposit, KIADB will issue primary notification under Sections 3(1), 1(3) and 28(1) of the KIADB Act.

2.4 History

The area around the Ganajur Gold Mining Project is known for ancient artisanal gold mining activity. Chinmulgund, located southeast of Haveri, shows evidence ancient mining activities (i.e. old workings, ancient shafts, adits, waste dumps and pounding marks). Pounding mark structures are also noticed on of Ganajur Main Gold Hill. Ancient rat hole-type of mining excavations, old workings and pounding marks have also been seen at the Karajgi Main prospect. The Geological Survey of India has explored and reported incidence of gold mineralisation in the Karajgi, Chinmulgund and Lakkikoppa areas of Haveri and Hanagal taluks of the Haveri district.

The Ganajur Main Prospect and the surrounding prospects are the discoveries of DESPL and no previous exploration was reported.

2.4.1 Tenement details

DESPL, under the Hanagal reconnaissance permit approved by the Ministry of Mines, Government of India in 2002, and granted by the State Government of Karnataka for 1,542 km^2 covering the Ganajur Main prospect, carried out initial exploration. This reconnaissance campaign included surface geological mapping, regional geochemical study of stream sediments and rock chips, channel sampling, limited RC and DTH hammer drilling. This exploration effort helped to identify and discover several gold-bearing prospects designated as Ganajur Main, South, South East, Central, Karajgi Main, Karajgi East and Hut.

Based on the results obtained during the RP period, DESPL applied for a prospecting licence over an area of 2.2 km^2 (Ganajur-Karajgi prospecting licence). The Government of Karnataka vide CI.157:MMM.2005 dated 10 September 2009 granted the prospecting licence and the same was executed on 25 September 2009. DESPL carried out detailed exploration as per International Standards using multi parametric exploration methods in the granted prospecting licence area. The prospecting licence area included the high potential Ganajur Main Gold Deposit and its adjoining seven gold prospects. The Ganajur Main Gold Prospect has now progressed into a mining project with economic grades of Au.

DESPL recognised the mining potential of the Ganajur Main Gold Prospect and submitted a mining lease application over an area of 29.14 ha covering the Ganajur Main Gold Deposit. The Ganajur mining lease application in Ganajur Village, Haveri Taluk and District in Karnataka was approved by the Ministry of Mines, Government of India vide letter no. 4/113/2010-MIV dated 24 July 2015. The approval is per Section 10(A)(2)(B) of the New MMDR Act 2015 and Section 5(1). Prior to this, the mining lease application for the Ganajur Main Project was recommended by the Government of Karnataka.

DESPL signed a memorandum of understanding with the Government of Karnataka during the Global Investors Meet at Bengaluru on 4 June 2010 for commencement of a 2,000 t/d gold mine and setting up of a 2,000 t/d processing plant at Ganajur Village. Prior to this, the KSHLCC approved the project on 24 May 2011.

2.4.2 Historic resource estimation

There is no record of historical sampling or exploration of the Ganajur Main Gold Deposit prior to that undertaken by DESPL. No historical resource estimation is available for the Ganajur Main Deposit which is a discovery of DESPL. In December 2009, DESPL appointed SRK Consulting (India) Ltd (SRK) to prepare an initial Mineral Resource estimate for the Ganajur Main Gold Deposit. SRK estimated Indicated Resources of 238,300 ounces (oz) and Inferred Resources of 26,700 oz. The Mineral Resource estimate was prepared following the guidelines of the JORC 2004.

DESPL once again commissioned SRK, an internationally acclaimed independent geological consulting firm to undertake a comprehensive scoping studies for the Ganajur Main Gold Project in 2010. The study involved an updating of the Mineral Resource estimate and undertaking a scoping study and preliminary economic assessment for the Ganajur Main Gold Deposit.

SRK submitted an updated Mineral Resource Statement in accordance with the Australasian Code for Reporting of Exploration (JORC 2004). SRK estimated a total of 308,000 oz of gold, out of which 255,000 oz were under the Indicated category. The resource has been estimated up to a depth of 120 m.

The scoping study addressed the mine optimisation, process design, capital expenditure estimates, operating expenditure estimates, project economics (including key risks and opportunities) and future work program.

The study revealed that project economics are technically and economically attractive but could be improved upon considering the fact there are a number of highly prospective gold-bearing targets in the prospecting licence block with distinct possibilities of adding to the existing Mineral Resource.

Encouraged by the outcome of scoping studies by SRK, in 2016 DESPL appointed internationally reputed geological and mining consultant, Snowden, based in Perth, for undertaking a Bankable Feasibility Study (FS) for the Ganajur Gold Project. The FS comprised several disciplines such as resource geology, mine planning, geochemistry, process and metallurgy, plant designing and engineering, tailings disposal and design of the tailing dump, geotechnical engineering, hydrology and hydrogeology, environmental studies etc. The FS is in the final stages of completion.

2.5 Local resources

The Ganajur Project is located in the Haveri District, which is a hub of new developments. It is a relatively new district and the State Government is promoting and supporting it for overall growth and development. Haveri has six industrial estates and 32 industrial sheds. The district has large reserves of chinaclay, sand and building stones. Haveri District is also known for good primary and higher educational institutions. Apart from regular colleges, there are two engineering colleges and one medical college in the district. Devagiri and Karajgi have primary health centres run by the Government of Karnataka. Haveri has a District General Hospital with good facilities.

Haveri and Hubli cities have abundant skilled manpower. Unskilled laborers can be sourced from Ganajur and nearby villages apart from Haveri. Most of the raw materials for the Project can be procured from Haveri and Hubli.

2.6 Adjacent properties

DESPL has submitted a prospecting licence application for gold over an area of 8 km^2 adjacent to the Ganajur mining lease. Apart from this, DESPL has also lodged a mining lease application for gold over an area of 2.8 km^2 which includes all other gold prospects adjacent to the Ganajur Main Gold Deposit. These applications are protected under Section 10A(2)(b) of the MMDR Act 1957 (amended in March 2015). These two tenements are being processed by the Karnataka State Government.

2.7 Legal aspects

The Ganajur Project is subject to the regulations of the Mines and Minerals (Regulation and Development) Act 1957 as amended in March 2015, the Mineral Concession Rules 1960, and the Mineral Conservation and Development Rules 1988 of India (as amended in March 2017).

2.8 Competent Persons

The Mineral Resources and Ore Reserves are reported under the guidelines for the JORC Code (2012. Competent Persons for the Mineral Resource and Ore Reserves are Lynn Olssen and Frank Blanchfield.

Table 2.2 Competent Persons for reporting Mineral Resource and Ore Reserves in accordance with the JORC Code (2012)

Reporting	Name	Qualifications	Company	Position
Mineral Resource	Lynn Olssen	BSc, Post Graduate Cert. Geostatistics, MAusIMM(CP), GAICD	Snowden	General Manager – Geosciences
Ore Reserve	Frank Blanchfield	BEng, FAusIMM	Snowden	Principal Consultant – Mining

2.9 Responsibility for Feasibility Study chapters

The following persons, contractors or consulting groups (Table 2.3) were involved in the completion of the Ganajur Gold Project FS.

Table 2.3 Main contributors to the Ganajur Gold Project FS

Company	Abbreviation	Contributors	Position
Deccan Exploration Services Private Limited	DESPL	Saradchandra Rao Peshwa S.B. Harish Kumar John Fodor	Director – Exploration Exploration Manager Process Consultant
Prime Resources (Pty) Ltd	Prime Resources	Peter Theron Stephan Geyer Dr Bronwyn Grover Jonathon van de Wouw	Director Consultant – Tailings Consultant – Geochemistry Principal Consultant
Snowden Mining Industry Consultants Pty Ltd	Snowden	Frank Blanchfield Tarrant Elkington Matt Cotterell Lynn Olssen Lindsay Farley John Elkington Kath McGuckin	Principal Consultant – Mining General Manager – Mine Planning Principal Consultant – Mining General Manager – Geosciences Senior Consultant – Geosciences General Manager – Corporate Services Principal Consultant – Corporate Services
CPC Project Design Pty Ltd	CPC	Dimitrios Felekis Drew Noble Jacqueline London	Study Manager/Principal Mechanical Engineer Manager Process Engineering Lead Process Engineer

Responsibility for the preparation of each chapter in the FS is listed in Table 2.4.

Table 2.4 Responsibility for FS chapters

Chapter	Description	Responsibility
1	Executive Summary	Snowden
2	Introduction	DESPL
3	Project Approvals	DESPL
4	Geology and Mineral Resource Estimates	Snowden
5	Mineral Processing, Metallurgical Testing and Recovery	John Fodor (DESPL)
6	Mining Engineering and Ore Reserves	Snowden
7	Process Plant and Description	John Fodor (DESPL)/CPC
8	Surface Geotechnical and Tailings Disposal	Prime Resources
9	Project Infrastructure	Prime Resources
10	Marketing Information	DESPL
11	Geochemistry	Prime Resources
12	Hydrogeology and Hydrology	Prime Resources
13	Environmental Studies, Permitting and Social or Community Impact	Prime Resources
14	Cost Analysis	CPC, Snowden, John Fodor (DESPL)
15	Economic Analysis	Snowden
16	Project Implementation	Snowden
17	Risks and Opportunities	Snowden
18	Recommendations, Interpretation and Conclusions	Snowden
19	References	Snowden

3 PROJECT APPROVALS

3.1 Introduction

3.1.1 Structure of minerals and mining in India

The Ministry of Mines (MoM), Government of India is responsible for the entire minerals and mining sector in the country that includes legislation, administration, policy formulation etc. in respect of all mines and minerals other than coal and lignite, natural gas and petroleum, but including offshore minerals. In India, the minerals are classified as minor minerals and major minerals. The power to frame policy and legislation relating to minor minerals is entirely delegated to the State Governments while policy and legislation relating to the major minerals is dealt by the MoM. All the mineral legislations in the country conform to the provisions of the Mines and Minerals (Development & Regulation) Act (MMDR Act), 1957. Indian Bureau of Mines (IBM), a subordinate office of the MoM is mainly responsible for regulation of mining in the country. Mineral concessions in India are granted to Indian nationals or entities incorporated in India only.

3.1.2 Mineral and mining sector legislation in India

India's mineral and mining sector operates under a federal structure wherein the Central Government formulates the legislation for all minerals except the minor minerals and the State Governments formulate legislation for minerals classified as minor minerals. Government of India permits 100% Foreign Direct Investment (FDI) in exploration, mining, mineral processing and metallurgy through the automatic route, by way of equity participation in a company incorporated in India, for all non-fuel and non-atomic minerals.

India has written legal and constitutional framework to manage the mineral sector. National Mineral Policy provides the direction for mineral sector. Management of mining sector is the responsibility of the Central Government and the State Governments. The State Governments are the owners of minerals occurring onshore. The Constitution bestows power to the Parliament to enact legislation relating to the mining and the States are bound by the Central legislation. The Mines and Mineral (Development and Regulation) Act 1957 is the central legislation in force for regulation of mining operations. The MMDR Act enables all the State Governments to exercise their powers within a uniform national framework. The State Governments, as owners of onshore minerals, grant mineral concessions and collect royalty, dead rent and fees as per the provisions of MMDR Act, 1957.

3.1.3 Legislation

The MMDR Act, 1957, is the principle legislation that governs the mineral and mining sector in India. The MMDR Act, 1957, together with the following rules and legislation comprises the legal framework for this sector (Figure 3.1).

Mineral Concession Rules (MCR), 1960

The MCR, 1960, defines the process of grant of mineral concessions as per the provisions of Section 13 of the MMDR Act, 1957. The rules lay down the process and timelines for grant of concessions, disposal and refusal of applications and the basic conduct of accounts, registers and information reports.

Mineral Conservation & Development Rules (MCDR), 1988

The MCDR, 1988, prescribes guidelines for the conservation and development of minerals as per the provisions of Section 18 of the MMDR Act, 1957. The rules prescribe procedures for carrying out prospecting and mining operations and the general requirements relating to preparation of mining and prospecting plans and filing of notices and returns. The rules also cover guidelines for protection of the environment.

Mines Act, 1952

The Mines Act, 1952 prescribes the laws relating to the regulation of labour safety in mines, regulations for carrying out mining operations and management of mines. It lays down the basic provisions for health and safety of people employed in mines and regulates their working conditions. It also has provisions relating to inspection of mines and procedure of reporting to be followed.

Mines Rules, 1955

The Mines Rules, 1955, defines the framework for medical examination of persons employed or to be employed in mines, basic health and sanitation provisions and welfare amenities for the miners and their families.

National Mineral Policy, 2008

The National Mineral Policy (NMP) was notified in 2008. It recommends measures like assured right to next stage mineral concession, transferability of mineral concessions and transparency in allotment of concessions, in order to reduce delays which are seen as impediments to investment and technology flows in the mining sector in India. The Mineral Policy also seeks to develop a Sustainable Development Framework for optimum utilisation of the country's natural mineral resources for the industrial growth in the country and at the same time improving the life of people living in the mining areas, which are generally located in the backward and tribal regions of the country.

Recent developments

The MMDR Act, 1957, was amended through the MMDR Amendment Act, 2015. The amendment that came into force on 12 January 2015 has ushered in the regime of transparent and non-discretionary grant of mineral concessions. The major features of the MMDR Amendment Act are:

- Removal of discretion; auction to be sole method of allotment.
- Impetus to the mining sector: Mining Leases (MLs) will now be granted for a term of 50 years.
- Removal of discretion: The mineral concessions will now be granted through auction process and will not be renewed after the expiry of the concession.
- The Central Government will prescribe the terms and conditions for grant of mineral concessions through competitive bidding.
- Reconnaissance Permits (RPs) are granted on non-exclusive basis. However, the rights of the RP holders prior to the amendment of this act are fully protected as per Section 10A(2)(b) of the Act.
- The Central Government has the authority to reserve mines for specific end uses at its discretion.
- Safeguarding interest of affected persons: District Mineral Foundation is to be set up in each mineral bearing district for local area development.
- Encouraging exploration and investment: National Mineral Exploration Trust is to be set up for regional and detailed exploration in the country.

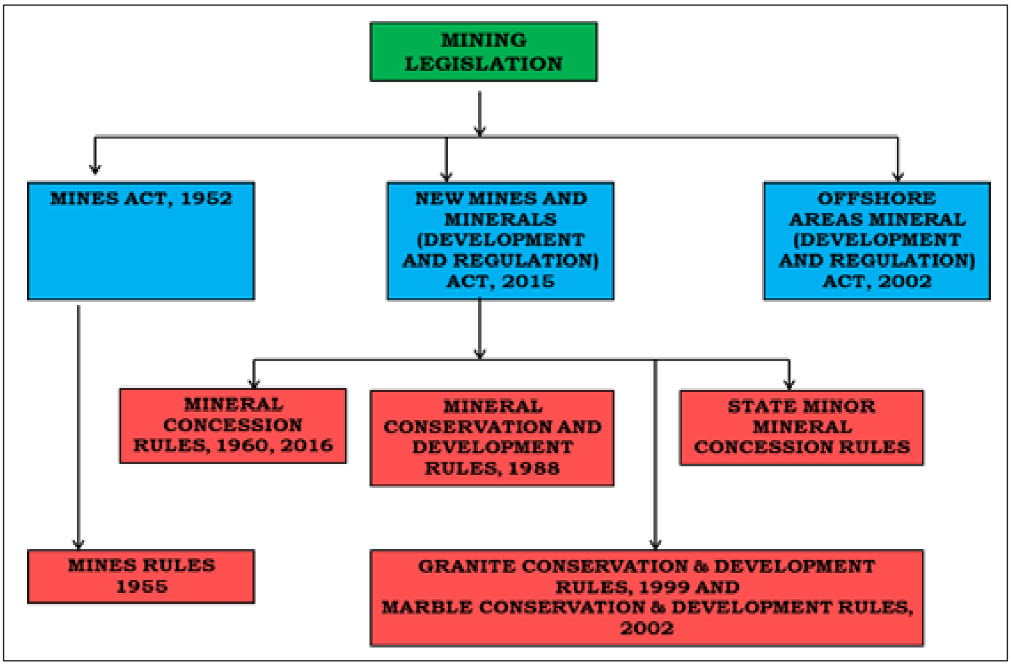
Government of India is in the process of simplifying and updating the subordinate legislation relating to the mineral and mining sector in India that includes necessary amendments to MCR, 1960, and MCDR, 1988. As a part of this initiative, the Central Government has notified the following rules for implementation of provisions of the MMDR Amendment Act, 2015:

1) **Minerals (Evidence of Mineral Contents) Rules, 2015**: Rules that prescribe procedures to be followed for conducting the exploration to determine mineral content so that the mineral blocks could be taken up for auction of mineral concessions.
2) **Mineral (Auction) Rules, 2015**: Rules that detail the process to be followed for auction with respect to grant of minerals concessions.
3) **Mineral (Non-exclusive Reconnaissance Permits) Rules, 2015**: Rules that detail the process to be followed for grant of Non-exclusive Reconnaissance Permit.

4) **National Mineral Exploration Trust Rules, 2015**: Rules that detail the objectives, functions, operations of the National Mineral Exploration Trust.

5) **The Minerals (other than Atomic and Hydro Carbons Energy Minerals) Concession Rules, 2016.**

Figure 3.1 Mineral Conservation and Development (Amendment) Rules, 2017



3.1.4 Classification of minerals in India

Gold is classified as a Major Mineral and is placed under Part C, First Schedule of MMDR Act 1957.

3.1.5 Surface rights and compensation

A mineral right holder is required to exercise his rights so as to minimise the impact on the interests of any lawful occupier of the land.

3.2 Ganajur Mining Lease application

Based on the above overview on the legislation regarding access to mineral rights in India, DESPL has secured a ML for the Ganajur Gold Project from the MoM, with final approval from the Kanataka State government pending. The historical process on obtaining the Mining Lease (ML) is outlined below:

1) DESPL executed the Hanagal RP over an area of 1,542 km^2 under RP Deed no. 24 dated 17 January 2003. Ganajur Main and other satellite prospects were discovered in the RP area.

2) DESPL applied for two Prospecting Licences (PLs): (a) 2.3 km^2, ack. no. 40 APL 03/13314 dated 20 October 2003; and (b) 8 km^2, ack. no. 08 APL 05/15340 dated 17 January 2005.

3) The first PL Deed no. 3370 was executed on 25 September 2009 for a 2.2 km^2 area. This PL includes the Ganajur Main Gold deposit.

4) DESPL files ML application on 8 June 2006 vide ack. no. 567 AML 06/3389.

5) 25 October 2010: Director, DMG, Karnataka recommended the ML application to the State Government.

6) 8 November 2010: The Government of Karnataka recommended to the MoM for approval of the ML.

7) 24 April 2015: Letter no. N-11011/20/93/-CCOM/1999 from Controller General, Indian Bureau of Mines (IBM) Nagpur to the Secretary Mines, MoM, Government of India indicating that the mineral resource satisfies UNFC guidelines and scientific Mining Plan can be prepared for the Ganajur Gold Deposit.

8) ML application was approved on 24 July 2015, vide letter no. 4/113/2010-M.IV by MoM, GOI approving the grant of Mining Lease over an area of 72 acres for a period of 50 years for the Ganajur Gold Project, DESPL. The approval is as per Section 10 A(2)(b) of the MMDR Act 2015.

9) Grant order/Letter of Intent from the Karnataka State Government is awaited.

10) State Government will issue ML grant order (As per section 5(2)(b)(ii) of the MMDR amendment Act 2015.

11) All the applications that were filed by DESPL have originated from the RP and hence are protected under Section 10A(2)(b) of the MMDR Act 1957 (amended in 2015).

12) As per this act (b) where before the commencement of the said Ordinance a RP or PL has been granted in respect of any land for any mineral, the .permit holder or the licensee shall have a right for obtaining a PL followed by a ML, or a ML, as the case may be, in respect of that mineral in that land, if the State Government is satisfied that the permit holder or the licensee, as the case maybe, has undertaken reconnaissance operations or prospecting operations, as the case may be, to establish the existence of mineral contents on such land in accordance with such parameters as may be prescribed by the Central Government.

3.3 Land acquisition

A total area of 255 acres of land has been currently estimated for the implementation of the Ganajur Project.

The State Government, through a Government Order (GO), has approved the acquisition of 200 acres of land via the submission of an application by DESPL to the KIADB in March 2013. The procedure for the acquisitions of lands for a Single Unit Complex (SUC) as per the KIAD Act is outlined below:

- Identification of the lands by the promoters for their proposed projects in tune with the Government guidelines
- Approval of the projects by the competent authority – State High Level Clearance Committee (SHLCC)/State Level Single Window Clearance Committee (SLSWCC)
- Filing the application to KIADB along with the required documents and also the details of the lands
- The Project proponents are required to remit 40% of the tentative cost of land along with Board service charges before submission of preliminary notifications U/s - 3(1), 1(3) and 28(1) of the KIAD Act, 1966 to the Government's Commerce and Industries Department for approval
- The Project proponents are required to remit balance (60%) of the tentative cost of land along with Board service charges before submission of final notification U/s 28(4) of the KIAD Act, 1966 to the Government's Commerce and Industries Department for approval
- After publication of final notification in the Karnataka Gazette, the Price Advisory Committee under the chairmanship of the Deputy Commissioner of the concerned District will fix the rate of compensation payable for the lands acquired, and approval of the KIAD Board will be obtained for the said rates.
- After payment of the difference of amount along with Board service charges as per the cost of land approved by the Board, land will be allotted on lease-cum-sale basis.

DESPL will be issuing an application for the additional 55 acres of land after obtaining approval from Karnataka Udyog Mitra and SHLCC. Most of the land is private agricultural land, with the remainder being government land. DESPL has obtained consent of more than 85% of the landowners that complies with the land acquisition procedure. DESPL's long-term lease agreement with landowners of the proposed gold mine will be an added advantage in the land acquisition process.

Recently, KIADB has processed the application for 200 acres of land and has issued a demand letter asking DESPL to remit 40% of the land cost. After payment of this deposit KIADB will issue primary notification under Sections 3(1), 1(3) and 28(1) of the KIADB Act. The final compensation will be fixed by the District Commissioner after consultations with the current landowners and the project proponent.

3.4 Environmental Clearance

The Ministry of Environment, Forest and Climate Change (MOEF) is the nodal agency in the administrative structure of the Central Government for the planning, promotion, coordination and overseeing the implementation of India's environmental and forestry policies and programs.

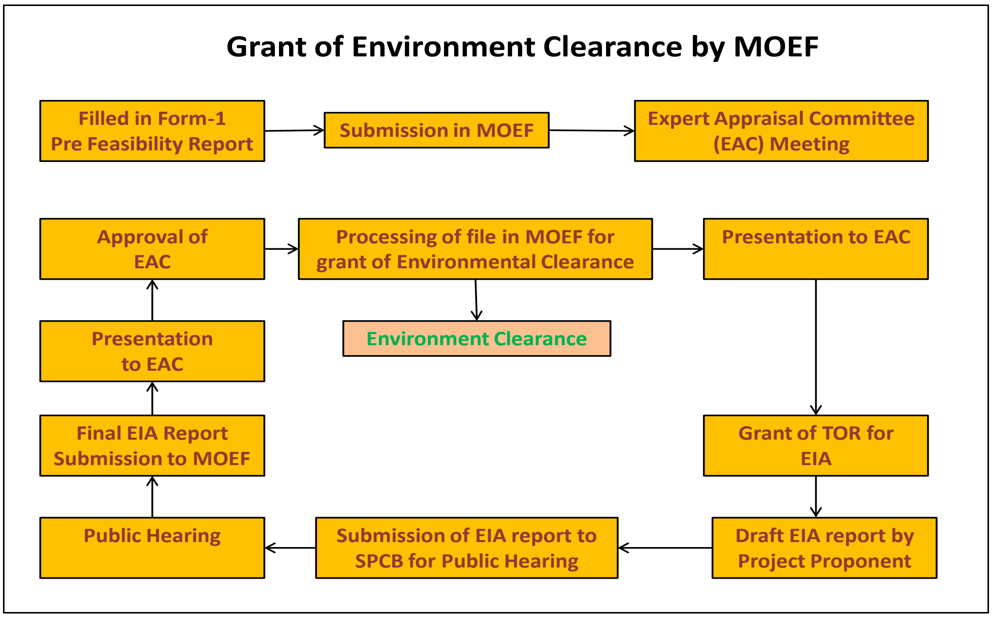
The primary concerns of the MOEF are the implementation of policies and programs relating to conservation of the country's natural resources including its lakes and rivers, its biodiversity, forests and wildlife, ensuring the welfare of animals, and the prevention and abatement of pollution. While implementing these policies and programs, the MOEF is guided by the principle of sustainable development and enhancement of human wellbeing.

The procedure for obtaining Environmental Clearance for a project is presented in Figure 3.2.

After obtaining the Environmental Clearance (EC), DESPL will then obtain a Consent for Establishment (CFE) and Consent for Operation (CFO) from the State Pollution Control Board.

At present, DESPL has submitted an application for obtaining the EC for the processing plant. The MOEF has granted the Terms of Reference (TOR) for the process plant.

Figure 3.2 Granting of Environmental Clearance by the MOEF



3.5 Miscellaneous approvals

The project infrastructure items listed below are currently being processed for approval by local and state government agencies responsible for the Ganajur Gold Project area:

- The grid power supply from the Kanataka State Power Authority
- The water abstraction from the Varada River and pipeline corridor to the plant site
- New road access north and south of the ML
- Pipeline in culvert beneath the railway line.

4 GEOLOGY AND MINERAL RESOURCE ESTIMATE

4.1 Introduction

Snowden carried out the Mineral Resource estimate for the Ganajur Main Gold Deposit during August 2016 on behalf of Deccan Exploration Services Private Limited (DESPL). The previous resource estimate was completed by SRK in August 2011 (SRK, 2011).

Lynn Olssen (General Manager Geosciences, Snowden) visited the Ganajur site in July 2016, observing the core yard and selected diamond drill core, outcropping mineralisation, diamond drill rig, drillhole collars and general site layout. Time was also spent in the DESPL office in Bangalore where geological maps, regional and local geology and procedures were discussed.

The main laboratory used for assaying, Shiva Analyticals (India) Private Limited in Bangalore (Shiva), was also visited by Lynn Olssen. The facilities were found to be well organised and procedures adequate. Snowden recommends that the use of a scoop for subsampling the pulps be revised and a small rotary splitter used instead. Given the nature of the mineralisation, this is not likely to cause any material issues with the quality of the data.

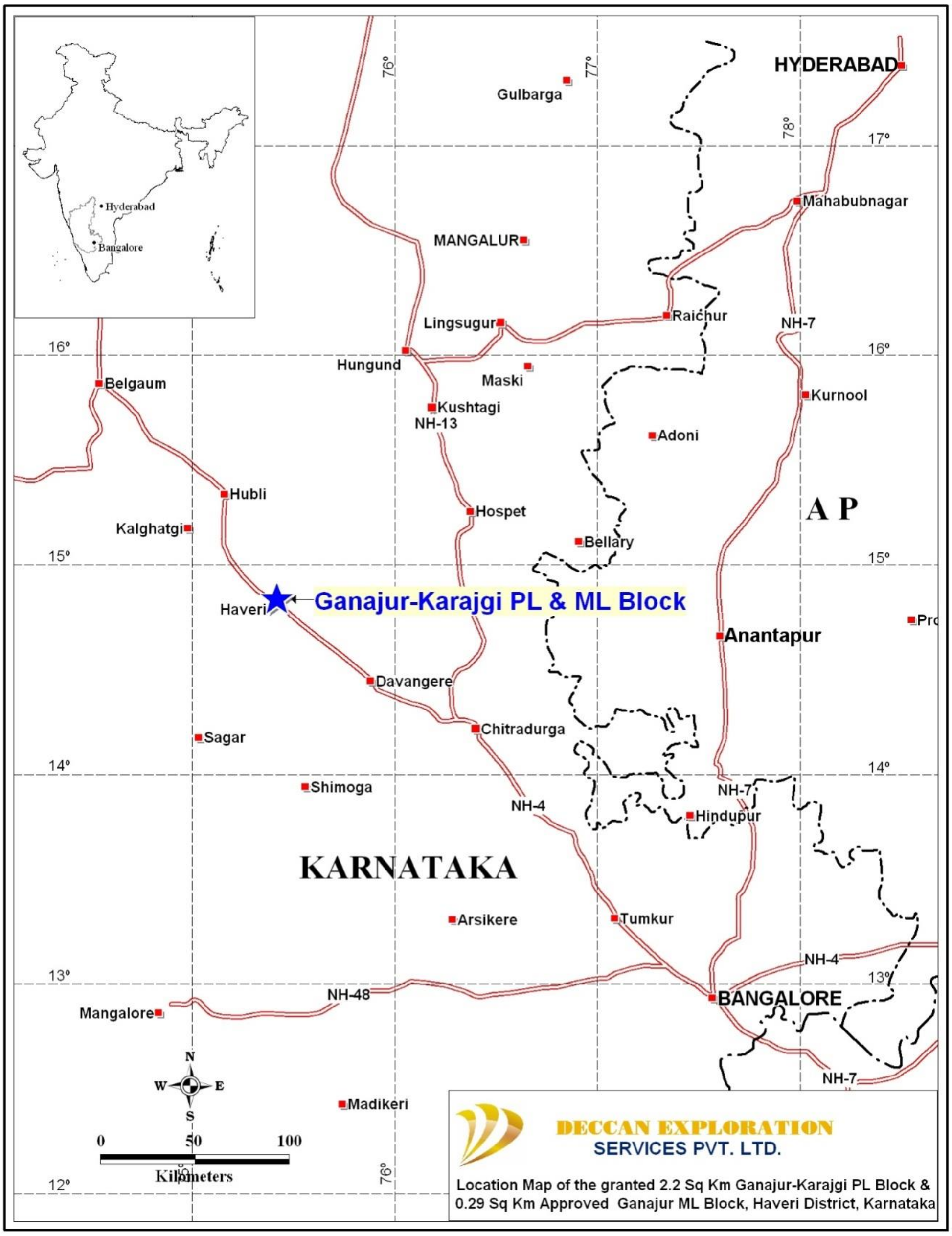
4.2 Project description and location

The Ganajur Main Gold Deposit is located within the Ganajur-Karajgi PL block which forms part of Survey of India topographic sheet no. 48 N/5 and falls in the jurisdiction of the Haveri taluk and district in the State of Karnataka. The Ganajur Karajgi PL was granted to DESPL in 2009. DESPL explored the PL block till 2014 and the PL tenure has since expired. DESPL has since applied for and been granted a ML over the Ganajur Main Gold Deposit area.

Haveri town is located on National Highway No. 4 and is 350 km northwest of Bangalore and 100 km south of Dharwar. The centre of the Ganajur-Karajgi PL block is located 6 km northeast of Haveri town and 1.5 km southeast of Ganajur Village. The Ganajur-Karajgi PL block is well connected by an all-weather metalled road from Haveri and Ganajur.

The Bangalore-Pune Broad gauge line on South Central Railway passes through Ganajur Village at a distance of 0.5 km from the northwest corner of the PL block. The nearest railway stations are Karajgi and Haveri situated on the South Central Railway and are located at a distance of 4 km and 6 km respectively from the centre of the PL. The location of the PL and ML block is shown in Figure 4.1.

Figure 4.1 Location map of Ganajur-Karajgi PL block and ML block



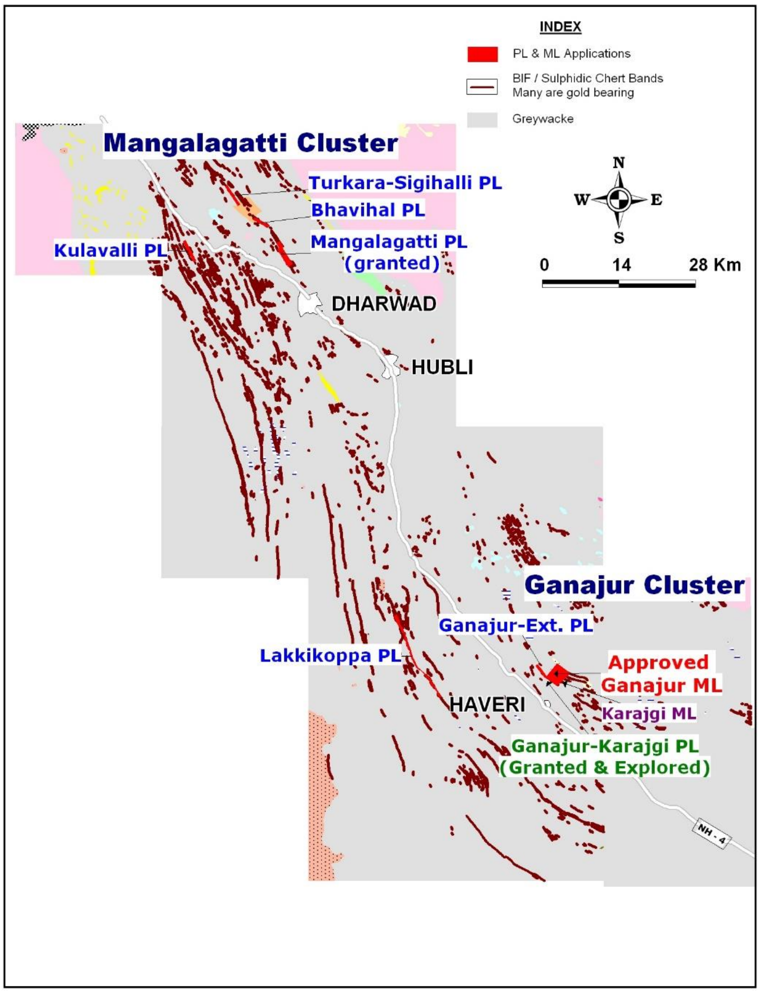
4.2.1 Tenement status and history

DESPL has been exploring the Ganajur-Karajgi block since the RP was granted in 2004 under different phases as per international norms. The exploration efforts of DESPL resulted in locating several significant gold prospects that include Ganajur Main Prospect, a discovery of DESPL, and the adjacent satellite gold prospects, viz., Ganajur East, Ganajur South, Ganajur South East, Ganajur Central, Karajgi Main, Karajgi East and Hut prospects collectively called as Ganajur-Karajgi cluster. DESPL recognised the mining potential of the Ganajur Main Prospect and other surrounding prospects and applied for a PL over an area of 2.2 km^2 which was granted by the Government of Karnataka vide CI.157:MMM.2005 dated 10 September 2009. After the grant of the PL, DESPL further explored the important Ganajur Main Gold Deposit and other prospects extensively. The tenure of the PL expired in 2014 (Figure 4.2).

DESPL subsequently applied for a ML over an area of 29 ha covering the Ganajur Main Gold Deposit. The application for the grant of ML was recommended by the State Government of Karnataka to the Ministry of Mines (MoM), Government of India, New Delhi for approval. DESPL, has also signed an MOU with the Government of Karnataka during the Global Investors Meet at Bangalore in 2010 for commencement of a 2,000 t/d gold mine and setting up of a 2,000 t/d processing plant at Ganajur Village. The KSHLCC has also approved the Project.

The Ganajur ML application was approved by the MoM, Government of India on 24 July 2015 over an area of 72 acres (29 ha or 0.29 km^2) for a period of 50 years. The final approval in the form of a Letter of Intent/Grant Letter is awaited from the Karnataka State Government.

Figure 4.2 Map showing Dharwar-Shimoga Basin, tenements and applications

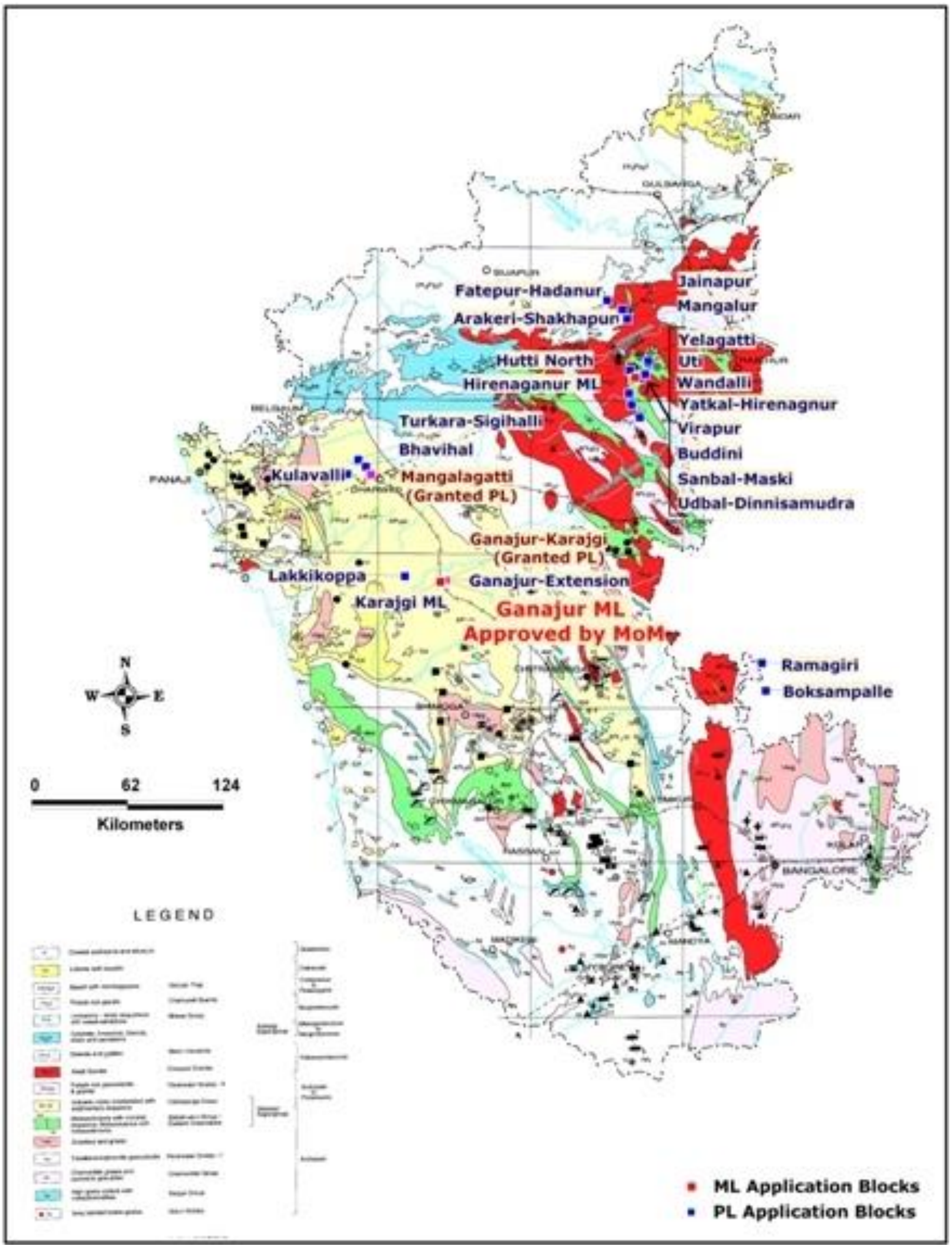


4.3 Geology

The following section is based on documentation provided by DESPL.

The Ganajur-Karajgi Block is part of Ranibennur group of the late Archaean-Dharwar-Shimoga greenstone belt in the western Dharwar Craton (Figure 4.3).

Figure 4.3 Geological map of the Karnataka State



Source: DESPL

4.3.1 Regional geology

The Ganajur-Karajgi Block area is located in the eastern part of the Dharwar-Shimoga (D-S) belt (also referred to as the Dharwar-Shimoga basin) in the foreland region of the Western Dharwar craton (WDC). The D-S basin constitutes the western part of the Late-Archaean Dharwar craton in South India. Ancient artisanal gold mining is known to have occurred within the D-S basin, such as the case of Chinmulgund and Karajgi which have been explored by the Geological Survey of India (GSI). It is one of the largest exploration grounds for banded iron formation (BIF) hosted gold mineralisation and contains a myriad of banded cherty iron formations amidst greywacke. The BIFs in the western part of the area are magnetite bearing, whereas in the eastern part they are sulphidic and also auriferous in places.

The WDC is occupied by vast areas of peninsular gneiss along with two prominent super belts. The volcano-sedimentary sequence (2.9 Ga to 2.6 Ga) in Bababudan-Western Ghats-Shimoga-Chitradurga-Gadag are collectively called the Dharwar Super Group. In the southern part, there is a group of narrow linear schist belts belonging to the older Sargur group. In general, the greenstone belts of WDC consist of ultramafic-mafic sequences, basalt-andesite-dacite-rhyolite (BADR), banded iron and manganese formations, cherts, conglomerates, quartzites, greywackes and phyllites. The regional stratigraphic sequence for the WDC is shown in Figure 4.4.

Dharwar Super Group consists of two groups; the lower Bababudan Group (BG) and the upper Chitradurga Group (CG). The super group is exposed in two large schist belts that may be called superbelts: (1) the Bababudan-Western Ghats-Shimoga; and (2) Chitradurga-Gadag. The BG includes oligomictic conglomerate, quartzite, phyllite, mafic-felsic volcanic rocks, tuffs and thick sequence of BIFs.

The CG comprises polymictic conglomerates, greywackes, argillites and limestones with intercalations of mafic-felsic volcanic rocks and BIFs (Chadwick *et al.,* 1991). The Ganajur Main Gold deposit and its adjoining prospects are part of the Ranibennur Formations belonging to the CG in the Shimoga greenstone belt.

The rocks of the Shimoga greenstone belt show evidence of pervasive low-grade greenschist facies metamorphism with relatively higher-grade greenschist-amphibolite transition facies located along particular tectonic axes. The overall mineralogical assemblage reflects regional metamorphism spanning from lower greenschist (chlorite-biotite) to upper greenschist-amphibolite facies (Swami Nath and Ramakrishnan, 1981).

The most important and characteristic feature of the Dharwar-Shimoga basin is the abundance of BIF units. There are several hundred BIF bands, most of which are sulphidic and generally poor in magnetite/hematite. The BIFS are found within the greywacke and ferruginous greywacke, with later dykes which have intruded these units.

The Ganajur-Karajgi block, located northwest of the Honnali dome of the Shimoga greenstone belt, is characterised by cherty and sulphidic BIFs, which are auriferous in places and are associated with deeply weathered greywackes.

The regional trends of bedding and foliation are parallel and vary from northwest to west-northwest with steep north-easterly dips. Rocks are deformed into isoclinal folds, with steeply dipping axial planes. The folds are defined by BIF bands, which act as marker horizons. A number of north-northwest to northwest trending shear zones occur within the basin. Shearing has brought about fracturing of the competent BIF bands and associated acid volcanics.

Figure 4.4 Regional stratigraphic sequence for the WDC

Proterozoic mafic dykes				
Chamockites (2500 Ma to 2600 Ma)				
Younger granites (2600 Ma)				
DHARWAR SUPERGROUP (2600 Ma to 2800 Ma)	**CHITRADURGA GRROUP**	Ranibennur Subgroup	Greywackes with BIF, polymict conglomerate, mafic-felsic volcanics	
		Vanivilas Subgroup	Mafic-felsic volcanics with BIF, phyllites (basin centre)	Manganese and iron formations, stromatolitic carbonates, biogeniccherts, pelites, quartzites and polymict conglomerates (basin margin)
			Talya/Kaldurga Conglomerate = Metabasalts and siliceous phyllites of Jagar valley	
	-------------------------Disconformity -------------------------			
	BABABUDAN GROUP	**Mulaingiri Formation**	BIF with phyllites and rare ultramafic-mafic sills	
		Santaveri Formation	Metabasalts, felsic volcanics (Galipuje felsite) ultramafic schists, layered basic complexes, siliceous phyllites, cross-bedded quartzite (Kaimara, Tanigebail)	
		Allampura Formation	Metabasalts, gabbros, ultramafic schists, local BIF, phyllites, cross-bedded quartzite (Lakya)	
		Kalasapura Formation	Metabasalts, gabbros, ultramafic schists, phyllites, quartzites, basal quartz pebble conglomerate (Kartikere Conglomerate)	
	-------------- Deformed angular unconformity ---------------------			
		Peninsular Gneiss with trondjhemite-granodiorite plutons (>3000 Ma)		
		-----------------Intrusive/Tectonic Contact ----------------		
SARGUR GROUP (3100 Ma to 3300 Ma)		Ultramafic-mafic layered complexes, tholeiitic amphibolites, komatiites, BIF		
		Quartzites, pelites, marbles and calc-silicate rocks		
		--------------- Intrusive/Tectonic Contact------------------		
		Gorur Gneiss (3300 Ma to 3400 Ma)		

Source: Geology of India (Volume 1) by M. Ramakrishana and R. Vaidyanadhan. Published by Geological Society of India, Bangalore, 2008.

4.3.2 Local geology of the Ganajur-Karajgi cluster

The Ganajur-Karajgi cluster predominantly comprises greywacke and interbedded banded sulphidic chert (i.e. BIF). The general strike direction of the banded sulphidic chert is northwest and dips at 35° to 50° towards the northeast. The litho-stratigraphy of the area is as follows:

- Dolerite dykes.
- Quartz veins.
- Basic intrusive (gabbro and dolerite).
- Late Archaean Dharwar Super group, Chitradurga Group/Ranibennur Formation:
  - greywacke (quartz-chlorite-biotite schist), sericite-chlorite schist/phyllite (“shale”)

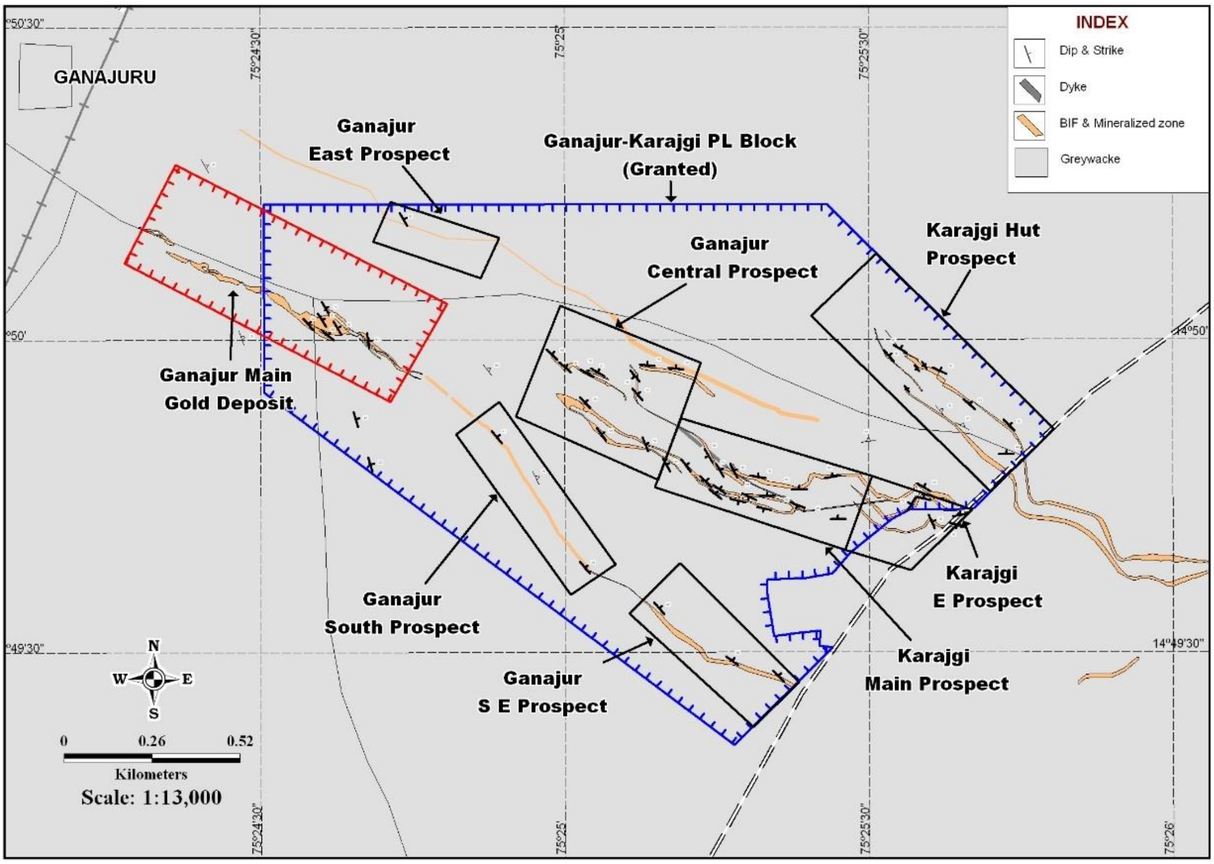
– banded sulphidic-magnetite-quartzite inter-banded with narrow bands and lenses of felsic, intermediate and basic volcanic rocks and greywacke.

Gold mineralisation is confined to the sulphide facies of the BIF, and the mineralisation is both syngenetic, stratabound and epigenetic. The gold mineralisation is associated with strong sulphidation, chlorite-sericite and carbonate alteration along with minor biotite. The mineralised zones are characterised by the presence of significant sulphide minerals (10% to 12%) such as pyrite, chalcopyrite and arsenopyrite. A positive correlation is noticed between gold values and the amount of sulphides.

The Ganajur Main Gold Deposit (Figure 4.5) is hosted in a banded cherty sulphidic iron formation located at the hinge portion of a north-westerly plunging regional antiformal structure. The mineralised body is bound by a thick sequence of greywacke.

The gold mineralisation in the Ganajur Main deposit is associated with a deformed iron formation hosted in a polydeformed paragneiss sequence. The gold mineralisation is characterised by intense shearing, strong sulphide mineralisation, silica breccia and quartz carbonate veining developed within a sulphidic chert unit. The intensity of shearing, hydrothermal alteration and gold and sulphide mineralisation is more towards the footwall contact. The main gold zones form a moderately to steeply dipping tabular body trending northwest to north-northwest and dipping northeast. Fractures are filled with remobilised silica or quartz carbonate veinlets. The amount of gold is directly proportional to the amount of sulphides. Mineralisation dominantly occurs as disseminations, lumps, nuggets or veinlets. However, sulphide enrichment up to 30% was noticed in several drillholes. The mineralised zone closer to the surface is highly oxidised, sheared, gossanised and shows a baking effect.

Figure 4.5 Geological map of the Ganajur-Karajgi cluster



Source: DESPL

Banded iron formations

Banded ferruginous chert (BFC) (Figure 4.6), banded sulphidic chert (BSC) and banded magnetite quartzite (BMQ) are collectively called “BIFs”. These units are important as far as gold mineralisation is concerned. The BFC units are discontinuously exposed in the area at many places for lengths of 10 m to 500 m. Visually the BIF exhibits fine to very fine box works due to the disintegration of sulphides and at places is traversed by an anastomising network of hair-like quartz veinlets. Ash grey coloured quartzite is exposed at places in the Ganajur Main Gold Deposit and is relatively poor in sulphides and gold mineralisation. BMQ is exposed towards the east and southeast of the Ganajur Main Gold Deposit. Contacts between greywacke and the BIF are generally characterised by the presence of intense brecciation due to shearing and quartz-carbonate veining.

Figure 4.6 BIF exposed in the Ganajur Main Prospect

Source: DESPL

Greywacke

Greywackes, the predominant country rock in the area, are seen outcropping at several places and contains minor sulphide disseminations (Figure 4.7). The rocks are occasionally intercalated with thin bands of acid volcanic tuffs, pink shales and white sedimentary clay bands and at places with BIF. Greywacke at the contact of BIF is sheared, brecciated and traversed by a network of quartz-carbonate veins. The strike of schistosity varies between northeast (030°) to north-northwest (300°) and dips at an angle of 35° to 80° east.

The greywacke consists of quartz, plagioclase feldspar and rock fragments (quartzite, chert, siltstone, shale, mica schist, intermediate and acid volcanics) as the framework with a host of other minerals such as chlorite, sericite, biotite, calcite, siderite, epidote, sphene, leucoxene, pyrite, occupying the matrix.

Figure 4.7 Greywacke exposed in the Varada River bank and Ganajur quarry section

Source: DESPL

Quartz veins

The project area has two types of quartz veins; white and blue opalescent quartz that shows rare to moderate presence of sulphides. The quartz veins are either parallel to the bedding/schistosity or discordant to the planar fabric. At times, mostly in the mineralised zones, thin, hair-like quartz veinlets are seen traversing the BIFs in all directions.

Dolerite

Narrow discontinuous exposures of weathered dolerite dykes are observed cutting across the major litho-units. The trend of the dykes is generally north-northwest to south-southeast.

4.3.3 Geological interpretation for the Ganajur Main Gold deposit

DESPL carried out the geological interpretation using the geological logging of the chert (BIF) domain, and the gold assays at a nominal 0.3 g/t Au cut-off to define the mineralised envelopes. The mineralised domain is typically restricted to the chert with 1 m to 2 m of halo mineralisation in places and occasional small areas of unmineralised chert.

In addition, DESPL used the geological logging to interpret the base for the oxidation. There is a relatively sharp boundary between the oxide and sulphide rock with a minor layer of transition zone that cannot be shown as a distinct unit and hence was included in the oxide zone for the interpretation.

4.4 Drilling, sampling and assaying

Snowden reviewed the written procedures for drilling, sampling, assaying and logging as provided by DESPL and observed the procedures being carried out on site. Procedures reviewed include:

- Diamond Core Drilling and Sampling Procedure (DESPL, 2016a)
- Reverse Circulation (R.C.) Drilling Procedures ‘R.C. (DESPL, 2016b)
- Shiva Analyticals Sample Preparation and Analysis Protocol (Shiva, 2016).

4.4.1 Drilling

DESPL has carried out the following types of drilling and sampling over the Ganajur lease:

- Diamond core drilling (DDH) – 83 drillholes for 5,121.64 m
- Reverse circulation drilling (RC) – 22 drillholes for 1,219 m
- Trenching – 59 trenches for 1,141.8 m
- Down-the-hole hammer drilling (DTH) – 12 drillholes for 649 m.

Diamond core drilling

Diamond core drilling at Ganajur was carried out with an Atlas Copco-CT-14 drill rig, and Boart Longyear made DB525, DB520 and Dynatech drill rig. Drilling was carried out using HQ3 sized wireline accessories with a triple tube core barrel to achieve maximum core recovery.

The azimuth of the diamond core drilling was fixed using a Brunton compass and a line drawn on the ground in the direction of the drilling to orient the drill rig (Figure 4.8). The rig was aligned along the line and the angle/inclination checked using a Brunton compass.

After completion of each drillhole, the casing was closed with a metal or PVC cap. The collar was then covered by a cement cap with the hole number written on it (Figure 4.9).

Figure 4.8 Diamond drill rig showing orientation mark-up

Figure 4.9 Drillhole location showing cement capping

Reverse circulation

The RC drilling program was carried out in two phases in 2005 and 2009 respectively. In 2005, DESPL used its own SPARR (Sparr Engineering, Bangalore) rig while the drilling in 2009 was undertaken by APC Drilling and Construction of Namakkal, India, using JCR Drillsol drilling equipment and compressor.

The RC drilling was oriented using the same process as the diamond core drilling described above and shown in Figure 4.8.

Trenching

Trench sampling was either based on channel sampling across an outcrop or shallow excavated trenches which where channel sampled along the floor/walls.

Trenches could not be viewed on site as these occur on agricultural land and as such have been back filled.

Several channel sample locations were observed on outcrop and the sampling appears to have been carried out well (Figure 4.10).

Figure 4.10 Channel sampling location observed in outcrop

Down-the-hole hammer

As discussed in the following section, DTH drilling results were not used for resource estimation as the assays appear to be biased.

Bias analysis

Snowden carried out comparisons between the various sample types within the oxide and sulphide material by creating quantile-quantile (Q-Q) plots to compare the distribution of grades. All Q-Q plots are located in Appendix 4B.

These comparisons indicate that the DTH samples are biased compared to the diamond and RC samples. As such the DTH samples were not used for further analysis or estimation.

Comparisons between the diamond, RC and trench samples show no indication of any material bias and all these samples were retained for estimation.

4.4.2 Collar and downhole surveying

Total station surveying was carried out at Ganajur, with all drillhole collar locations surveyed prior to drilling. For precision and accuracy of the coordinates, control points were established using Sokkia Radian IS DGPS (Differential GPS) which is considered highly accurate.

Vertical survey control of the elevation (RL) of the control points was carried out by levelling using auto levels (Sokkia make). Survey of India Bench Mark (RL 562.5 m) at the PWD office, Haveri was considered as the base for the elevation. Levelling was carried out from the base point and connected to all the control points established by DGPS.

Collars were again surveyed after drilling to check any possible change in the drillhole location.

Run wise continuous core orientation survey was carried out for each diamond drillhole using a Reflex ACT and ACT II Core orientation tool. Downhole surveys were carried out at regular intervals (every 24 m or 18 m) using a Reflex borehole single-shot camera to assess the deviation of the drillhole.

4.4.3 Sampling

A nominal 1.0 m and 0.5 m sample interval was used in the mineralised zone for RC drilling and diamond core drilling respectively. Samples were also collected outside the mineralised zone for a length of up to 5 m. However, the unmineralised portion between the two mineralised zones was sampled fully. In cases where the last sample in the unmineralised zone assayed gold values greater than 0.2 g/t Au, additional samples were collected to ensure that all mineralisation was sampled.

For RC samples, the bulk sample of a nominal 20 kg to 25 kg weight was reduced in size by riffle splitting using a Jones riffle splitter to about 2.5 kg and then placed in pre-numbered sample bags for dispatch to the analytical laboratory. In the case of wet drilling, the bulk sample was collected in a plastic bag, excess water drained and dried and then the assay sample was split.

Diamond drillholes were sampled using half core samples, cut with a diamond saw (Figure 4.11). Care was taken to preserve the same side of the split core for consistency. All diamond core was photographed dry and wet prior to cutting, and wet after cutting. Each piece of diamond core was marked with the sample number; where multiple fragments of core fell into one sample, each fragment was individually labelled (Figure 4.12).

Trench samples were either based on channel sampling across outcrop, or shallow, hand excavated trenches which were channel sampled along the walls and floors at a 0.5 m to 1 m interval.

Figure 4.11 Diamond core cutter

Figure 4.12 Example of sample numbering on diamond core

4.4.4 Geological logging

Diamond core

Geotechnical logging was carried out at the drill site before transporting the core boxes to the core shed (Figure 4.13). Geotechnical logging included RQD (Rock Quality Designation) measurements, core recovery, weathering, rock strength and description of discontinuity spacing.

All the natural breaks in the core were marked by putting a cross mark on the core with green marker pencil, and induced breaks were marked by putting a cross mark with red marker pencil/pen.

Litho-structural logging was carried out at the core yard before cutting which included logging of colour, grain size, alteration, shearing, bedding, foliation, rock type and amount of sulphides present. Structural logging included measurement of internal core angles (alpha (α) and beta angles (β)) which were carried out on planar features such as bedding plane, foliation, shear fractures and foliation, lithology contacts and mineral veins (e.g. quartz). Each point on the core, where the structural data was taken was marked with a tick mark using (red or blue) permanent marker pen. All core was orientated, allowing the calculation of dip and strike for each feature.

Figure 4.13 DESPL core yard showing stored diamond core trays

Reverse circulation

The RC bulk sample was sieved to obtain about 200 g of chips. The chips were washed to remove dust and spread sequentially on a plastic sheet spread on the ground close to the drillhole (Figure 4.14).

The rock description for each interval was entered on to a logging sheet/register. Logging consisted of a metre by metre description of the cuttings. The detailed observation was done with the help of handheld lens (10x to 40x) or using 45x polarising microscope. Information logged included mineralogy, grain size, colour, texture, and lithology. Other attributes commonly recorded include the percentage of quartz and sulphides, degree and type of alteration, degree of oxidation, and depth to water table.

After logging, the chips were preserved in self-locking plastic covers and RC chip trays for future reference (Figure 4.15).

Figure 4.14 Geological logging of RC cuttings

Source: DESPL

Figure 4.15 RC chip trays

4.4.5 Laboratory sample preparation and assaying

Laboratory sample preparation and assaying was conducted at Shiva Analyticals India Ltd (Shiva) laboratories in Bangalore which is ISO/IEC 17025:2005 accredited.

Shiva employs a quality control program covering both sample preparation and analysis. Sample preparation procedures include the use of barren rock/quartz flush of sample preparation equipment between each sample to avoid contamination and sizing of pulps is conducted to ensure conformity to the required 200 mesh (75 µm) size as per DESPL's requirement.

Shiva uses certified reference materials (CRMs) to monitor analytical accuracy and replicate analysis to monitor precision. Shiva protocol requires that each batch of samples analysed include a reagent blank, replicate determinations and CRMs of different concentrations.

Shiva also participates in international round robin and proficiency testing programs conducted all over the world as well as inter-laboratory test programs covering all the services rendered to the clients in the country.

The sample preparation and assaying process for the DESPL samples included:

- Drying
- Primary crush to a nominal top size of 10 mm for RC and 6 mm for diamond core using an Essa jaw crusher (Figure 4.16, top left)
- Secondary crush to a nominal top size of 2 mm for RC and 1 mm for diamond core using an Essa rolls crusher (Figure 4.16, top right)
- Split by rotary divider to a maximum 1.2 kg (Figure 4.16, bottom left)
- Pulverising to 90% passing 75 µm using an Essa LM2 pulveriser (Figure 4.16, bottom right)
- Scoop sample of pulp of 50 g (Figure 4.17)
- Assaying for Au using fire assay (Figure 4.18) with an AAS finish
- Assaying for base metals using ICPOES.

The sample sizes are considered reasonable to correctly represent the mineralisation based on the style of mineralisation, the thickness and consistency of intersections and the drilling methodology.

Snowden recommend that the use of a scoop for subsampling the pulps be revised and a small rotary splitter used instead. Given the nature of the mineralisation this is not likely to cause any material issues with the quality of the data.

Figure 4.16 Essa jaw crusher (top left); rolls crusher (top right); rotary sample divider (bottom left); and LM2 pulveriser (bottom right)

Figure 4.17 Scoop sampling the pulp

Figure 4.18 Fire assay

4.4.6 Quality assurance and quality control

DESPL's quality assurance and quality control (QAQC) program includes:

- Standards (mix of commercial CRMs and uncertified in-house standards) inserted at a nominal rate of one standard every 20 samples
- 5% pulp duplicates for the diamond drilling, carried out by Shiva
- Non-certified blanks at a rate of one in every 25 samples
- Check analysis of pulp rejects from the diamond drilling at an umpire laboratory
- Grind sizing analysis carried out by Shiva.

No QAQC was carried out for the RC drilling or trench sampling.

Standards and certified reference material

DESPL inserts standards into the sample batches at a nominal rate of one standard every 20 samples to assess the analytical accuracy of the laboratory assays for gold. Most of the standards are in house standards prepared by DESPL that do not have certified expected values or standard deviations. Snowden has not reviewed the procedure for certifying the standards and cannot comment on the quality of the process. The commercial standards (CRMs) used are sourced from ROCKLABS Limited based in New Zealand.

The standards are listed in Table 4.1, along with the expected gold value and standard deviations.

Table 4.1 Standard sample expected values and standard deviation

Standard ID	Expected value (Au g/t)	Standard deviation (Au g/t)	Source
87302	0.990	0.117	DESPL
87303	0.090	0.018	DESPL
87304	0.140	0.039	DESPL
97305	3.970	0.031	DESPL
87307	0.120	0.013	DESPL
87308	0.640	0.046	DESPL
87310	0.940	0.158	DESPL
98901	0.120	0.012	DESPL
98902	0.260	0.031	DESPL
98905	0.150	0.043	DESPL
98908	0.120	0.016	DESPL
98910	1.830	0.067	DESPL
98911	0.330	0.009	DESPL
98912	0.040	0.011	DESPL
OxE113	0.609	0.014	ROCKLABS
OxK110	3.602	0.053	ROCKLABS
SE44	0.606	0.017	ROCKLABS
SG66	1.086	0.032	ROCKLABS
Si64	1.780	0.042	ROCKLABS
SK78	4.134	0.138	ROCKLABS
SL46	5.876	0.170	ROCKLABS

Summary statistics for all standards are presented in Table 4.2. Control charts for all standards, are provided in Appendix 4A. All standards have fewer than 20 records with more than half having less than 10 records.

What is apparent is that many assays are falling outside of the three standard deviation control limit, indicating poor analytical accuracy. The ROCKLABS standards tend to consistently under-report the gold grade. The poor results from the standards are likely a result of poor matrix matching or poor certification. Snowden recommends that in the future, DESPL only source standards that have certified expected values and standard deviations and that they trial standards from a different source to determine whether the poor performance is an analytical issue or a result of inappropriate standards.

Table 4.2 Standard summary statistics

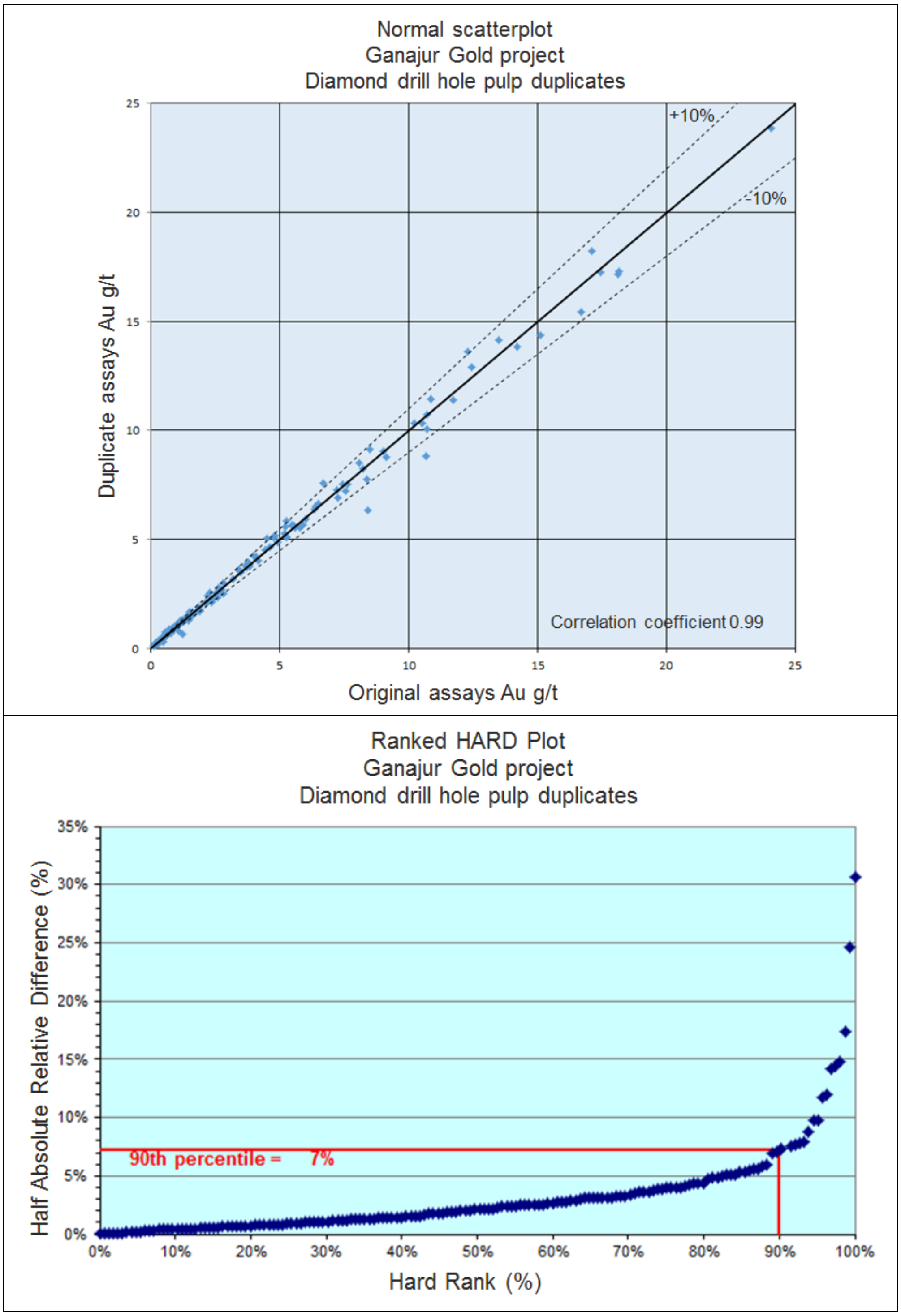
Standard	Statistic							
	Expected value	No. of assays	Mean	Minimum	Maximum	Standard deviation	% outside 3 standard deviation limits	Difference of mean to expected
87302	0.990	11	0.971	0.581	1.133	0.152	9.1%	-1.9%
87303	0.090	12	0.099	0.011	0.229	0.052	25.0%	10.0%
87304	0.140	1	0.097	N/A	N/A	N/A	0.0%	-30.7%
97305	3.970	7	4.023	2.371	6.760	1.431	85.7%	1.3%
87307	0.120	11	0.114	0.076	0.136	0.019	9.1%	-5.0%
87308	0.640	4	0.699	0.505	0.893	0.158	25.0%	9.2%
87310	0.940	6	0.744	0.561	0.880	0.131	0.0%	-20.9%
98901	0.120	6	0.134	0.118	0.144	0.009	0.0%	11.7%
98902	0.260	10	0.392	0.226	1.315	0.329	30.0%	50.8%
98905	0.150	9	0.204	0.163	0.285	0.039	11.1%	36.0%
98908	0.120	7	0.142	0.119	0.196	0.027	14.3%	18.3%
98910	1.830	7	1.468	0.163	1.966	0.590	57.2%	-19.8%
98911	0.330	5	0.333	0.320	0.351	0.011	0.0%	0.9%
98912	0.040	9	0.063	0.044	0.087	0.015	23.2%	57.5%
OxE113	0.609	9	0.635	0.608	0.671	0.023	23.2%	4.3%
OxK110	3.602	12	3.412	2.023	3.889	0.490	66.7%	-5.3%
SE44	0.606	19	0.559	0.352	0.651	0.072	26.3%	-7.8%
SG66	1.086	8	1.081	1.045	1.132	0.026	0.0%	-0.5%
Si64	1.780	10	1.744	1.495	2.003	0.154	60.0%	-2.0%
SK78	4.134	9	4.044	3.693	4.259	0.170	11.1%	-2.2%
SL46	5.876	19	5.298	3.075	6.522	0.978	47.4%	-9.8%

Duplicates

Only pulp duplicates from the diamond drilling have been collected. There has been no testing of field or crush duplicates or testing of duplicates from any other drilling or sampling type.

For the diamond drilling, 231 duplicate pulp samples (around 5%) were selected by Shiva and assayed with the results shown below in Figure 4.19. The scatterplot shows that the majority of results plot within plus or minus 10% and that there is also a high correlation coefficient of 0.99. The ranked HARD (half absolute relative difference) plot shows 90% of the duplicate pairs have a HARD of less than 7%. All results show that the precision of the pulp duplicate data is good with no evidence of bias.

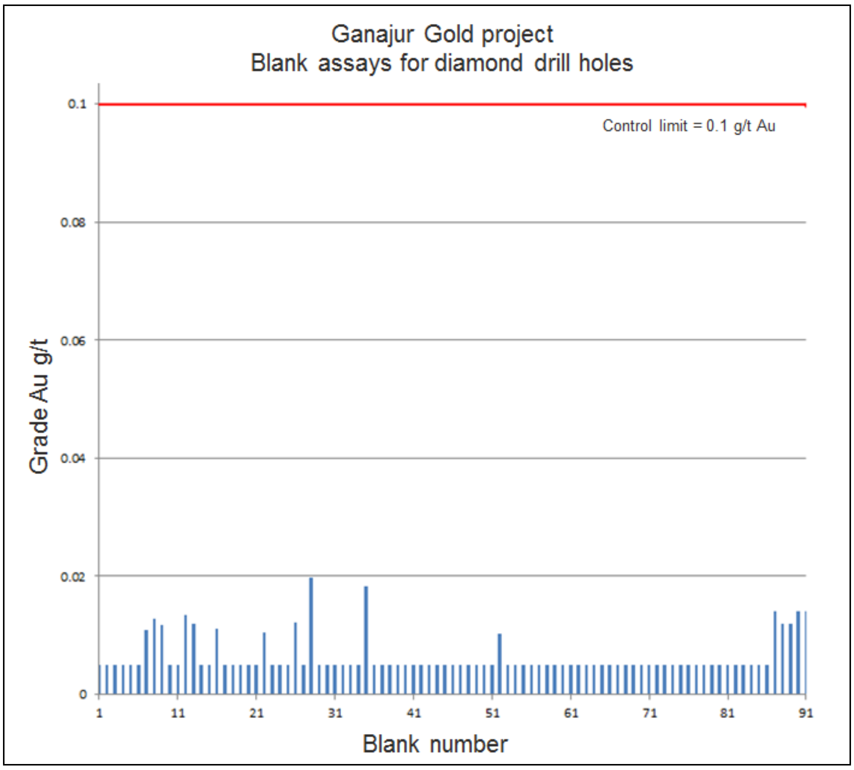
Figure 4.19 Scatterplot (top) and ranked HARD plot (bottom) for diamond drilling pulp duplicates



Blanks

Non-certified blanks were prepared and used by DESPL to test for contamination. A total of 91 blanks were inserted into the sample stream for the diamond drilling and show that all results are well within the acceptable limit of 0.1 g/t Au, as shown in Figure 4.20. These results are as expected as the pulverisers are cleaned with a barren quartz flush after every sample.

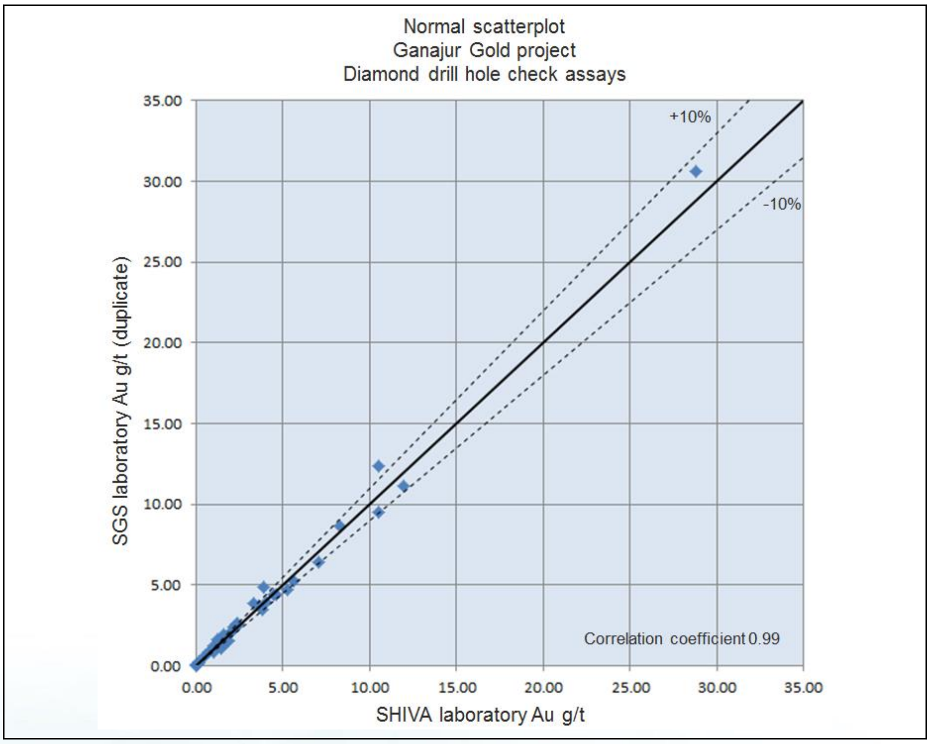
Figure 4.20 Blank assay results



Check assays

A total of 40 pulp rejects from the diamond drilling were sent to SGS in Chennai for analysis. A scatterplot in Figure 4.21 shows that results plot well about the 1:1 line with only a small number falling outside of the 10% control lines. A correlation coefficient of 0.99 also supports the good results for the umpire laboratory check analysis. No bias is evident in the results.

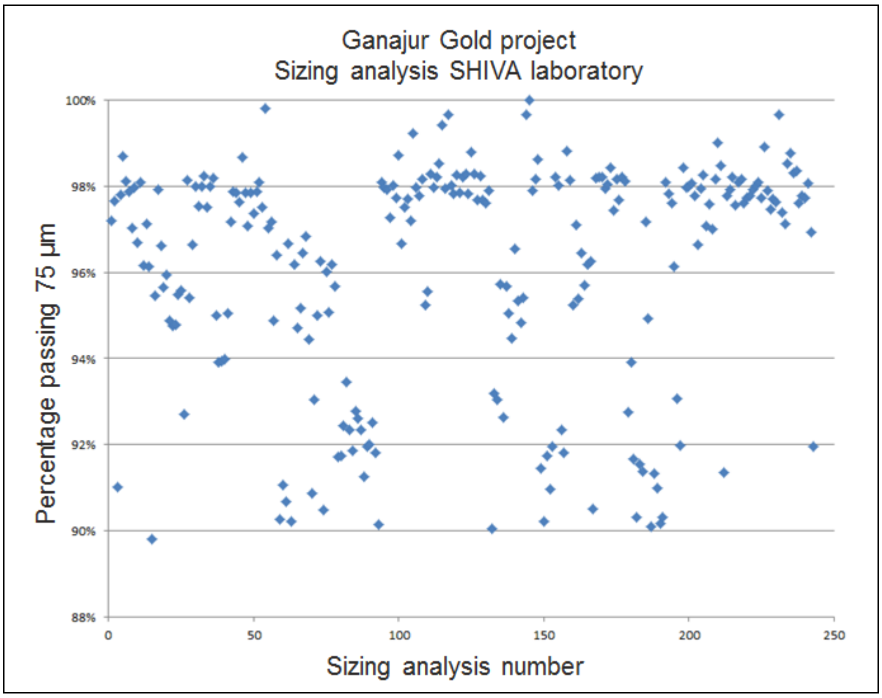
Figure 4.21 Scatterplot of check assays – Au



Sizing analysis

A total of 243 sizing analyses were completed by Shiva during 2016 with an average of 96% passing 75 µm. The results below in Figure 4.22 show that all but one analysis exceeds the 90% passing 75 µm which is a very good result.

Figure 4.22 Sizing analysis – Shiva laboratory



Author's opinion

Given the good results from the check assays and duplicate data, Snowden considers that the data is suitable for estimation, however the standard results indicate poor analytical accuracy with many standard assays falling outside of the three standard deviation control limit. The ROCKLABS standards tend to consistently under-report the gold grade. The poor results from the standards are likely a result of poor matrix matching or poor certification. Snowden recommends that in the future, DESPL only source standards that have certified expected values and standard deviations and that they trial standards from a different source to determine whether the poor performance is an analytical issue or a result of inappropriate standards.

While there is no QAQC for the RC or trench sampling, there is no indication of any bias when compared to the diamond samples (see the "Bias analysis" subsection of Section 4.4.1).

Snowden recommends a review of the QAQC procedures for future drilling programs, including:

- The use of fully certified standards (CRMs) and trialling CRMs from another source
- The inclusion of fields duplicates for RC drilling and coarse crush duplicates for diamond core
- The inclusion of QAQC samples for all drilling/sample types.

In addition, Snowden recommends that the use of a scoop for subsampling the pulps be revised and a small rotary splitter used instead. Given the nature of the mineralisation, this is not likely to cause any material issues with the quality of the data.

4.4.7 Sample security

All samples were collected by DESPL geologists. Samples collected were fully under the control of the geologist or their designated representatives until shipped or delivered to the laboratory responsible for sample preparation or analysis. All samples that remained on site, prior to delivery to the laboratory, were kept in a secure location not accessible by anyone other than approved personnel.

The samples were sealed in tough polyurethane bags and dispatched through a courier service to Bangalore. Along with the samples, the sample dispatch sheet (PX sheet) and other documents were also couriered to Shiva at Bangalore. After dispatching the samples, the site geologist would call Shiva staff on a mobile phone and inform about the dispatch and give details such as receipt number (LR number) and the PX number. The Shiva staff would then collect the samples from the courier office on the morning of the next day and transport the samples to their laboratory. Often the samples would also be transported by the DESPL company vehicle and delivered directly to Shiva.

4.4.8 Sample storage

Diamond core trays and RC chip trays are stored in the DESPL core yard. Shiva returns pulp samples which are also stored in the core yard. Snowden notes that older pulps were returned in well-structured and labelled boxes (Figure 4.23) while more recent pulps have been returned in poorly-structured, recycled boxed which are harder to store and not as robust (Figure 4.24). Snowden recommended that DESPL request a return to the original boxes for pulp sample returns, which has subsequently been implemented.

Figure 4.23 DESPL core yard with stored return pulp samples – historical samples

Figure 4.24 DESPL core yard with stored return pulp samples – recent samples

4.5 Input data

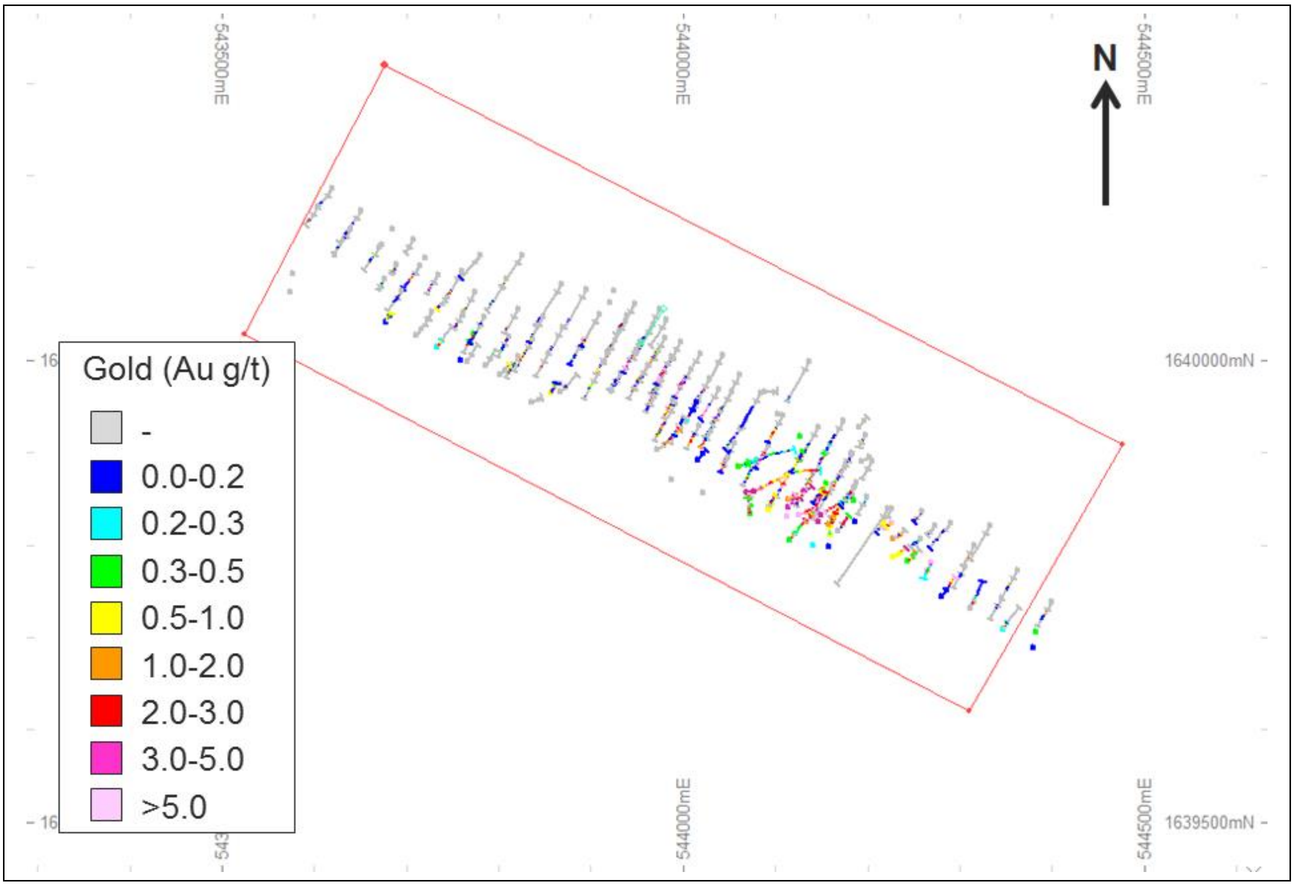
4.5.1 Drillhole data

The input data for the Ganajur Mineral Resource estimate includes:

- Diamond core – 83 drillholes for 5,121.64 m
- Reverse circulation – 22 drillholes for 1,219 m
- Trench – 59 trenches for 1,141.8 m.

The drilling was completed along a set of north-northeast trending sections (striking 030°). The drill spacing is 20 m x 20 m for the majority of the deposit.

Figure 4.25 Input drillhole data used for the Ganajur resource estimate



4.5.2 Collar and survey adjustments

Some trench data surveys lie just above topography as a result of precision issues between the surveys and topography. These surveys were not adjusted; however, the data coding was carried out to ensure that these samples were included in the oxide coding.

4.5.3 Data validation

All data and logging is entered in a series of paper based registers and then transposed into a Microsoft Excel spreadsheet.

Snowden carried out a basic validation of the database and found no material issues.

Snowden recommends that an industry standard database be utilised going forward and that DESPL assess the use of a digital logging system (e.g. onto tablets) with digital data transfer to avoid transcription errors.

4.5.4 Topography and depletion surfaces

The grid is based on a local UTM grid coordinate system (Zone 43N) based on the Everest (1830) datum.

The initial topographic survey was undertaken by DESPL in 2004 at a scale of 1:1000 with 1 m contour interval. In 2010, DGML again commissioned a topographical survey using DGPS and total station in and around the Ganajur Main prospect. The 2010 topographic survey was further checked in 2016.

Snowden used the 2010 topography (topo1-31-8_snotr.dm/topo1-31-8_snopt.dm). This surface did not quite cover the entire area and was subsequently extended to ensure full coverage of the tenement.

4.5.5 Bulk density

Bulk density of core samples was initially measured using a Double Beam Physical Balance to measure the weight of the sample in water and the weight in air. Subsequently, to speed up the process as well as increase the weight of the samples measured, a specially designed electronic balance was attached to the set up for measuring the bulk density (Figure 4.26).

Bulk density was measured for every alternate sample collected. For drillholes GMC-11 to GMC-34, a bulk density measurement was taken for each sample sent for gold analysis.

A representative piece of the core sample (about 15 cm to 20 cm) was taken from the sample bag for bulk density measurement. After the bulk density was measured, the sample piece was returned into the same sample bag from where it was taken.

The weight of the sample was measured using the electronic or physical balance to give the weight of the sample in air. The sample was then immersed in water and the weight measured to give the weight of the sample in water. The bulk density of the sample was obtained by the formula:

Bulk Density = Weight in Air / (Weight in Air – Weight in Water)

For each bulk density measurement, a sample description was recorded in a register, including colour, grain size, rock type, structure, alteration and amount of sulphides observed.

No coating was applied to the samples for bulk density measurement and hence there is the possibility that the bulk density may overstate in areas of weathered rock due to porosity. Snowden recommends a test program using wax coating of the samples to test this.

Figure 4.26 Bulk density measurement with electronic balance

4.6 Resource estimation

4.6.1 Domain coding

Snowden coded the validated drillhole data within the mineralised domains and weathering domains as detailed in Table 4.3.

Table 4.3 Drillhole domain coding

Field name	Value	Description
ORE	0	Non-mineralised
	1	Mineralised
OXIDE	1000	Oxide
	2000	Sulphide

4.6.2 Compositing

The drillhole data was composited downhole prior to running the estimation process using a 1 m sample interval to minimise any bias due to sample length. Around 90% of the mineralised samples have been taken at 0.5 m or 1 m intervals.

The compositing was run within the weathering and mineralisation fields to ensure that no composite intervals crossed any boundaries. To allow for uneven sample lengths within each of the domains, the composite process was run using the variable sample length method. This adjusts the sample intervals, where necessary, to ensure all samples are included in the composite file (i.e. no residuals) while keeping the composite interval as close to the desired interval as possible.

4.6.3 Statistical analysis

Histograms of composited gold grades (Au g/t) for the mineralised domain, within the oxide and sulphide, are presented in Figure 4.27 and Figure 4.28. Histograms for arsenic (As ppm), copper (Cu ppm), lead (Pb ppm), sulphide sulphur (SS ppm) and zinc (Zn ppm) are presented in Appendix 4C. Summary statistics for all variables are provided in Table 4.4 and Table 4.5.

The gold coefficient of variation (CV) of 1.05 for the oxide mineralised domain and 1.08 for the sulphide mineralised domain indicates that the distributions have a low to moderate positive skew with no extreme outliers. As a result, ordinary kriging with no top cuts was considered suitable for estimation.

For the other variables, the sulphide sulphur within the oxide domains and arsenic in the sulphide domain show elevated CV values with indications of outliers which may cause local overestimation when using ordinary kriging. As a result, top cuts were applied to these variables prior to estimation. Top cut statistics are presented in Table 4.6.

Figure 4.27 Histogram for gold – oxide mineralised domain

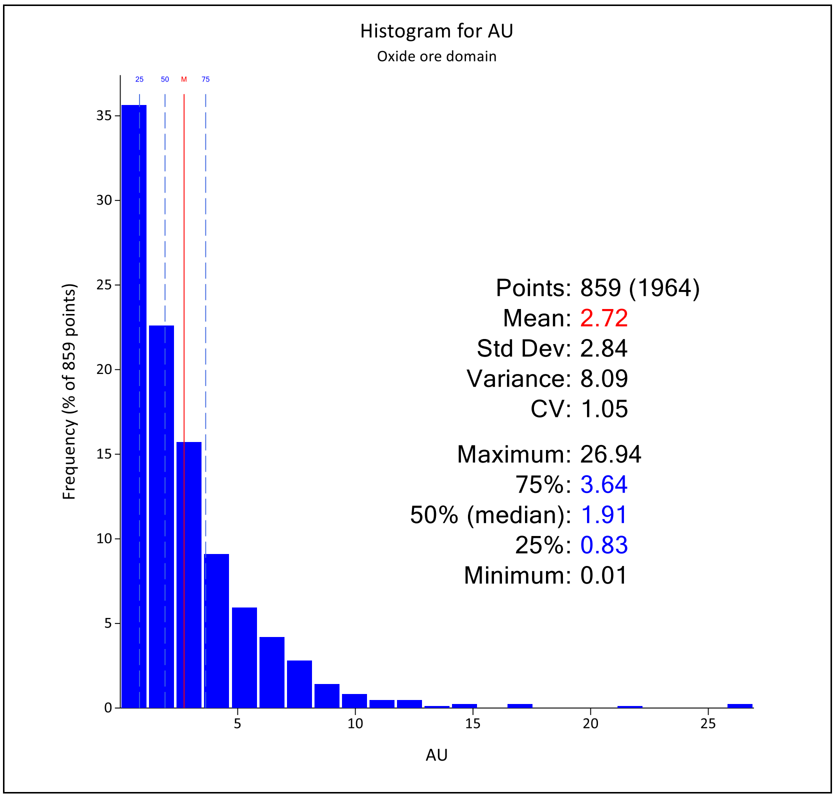


Figure 4.28 Histogram for gold – sulphide mineralised domain

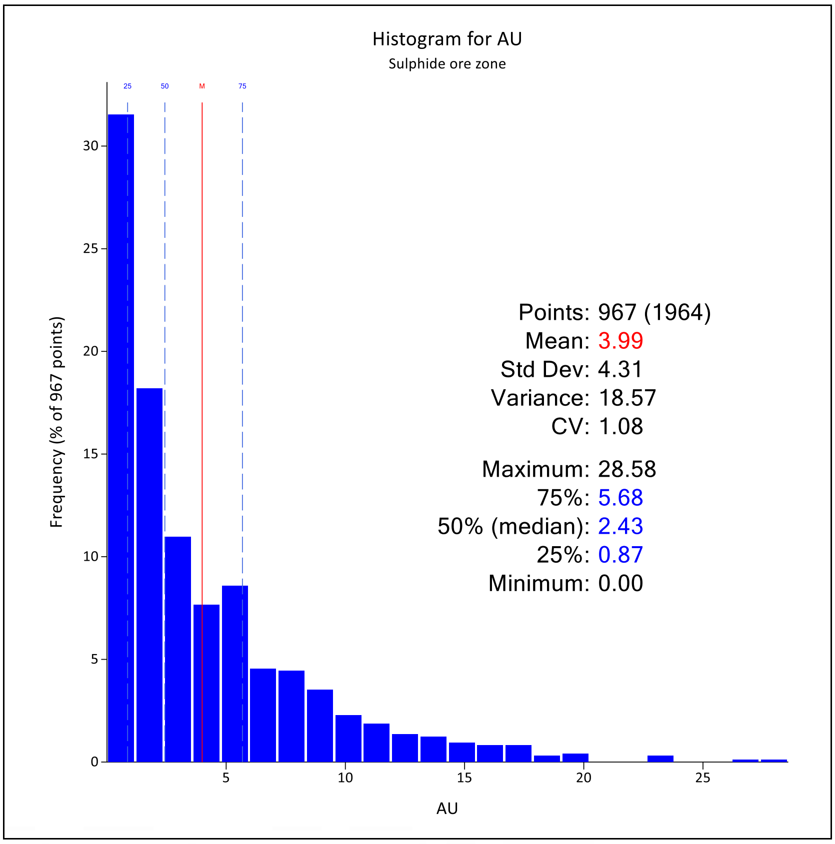


Table 4.4 Composite summary statistics for the oxide mineralised domain

Statistic	Au (g/t)	As (ppm)	Cu (ppm)	Pb (ppm)	SS (ppm)	Zn (ppm)
No. of samples	859	599	175	175	74	175
Minimum	0.01	61.00	2.00	2.00	50.00	19.00
Maximum	26.94	23,492.08	269.03	293.50	46,014.00	174.32
Mean	**2.72**	**1,611.19**	**36.23**	**37.25**	**2,297.27**	**62.75**
Standard deviation	2.84	1,722.79	36.79	25.13	7,208.85	25.75
Median	1.91	1,1210.24	23.44	33.26	562.00	58.79
CV	**1.05**	**1.07**	**1.02**	**0.67**	**3.14**	**0.41**
Variance	8.09	2,968,022	1,353.64	631.34	51,967,526	663.28
Skewness	3.01	5.69	2.95	6.40	4.67	1.13

Table 4.5 Composite summary statistics for the sulphide mineralised domain

Statistic	Au (g/t)	As (ppm)	Cu (ppm)	Pb (ppm)	SS (ppm)	Zn (ppm)
No. of samples	967	632	632	632	141	632
Minimum	0.001	30.00	2.00	2.00	172.00	15.00
Maximum	28.58	59,452	620.00	109.85	208,029.00	149.00
Mean	**3.99**	**4,758.27**	**44.45**	**31.39**	**43,681.76**	**58.52**
Standard deviation	4.31	8,438.15	52.01	15.00	39,024.36	19.12
Median	2.43	1,482.80	34.00	31.01	35,778.00	58.28
CV	**1.08**	**1.77**	**1.17**	**0.48**	**0.89**	**0.33**
Variance	18.57	71,202,359	2,705.24	225.10	1,522,900,292	365.41
Skewness	1.79	3.55	5.02	0.56	1.28	0.67

Table 4.6 Top cut statistics

Domain	Field	Top cut value	No. of samples	No. of samples top cut	Percent of samples top cut	CV		Mean	
						Raw	Top cut	Raw	Top cut
Oxide	SS	8,000	74	4	5.4	3.14	1.66	2,297	1,072
Sulphide	As	40,000	632	9	1.4	1.77	1.64	4,758	4,585

Correlations (as measured by the Pearson correlation coefficient) between elements are presented in a correlation matrix for the oxide and sulphide mineralised domains in Table 4.7 and Table 4.8.

Results show that the majority of correlations in the oxide mineralised domain are weakly positive, with only gold to arsenic and copper to arsenic showing strong correlations of 0.59 and 0.61 respectively. The sulphide mineralised domain has a majority of moderate to strong correlations (between 0.36 and 0.73) with the exception of correlations to zinc which are all poor, and lead to copper.

Table 4.7 Correlation matrix of composites in the oxide mineralised domain

Element	Au	As	Cu	Pb	SS	Zn
Au	1	0.59	0.49	0.14	0.08	0.00
As		1	0.61	0.11	0.14	0.13
Cu			1	0.00	0.24	0.37
Pb				1	0.12	0.19
SS					1	-0.12
Zn						1

Note: Green colours indicate positive correlation; red colours indicate negative correlation; darker colours indicate stronger correlation

Table 4.8 Correlation matrix of composites in the sulphide mineralised domain

Element	Au	As	Cu	Pb	SS	Zn
Au	1	0.63	0.36	0.52	0.73	0.04
As		1	0.49	0.37	0.50	-0.01
Cu			1	0.19	0.57	0.23
Pb				1	0.71	-0.04
SS					1	0.05
Zn						1

Note: Green colours indicate positive correlation; red colours indicate negative correlation; darker colours indicate stronger correlation

4.6.4 Variography

Normal scores experimental variograms were generated for all grade attributes for both the oxide and sulphide mineralised domains. Variograms were modelled using Snowden Supervisor software using the following general approach:

- All variograms were standardised to a sill of one.
- The nugget effect was modelled from the true downhole variogram.
- Variograms were modelled using two nested spherical variograms.
- The variograms were evaluated using normal scores variograms rather than traditional variograms. This method produces a clearer image of the ranges of continuity, especially in skewed datasets. The nugget and sill values were then back-transformed (using Supervisor software) to traditional variograms using the discrete Gaussian polynomials technique.

The maximum direction of continuity was interpreted to strike at 295°, dipping 35° to the north with no plunge. For gold, the nugget effect ranges from approximately 13% of the total variance in the oxide mineralised domain to 19% of the total variance in the sulphide mineralised domain. This is quite low for a gold deposit and Snowden believes this is due to the low variance of the gold grades and absence of grade outliers.

The maximum range of continuity in the oxide mineralised domain for gold is in the order of 60 m, while in the intermediate direction the range is approximately 35 m. In the minor direction, which is oriented orthogonal to the dip plane, the range of continuity is 9 m. In the sulphide mineralised domain, the maximum range of continuity for Au is in the order of 30 m, while in the intermediate direction the range is approximately 15 m. In the minor direction, which is oriented orthogonal to the dip plane, the range of continuity is 9 m.

Back-transformed variogram model parameters are presented in Table 4.9 and Table 4.10 for the oxide and sulphide mineralised domains, respectively. The normal scores variogram models are provided in Appendix 4D.

Table 4.9 Back-transformed variogram model parameters for the oxide mineralised domain

Variable	Directions of continuity			Nugget effect	First structure				Second structure			
	Direction 1	Direction 2	Direction 3		Sill	Range 1	Range 2	Range 3	Sill	Range 1	Range 2	Range 3
Au	00°→295°	-35°→025°	-55°→205°	0.13	0.54	20	20	4	0.33	60	35	9
As	00°→295°	-35°→025°	-55°→205°	0.02	0.53	11	11	4	0.45	30	15	6
Cu	00°→295°	-35°→025°	-55°→205°	0.01	0.49	20	20	3	0.50	60	25	18
Pb	00°→295°	-35°→025°	-55°→205°	0.44	0.24	6	6	6	0.32	30	30	10
SS	00°→295°	-35°→025°	-55°→205°	0.03	0.80	13	13	4	0.17	20	20	12
Zn	00°→295°	-35°→025°	-55°→205°	0.16	0.49	9	9	8	0.35	15	15	9

Table 4.10 Back-transformed variogram model parameters for the sulphide mineralised domain

Variable	Directions of continuity			Nugget effect	First structure				Second structure			
	Direction 1	Direction 2	Direction 3		Sill	Range 1	Range 2	Range 3	Sill	Range 1	Range 2	Range 3
Au	00°→295°	-35°→025°	-55°→205°	0.19	0.34	25	10	4	0.47	30	15	9
As	00°→295°	-35°→025°	-55°→205°	0.02	0.57	4	4	4	0.42	20	20	10
Cu	00°→295°	-35°→025°	-55°→205°	0.03	0.76	25	25	3	0.21	45	45	15
Pb	00°→295°	-35°→025°	-55°→205°	0.09	0.36	22	22	4	0.55	27	27	5
SS	00°→295°	-35°→025°	-55°→205°	0.01	0.54	23	23	6	0.45	35	35	20
Zn	00°→295°	-35°→025°	-55°→205°	0.08	0.24	23	23	2	0.68	50	50	50

4.6.5 Kriging neighbourhood analysis

A kriging neighbourhood analysis (KNA) was completed to determine the optimal parameters for estimation. A KNA calculates statistics for the kriging efficiency and slope of regression, together with the number of negative weights, which can then be compared for each scenario to determine which gives the optimal results. A kriging efficiency of better than 80% and a slope of regression better than 0.9 indicate very good results. This was performed on both the oxide and sulphide mineralised domains using Snowden's Supervisor software.

Selection of block size

Several different block sizes were tested for both the oxide and sulphide mineralised domains; results are show below in Figure 4.29 and Figure 4.30, respectively.

Based on these results along with consideration of the geometry of the mineralisation and the nominal drilling density of 20 mE x 20 mN, Snowden has opted for a block size of 10 mN x 10 mE x 5 mRL as being suitable for estimation. The vertical block dimension was chosen to match the current open pit mining plan which anticipates a bench height of 2.5 m.

Figure 4.29 KNA results for block size in the oxide mineralised domain

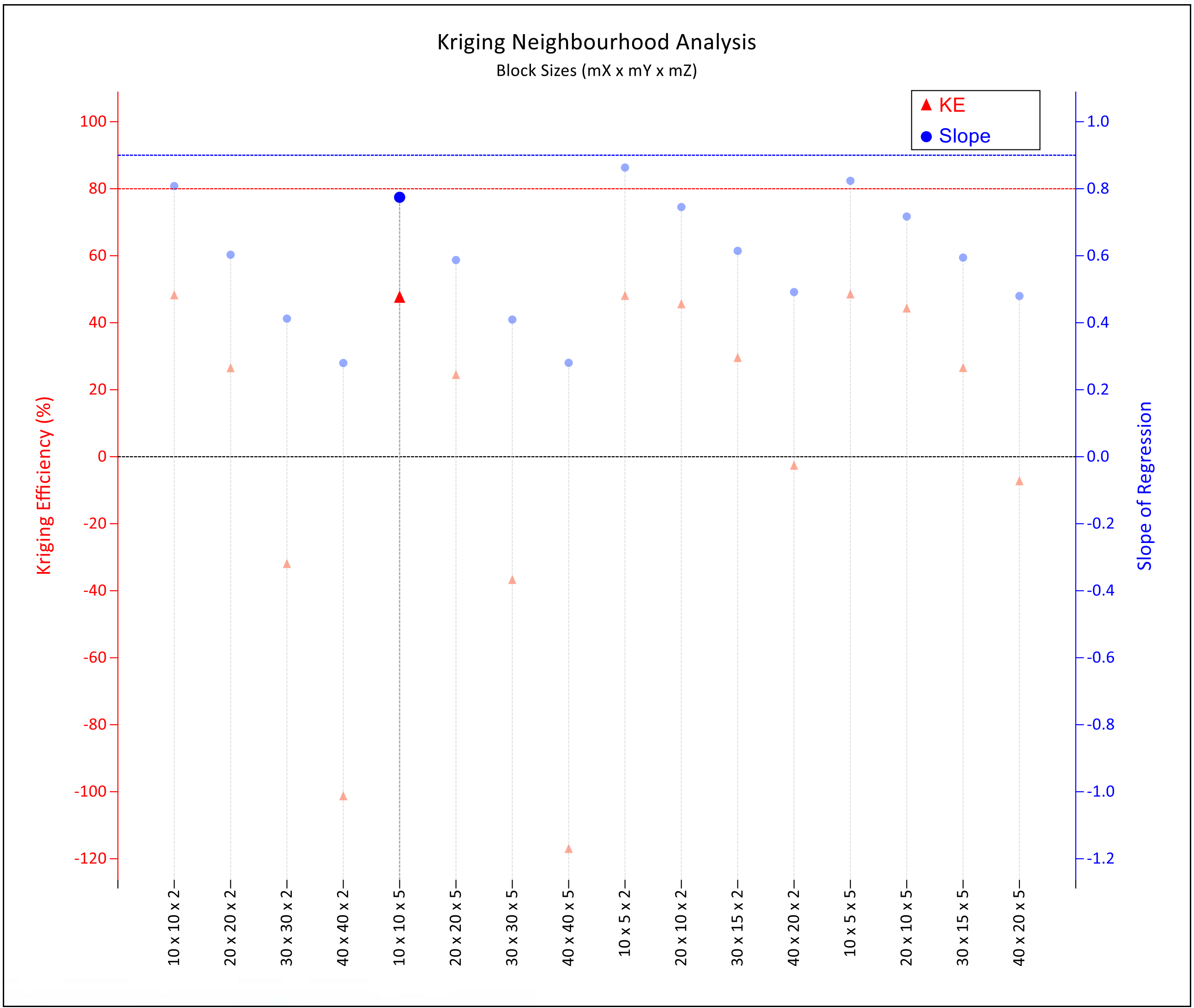
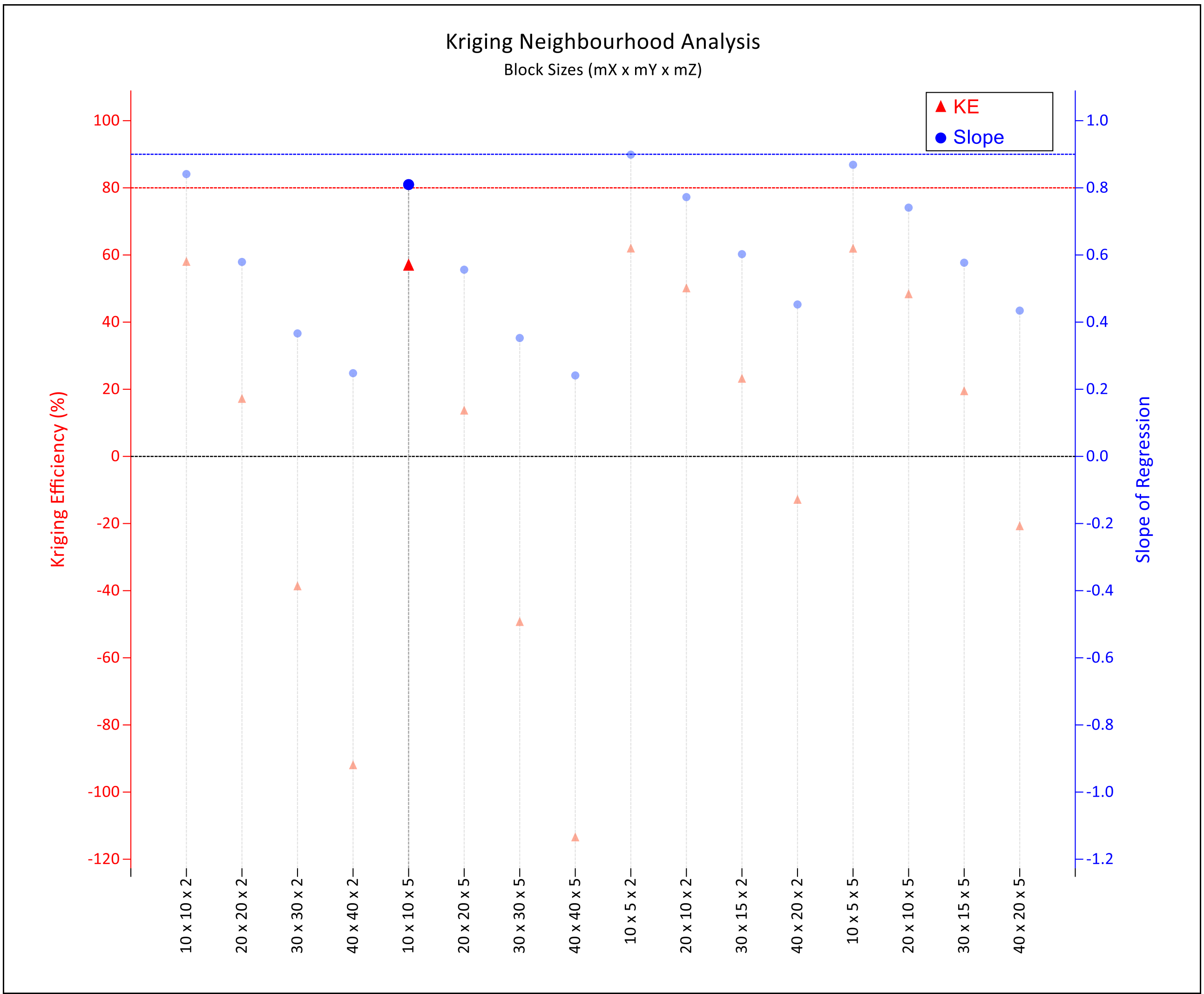


Figure 4.30 KNA results for block size in the sulphide mineralised domain



Selection of number of informing samples

A KNA was run to determine the optimal minimum and maximum number of samples for estimation. Figure 4.31 and Figure 4.32 show the results for the oxide and sulphide mineralised domains respectively. For the oxide mineralised domain, a minimum of eight and a maximum of 17 samples and for the sulphide mineralised domain a minimum of eight and a maximum of 24 samples were chosen for estimation. For the oxide mineralised domain, a maximum of 17 was chosen as a large number of negative weights were produced when using more than 17 samples as shown in Figure 4.33.

Figure 4.31 KNA results for number of samples in the oxide mineralised domain

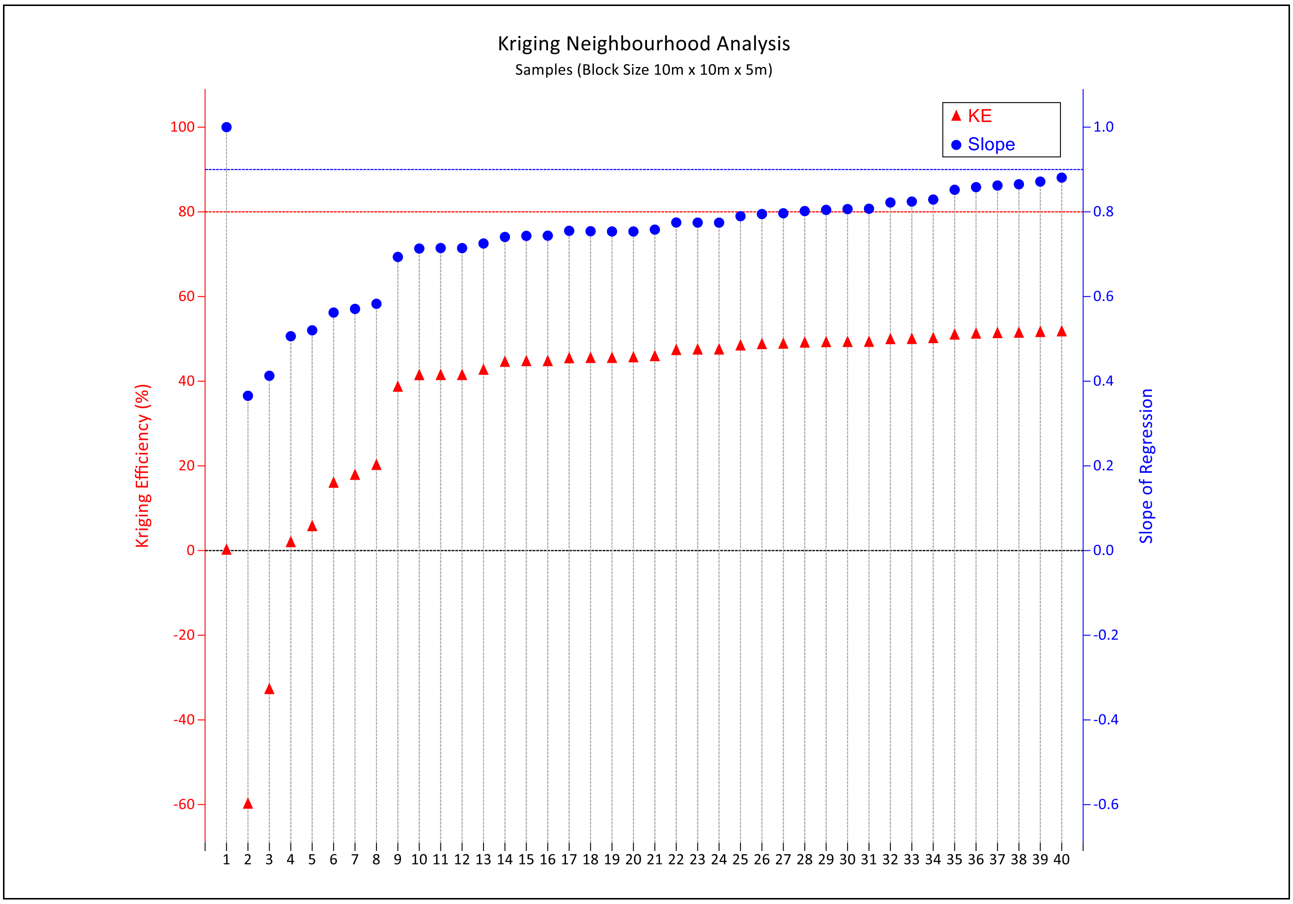


Figure 4.32 KNA results for number of samples in the sulphide mineralised domain

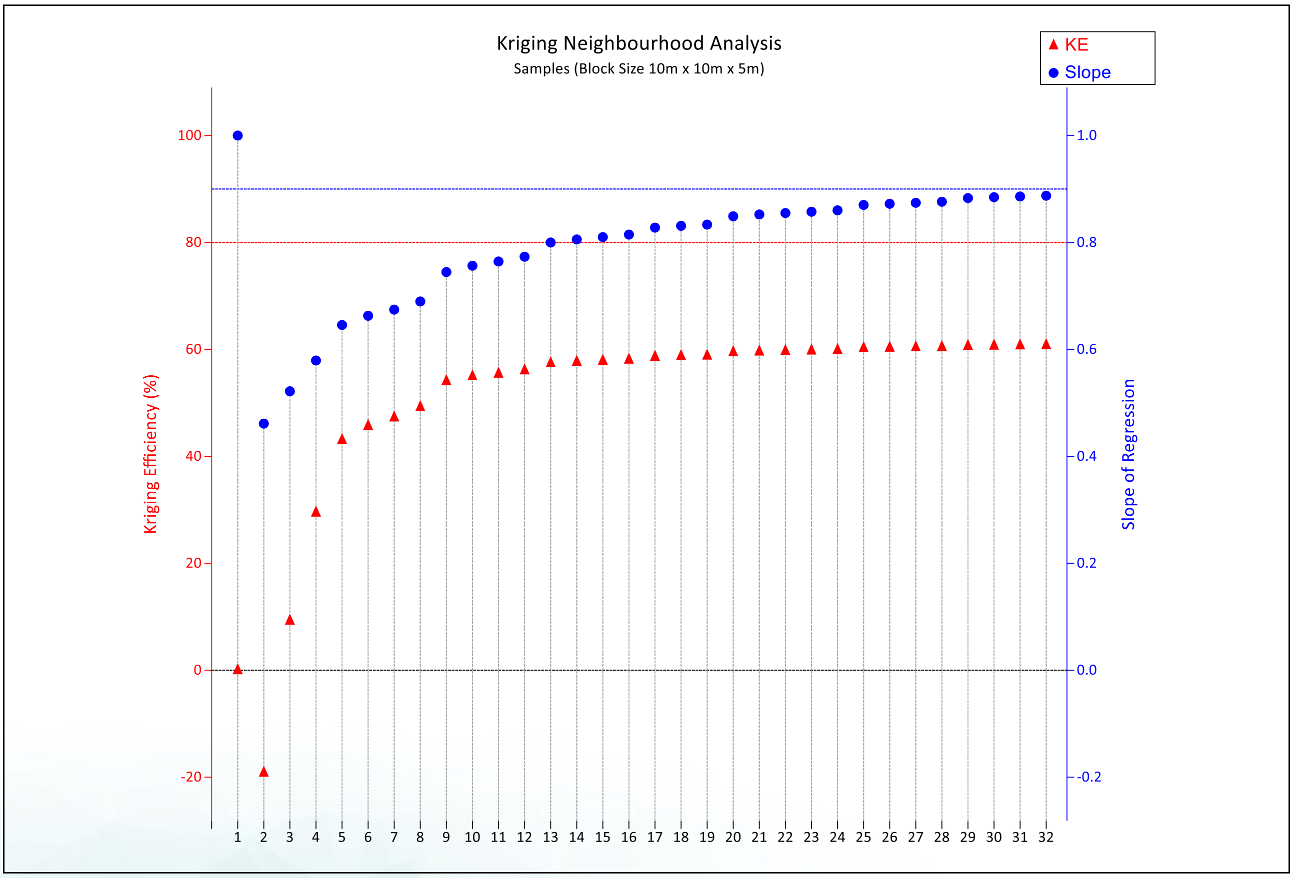
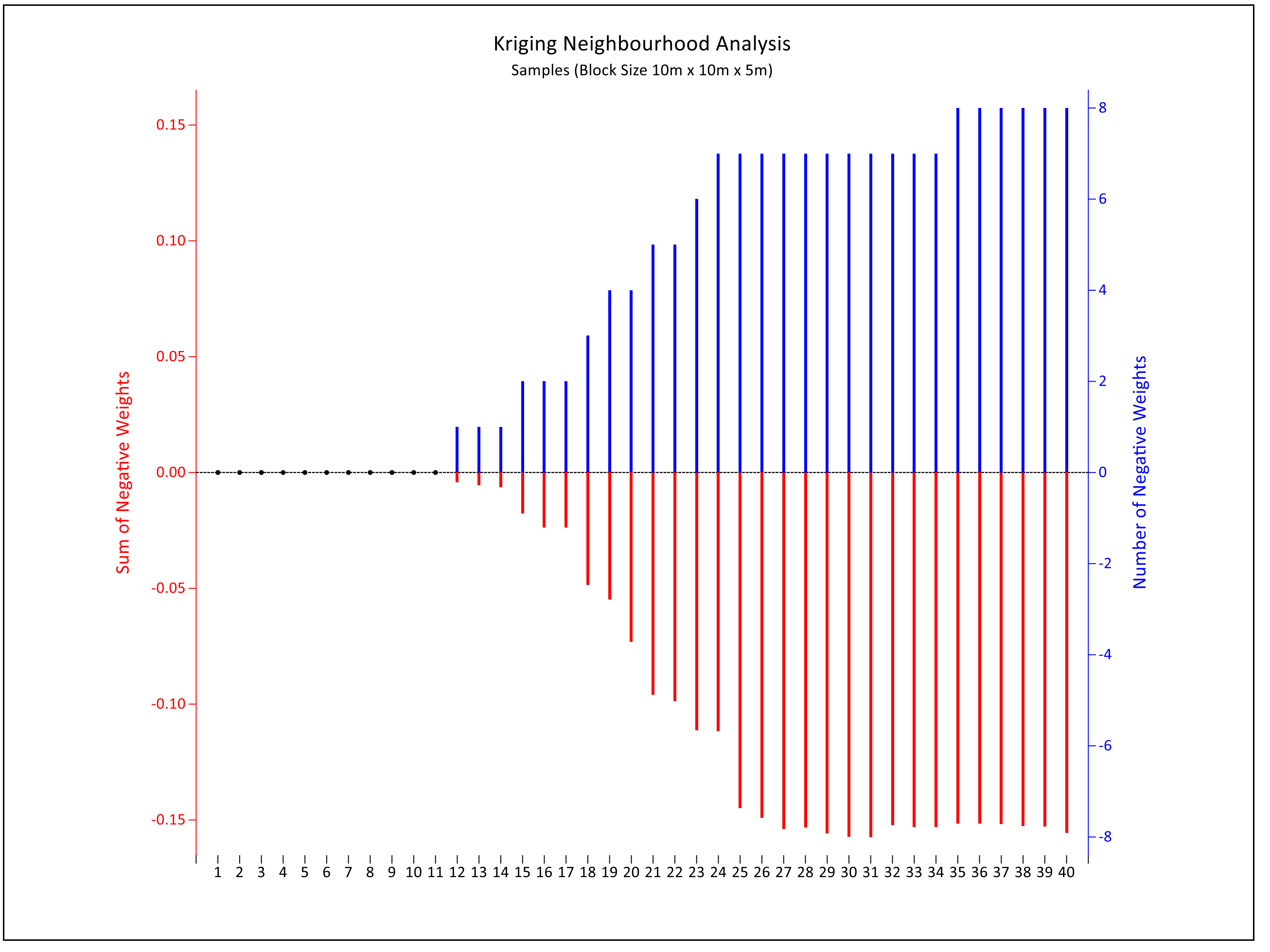


Figure 4.33 KNA – number of negative weights for the oxide mineralised domain



4.6.6 Block modelling

A block model was constructed based on a parent block size of 10 mE x 10 mN x 5 mRL, with a minimum sub-block size of 2.5 mE x 2.5 mN x 1.25 mRL.

Snowden created a block model with extents slightly larger than the tenement boundary supplied by DESPL, but limited the reported resource to within the tenement boundaries.

The final block model extents and parent and sub-cell sizes are detailed in Table 4.11.

Table 4.11 Block model prototype settings

	Easting (E)	Northing (N)	Elevation (RL)
Origin	543,450	1,639,500	350
Limit	544,550	1,640,400	600
Parent block size (m)	10	10	5
Minimum sub-cell size (m)	2.5	2.5	1.25

The model was coded with the mineralised domains (ORE) and weathering domains (OXIDE) as per the drillhole data coding detailed in Table 4.3. In addition, the tenement limits were coded (Table 4.12).

Figure 4.34 to Figure 4.36 illustrate example section views of the mineralised domain and weathering domain coding and a plan view of the tenement coding.

Table 4.12 Block model domain coding

Field name	Value	Description
LEASE	0	Outside tenement boundary
	1	Inside tenement boundary

Figure 4.34 Example section (544,125 mE) showing mineralised domain coding

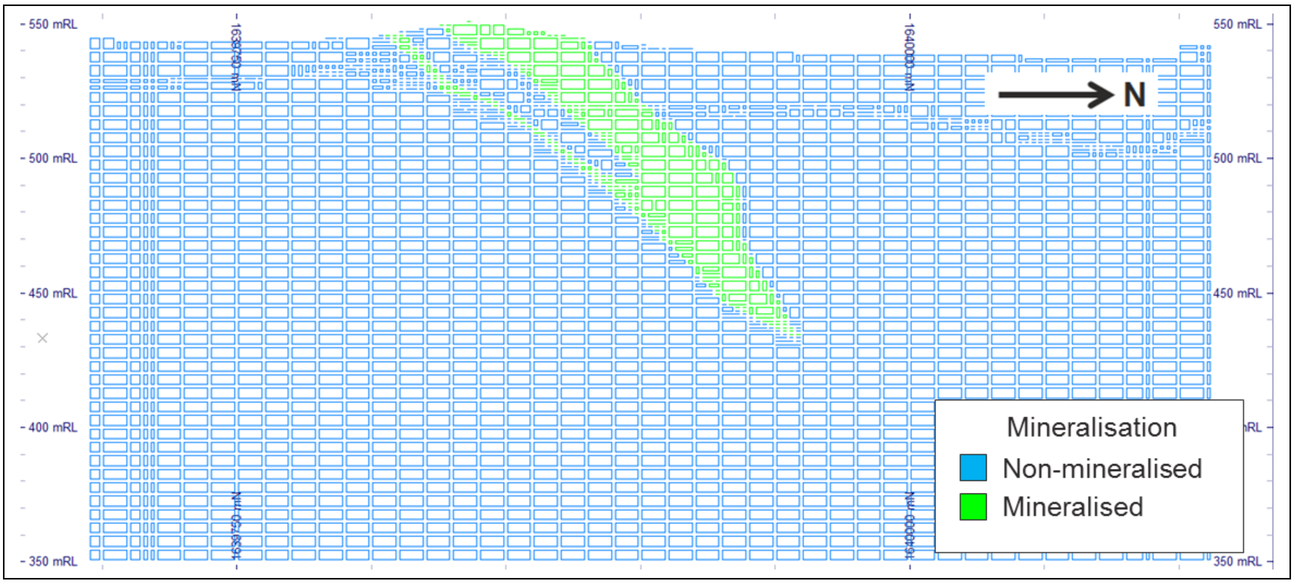


Figure 4.35 Example section (544,125 mE) showing weathering domain coding

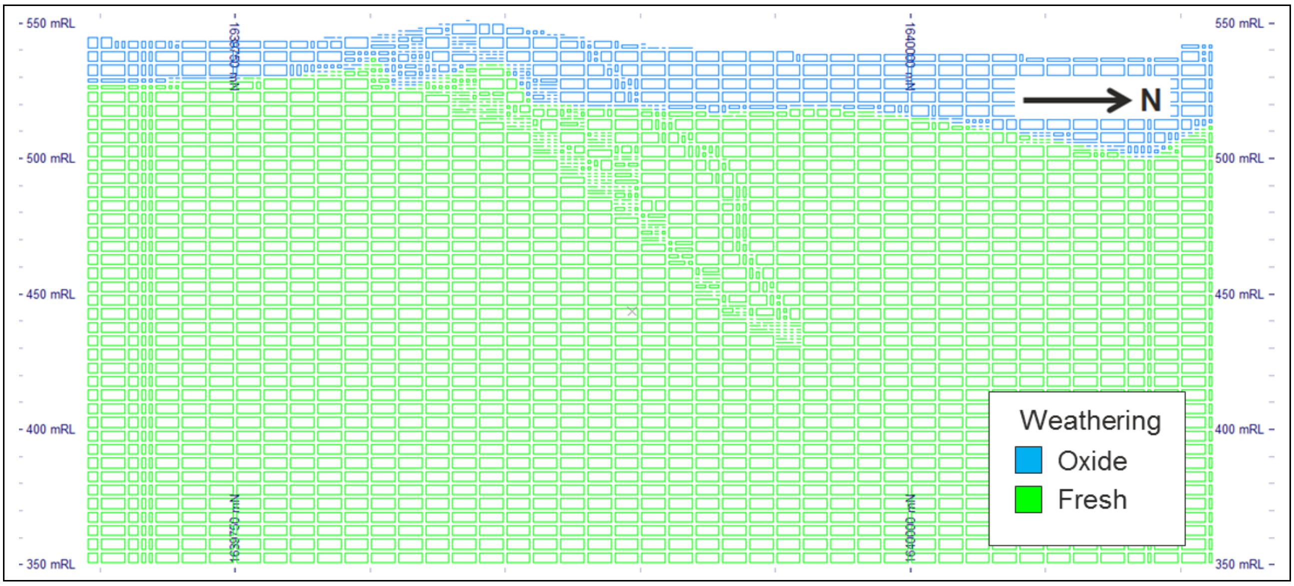
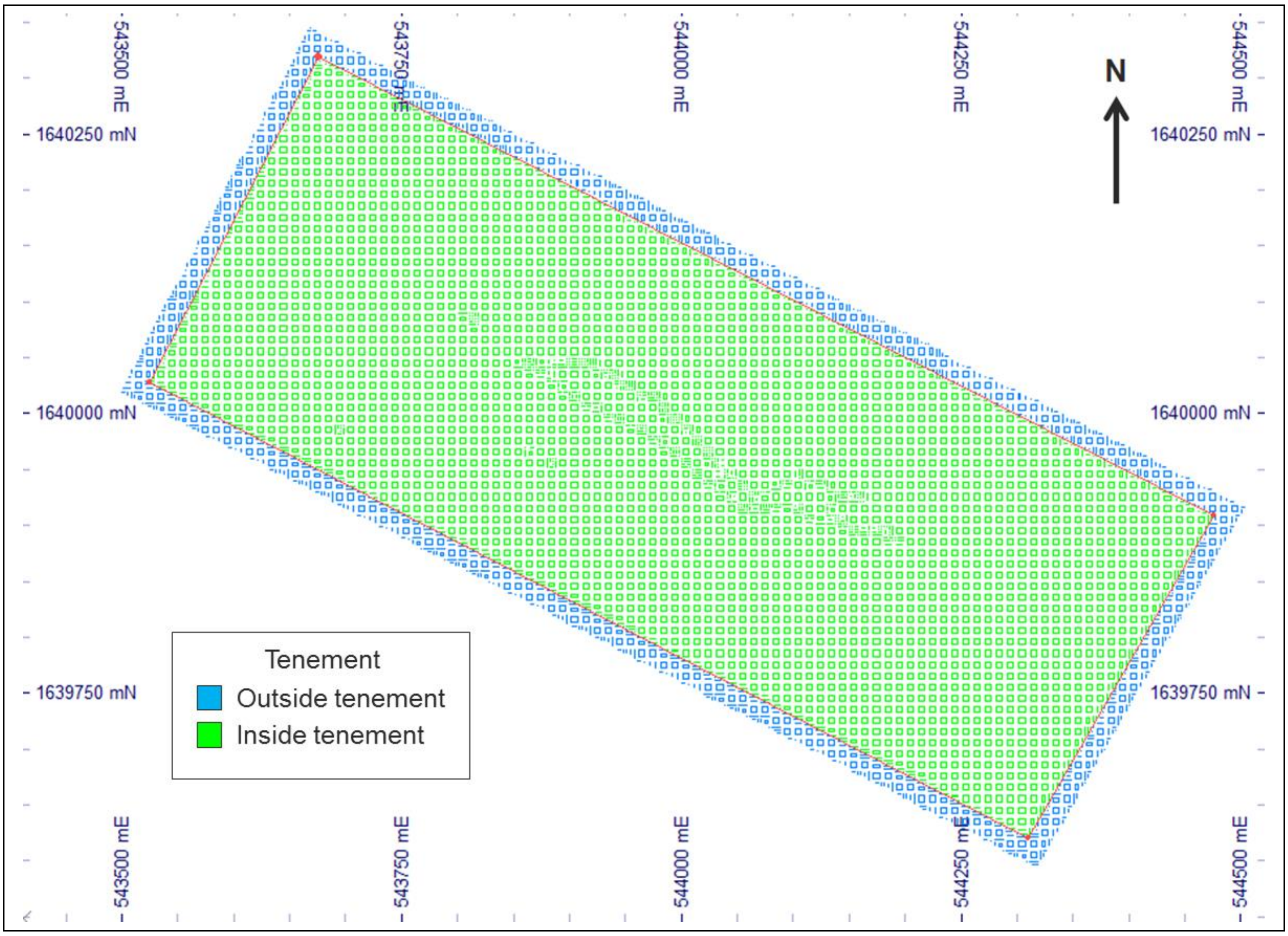


Figure 4.36 Example plan (500 mRL) showing tenement coding with boundary supplied by DESPL in red



4.6.7 Grade estimation

Snowden estimated gold, arsenic, copper, lead, sulphide sulphur and zinc grades using ordinary block kriging (parent cell estimates) using CAE Datamine software. Grades were only estimated within the mineralised domain and were estimated for the oxide and sulphide domains independently using hard boundaries.

Different search ellipse ranges were used for estimates in the oxide and sulphide mineralised domains. However, the same search ellipse ranges were used for all variables within each of the domains in order to keep the ratios of the various variables (i.e. correlation) as consistent as possible.

In the oxide mineralised domain, an initial search ellipse of 40 m x 23 m x 6 m was defined after assessing the data coverage and ranges of continuity seen in the variography. A minimum of eight and maximum of 17 samples was used for the initial search pass based on the KNA, with no more than four composites allowed per drillhole, meaning that at least two drillholes were required for estimation. A second search pass utilised 1.5 times the search ellipse radii (i.e. 60 m x 35 m x 9 m) with the same minimum and maximum number of samples. For the third search pass, the search ellipse radii were multiplied by five (i.e. 200 m x 115 m x 30 m) and the minimum number of samples reduced to two so that all remaining blocks were estimated.

In the sulphide mineralised domain, an initial search ellipse of 20 m x 10 m x 6 m was defined after assessing the data coverage and variography. A minimum of eight and maximum of 24 samples was used for the initial search pass based on the KNA, with no more than four composites allowed per drillhole, meaning that at least two drillholes were required for estimation. A second search pass utilised 1.5 times the search ellipse radii (i.e. 30 m x 15 m x 9 m) with the same minimum and maximum number of samples. For the third search pass, the search ellipse radii were multiplied by seven (i.e. 140 m x 70 m x 42 m) and the minimum number of samples reduced to two so that all blocks were estimated.

The key search ellipse and estimation parameters are summarised in Table 4.13.

Table 4.13 Estimation parameters

Estimation setting	Description/setting
Final model name	gan1608v1.dm
Drillhole file	Ganhole_est.dm (coded drillhole composite data in Datamine format with top cuts. Non-mineralised domains and DTH holes removed)
Boundary conditions	Hard domain boundaries for all estimates
Top cuts	See Section 4.6.3
Search ellipsoid	See Section 4.6.7
Method	Ordinary kriging (parent cell estimation)
Variograms	See Section 4.6.4
Dynamic search volumes used	Yes
Minimum no. of samples – volume 1 (oxide)	8
Maximum no. of samples – volume 1 (oxide)	17
Search volume 2 factor (oxide)	1.5
Minimum no. of samples – volume 2 (oxide)	8
Maximum no. of samples – volume 2 (oxide)	17
Search volume 3 factor (oxide)	5
Minimum no. of samples – volume 3 (oxide)	2
Maximum no. of samples – volume 3 (oxide)	17
Minimum no. of samples – volume 1 (sulphide)	8
Maximum no. of samples – volume 1 (sulphide)	24
Search volume 2 factor (sulphide)	1.5
Minimum no. of samples – volume 2 (sulphide)	8
Maximum no. of samples – volume 2 (sulphide)	24
Search volume 3 factor (sulphide)	7
Minimum no. of samples – volume 3 (sulphide)	2
Maximum no. of samples – volume 3 (sulphide)	24
Octant searching used	No
Block discretisation (XYZ)	4 x 4 x 4 (i.e. 64 points per parent cell)
Maxkey	4 (maximum number of composites per drillhole)

Density estimation and assignment

Bulk density was estimated using ordinary kriging into both the oxide and sulphide mineralised domains using a hard boundary. Variograms and the grade trend plots for both domains are presented in Appendix 4E.

Parameters used for estimation are the same as those set out in Table 4.13 for both the oxide and sulphide mineralised domains. The summary statistics for both the oxide and sulphide mineralised domains are presented in Table 4.14; estimate values and input composite statistics are provided.

Results show that the density estimates validate well with a difference of <1% in both the oxide and sulphide mineralised domains. Where estimates were not possible an average of 2.75 t/m^3 and 3.08 t/m^3 were used for the oxide and sulphide mineralised domains, respectively.

Table 4.14 Model and drilling statistics for bulk density

Domain	Estimate			Drilling			Difference (%)
	Minimum	Maximum	Mean	Minimum	Maximum	Mean	Mean
Oxide	2.46	3.21	2.73	2.36	3.70	2.75	-0.7
Sulphide	2.62	3.39	3.06	2.36	4.02	3.08	-0.7

Non-mineralised blocks

For the non-mineralised areas, top cut average grades were assigned to blocks as detailed in Table 4.15 below.

Table 4.15 Grades assigned to the non-mineralised domain

Domain	Au (g/t)	As (ppm)	Cu (ppm)	Pb (ppm)	SS (ppm)	Zn (ppm)
Oxide	0.12	354.0	39.0	12.5	138.0	66.0
Sulphide	0.12	301.0	37.0	16.0	1,316.0	65.0

Prior mining

There has been no prior mining at Ganajur.

Model fields

A full list of the fields in the final model file (gan1608v1.dm) is given in Table 4.16.

Table 4.16 Ganajur block model (gan1608v1.dm) fields

Field name	Value	Description
ORE	0	Non-mineralised
	1	Mineralised
OXIDE	1000	Oxide
	2000	Sulphide
LEASE	0	Outside tenement boundary
	1	Inside tenement boundary
AU_OK		Gold grade estimate (Au g/t)
AS_OK		Arsenic grade estimate (As ppm)
CU_OK		Copper grade estimate (Cu ppm)
PB_OK		Lead grade estimate (Pb ppm)
SS_OK		Sulphide sulphur grade estimate (SS ppm)
ZN_OK		Zinc grade estimate (Zn ppm)
NSAMAU	2-24	Number of informing sample for gold estimate (-4 where assigned grade)
SVOL_AU	1-3	Search pass for gold estimate (-4 where assigned grade)
VARAU		Kriging variance for gold estimate (-4 where assigned grade)
BD_OK		In-situ dry bulk density (t/m^3)
RESCAT	1	Measured classification
	2	Indicated classification
	3	Inferred classification
	4	Unclassified

4.6.8 Model validation

The block grade estimates were validated using:

- A global comparison of the average composite grades (naïve and de-clustered) and estimated block grades
- A visual comparison of block grade estimates and the input drillhole data
- Moving window averages comparing the mean block grades to the composite grades
- Comparison of correlation matrices for the block grades and the composites.

Global comparisons

The summary statistics for the oxide and sulphide mineralised domains are presented in Table 4.17 and Table 4.18, respectively; estimates, naïve composite and de-clustered composite statistics are provided. The results show that the gold grades validate well with the de-clustered percentage difference being -4.3% for the sulphide mineralised domain and -0.4% for the oxide mineralised domain.

Other elements generally validate well with all naïve and de-clustered differences in the oxide mineralised domain being within ±10% except for copper, which has a de-clustered difference of -15.4%. The sulphide mineralised domain shows that copper, lead and zinc validate well with all naïve and de-clustered differences within ±5%. The validation of arsenic and sulphide sulphur show de-clustered differences of -11.5% and 15.1%, respectively, which are considered somewhat high, however the estimation of sulphide sulphur is hindered due to the lack of informing data (141 composites).

Table 4.17 Estimate vs. input composite statistics for oxide mineralised domain

Variable	Estimate			Drilling		Naïve drilling	De-clustered drilling	Naïve difference (%)	De-clustered difference (%)
	Min.	Max.	Mean	Min.	Max.	Mean	Mean	Mean	Mean
Au	0.40	11.12	2.54	0.01	26.94	2.72	2.55	-6.5	-0.4
As	199	10,856	1,518	61.00	23,492	1,611	1,686	-5.8	-10.0
Cu	5.20	124.21	34.02	2.00	269.03	36.23	40.20	-6.1	-15.4
Pb	20.19	74.31	35.03	2.00	293.50	37.25	34.78	-6.0	0.7
SS	282	5,510	1,161	50.00	8000	1,072	1,238	8.3	-6.2
Zn	31.35	101.93	61.37	19.00	174.32	62.75	61.98	-2.2	-1.0

Table 4.18 Estimate vs. input composite statistics for sulphide mineralised domain

Variable	Estimate			Drilling		Naïve drilling	De-clustered drilling	Naïve difference (%)	De-clustered difference (%)
	Min.	Max.	Mean	Min.	Max.	Mean	Mean	Mean	Mean
Au	0.42	17.17	3.52	0.00	28.58	3.99	3.68	-11.9	-4.3
As	74	23,713	3,794	30	40,000	4,585	4,288	-17.3	-11.5
Cu	2.41	188.41	45.72	2.00	620.00	44.45	44.17	2.9	3.5
Pb	9.59	58.88	30.00	2.00	109.85	31.39	30.80	-4.4	-2.6
SS	727	106,133	50,333	172	208,029	43,682	43,717	15.2	15.1
Zn	25.20	112.79	59.25	15.00	149.00	58.52	58.46	1.3	1.4

Visual comparisons

Visual comparison of composite sample grades and estimated block grade was conducted in cross section and in plan. Visually the model spatially reflects the composite grades as shown in the example cross sections for gold (Figure 4.37, Figure 4.38 and Figure 4.39).

Figure 4.37 Ganajur location plan showing section lines

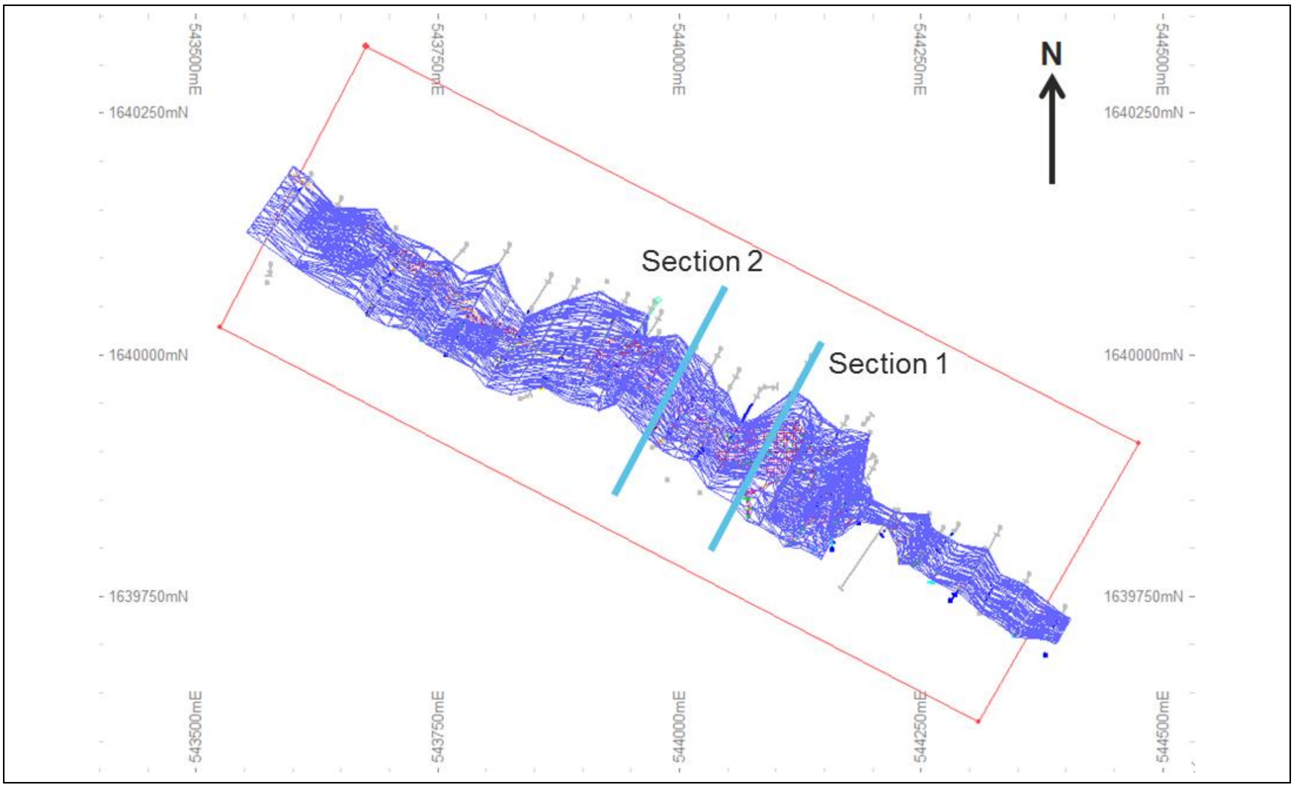


Figure 4.38 Ganajur oblique cross sections showing estimated grades and drillhole composites for gold – section 1 (+/-10 m clipping)

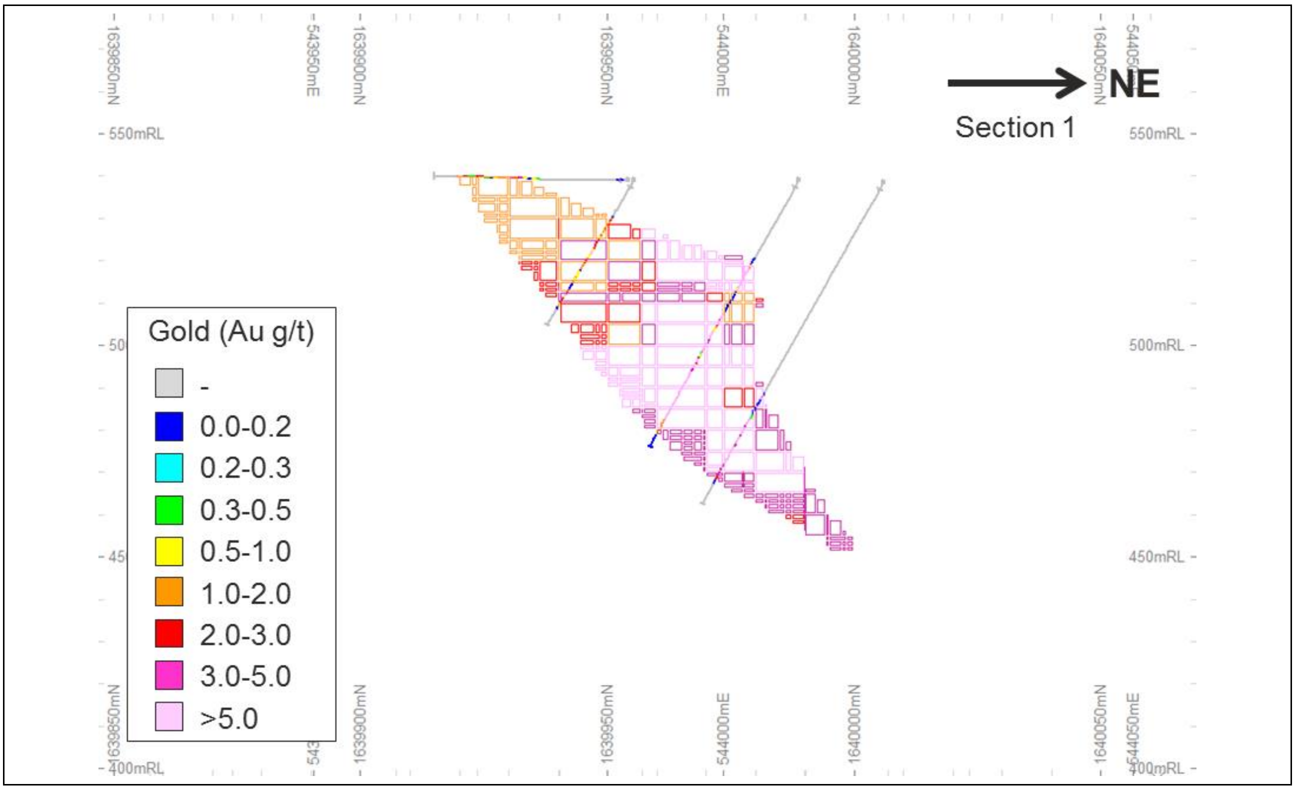
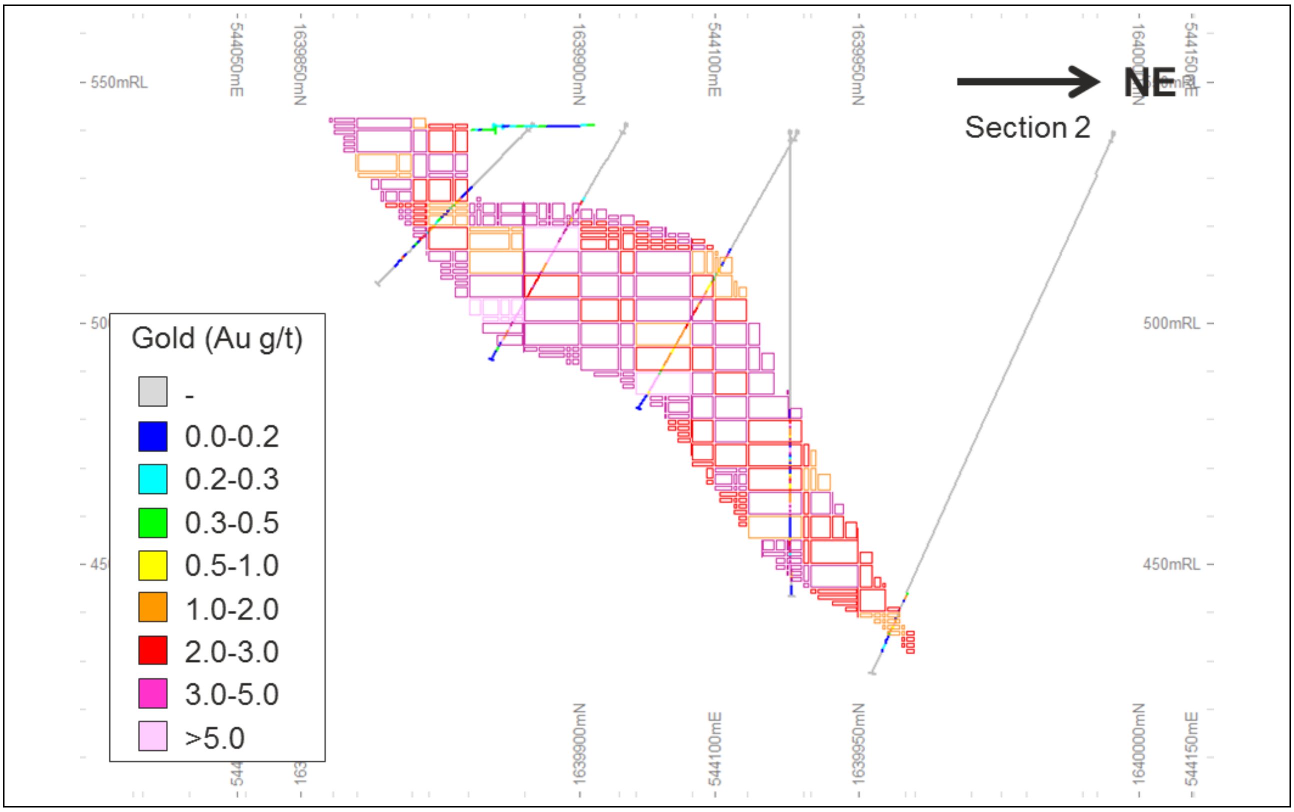


Figure 4.39 Ganajur oblique cross sections showing estimated grades and drillhole composites for gold – section 2 (+/-10 m clipping)



Grade trend plots

Grade trend plots were created for each variable in the oxide and sulphide mineralised domains and are shown below in Appendix 4F (plots generated on 10 m easting slices, 10 m northing slices and 5 m elevation slices).

The grade trend plots in the oxide mineralised domain follow the grade trends well except for arsenic and copper which show some underestimation below the 520 mRL where there are few samples. This is reflected in the statistics presented in the previous section which show a -10% and -15% difference between the estimated grade and the de-clustered composites for arsenic and copper.

Grade trend plots in the sulphide mineralised domain follow the grade trends well except for sulphide sulphur which shows poor validation below the 460 mRL and near surface where there are few samples. This is reflected in the poor global validation with +15% difference between the estimated grade and the de-clustered composites for sulphide sulphur in this domain.

Correlations

Correlations (as measured by the Pearson correlation coefficient) between estimated variables in the block model are presented in a correlation matrix for the oxide and sulphide mineralised domains in Table 4.19 and Table 4.20, respectively. Snowden compared the correlations within the block model to the correlations within the composite file to ensure that correlations are maintained within the block estimates.

A comparison of the correlations shows that the majority, including all gold correlations have a difference of ±0.25 in both the oxide and sulphide mineralised domains. Within the oxide mineralised domain, only the arsenic to lead correlation falls outside of this with an increased correlation from 0.11 to 0.44. In the sulphide mineralised domain, the correlation for copper to gold has decreased from 0.36 to 0.03. The correlations comparisons for sulphide sulphur are poor, however the sulphide sulphur estimate does not validate well and is based on very little data.

Snowden recommends that in future drilling campaigns that all variables are assayed so that there is more data for analysis.

Table 4.19 Correlation matrix of blocks in the oxide mineralised domain

Element	Au	As	Cu	Pb	SS	Zn
Au	1	0.72	0.37	0.30	0.15	0.26
As		1	0.38	0.44	0.20	0.21
Cu			1	-0.02	0.19	0.48
Pb				1	-0.10	0.40
SS					1	-0.04
Zn						1

Note: Green colours indicate positive correlation; red colours indicate negative correlation; darker colours indicate stronger correlation

Table 4.20 Correlation matrix of blocks in the sulphide mineralised domain

Element	Au	As	Cu	Pb	SS	Zn
Au	1	0.68	0.03	0.34	0.64	-0.05
As		1	0.32	0.33	0.46	-0.14
Cu			1	-0.08	0.02	0.27
Pb				1	0.26	0.11
SS					1	-0.06
Zn						1

Note: Green colours indicate positive correlation; red colours indicate negative correlation; darker colours indicate stronger correlation

4.6.9 Resource classification

The August 2016 Ganajur Mineral Resource estimate was classified and reported in accordance with the 2012 JORC Code. Snowden's assessment of the Table 1 criteria that were considered when classifying the Ganajur Mineral Resource estimate in accordance with the 2012 JORC Code guidelines are detailed in Appendix 4G.

The Mineral Resource has been classified as a combination of Measured, Indicated and Inferred Resources using the following criteria:

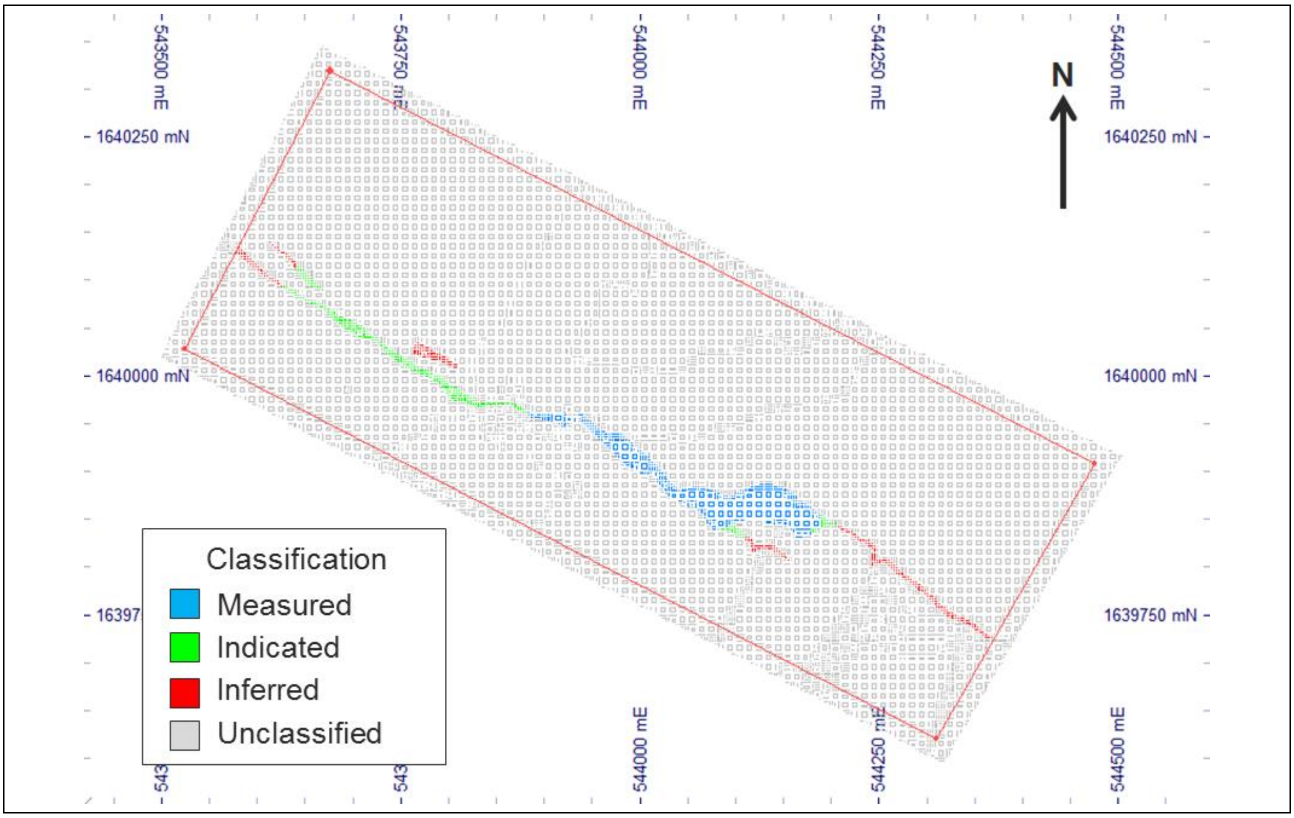
1) Measured Resources – Restricted to within the mineralised wireframe where drilling is approximately 20 mN x 20 mE or better, geological and grade continuity is confirmed and the mineralised body is at its thickest, typically 20 m to 50 m thick.
2) Indicated Resource – Restricted to within the mineralised wireframe where drilling is approximately 20 mN x 20 mE or better, geological and grade continuity is assumed. This has been restricted to areas where the mineralised body is typically less than 20 m thick.
3) Inferred Resource – Mineralisation with poor geological and grade continuity or which is defined by drilling on a grid greater than 20 mE x 20 mN.

Reporting of the Mineral Resource estimate has been restricted to within the lease boundary. Any mineralisation that has been interpreted as being outside of the lease is "unclassified" and excluded from the Mineral Resource. Classification is based on the confidence in the gold grade estimate. The Measured classification assumes that mining will be at around a 0.8 g/t Au cut-off and hence will mine the majority of the mineralisation, non-selectively.

Snowden generated wireframe solids to assign portions of each deposit as either Indicated, Inferred or unclassified. The resource classification scheme for the August 2016 Ganajur Mineral Resource estimate is shown in an example plan in Figure 4.40.

Note the classification is based on the gold grade estimate. Given the lesser amount of data for the other variables, particularly sulphide sulphur, these should be considered of a lower confidence.

Figure 4.40 2016 Ganajur Mineral Resource classification scheme



4.7 Mineral Resource reporting

4.7.1 Mineral Resource

The total Measured and Indicated Mineral Resource for the Ganajur gold deposit, reported above a 0.8 g/t Au cut-off grade, is estimated to be 2,700 kt grading at 3.40 g/t Au as detailed below in Table 4.21. The 0.8 g/t cut-off is based on preliminary work carried out as part of the FS and includes the majority of the mineralisation.

Appendix 4H details the Ganajur resource estimate reported at multiple cut-offs.

Table 4.21 Ganajur Mineral Resource as at August 2016, reported above 0.8 g/t Au cut-off

Classification	Deposit	Tonnes (kt)	Au (g/t)
	Oxide	580	2.82
Measured	Sulphide	1,690	3.96
	Total Measured	2,300	3.67
	Oxide	130	1.85
Indicated	Sulphide	330	2.13
	Total Indicated	450	2.05
Measured + Indicated	**Total Measured and Indicated**	**2,700**	**3.40**
	Oxide	110	2.30
Inferred	Sulphide	110	2.29
	Total Inferred	210	2.30

Note: Small discrepancies may occur due to rounding

4.7.2 Competent Person's statement

The information in this report that relates to the Ganajur Mineral Resource estimate is based on information compiled by Lynn Olssen who is a Chartered Professional (Geology) and a Member of the Australasian Institute of Mining and Metallurgy (MAusIMM(CP)) and has sufficient experience which is relevant to the style of mineralisation and type of deposit under consideration and to the activity to which she is undertaking to qualify as a Competent Person as defined in the 2012 edition of the "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves". Lynn Olssen is a full-time employee of Snowden Mining Industry Consultants Pty Ltd and consents to the inclusion in the report of the matters based on this information in the form and context in which it appears.

4.7.3 Comparison to previous estimate

The previous resource estimate was completed by SRK in August 2011. The estimate was reported at a 1 g/t cut-off as detailed below in Table 4.22. To provide a valid comparison Snowden reported its August 2016 resource estimate at a 1 g/t Au cut-off in Table 4.23.

Comparison of the estimates shows that there is a 125 kt increase in tonnes and a slight decrease in grade for the total Measured and Indicated Resources at a 1 g/t Au cut-off. The updated Mineral Resource has converted the majority of the previously reported Indicated Resources to Measured Resources. This is due to the extra drilling which has increased confidence in geological knowledge and grade continuity of the deposit.

There is also an increase in the Inferred tonnes and grade due to extra drilling completed to increase the size of the deposit and better define the mineralised boundaries.

Table 4.22 Previous Mineral Resource estimate – SRK 2011 reported at a 1.0 g/t Au cut-off

Classification	Deposit	Tonnes (kt)	Au (g/t)
Indicated	Oxide	631	3.19
	Sulphide	1,921	3.83
	Total Indicated	2,552	3.67
Inferred	Oxide	17	3.26
	Sulphide	93	1.82
	Total Inferred	109	2.06

Note: Small discrepancies may occur due to rounding

Table 4.23 Ganajur resource estimate as at August 2016, reported above 1.0 g/t Au cut-off

Classification	Deposit	Tonnes (kt)	Au (g/t)
Measured	Oxide	569	2.85
	Sulphide	1,671	3.99
	Total Measured	2,240	3.70
Indicated	Oxide	117	1.96
	Sulphide	321	2.14
	Total Indicated	438	2.09
Measured + Indicated	**Total Measured and Indicated**	**2,678**	**3.44**
Inferred	Oxide	103	2.32
	Sulphide	96	2.46
	Total Inferred	199	2.39

Note: Small discrepancies may occur due to rounding

4.8 Recommendations

Snowden makes the following recommendations for future drilling and resource estimation:

- Create a geological model based on lithology (i.e. chert) as well as the mineralised interpretation. This will aid with the definition of bulk density.
- In future drilling campaigns, assay all variables so that there is more data for analysis. This is particularly relevant for any variables identified during the Feasibility Study as being important for processing.
- Look at using an industry standard database going forward and assess the use of a digital logging system (e.g. onto tablets) with digital data transfer.
- Review the QAQC procedures for future drilling programs including:
  - use of fully certified standards (CRMs) and trialling CRMs from another source
  - inclusion of fields duplicates
  - inclusion of QAQC samples for all sample types.
- Review the use of a scoop for subsampling the pulps and implement the use of a small rotary splitter instead. Given the nature of the mineralisation this is not likely to cause any material issues with the quality of the data.
- Older pulps samples were returned from the laboratory in well-structured and labelled boxes while more recent pulps have been returned in poorly-structured, recycled boxed which are harder to store and not as robust. Snowden recommended that DESPL request a return to the original boxes for pulp sample returns, which has subsequently been initiated.
- Ensure sufficient bulk density measurements are taken in the non-mineralised domains to provide more confidence in these values.
- Carry out a test bulk density program using wax coating of the samples to test the impact of porosity, particularly on the weathered samples.
- Samples within the unmineralised zones surrounding the mineralisation should ideally be assayed, perhaps using a larger sample interval (e.g. 2 m) to allow estimation of the waste grades for mining dilution and waste rock characterisation.
- The Ganajur Main Gold deposit is surrounded by a number of satellite prospects designated as Ganajur South, South East, Central, Karajgi Main and Hut etc. Exploration carried out by DESPL in these prospects to date using surface geological mapping, trenching, drilling and IP geophysical survey indicates significant gold mineralisation and the possibility of adding additional resources across the entire Ganajur-Karajgi Block. It is recommended that DESPL continue its exploration in the satellite prospects in pursuit of augmenting additional gold resources.

4.9 References

DESPL, 2016a, *Ganajur Gold Project, Diamond Core Drilling and Sampling Procedure* 'QA-QC SAMPLING PROCEDURE-1-8-16.pdf', internal procedure – Deccan Gold Mines

DESPL, 2016b, *Deccan Exploration Services PVT PD, Reverse Circulation (R.C.) Drilling Procedures* 'R.C. Drilling and sampling Snowden.pdf', internal procedure – Deccan Gold Mines

Shiva, 2016, *Shiva Sample Prep and Analysis Protocol* 'Shiva Protocol - Ganajur(1).pdf', Shiva Analyticals (India) Ltd., laboratory procedures for Deccan Gold Mines

SRK, 2011, *Updated Mineral Resource Estimate of Ganajur Main Gold Project*, Internal memorandum prepared by SRK Consulting for Deccan Gold Mines Limited, 3 September 2011.

5 MINERAL PROCESSING AND METALLURGICAL TESTING AND RECOVERY

5.1 Introduction and background

Prior to the commencement of the Feasibility Study (FS) for the Ganajur Gold Project, a Scoping Study level metallurgical assessment was performed by SRK in 2011/2012. The key findings from this work were:

- The sulphide resource gold mineralisation consisted of free gold and discrete fine gold locked in pyrite
- Direct cyanidation from a bulk sulphide composite sample achieved gold recovery in the 75% to 77% range
- High gold recovery (+90%) could be achieved via flotation for the recovery of the sulphides followed by roasting and cyanidation of the calcine.

An oxide bulk composite sample was tested in mid-2015 and high gold recovery (+90%) via direct cyanidation leach was achieved.

The metallurgical testwork for the Scoping Study phase was performed at the ALS Metallurgical Laboratory located in Perth, Western Australia. The ongoing testwork for the FS phase continued at the ALS Laboratory.

5.2 Geological Mineral Resource characteristics and metallurgical sample selection

The Geology and Mineral Resource Estimate section of the Ganajur Gold Project Feasibility Study report provides a comprehensive description of the Ganajur Main Gold deposit resource.

In summary, the key geological characteristics of the Ganajur Main Gold deposit Mineral Resource that influences the metallurgical response can be described as:

- The Ganajur Main Gold deposit gold mineralised resource is epigenetic in nature and is strata-bound within a cherty iron bound formation. The gold mineralisation is within two lithologies, the oxide ore is present in a Brecciated Ferruginous Chert (BFG) and the sulphide ore is present in the Brecciated Sulphidic Chert (BSC).
- The hanging and footwall waste lithology consists of predominately of quartzite and greywacke material.
- The resource is approximately 800 m in strike length, varies from 15 m to 40 m in thickness and dips at 60° to 70°. The known gold mineralisation occurs at the surface and continues to a depth of 80 m to 100 m.
- The total Ganajur Main Gold deposit Ore Reserve has been estimated to contain 0.7 Million tonnes (Mt) of oxide ore at 2.6 g/t gold and 1.8 Mt of sulphide ore at 3.7 g/t gold.
- The majority of the gold mineralisation is associated with sulphides (predominately pyrite and minor arsenopyrite); however, the first 20 m from the surface, the sulphides have been oxidised. No transition or supergene zone exists between the oxide and sulphide zones.

Figure 5.1 presents a typical cross-section located within the widest and central part of the orebody (40 m width), which contains approximately 70% of the resource tonnage over a 300 m strike length.

Table 5.1 and Table 5.2 present the samples that were selected for metallurgical testwork from the two main lithology units of the Ganajur Main Gold deposit resource, the Brecciated Ferruginous Chert (BFC – oxide ore) and Brecciated Sulphidic Chert (BSC – sulphide ore). The cross sections that the drill core samples were selected from are shown in Figure 5.2.

Table 5.1 BFC oxide variability samples

Sample no.	Section	Drillholes	Interval (mdh)	Sample Wt (kg)	Au (g/t)	As (ppm)
1	H-H'	GMC 50	7.60	9.2	2.1	1820
2	L-L'	GMC 60	17.00	22.5	4.8	2280

Table 5.2 BSC sulphide variability samples

Sample no.	Section	Drillholes	Interval (mdh)	Sample Wt (kg)	Au (g/t)	As (ppm)
GM1	C-C'	GMC-6	13.8	27.6	4.0	1053
		GMC-15	20.50	41	2.8	3795
		Total	**34.3**	**68.6**	**3.3**	**2692**
GM2	F-F'	GMC-16	32.44	64.88	8.9	8114
GM3	F-F'	GMC-18	27.00	54	4.6	5500
GM4	G-G'	GMC-19	25.10	50.2	3.9	3780
GM5	H-H'	GMC-17A	30.35	60.7	7.1	7491
GM6	I-I'	GMC-12	3.50	7	3.0	3921
		GMC-13	17.05	34.1	2.8	3320
		GMC-14	6.20	12.4	2.4	365
		Total	**26.75**	**53.50**	**2.7**	**2714**

Figure 5.1 Ganajur Main Gold deposit cross section G-G’

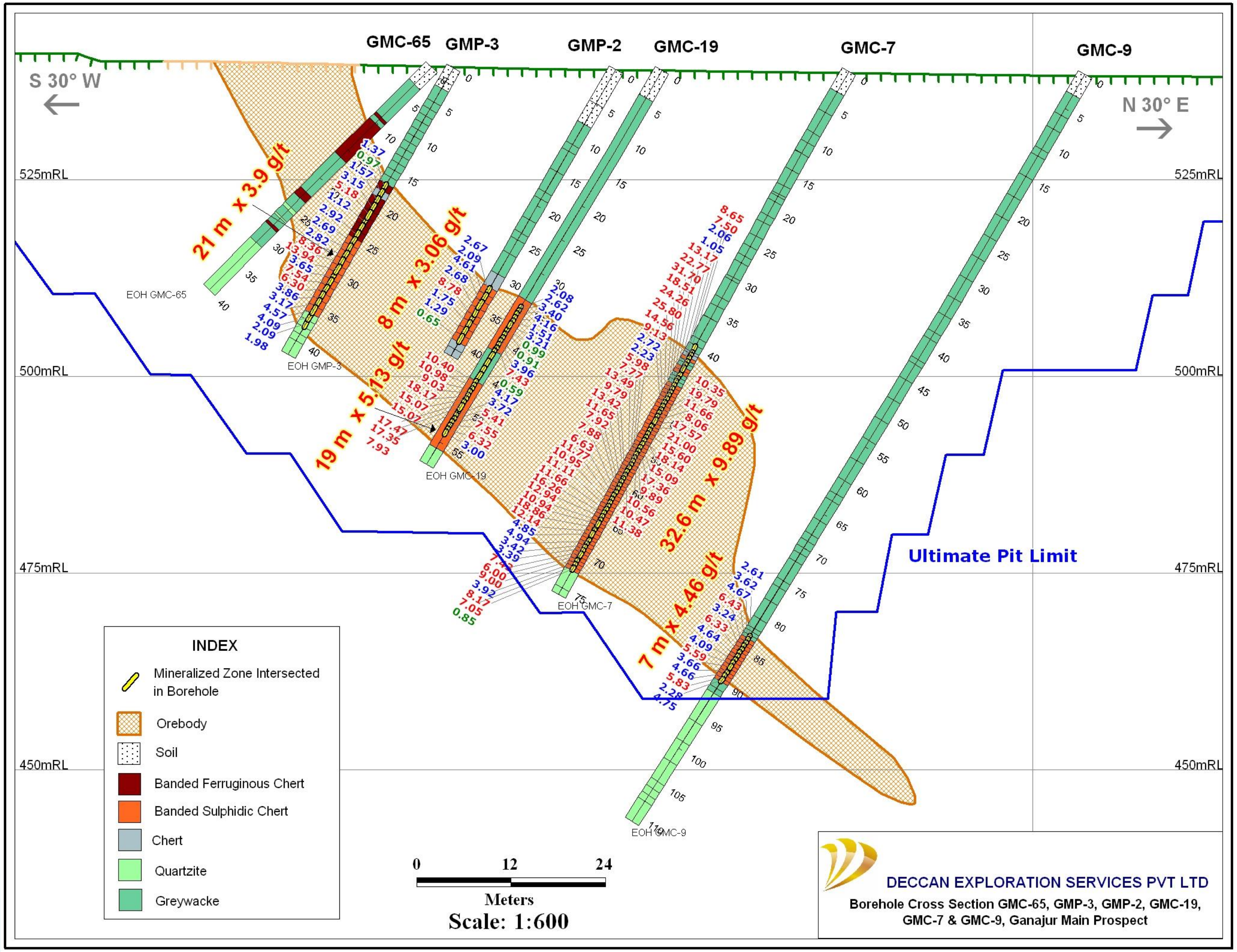
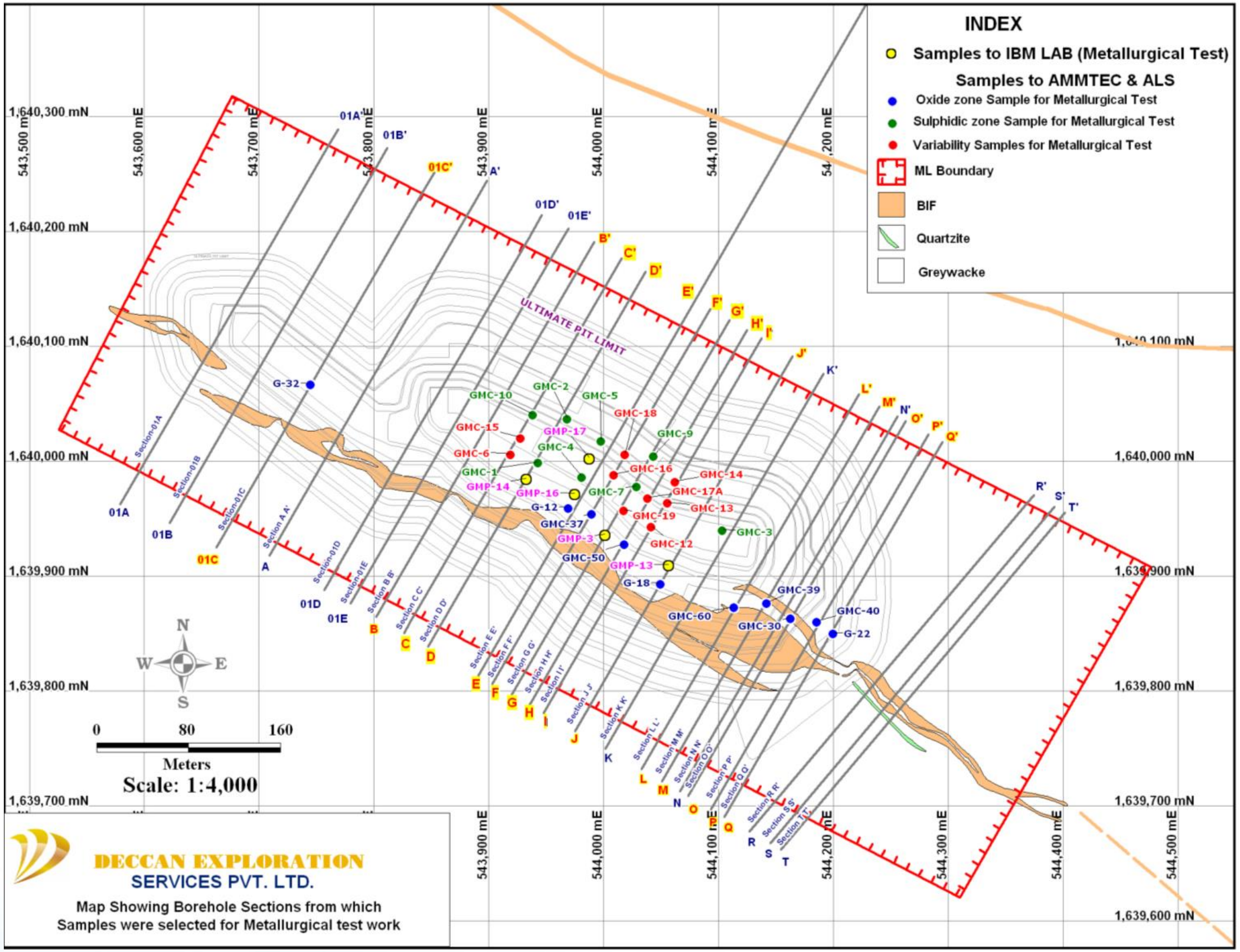


Figure 5.2 Variability sample location from cross-sections



For reference, the bulk oxide and sulphide composite sample details that were selected for metallurgical testwork prior to the above variability samples, are shown in Appendix 5A.

The rationale for the selection of the variability samples from the Ganajur Main Gold deposit resource is summarised as follows:

- The sulphide variability samples were sourced from the area of the resource where the ore widths are within the thickest (approximately 30 m to 40 m) part of the orebody – cross section DD' to KK'. Hence these samples are representative of a significant proportion of the total Ganajur Main Gold deposit resource (approximately 70% to 80%).
- The only spatial sulphide variability sample selected outside of the high ore width area was GM 1 on cross section CC'. The core library inventory was recorded as half PQ for the drillholes selected, however on retrieval of the core trays, the residual core was only quarter diamond core. Deccan Exploration was not prepared to release the residual quarter core.
- The Ganajur Main Gold deposit mineralisation is postulated as a single epigenetic event and is tightly strata-bound within a cherty iron bound formation. The DESPL geological team indicated a high degree of confidence that the gold mineralisation is consistent and uniform across the strike length and at depth of the sulphide resource. It was on this basis that no further samples were selected.
- Generally, both hanging and footwall waste dilution has been included (1 m) within the mineralised variability samples as well as internal waste if less than 3 m in width. It has been assumed that grade control could selectively mine the internal waste if greater than 3 m in width due to the high visual difference between the ore and waste material.
- The drill core selected for the variability samples provided a range of gold head grades with varying arsenic levels, which provides the opportunity for developing recovery models via regression analyses if significant relationships are evident.
- The wide ore widths enabled the initial metallurgical tests to commence with comminution characterisation followed by flotation/carbon-in-leach (CIL) on the comminution products (SMC and BBWi).

5.3 Testwork program outline

The main testwork flowsheets that were developed for the metallurgical testwork program as well as the testwork data results are presented in ALS Report No A171105. This report is shown in Appendix 5B. The Scoping Study testwork indicated that Ultra Fine Grinding (UFG) and CIL on the sulphide concentrates from the Ganajur Main Gold deposit resource offered the most likely opportunity for achieving a gold recovery that provided an optimum financial outcome for the project. It was on this basis that the testwork program was developed and progressed for the FS.

The metallurgical testwork was performed via the following key tests and sequence:

- Three (GM3, GM5 and GM6) of the five variability samples were comminution characterised by SAG Mill Competency (SMC), Bond Ball Mill Work Index (BBWi) and the Bond abrasion index testwork.
- Unconfined compressive tests (UCS) were performed from geotechnical drill holes that passed thru the BSC ore.
- Direct cyanidation leach was performed on samples GM3, GM5 and GM6 so as to compare the gold recovery differential against the flotation, UFG/CIL route.
- Direct cyanidation leach was performed on the oxide variability samples GMO1 and GMO2.
- The determination of the optimum grind liberation size for gold and sulphide sulphur recovery via flotation was performed on samples GM3, GM5 and GM6.

- The optimised flotation reagent, time and grind size obtained from the sighter tests progressed to bulk flotation tests. The bulk tests used 17 kg of sample in a 40-litre cell which enabled a large mass of sulphide concentrate to be produced for the following UFG and CIL testwork.
- The sulphide concentrates from the five variability samples were UFG to a P_{80} of 10 microns and then CIL leached for 48 to 72 hours.
- The bulk sulphide composite sample (BSC) from the Scoping Study was used to produce the high mass of sulphide concentrate necessary to perform the IsaMill Signature plot testwork (UFG specific power consumption), the Cyanide Destruct, Arsenic Removal, thickener settling tests and rheology stages of testwork.

5.4 Head analysis and ore mineralogy

Each of the variability samples after the comminution testwork was completed were assayed via fire assay for gold, Leco analyses for sulphur and carbon and ICP-EOS for a suite of cations/anions and metals. The sulphide variability samples were also x-ray diffraction (XRD) analysed for the determination of the major minerals present.

The chemical and XRD analyses for the sulphide variability samples are presented in Table 5.3 and Table 5.4, respectively. The chemical analyses for the oxide variability samples are presented in Table 5.5.

Table 5.3 Chemical analyses on sulphide variability samples

Analyte	Unit	Ganajur – Variability comp.				
		GM2 Sulphide	GM3 Sulphide	GM4 Sulphide	GM5 Sulphide	GM6 Sulphide
Au_1	g/t	10.2	5.16	3.77	8.27	2.52
Au_2	g/t	10.6	4.95	4.13	7.59	2.65
Au (Average)	g/t	10.4	5.06	3.95	7.93	2.59
Ag	g/t	1.2	0.6	0.6	1.5	0.6
Al	%	2.00	2.60	4.00	2.88	4.76
As	ppm	8600	6790	4400	8590	3130
Ba	ppm	100	160	240	140	280
Be	ppm	<20	<20	<20	<20	<20
Bi	ppm	<25	<25	<25	<25	<25
C_{TOTAL}	%	3.87	3.93	3.21	3.75	3.39
$C_{ORGANIC}$	%	0.39	0.33	0.33	0.39	0.27
$C_{CARBONATE}$	%	17.4	18.0	14.4	16.8	15.6
Ca	%	1.90	1.74	2.00	1.91	2.03
Cd	ppm	20	<20	<20	<20	<20
Co	ppm	<20	<20	<20	<20	<20
Cr	ppm	75	50.00	50	50.00	75
Cu	ppm	40	36	20	72	34
Fe	%	19.7	16.0	14.7	16.7	12.0
Hg	ppm	<0.1	<0.1	<0.1	<0.1	<0.1
K	%	0.4750	1.0000	1.10	0.8750	1.53
Li	ppm	<20	<20	<20	<20	<20
Mg	%	1.16	1.20	1.28	1.32	1.56
Mn	ppm	800	580	800	760	780
Mo	ppm	<20	<20	<20	<20	<20
Na	ppm	2650	3400	7700	3100	5550
Ni	ppm	40	40	40	40	40
P	ppm	1000	1000	750	1000	750
Pb	ppm	40	<20	20	20.00	<20
S_{TOTAL}	%	4.98	3.78	2.44	5.40	2.52
$S_{SULPHIDE}$	%	4.80	3.76	2.14	5.24	2.60
SiO_2	%	44.6	45.0	54.2	47.0	49.0
Sb	ppm	7.8	5.0	4.2	6.6	3.0
Sr	ppm	150	120	185	145	190
Te	ppm	0.4	<0.2	0.4	0.4	<0.2
Ti	ppm	800	1200	1600	1400	2200
V	ppm	25	40	45	40	65
Y	ppm	<100	<100	<100	<100	<100
Zn	ppm	60	60	65	65	50
True SG	**g/mL**	**3.209**	**3.121**	**3.038**	**3.158**	**3.018**

Table 5.4 XRD analyses on sulphide variability samples

Mineral ID	Ganajur – Mass % – Variability comp.				
	GM2 Sulphide	GM3 Sulphide	GM4 Sulphide	GM5 Sulphide	GM6 Sulphide
Clay mineral	1	1	1	1	2
Kaolinite	0	<1	0	0	<1
Serpentine	0	0	<1	0	0
Chlorite	0	1	<1	<1	<1
Talc	0	<1	0	0	0
Biotite – annite	<1	2	1	1	1
Muscovite – paragonite	6	5	7	5	8
Amphibole	0	0	1	0	0
Microcline – rutile – titanate	<1	1	<1	1	1
Sodic and/or calcic plagioclase	1	1	4	1	1
Alpha quartz	48	48	49	50	51
Dolomite – ankerite	8	7	9	8	12
Siderite	*26	*24	*22	*21	*18
Jarosite	<1	1	<1	<1	<1
Gypsum	0	<1	0	0	0
Pyrite	9	9	5	12	5
Arsenopyrite	1	1	1	1	1

Table 5.5 Chemical analyses on oxide variability samples

Analyte	Unit	Ganajur GMO1 Oxide – Variability comp.	Ganajur GMO2 Oxide – Variability comp.
Au_1	g/t	2.07	4.83
Au_2	g/t	2.17	4.77
Au (Average)	g/t	2.12	4.80
Ag	g/t	0.3	0.3
As	ppm	1820	2280
Bi	ppm	<25	<25
C_{TOTAL}	**%**	**0.18**	**0.09**
$C_{ORGANIC}$	**%**	**0.15**	**0.06**
$C_{CARBONATE}$	%	0.15	0.15
Cd	ppm	<20	<20
Co	ppm	<20	<20
Cr	ppm	50	25
Cu	ppm	5	20
Fe	%	22.1	25.7
Hg	ppm	< 0.1	< 0.1
Ni	ppm	40	40
Pb	ppm	40	40
S_{TOTAL}	**%**	**0.06**	**0.12**
$S_{SULPHIDE}$	**%**	**<0.02**	**0.04**
SiO_2	%	57.4	54.6
Sb	ppm	4.6	3.4
Se	ppm	<5	<5
Te	ppm	<0.2	0.2
Zn	ppm	55	45
True SG	**g/mL**	**3.079**	**3.174**

5.5 Comminution

As presented in Table 5.6, the comminution characteristics for the Ganajur Main Gold deposit resource have a moderate to high competency with a moderate resistance to abrasion.

Of note is that there is no significant difference in comminution characteristics between the oxide and the sulphide ores, which is logical in that the only key difference between the two ore types is that the sulphide content (approximately 5% to 10% content) has been converted to hematite/goethite.

The 85th percentile of these results have been selected for the process design criteria in the equipment sizing of the ball mill as well as for the three-stage crushing circuit.

5.6 Flotation

The determination of the optimum grind liberation size for flotation was performed at P_{80} 75, 106 and 125 microns on the variability sulphide samples GM3, GM5 and GM6. The results from this testwork are presented in Table 5.7.

All the sighter flotation tests that were performed on the sulphide variability samples and the BSC sample at the optimum grind liberation size of P_{80} 75 microns are presented in Table 5.8. These sighter tests were carried out to confirm that high gold and sulphide sulphur recovery into a rougher concentrate stream could be achieved for all the sulphide samples. The sighter flotation tests were performed on 2 kg sample lots in a 4.4-litre Agitair laboratory flotation cell.

Flotation testing progressed to a large 40-litre cell using 17 kg of sample per test, which enabled a high mass of rougher concentrate to be produced for subsequent UFG and CIL testwork.

One of the potential mine and mill production schedules under consideration for the Ganajur project, required a mill feed blend of approximately 30/70% of oxide ore to sulphide ore. A series of further flotation tests were carried out at varying oxide to sulphide blend ratios to confirm the flotation response, with results presented in Table 5.10.

The key outcomes from all the flotation testwork that was performed are summarised below;

- The optimum grind liberation size for flotation is at a P_{80} of 75 microns or less.
- The sighter rougher flotation tests at 20 minutes' retention time, with the staged addition of potassium amyl xanthate and copper sulphate achieved a recovery range of 93% to 96% for gold and 94% to 97% for sulphide sulphur (%SS). The full test results are presented in Table 5.8.
- The bulk rougher flotation results achieved a higher recovery (1% to 3%) for both gold and sulphide sulphur due primarily to a higher mass pull compared to the sighter tests, whereby the rougher concentrate sulphide sulphur grade was in the 20% to 24% range.
- The bulk rougher flotation tests on the sulphide variability samples demonstrated that a gold recovery of 95% and sulphide sulphur recovery at 97% can be achieved by a mass pull targeting a sulphide sulphur concentrate grade in the 20% to 22% range.
- The oxide to sulphide blend testwork demonstrated that flotation recoveries similar to the sulphide ore samples could not be achieved, despite an increase in reagent dosage and flotation time.

Table 5.6 Comminution results on variability and bulk composite samples

Sample no.	Sample ref.	DWi	A	b	A*b	SG	RWi	BWi	Ai	Mia	Mib	Mic
Oxide bulk composite	2015	4.69	70.4	0.86	60.5	2.8	19.1	18.4	0.3023	14.0	25.8	5.0
Sulphide bulk composite	2011						21.5	16.9	0.5006			
Sulphide variability samples												
GM 3	GMC-18	8.01	90.8	0.41	37.2	3.00		17.3	0.3861	20.2	24.3	8.0
GM 5	GMC-17A	8.98	100.0	0.34	34.0	3.06		16.8	0.3779	21.7	23.7	8.8
GM 6	GMC-12, GMC-13, GMC-14	6.51	74.9	0.61	45.7	2.99		16.6	0.3002	17.2	23.0	6.5
Minimum		6.5			34.0	3.0		16.6	0.300	17.2	23.0	6.5
Maximum		9.0			45.7	3.1		17.3	0.386	21.7	24.3	8.8
Average		7.8			39.0	3.0		16.9	0.355	19.7	23.7	7.8
85th percentile		8.7			43.2	3.0		17.2	0.384	21.3	24.1	8.6

Table 5.7 Grind optimisation testwork

Sample	Concentrate			Tailings			% recovery		Assays			
	wt%	Au g/t	SS %	wt%	Au g/t	SS %	Au	SS	Au-CH	Au-AH	SS-CH	SS-AH
GM3												
P_{80} - 125 micron	15.1	34.4	24.1	84.9	0.43	0.24	93.5	94.7	5.5	5.1	3.8	3.8
P_{80} - 106 micron	15.9	31.8	22.9	84.1	0.40	0.18	93.8	96.0	5.4	5.1	3.8	3.8
P_{80} - 75 micron	16.5	31.4	22.5	83.5	0.27	0.14	95.8	96.9	5.4	5.1	3.8	3.8
GM5												
P_{80} - 125 micron	18.5	36.8	26.3	81.5	0.94	0.36	89.9	94.3	7.6	7.9	5.2	5.2
P_{80} - 106 micron	19.4	38.2	27.7	80.6	0.72	0.30	92.7	95.7	8.0	7.9	5.6	5.2
P_{80} - 75 micron	20.3	34.2	24.3	79.7	0.72	0.22	94.1	96.6	7.4	7.9	5.1	5.2
GM6												
P_{80} - 125 micron	12.1	21.3	19.2	87.9	0.34	0.24	89.6	91.7	2.9	2.6	2.5	2.6
P_{80} - 106 micron	11.6	26.3	20.5	88.4	0.33	0.20	91.3	93.0	3.3	2.6	2.5	2.6
P_{80} - 75 micron	11.6	24.0	20.0	88.4	0.24	0.18	92.9	93.6	3.0	2.6	2.5	2.6

Table 5.8 Sighter flotation tests on sulphide samples

Sample	Test no.	Concentrate			Tailings			% recovery		Assays			
		wt%	Au g/t	SS %	wt%	Au g/t	SS %	Au	SS	Au-CH	Au-AH	SS-CH	SS-AH
BSC	6573	15.7	37.5	30.2	84.3	0.45	0.16	93.9	97.2	6.3	7.0	4.9	3.9
	6574	15.3	38.4	30.3	84.7	0.43	0.22	94.2	96.1	6.2	7.0	4.8	3.9
	6775	16.0	38.4	29.9	84.0	0.44	0.18	94.4	96.9	6.5	7.0	4.9	3.9
GM2	7226	24.1	35.6		75.9	0.6		95.0		9.0	10.4		
GM3	6745	16.5	31.4	22.5	83.5	0.27	0.14	95.8	96.9	5.4	5.1	3.8	3.8
GM4	7227	17.6	22.3		82.4	0.32		93.8		4.2	4.0		
GM5	6748	20.3	34.2	24.3	79.7	0.55	0.22	94.1	96.6	7.4	7.9	5.1	5.2
GM6	6751	11.6	24.0	20.0	88.4	0.24	0.18	92.9	93.6	3.0	2.6	2.5	2.6

Table 5.9 Bulk flotation tests on sulphide samples

Sample	Test no.	Concentrate			Tailings			% recovery		Assays			
		wt%	Au g/t	SS %	wt%	Au g/t	SS %	Au	SS	Au-CH	Au-AH	SS-CH	SS - AH
BSC	7041-7047	18.4	31.2	24.5	81.6	0.32	0.10	95.7	98.2	6.0	5.7	4.6	4.9
GM2	6852	18.1	49.5	24.9	81.9	0.73	0.12	93.8	97.9	9.6	10.4	4.6	4.8
GM3	6752	15.6	34.3	22.2	84.4	0.27	0.12	95.9	97.2	3.6	5.1	3.6	3.8
GM4	6853	11.4	32.8	19.8	88.6	0.36	0.12	91.2	97.9	4.1	4.0	2.4	2.1
GM5	6753	20.3	35.0	21.5	79.7	0.23	0.16	97.5	97.2	7.3	7.9	5.2	4.5
GM6	6754	11.5	28.1	22.3	88.5	0.19	0.14	95.2	95.4	3.4	2.6	2.7	2.7

Table 5.10 Oxide/Sulphide blend sighter flotation testwork at P_{80} 75 microns

Sample	Test no.	Concentrate			Tailings			% recovery		Assays			
		wt%	Au g/t	SS %	wt%	Au g/t	SS %	Au	SS	Au-CH	Au-AH	SS-CH	SS-AH
BOC + BSC - 20/80 % Blend	6854	12.6	38.3	28.6	87.4	0.55	0.22	91.0	94.9	5.3	5.2	3.8	3.7
BOC + BSC - 40/60 % Blend	6855	11.4	38.7	25.1	88.6	0.55	0.16	90.1	95.3	4.9	4.7	3.0	2.9
GMO2 + GM5 - 30/70 % Blend STD	7570	12.3	38.5	28.5	87.7	1.08	0.14	83.4	95.7	6.7	6.1	3.7	3.4
GMO2 + GM5 - 30/70 % Blend MOD	7571	14.8	32.8	23.8	85.2	0.99	0.14	85.1	96.7	6.8	6.1	3.6	3.4
GMO1 - 100% Oxide	7223	4.7	10.9		95.3	0.91		58.8		2.1	2.1		
GMO2 - 100% Oxide	7224	5.3	20.4		94.7	2.15		57.6		4.8	4.8		

5.7 UFG and CIL on sulphide concentrates

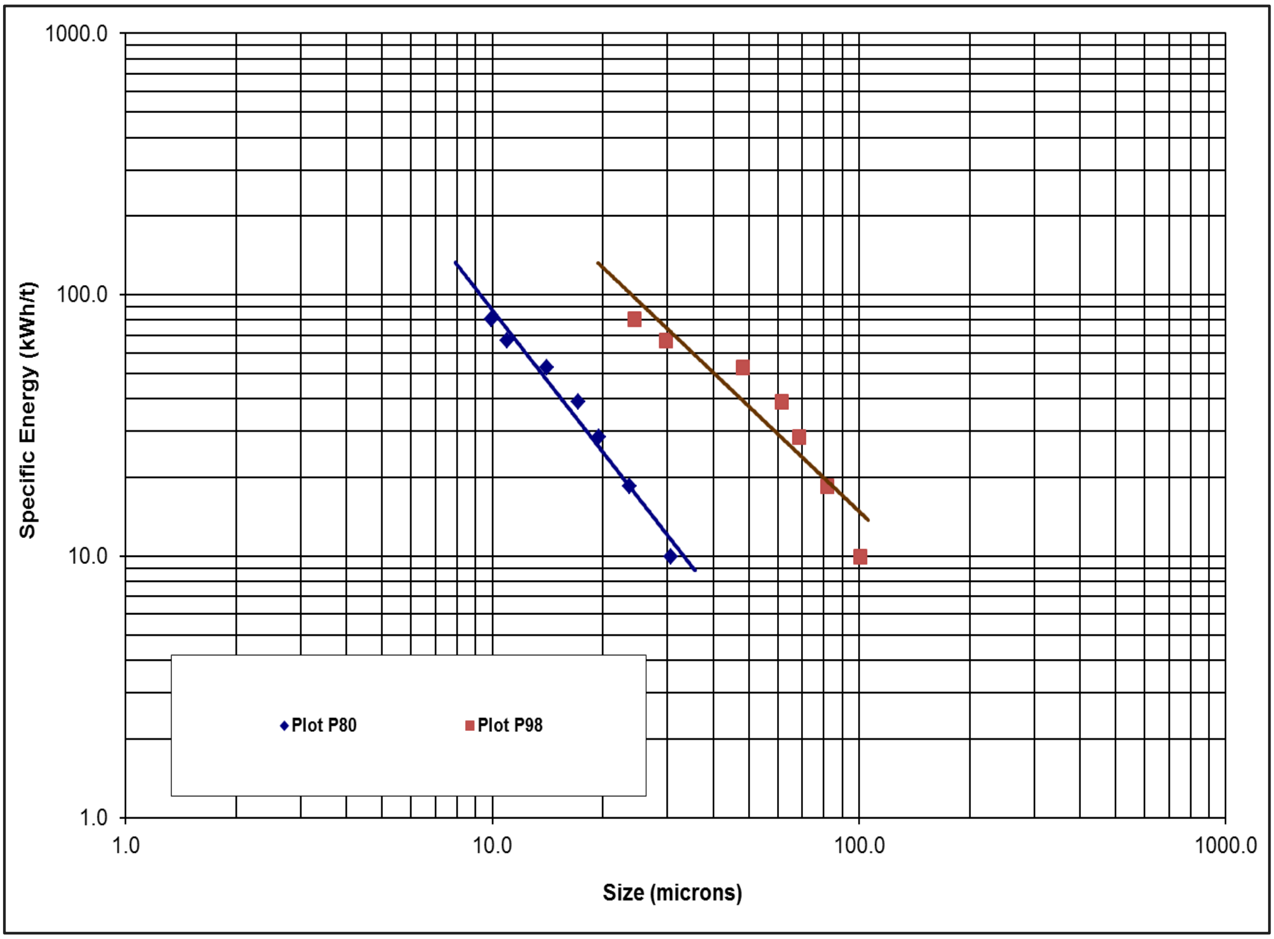
The sulphide concentrates from the bulk flotation tests on the sulphide variability samples were UFG to a P_{80} of 10 microns (via laser sizing analyses). The estimated power consumption to achieve this grind size is presented in Figure 5.3.

The CIL tests were performed under the following test conditions:

- An initial cyanidation concentration of 0.3% (wt%/vol) NaCN, pH 10.5 with hydrated lime addition, 0.10 kg/t of Lead Nitrate and 20 gpl of activated carbon addition.
- The oxygen concentration in slurry was targeted at 10 ppm to 20 ppm, the cyanide levels were maintained above 0.2% and the total leach retention time was 72 hours. These CIL conditions were adopted as the starting point of determining the optimum leach gold recovery from the sulphide concentrates.
- Preliminary Scanning Electron Microscopic (SEM) work during the Scoping Study identified that the gold particles encapsulated in the sulphides were in the 5 micron to 10 micron size range, however finer gold particles than 5 microns were identified.

The leach recovery results (ALS test no. 6849-51, 6894-95) from these initial test conditions as presented in Table 5.11 were variable with the CIL residues assaying from 4 g/t Au to 7.6 g/t Au. The CIL tests (ALS test no. 6923-24) were repeated for samples GM2 and GM4 due to the significant difference of the gold residue grades. To confirm if the organic carbon present in the concentrates could be responsible for the gold residue variability, higher concentrations of carbon were tested on samples GM3 and GM4 (ALS test no. 6951-53). A slight improvement in gold recovery was achieved as the CIL residues decreased by approximately 0.5 g/t.

Figure 5.3 IsaMill signature plot



To confirm the reasons for the variable CIL residue gold grades, samples GM2, GM3 and GM4 were sent to the mineralogical laboratory AMTEL in Canada for a deportment assessment of the gold contained in the CIL sulphide residues and the flotation concentrate. The key findings from the investigation performed by AMTEL were quoted as:

- Water soluble gold salts are insignificant (<0.01 g/t Au) in all three UFG/CIL residues. This indicates good activated carbon management.
- Residual free and attached gold grains which failed to dissolve make up 4% of each the GM2 and GM3 UFG/CIL residue grade; and a significant greater proportion (28%) of the GM4 UFG/CIL residue grade:
  - residual free gold grains are primarily <7 µm, likely remnants of incompletely dissolved larger particles.
  - surface analysis of free gold grains from the GM4 FC and UFG/CIL residue revealed a build-up of Ag, likely as Ag(S,SCN,I) compound, which hampered the complete dissolution of gold.
- Enclosed gold grains accounts for 7% to 12% of the UFG/CIL residue grade; corresponding to 0.3 g/t Au to 0.8 g/t Au.
- Size-by-size grade of gold enclosed in sulphides indicates a liberation target of 10 µm for the UFG. However, there are tiny gold inclusions (<1 µm) some of which will not liberate even after UFG.
- Enclosed gold grains in sulphide particles >10 µm account for 2% of the leach residue grade.
- Sub-microscopic gold in pyrite and arsenopyrite sets the absolute minimum grade of the UFG/CIL tails as this gold is refractory to direct cyanidation. For the GM2, GM3 and GM4 UFG/CIL, this is 3.1 g/t Au, 4.3 g/t Au and 4.0 g/t Au, respectively.

- Surface gold on C-matter carries the remaining 4% to 5% of the residual gold in the UFG/CIL residues or 0.2 g/t Au to 0.3 g/t Au. This indicates that the C-matter is a weak preg-robber.

The full AMTEL report is presented in Appendix 5C.

Whilst the above gold deportment assessment was in progress and on the assumption that some of the gold losses could be attributed to organic carbon and hence preg-robbing, some further CIL tests with blinding agents (diesel and kerosene) were carried out. The results for these tests showed that no improvement in gold recovery and hence supported the findings of the AMTEL deportment assessment. The details of these tests and the overall FS testwork performed at ALS are presented in Appendix 5B.

Based on the AMTEL report findings a CIL optimisation test regime was developed to counter the film formation by significantly increasing the lead nitrate addition rate. Also, a pre-aeration stage was introduced to confirm if a reduction in cyanide consumption could be achieved by oxidising the ferrous ions to ferric and thereby precipitating the iron as a hydroxide.

The optimised CIL test regime results based on the sulphide concentrate generated from the BSC are presented in Table 5.12 and are shown from ALS test no 7455 to 7460. The results indicated higher gold residue losses compared to the test result achieved from the initial CIL test conditions as per ALS test no. 7128.

On review of technical literature on the leaching of UFG sulphide concentrates it became evident that some of the CIL test procedure adopted for this optimisation regime could negatively impact gold recovery, which these tests have demonstrated.

A revised CIL optimisation regime was developed which included changes in the test procedure as identified from the technical literature findings. Due to the shortage of the test sample inventory, sample GM5 was selected to progress this phase of work. The key changes for this revised phase of CIL optimisation testwork are outlined below as well as the outcomes:

- No pre-aeration stage was incorporated, however a 15-minute pre-conditioning stage for the addition of the lead nitrate and hydrated lime addition for pH control was adopted
- The CIL tests were performed immediately after the UFG stage so as to eliminate any potential aging affect
- The initial cyanide concentration dosages were progressively decreased from 0.3% to 0.20% to confirm if cyanide consumption could be reduced without impacting gold recovery as well as limit the amount of iron and arsenic release into the leach solution
- The CIL gold grade residues did not decrease and are in alignment with the CIL residue achieved from the initial CIL test conditions
- The cyanide consumption decreased by 50% from the initial CIL test conditions and significant reductions in both iron and arsenic dissolution were also achieved (70% to 80% decrease).

Table 5.11 Initial CIL tests on sulphide variability samples

Sample	ALS test no.	Pre-leach							Post-leach						
		Au AH ppm	Au CH ppm	As %	Sulphide sulphur %	RT hrs	Solids slurry %	Carbon addition g/L	$Pb(NO)_3$ kg/t	Fe solution ppm	As solution ppm	Au residue ppm	Au % recovery	NaCN kg/t	Lime kg/t
GM3	6849	32.1	32.1	3.84	22.2	72	40	20	0.1	810	551	5.69	82.3	10.0	2.4
GM5	6850	35.5	33.9	3.5	25.1	72	40	20	0.1	944	646	4.77	85.9	10.8	2.3
GM6	6851	23.8	20.4	2.36	22.3	72	40	20	0.1	1282	1199	4.91	76.0	11.8	1.0
GM2	6894	61.4	35.9	4.48	24.9	72	40	20	0.1	1153	1074	4.03	88.8	12.5	1.5
GM4	6895	34.8	32.4	4.28	19.8	72	40	20	0.1	1251	1261	7.66	76.3	12.2	1.3
GM2	6923	61.4	51.9	4.48	24.9	72	40	30	0.1	1065	466	4.03	92.2	13.4	4.1
GM4	6924	34.8	32.9	4.28	19.8	72	40	30	0.1	802	442	7.45	71.1	12.8	3.4
GM3	6951	32.1	30.4	3.84	22.2	72	25	40	0.1	1455	1335	4.82	84.2	28.3	1.6
GM3	6952	32.1	31.2	3.84	22.2	72	25	60	0.1	1113	1189	5.01	84.0	28.4	2.5
GM4	6953	34.8	32.6	4.28	19.8	72	25	60	0.1	1244	1447	6.87	79.0	28.4	1.3

Table 5.12 Optimisation CIL tests on sulphide variability samples

Sample	ALS test no.	Pre-leach							Post-leach						
		Au AH ppm	Au CH ppm	As %	Sulphide sulphur %	RT hrs	Solids slurry %	Carbon addition g/L	$Pb(NO)_3$ kg/t	Fe solution ppm	As solution ppm	Au residue ppm	Au % recovery	NaCN kg/t	Lime kg/t
BSC	6584	38.1	38.2			24	17.45	50				4.82	87.4	99.8	0.9
BSC	7128	31.2	30.9	2.7	24.5	72	30	60	0.1	845	824	5.00	83.8	25.2	3.3
BSC	7455	31.2	30.9	2.7	24.5	42	40	50	0.5	805	25	5.80	81.3	11.6	10.5
BSC	7456	31.2	31.3	2.7	24.5	42	40	50	1.0	868	20	5.84	81.4	11.7	9.5
BSC	7457	31.2	32.1	2.7	24.5	42	40	50	2.0	892	35	5.73	82.2	11.9	10.6
BSC	7458	31.2	31.3	2.7	24.5	42	40	50	0.5	837	62	5.96	81.0	12.5	6.0
BSC	7459	31.2	31.2	2.7	24.5	42	40	50	1.0	848	75	5.59	82.1	12.5	5.8
BSC	7460	31.2	31.6	2.7	24.5	42	40	50	2.0	855	60	6.1	80.7	12.5	6.3
GM5	7549	35.5	33.5	3.5	25.1	48	35	50	1.0	594	484	4.85	85.5	9.7	2.2
GM5	7550	35.5	34.4	3.5	25.1	48	35	50	1.0	446	354	4.97	85.5	8.2	2.2
GM5	7551	35.5	34.7	3.5	25.1	48	35	50	1.0	250	166	4.80	86.2	6.1	2.5
GM5	7552	35.5	32.9	3.5	25.1	72	40	20	0.1	1226	905	5.11	84.5	12.0	2.2

5.8 Cyanide Destruct and Arsenic Removal

To comply with international and Indian EPA environmental standards on cyanide control and management, Cyanide Destruct testwork was performed on the CIL residue stream.

The SO_2/oxygen method for cyanide destruction was selected due primarily to the operational ease and safety aspects of this technology compared to the Caro's Acid method.

As shown in Table 5.11, arsenic dissolves during the CIL process and requires removal from solution and stabilisation as a solid, prior to disposal into the tailings storage facility (TSF). The removal of arsenic from solutions via the addition of ferric sulphate is recommended by the US EPA as the Best Demonstrated Available Technology (BDAT).

Hence, testing with ferric sulphate addition after the completion of the Cyanide Destruct tests were then performed.

The key outcomes from the Cyanide Destruct and Arsenic Removal testwork are summarised below:

- Due to the high residual WAD cyanide levels (+500 ppm) after CIL, a two-stage Cyanide Destruct test with 60 minutes/stage retention was selected as the optimum operating strategy.
- The testwork determined that the optimum SO_2/WAD cyanide ratio for reducing the WAD cyanide levels to less than 10 ppm was 4.7:1, which is equivalent to a SMBS/g WAD cyanide ratio of 6.9. This equates to a SMBS consumption of 5.2 kg/t of concentrate.
- Due to the high iron dissolution during CIL, the amount of copper sulphate required to act as a catalyst is high at 3.2 kg/t, which is based on a ferrous ion content of 250 ppm in solution.
- Hydrated lime consumption to maintain the pH in the range of 8 to 9 is low at 1 kg/t of sulphide concentrate.
- A stoichiometric ratio of 3.6:1 for Fe:As was successful in reducing the arsenic levels from 680 ppm to less than 0.4 ppm with 60 minutes of retention time.
- Based on the optimised CIL conditions, a ferric sulphate consumption of 6 kg/t of sulphide concentrate will be required to remove 250 ppm of arsenic in solution. A sulphuric acid consumption of 8.7 kg/t of concentrate is necessary to maintain the pH at 6.0.

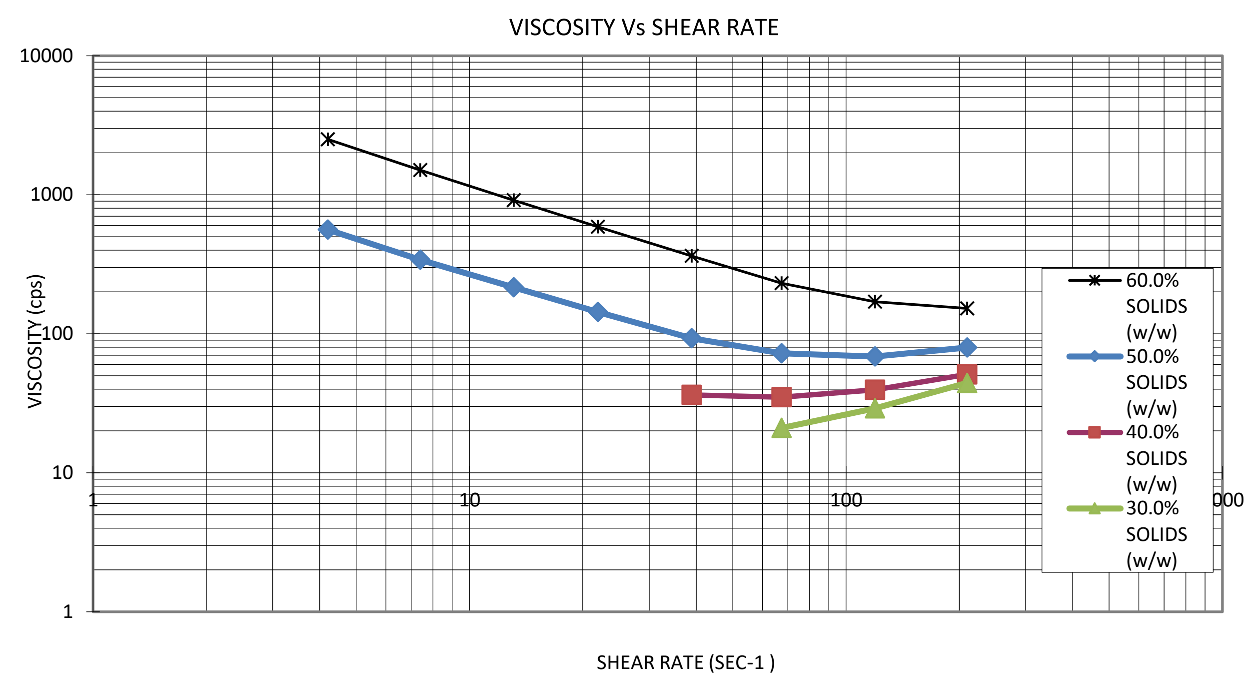
5.9 Ancillary testwork

Slurry rheology and oxygen uptake rate testwork was performed on the BSC sulphide concentrate sample and thickener settling tests on both the concentrate and flotation tail originating from the BSC sample were also carried out.

The key outcomes and results from this work are outlined below:

- At pH 10, the viscosity of the flotation sulphide concentrate with slurry % solids between 40% and 60% is below 1000 cps over the typical shear rates encountered in the process plant operation, as shown by Figure 5.4.
- The oxygen uptake tests indicated that the oxygen consumption during the CIL of the sulphide concentrates will be low at less than 0.5 t/day.
- The settling testwork performed by Outotec for the flotation concentrate after UFG and the flotation tailings streams produced the following test results:
  - the UFG sulphide concentrate settling rate was 0.25t/m^2/hr and produced a U/F slurry density of 60% with a flocculant consumption of 15 g/t to 20 g/t
  - the flotation tailings settling rate was 1.5t/m^2/hr and produced a U/F slurry density of 60% with a flocculant consumption of 15 g/t to 20 g/t.

Figure 5.4 Rheology results on sulphide concentrate



5.10 Cyanidation leach – whole ore

Direct CIL grind liberation size vs. gold recovery tests were performed on selected sulphide and oxide samples. The key test details are summarised below:

- The grind liberation size for sulphide samples were at P_{80}: 75, 53 and 38 microns and the oxide samples at P_{80}: 106, 75 and 38 microns.
- The sulphide samples recovery differential between the grind sizes was minor, with the P_{80} at 38 microns achieving the highest gold recovery. The CIL test results at this grind size are presented in Table 5.14.
- The sulphide sample CIL test conditions were performed at 40% slurry density, 48 hours of leach retention with the initial dosage of cyanide at 0.25% and 20 gpl of activated carbon.
- The oxide samples recovery differential between the grind sizes was minor, with the P_{80} at 38 microns achieving the highest gold recovery. The CIL test results at the P_{80} 75 microns grind size results are selected and presented in Table 5.14 so as to provide a consistent comparison between all three samples (BOC sample not tested at P_{80} 38 microns).
- The oxide sample CIL test conditions were performed at 40% slurry density, 48 hours of leach retention with the initial dosage of cyanide at 0.1% to 0.25% and 20 gpl to 50 gpl of activated carbon.

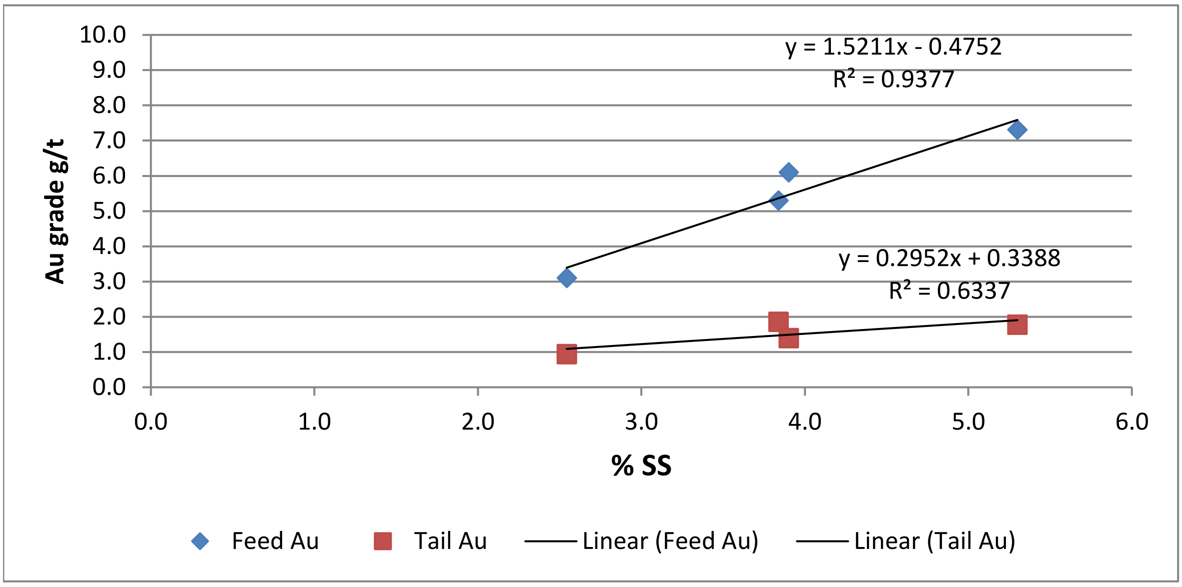
Table 5.13 Direct CIL on sulphide samples

Sample	CIL feed			CIL tail Au g/t	Au % recovery
	AH-Au g/t	CH-Au g/t	SS%		
BSC	6.1	7.0	3.9	1.39	77.2
GM3	5.3	5.1	3.8	1.85	65.1
GM5	7.3	7.9	5.3	1.78	75.7
GM6	3.1	2.6	2.5	0.93	70.0

Table 5.14 Direct CIL on oxide samples

Sample	CIL feed			CIL tail Au g/t	Au % recovery
	AH-Au g/t	CH-Au g/t	SS%		
BOC	2.0	2.0	0.34	0.25	87.5
GMO1	2.1	2.0	0.02	0.20	90.2
GMO2	4.8	4.7	0.04	0.35	92.6

Figure 5.5 Au vs. SS relationship for sulphide samples



As shown in Figure 5.5, there is a strong trend relationship between gold and sulphide sulphur content for the sulphide samples for both the feed and residue streams.

5.11 Recovery model

Due to the high correlation relationship between gold and sulphide sulphur coupled with the positive flotation response for the GM sulphide resource, these factors provided the opportunity to develop a gold recovery model. However, during the exploration and resource definition drill out phase, sulphur analyses were not performed. The key geological analyses performed for the GM resource were Au, Ag, Fe, As, Pb, Zn and Cu.

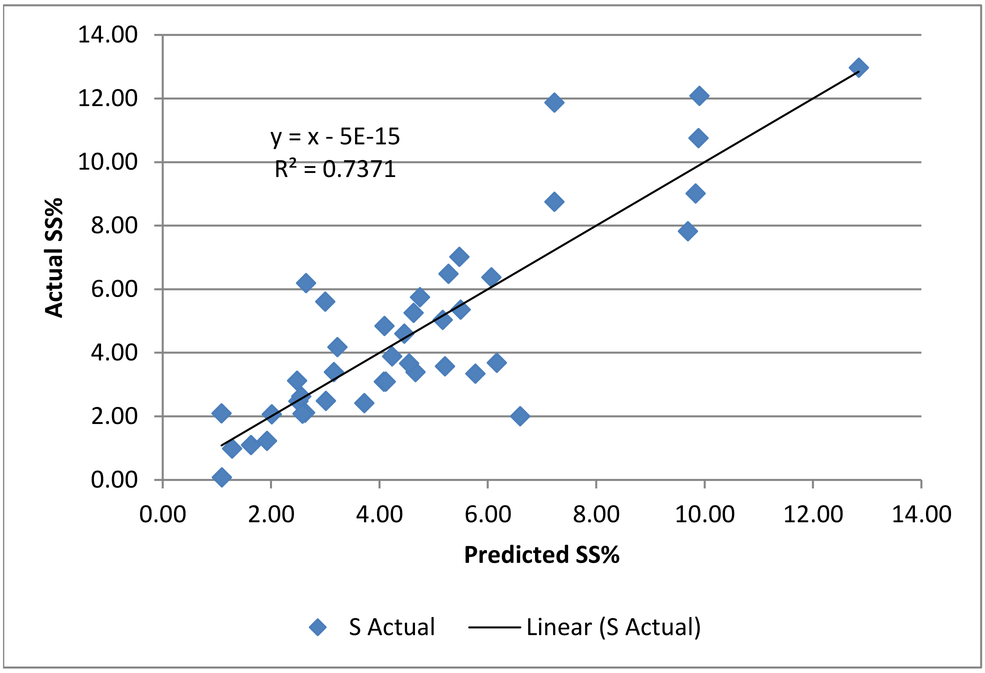
A multivariate regression assessment on an infill drill programme (with diamond core analysed for SS) was successful in identifying significant predictors (Au, As and base metals, Pb, Zn and Cu) that enabled the sulphide sulphur content to be estimated for the overall GM sulphide resource.

The developed regression equation and the graphical plot of the actual sulphur analyses versus the predicted values (Figure 5.6) are presented below:

- % SS = 1.0074+0.6627*Au-1.3674*Base
- Base = As, Pb, Zn, Cu ppm
- Au = g/t.

The supporting details for the multivariate regression assessment are shown in Appendix 5D.

Figure 5.6 Actual vs. predicted SS%



Based on the SS regression equation and the flotation results achieved from the testwork program, the overall gold recovery algorithm that was developed for estimating the gold recovery for the GM sulphide resource is presented below:

- Determine % SS via regression model = SS % = 1.0074 + 0.6627 x Au - 1.367x Base (Base = As, Pb, Zn and Cu combined content)
- Calculate Wt% Conc produced via flotation = (%SS - %SS*0.04) / (22-%SS*0.04)
- Calculate Final Residue (FR) grade = (100 - Wt %)*0.27)/100) + Wt%/100*4.5
- Calculate Gold Recovery = (Au - FR)/Au*100.

Applying the above overall gold recovery algorithm against the testwork results achieved for the sulphide variability samples, the comparison between the actual gold recoveries versus predicted are presented in Table 5.15. The direct cyanidation CIL results for the samples tested are also presented in the table. The variability sample GM 4 CIL recovery has been adjusted as per the AMTEL findings.

The recovery model results are in close agreement with the actual gold recoveries achieved from the testwork (with a slight low bias) except for sample GM 2. The AMTEL report identified that this sample contained the lowest proportion of sub-microscopic gold content, which is the reason for high actual gold recovery.

Table 5.15 Predicted vs. actual gold recovery

Samples	Predicted Au recovery %	Actual Au recovery %	Whole ore leach recovery
BSC	82.4	84.7	77.2
GM2	85.2	94.6	
GM3	82.6	87.4	65.1
GM4	78.9	79.5	
GM5	84.4	87.0	75.7
GM6	76.3	80.5	70.0

5.12 Summary and conclusions from metallurgical testwork

The key outcomes and conclusions that can be derived from the FS metallurgical testwork program that has been performed on the Ganajur Main Gold deposit geological resource samples are presented as follows:

- The sulphide variability samples are representative of the major tonnage area ("Belly" portion) of the Ganajur Main Gold deposit sulphide resource.
- The sulphide samples contained variable gold (2.6 g/t to 10.4 g/t), sulphide sulphur (SS) (2.14% to 5.24%) and arsenic levels (0.3 to 0.8%) against the average sulphide resource grades of 3.8 g/t Au, 2.8% SS and 0.33% As.
- The sulphide samples contain organic carbon (0.3%) which necessitates the addition of activated carbon in the cyanidation leach testwork so as to negate the mild preg-robbing properties.
- The XRD analyses confirmed that the major sulphide present is pyrite (9% to 12%) with minor arsenopyrite (1%), with silica levels at 50% and the carbonate minerals, siderite (20% to 25%), dolomite/ankerite (7% to 12%) representing the major gangue components.
- The oxide variability samples and bulk oxide composite (BOC) are similar in mineralogy to the sulphide samples except that the sulphide content has been oxidised to goethite and hematite.
- The comminution characteristics for the sulphide and oxide samples are similar and are considered to have a moderate to high competency with a moderate resistance to wear (abrasion).
- The flotation response on the sulphide resource samples is positive with 95% gold and 97% sulphide sulphur recovery achieved into a low weight rougher concentrate stream (10% to 20%). These recoveries are attained by targeting a 20% to 22% sulphide sulphur concentrate grade, which results in a flotation tailings stream grading approximately 0.27 g/t Au and 0.13% SS, which reports directly to the TSF.
- The flotation response on the oxide resource samples and oxide blends with the sulphide ore is negative compared to the gold recoveries achieved on the sulphide resource samples.
- The optimum grind liberation size for the flotation of the sulphide resource is at a P_{80} of 75 microns.
- The optimum UFG grind liberation size and CIL conditions to attain high gold recovery from the sulphide concentrates are:
  - a P_{80} of 10 microns, which requires an energy consumption of 90 kWh/t
  - a leach retention time of 48 hours, 0.20% initial cyanide dosage, 1 kg/t of lead nitrate, pH at 10.5 and 50 gpl of activated carbon addition
  - the reagent consumptions are 6 kg/t of NaCN, 1 kg/t of lead nitrate and 2.5 kg/t of hydrated lime.
- A comprehensive gold deportment analyses on the sulphide CIL residues confirmed that the predominant gold losses were due to:
  - Sub-microscopic gold with a gold content ranging between 3 g/t and 4 g/t. This residual gold cannot be recovered by CIL and sets the minimum gold residue grades.
  - The remainder of the gold losses occur as unliberated fine gold particles locked in sulphides and equates to approximately 7% to 12% or 0.3 g/t to 0.8 g/t of the gold content.
  - Based on this assessment, an average of 4.5 g/t Au has been selected as the average sulphide residue gold grade that can be achieved from the CIL of the sulphide concentrates recovered via flotation.
  - The GM4 sample CIL residue at 7 g/t Au was due to a film/coating from AgS or SCN which prevented further cyanidation leaching. The use of high concentrations of lead nitrate will negate this issue from arising during CIL. The estimated CIL residue grade for GM4 without the film/coating effect is 4.7 g/t Au.
- Comparing the gold recoveries achieved from direct CIL vs. the flotation/UFG/CIL route on the sulphide samples is presented in Table 5.16. As shown, the flotation/UFG/CIL route provides an

incremental gold recovery increase ranging from 7.4% to 22.3% compared to direct CIL on the sulphide samples. Due to this significant gold recovery differential, it is recommended that the Ganajur Main Gold deposit project incorporate a flotation/UFG/CIL circuit for the processing of the sulphide resource.

- A gold recovery model for the sulphide resource has been developed via the combination of a multivariate regression analyses (to predict the sulphide sulphur content of the GM resource) and the interpreted results achieved from the flotation and UFG/CIL testwork program. The predicted versus the actual gold recoveries achieved from testwork are in close agreement.
- A gold recovery of 90% for the oxide resource can be achieved via a P_{80} grind liberation size of 75 microns and 24-hour leach retention. The estimated key reagent consumptions are 0.5 kg/t of cyanide and 0.80 kg/t of lime.
- The WAD cyanide content after CIL on the sulphide concentrates can be successfully decreased to below 10 ppm via two stages of the SO_2/O_2 method (Inco) for cyanide destruction. The SMBS consumption is moderate at 5.2 kg/t of concentrate. Due to the high iron content in solution after CIL, the copper sulphate consumption is high at 3.2 kg/t of concentrate.
- Removal of soluble Arsenic prior to discharge into the TSF has been achieved via the addition of ferric sulphate which is the Best Demonstrated Available Technology (BDAT) as recommended by the US EPA. A moderate consumption of ferric sulphate at 6 kg/t is estimated for this stage of the overall Ganajur flowsheet.
- The rheology and settling properties for the Ganajur sulphide concentrate and final tailings stream are benign which should result in minimal slurry pumping issues, inter-tank screen head losses in CIL or thickener area requirements in order to achieve high thickener underflow slurry densities and clear overflow streams.

Table 5.16 Direct CIL vs. flotation/UFG/CIL

Sample	Feed		UFG/CIL		Direct CIL		Recovery
			Tail		Tail		Variance %
	Au	SS	Au	% recovery	Au	% recovery	
GM2	11.0	4.6	0.73	94.6			
GM3	5.2	3.8	0.78	87.4	1.85	65.1	22.3
GM4	3.7	2.4	0.86	79.5			
GM5	7.1	5.3	1.15	87.0	1.78	75.7	11.3
GM6	3.3	2.5	0.73	80.5	0.93	70.0	10.5
BSC	6.2	3.9	1.13	84.7	1.39	77.2	7.4

5.13 Risks and opportunities

The following key risks and opportunities have been identified from the metallurgical testwork program:

- The developed model for estimating sulphide sulphur is valid for the gold, sulphide sulphur, arsenic and base values in the geological data set that was evaluated for the regression assessment. The gold recovery model for the sulphide resource is valid based on the interpretation of the testwork data derived from the sulphide bulk composite and sulphide variability samples.
- The flotation testwork has been performed with fresh water sourced from the potable water distribution system in Perth, Western Australia. The proposed fresh water sourced from the Varada River has a lower TDS (total dissolved salt) content compared to Perth; however, no recycle water to simulate water recovery from the TSF has been used in the flotation testwork. The risk of a potential impact of water quality issues on flotation response is considered minor.
- The CIL gold recovery for the sulphide resource located to the west of the “belly” portion has the potential to have a higher gold recovery. This is due to the interpreted lower sub-microscopic gold content for the ore located in the western area of the Ganajur resource. Further metallurgical testwork on representative samples are necessary to realise this potential.

6 MINING ENGINEERING AND ORE RESERVES

6.1 Executive summary

Snowden completed the mining section of the FS for the planned Ganajur Gold Project, Haveri district, Karnataka.

Following a study to identify favourable geotechnical conditions for the Ganajur rock mass and a process of identification of an ore inventory, pit design, production scheduling and economic evaluation, Snowden produced a mine plan for the Ganajur deposit resulting in the estimation of Proved and Probable Ore Reserves. The Ore Reserves were defined using the 2012 edition of the "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves" (JORC Code 2012).

6.2 Geotechnical assessment for open pit mining

6.2.1 Introduction

SGES was contracted by DESPL in June 2016 to provide geotechnical investigation and design services encompassing the mine pit area, tailings dam and processing plant infrastructure for the FS for the planned Ganajur Gold Mine Project, Haveri district, Karnataka.

The geotechnical engineering report was prepared by SGES in support of the FS, with technical oversight and review provided by Snowden (SGES Report and Snowden Design Memorandum are provided in Appendix 6A).

The Ganajur area comprises a gently undulating, intensely cultivated plain with isolated northwest-southeast trending low ridges of outcropping BIF bedrock. The average elevation difference is of the order of 50 m to 60 m and the landmass of the area is situated between the elevations of 515 mamsl to 570 mamsl.

The Ganajur Main Gold deposit is hosted dominantly by greywacke and inter-bedded auriferous, banded, ferruginous and sulphidic chert (the BIF), which are the part of the Archaean-age Dharwar-Shimoga greenstone belt. The general strike direction of the Ganajur BIF zone varies in azimuth between 300° and 320° and dips at 35° to 50° towards the northeast.

The BIFs pinch and swell due to "S-type" folding contains a number of healed shear zones with mylonitic infill. The main mining targets, containing the bulk of the resource, are associated with local swellings in the BIFs; where locally intense folding and brecciation are observed.

The greywacke country rocks comprise predominantly fine-grained sediments, typically siltstones. Bedding is highly persistent with schistose foliation parallel to the bedding resulting in a "flagstone" like material, occasionally quarried for paving and building materials. Pit walls will be largely formed in these country rocks.

Both BIF and greywacke units contain sub-horizontal and sub-vertical joint sets, and both contain intense micro-veining (carbonate, quartz) which may influence intact rock strengths.

Weathering has extended to at least 25 m depth in the greywackes, but appears to have altered the highly siliceous BIF rocks substantially less than the slightly argillic greywackes.

6.2.2 Investigation program

The FS investigation program comprised the following activities in support of open pit development:

- Geotechnical and geological logging of four boreholes of 84 m to 90 m depth
- Field permeability tests at specified depths for each of the drilled boreholes
- Laboratory tests on soil and rock samples.

In addition to the FS geotechnical investigations, geotechnical and structural logging was undertaken on core from exploration drillholes.

Field investigation by diamond core drilling was performed by M/s South West Pinnacle during 22 July to 8 August 2016. The investigation consisted of sinking four exploratory boreholes (viz., GMC-GT01, GMC-GT02, GMC-GT03 and GMC-GT04) using a diamond core drill rig with a 61.1 mm core diameter. he boreholes were drilled to depths ranging from 84.0 m to 90.0 m from the ground surface level, inclined at 70° to horizontal, using oriented core drilling techniques. SGES undertook geotechnical and geological core logging, sample selection, packer testing for hydrogeological information, and soil and rock strength testing.

The geotechnical model for the deposit comprises the following units:

- Overburden residual soil depths vary from 3 m to 6 m
- Highly to moderately weathered greywacke rock from 6 m to 30 m depth
- Fresh, high strength greywacke rock present from 30 m to >90 m depth
- BIF zone, host rock for gold mineralisation.

The groundwater level was located at approximately 20 m depth below surface at the time of investigation.

Rock density and strength test data are summarised in Table 6.1.

Table 6.1 Summary of Ganajur unit weight and uniaxial compressive strength test data

Rock type	Unit weight (t/m^3)			UCS (MPa)		
	Minimum	Maximum	Mean	Minimum	Maximum	Mean
Highly weathered greywacke	2.25	2.37	2.31	0.67*	30.8*	8.9*
Moderately weathered greywacke	2.14	2.94	2.59	12.46	136	42.4
Fresh greywacke	2.74	3.08	2.81	13.82	209.6	72.0

**Derived from point load tests*

Rock mass rating (RMR) values derived from the borehole geotechnical logs ranged from 20 to 74 and rock mass conditions are classified as “Poor” to “Good”. The results are summarised in Table 6.2.

Table 6.2 Rock mass classification sorted by lithology

Rock type	RMR (Bieniawski 1973, 1993)			Rock mass class	Description
	Minimum	Maximum	Mean		
Highly weathered greywacke	20	29	23.25	IV	Poor rock
Moderately weathered greywacke	34	47	39.25	IV	Poor rock
Slightly weathered greywacke	52	70	63.25	II	Good rock
Fresh greywacke	62	79	69.75	II	Good rock

6.2.3 Design program

The geotechnical design program undertaken by SGES comprised the following tasks:

- Stereographic projection and kinematic stability analysis of slope sections for batter design
- Analyses of critical pit slope sections for 100 m deep (vertical) open pit slopes by limit equilibrium methods using the Geoslope and Slide analysis software
- Development of pit slope design recommendations
- Recommendations for slope monitoring systems.

The material design parameters used by SGES for the stability analyses were based on the RMR values and are summarised in Table 6.3.

Table 6.3 Material design parameters

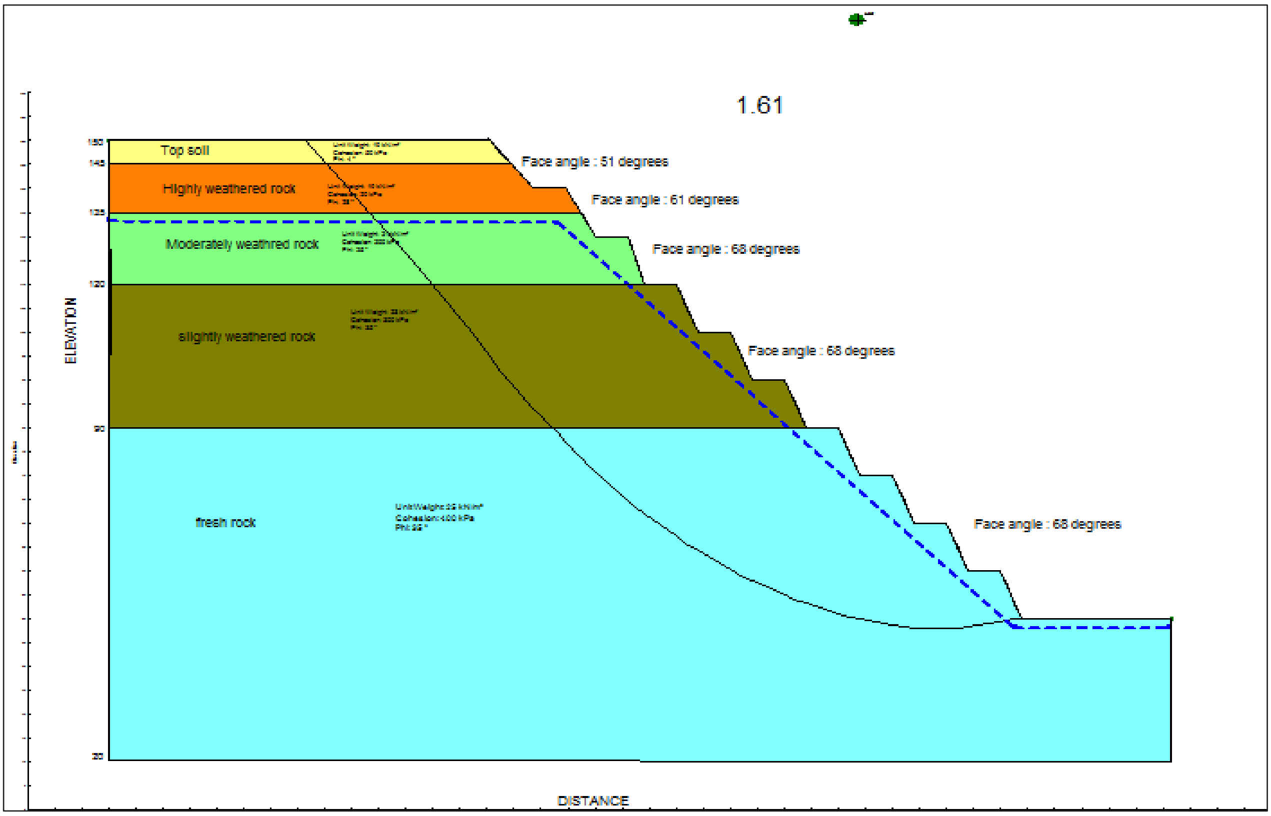
Depth (m)	Description	Total unit weight (kN/m^3)	Cohesion (kPa)	Friction angle (°)
0 to 5	Sandy clay	19	50	4
5 to15	Highly weathered rock	19	20	28
15 to 30	Moderately to slightly weathered rock	21	200	25
30 to 60	Slightly weathered to hard rock	23	300	34
60 to100	Hard rock	25	400	34

The results of rotational limit equilibrium analyses confirmed that the overall pit wall slopes will have a high factor of safety against rock mass shearing modes of failure for both static and seismic conditions with the results are summarised in Table 6.4 with the analysis for Section 1 illustrated in Figure 6.1.

Table 6.4 Factors of Safety for static and seismic conditions

Design section	Bench face angle (°)			Maximum bench height (m)	Berm width (m)	Factor of Safety *	
	0 to 10 m	10 m to 20 m	20 m to 100 m			Static	Seismic
1	51	61	68	10	6	1.61	1.23
2	51	61	68	10	6	1.66	1.13
3	51	61	68	10	6	1.63	1.20

Figure 6.1 Critical slip surface – footwall section 1 (static loading)



Rock mass structural conditions were identified as the main control on pit wall stability, and provide the principal mechanisms for slope failure. Snowden reviewed the SGES findings and slope design recommendations and identified modifications to the SGES recommendations based on adopting an acceptable, moderate level of risk of batter scale failure.

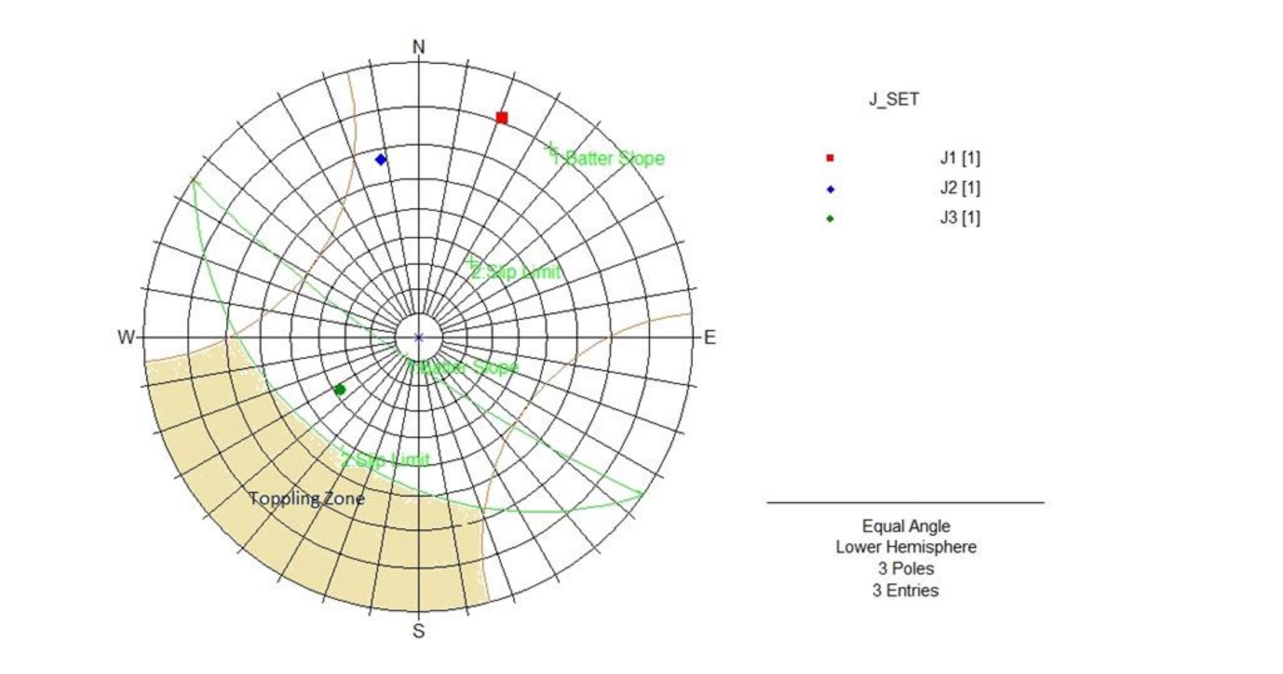
Key points identified by the SGES and Snowden geotechnical assessments were:

- Bedding is the dominant structural set present with average orientation of 38°/056°.

- There is substantial variability in the orientation of the bedding structures (both dip and dip direction), suggesting localised folding is present; also, bedding dips are lower in the southeast sector of the deposit than in the northwest sector (see Figure 6.2).
- Typically, there is at least one and generally two sets of joints also present, with average orientations of 81°/201° and 67°/168°; both sets have substantial variability in orientation.
- Based on site observations of surface planarity of greywacke bedding planes the estimated friction angle of bedding and joint structures will be greater than 43 for a range of normal stresses up to the equivalent of 10 m cover depth.
- The most likely slope failure mechanisms were confirmed as:
  - slabs sliding on bedding planes dipping out of footwall batters
  - toppling slabs on bedding planes on hangingwall batters.
- The probability of sliding wedges forming between the main structure sets on any wall is low as they have approximately parallel strike orientations;
- The proportion of bedding planes with dip angles between the proposed 55° or 60° footwall batter angles and minimum friction angle of 43° and is acceptable in terms of the probability of occurrence and potential volume of sliding failures.

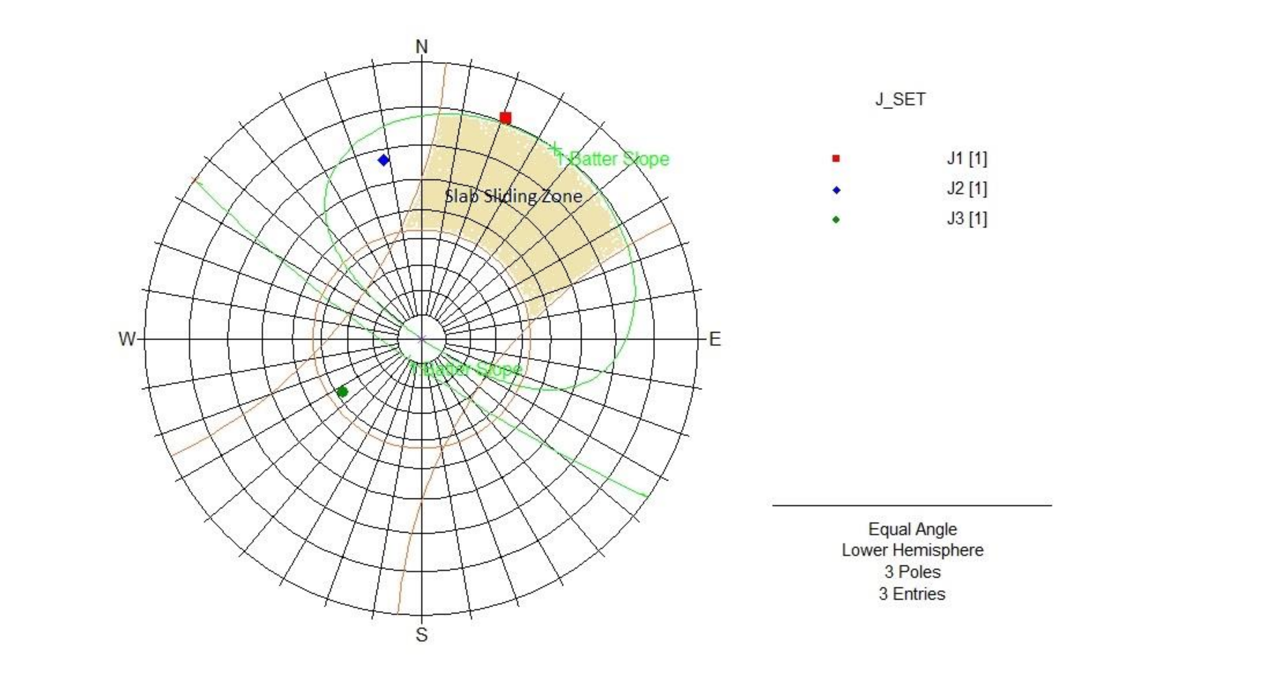
SGES concluded from their studies that the hangingwall batter angles should be 75°. Snowden undertook kinematic stability analyses using the Dips software for toppling, planar sliding and wedge sliding to assess the risk of increasing the batter slope angle to 80° within the hangingwall domain, using the minimum friction angle of 43° reported above. The results are illustrated in Figure 6.2 to Figure 6.4.

Figure 6.2 Toppling analysis for hangingwall



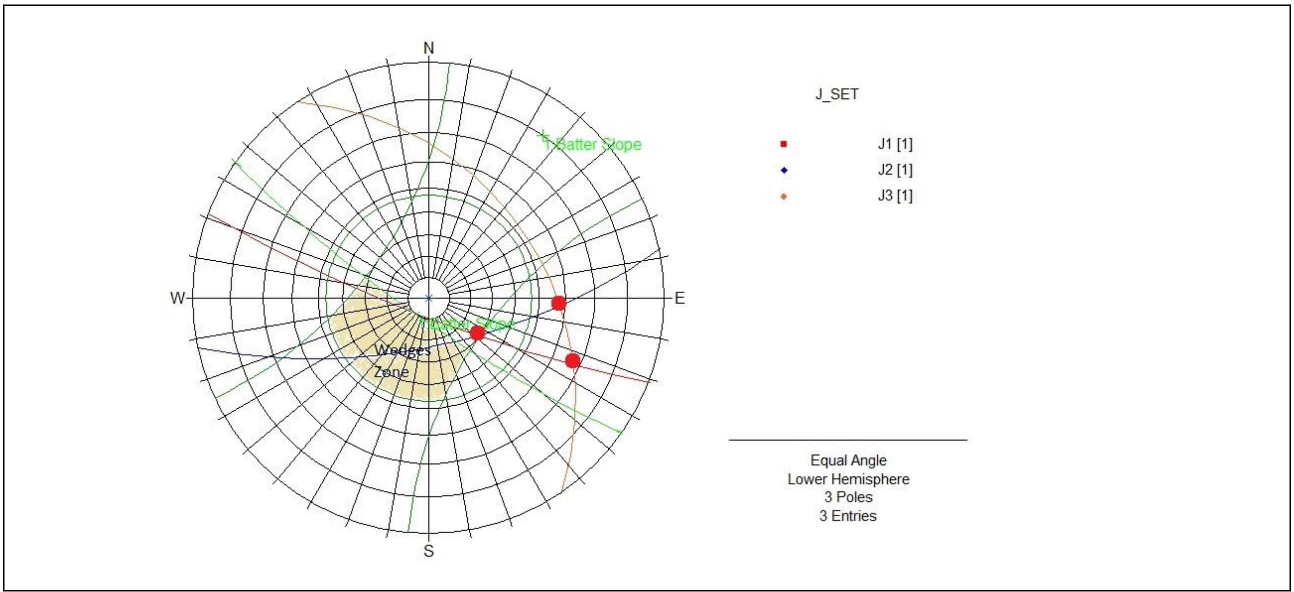
The analysis illustrates that bedding dip is too shallow to initiate toppling failure for hangingwall batter faces.

Figure 6.3 Slab sliding analysis



The analysis illustrates that the average dip of the J1 set lies outside the risk zone. It is likely that a significant proportion of the joints within the set will have slightly lower dip and come within the risk zone. However, these structures have low persistence and are unlikely to form more than minor, localised, thin sliding wedges along crest lines, easily removed by routine face scaling during excavation.

Figure 6.4 Sliding wedge analysis (wedge intersections shown by red dots)



6.2.4 Design recommendations

Based on the geotechnical studies, pit slope design recommendations were developed with an acceptable level of risk of small-scale failures developing on batter faces. The recommended slope design angles are summarised in Table 6.5 and design sectors provided in Figure 6.5.

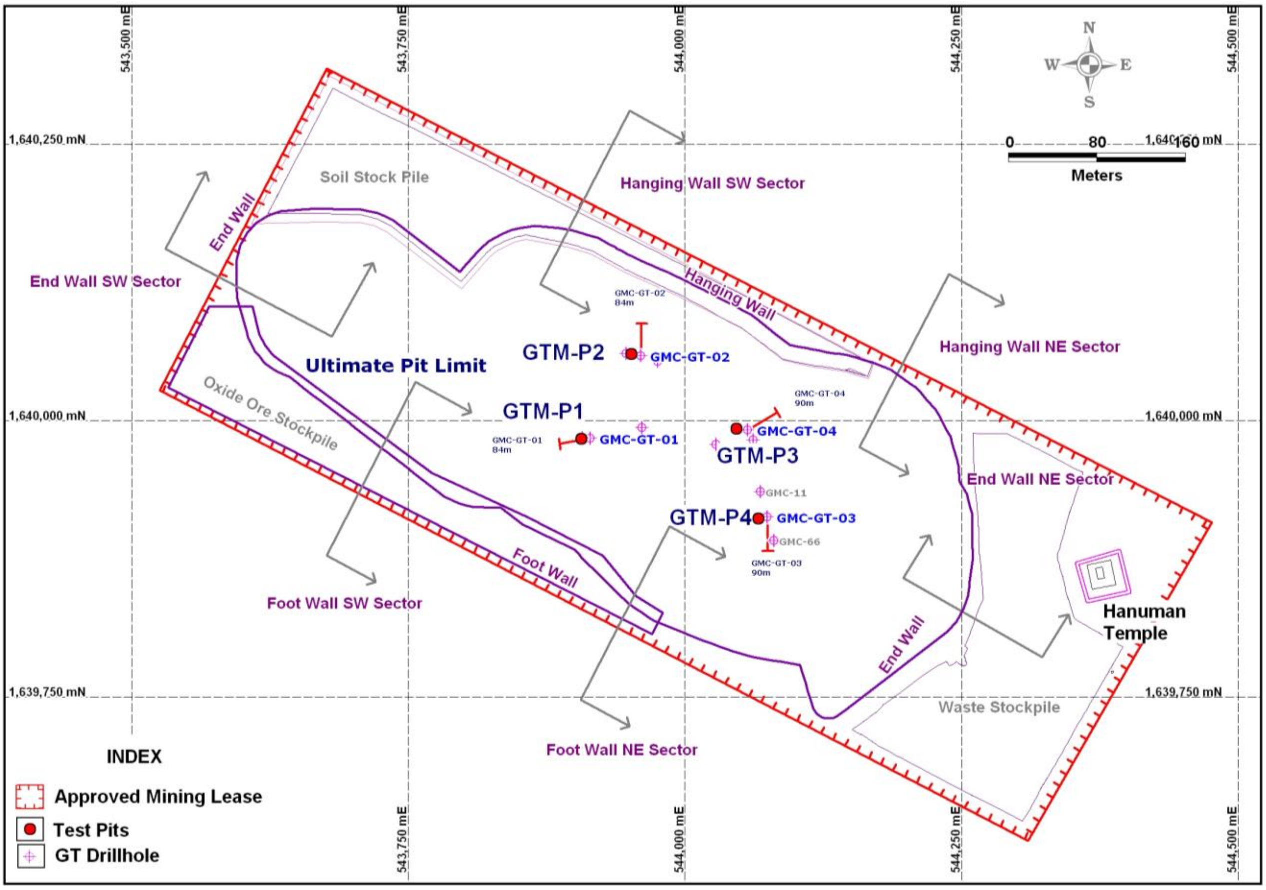
Table 6.5 Pit slope design recommendations

Sector	Wall	Batter angles (°)		Maximum inter-ramp angle (°)	
		Weathered	Fresh	0 to 50 m	50 m to 100 m
Northwest	Footwall	45	60	46	51
	Hangingwall	45	80	48	56
	End wall	45	75	46	56
Southeast	Footwall	45	55	46	51
	Hangingwall	45	80	48	56
	End wall	45	75	46	56

Notes to design recommendations:

- "Weathered" applies to top two 10 m benches
- "Fresh" applies to all benches more than 20 m below surface
- All batters to be maximum 10 m height
- All berms to be minimum 5.0 m width.

Figure 6.5 Pit design sectors



The overall slope angle (crest to toe) of the footwall slope should not exceed 40° for a maximum bench stack height of 50 m. If the ramp is not present on the slope, a double-width berm (10 m) should be incorporated at 50 m depth.

Operational controls to mitigate batter-scale hazards should include the following:

- Weekly inspections of pit walls to identify and map geotechnical hazards
- Final batter faces should be mined with controlled blasting techniques

- Loose rock should be scaled off batter faces with excavator buckets after blasting
- Loose rock on berms and against batter toes should be cleaned up before the subsequent bench is taken.

6.3 Mining concept

As the Ganajur deposit is near to the surface it will be mined using open pit methods. The majority of mining activities are planned to be undertaken by a suitable mining contractor.

6.3.1 Clearing and soil stripping

Mining areas will be cleared of vegetation using track dozers. Vegetation will be stockpiled for spreading on rehabilitated slopes.

Soil will be stripped from cleared areas (with exception of soil stockpile areas), to a depth of about 140 mm, using graders and track dozers forming windrows which will be reclaimed using front-end loaders (FELs) and hauled in trucks. It will be re-spread over disturbed areas and any waste dumps after mining activities are completed.

6.3.2 Drilling

Grade control

The ore rock mass requires drilling for grade control purposes. In addition, an amount corresponding to 10% of the ore will be considered for grade control. All blast hole drilling will be sampled for grade control. There will be no additional drilling for grade control.

Production

Drill and blast design for standard ore and waste drill and blast design patterns were recommended on 10 March 2017 by Deepak Vidyarthi, Advisor (Mining) for DESPL. Table 6.6 summarises the production drill patterns by Vidyarthi. Production drilling will be undertaken by rotary blast hole drills. Around 30 m below surface most blast holes will have some groundwater present.

Table 6.6 Production drill pattern parameters

Material type	Bench height (m)	Hole diameter (mm)	Burden (m)	Spacing (m)	Depth (m)	Sub-drill (m)	Angle (°)	Hole length (m)	Volume per hole (bcm/hole)
Ore	10	150	4	5	10	1	90	11	200
Waste	10	150	5	6	10	1	90	11	300

Wall control

In order to minimise damage to the pit walls in the competent rock 30 m below the surface, a modified production blast or pre-split may be utilised. Initially the unmodified blasts will be assessed to see if there is an acceptable amount of damage to these walls. Further control to minimise damage to the pit walls will be considered after mining commences in the competent rock.

6.3.3 Excavation guidance

In general excavation guidance will involve the following steps:

- Grade control data collection:
  - Collection of grade control samples from blast hole drilling
  - Sample assaying (the samples are analysed in the on-site laboratory designated for this purpose).

- Ore/Waste determination:
  - Grade control block modelling
  - Dig plan creation
  - Dig plan information provided to drill and blast engineer before blasting to ensure blast design optimises ore breakage and minimises movement.
- Ore control:
  - Mark out by surveyors
  - Visual identification by the grade control geologists and the excavator operators
  - Assessment of blast movement and adjustment of mark out by geologists
  - Ore spotting, if required (the ore is structurally controlled and there will be visual control of the ore).
- Reconciliation:
  - Comparison of mined and processed with grade control and resource block models.

6.3.4 Blasting

The preliminary geotechnical investigation determined that the rock mass is altered to varying degrees. The alteration indicates that not all material will require blasting. The entire rock mass within the resource model was coded for geotechnical logging. The rock type of completely weathered and highly weathered are considered to be free dig. Below the highly-weathered rock is moderately-weathered rock and this will be ripped with a bulldozer before excavation. It is assumed for the FS that 73% of the ore and waste will require blasting.

The explosive used is in the form of bulk emulsion as groundwater is present. The forecast average powder factors provided by Vidyarthi are:

- 0.22 kg of explosives per tonne of ore rock
- 0.19 kg of explosives per tonne of waste rock.

It is proposed to use non-electric initiation for blasts. A drill and blast design report was prepared for DESPL by Deepak Vidyarthi, Advisor (Mining) and this is supplied as Appendix 6B.

6.3.5 Loading

Mining is anticipated to be undertaken using small excavators (operating weight around 40 t with a bucket capacity of about 2 m^3) as the orebody is generally around 30 m wide (varying between 5 m and 45 m) and annual ex-pit movement is expected to peak at around 700 kt/a. This size excavator will allow the 10 m drilled and blasted benches to be mined in either two or three passes to help minimise dilution and ore loss.

6.3.6 Hauling

Trafficability was not assessed. Snowden recommends that this be included in future studies so that sheeting requirements can be determined.

All ex-pit material will be hauled using multi-axle tipper trucks or articulated haul trucks. These trucks are anticipated to have a nominal payload of about 25 t.

6.3.7 Tipping

Waste rock

There is insufficient space within the mining lease to store the waste generated by the mine. The mine waste (greywacke) is considered to be of suitable quality for sale to construction companies identified by DESPL in responses to an expression of interest for the purchase of the waste rock. Waste from the mine will be placed on a waste buffer stockpile, and made available for transport. Management of reclaimed (sold) quantities will need to be managed carefully to ensure that mine production can continue (i.e. there is always space to dump waste).

The dumps will be progressed by tipping from a higher level against a windrow and progressively pushing the waste out with a dozer.

Ore

The processing of ore in the Ganajur plant will be either oxide or fresh as these materials are not able to be blended. As a result, ore will be tipped on the run of mine (ROM) or in long term oxide stockpiles for later rehandle. When blending constraints permit, ore may also be direct tipped into the primary crusher.

6.3.8 Rehandle

Long term stockpiles

Rehandle from long term stockpiles was assumed to be tipped on the ROM for further rehandle into the primary crusher. This will be undertaken using the same load and haul fleet as used for ex-pit movement.

Run of mine stockpiles

Rehandle from ROM stockpile to the primary crusher will be undertaken using a front-end loader (FEL).

Ore rehandle is required to ensure that sulphidic sulphur grades are maintained below 2.8% as this maximises concentrate production.

During normal operations, the ore feed is achieved by a combination of ore direct tipped from the pit into the ROM bin by the haul trucks with the ROM loader adding the makeup from ROM stockpiles. Approximately 40% of all ore sent directly to the ROM pad is stockpiled and rehandled due to ore blending and scheduling requirements.

6.3.9 Rehabilitation

Although it is likely that all waste rock, and stockpiled oxide ore on the mining lease will be removed by the end of mining, any disturbed areas will be progressively rehabilitated with topsoil, where possible. Surfaces of dumps shall be contoured to minimise batter scour and ripped at 1.5 m centres to a depth of 400 mm, where practicable. Rock-lined drains are constructed, where required, to ensure excess runoff is controlled and directed down to sediment traps.

6.3.10 Other mining activities

Ancillary equipment

The load and haul fleet is supported by ancillary equipment; including dozers, graders, water trucks and service trucks. The ancillary equipment will:

- Construct and maintain roads and ramps
- Maintain pit, stockpile and waste dump floors
- Prepare drill pattern areas

- Clean-up spillage around working areas and roads.

The Ganajur Gold Project ROM pad works dayshifts and nightshifts. Portable lighting towers will be used where there is a lack of permanent lighting. Permanent lighting is planned to be installed at the crusher and road intersections adjacent to the crusher pad where there is access main power supplies.

Mine services

The main function of mine services is to provide assistance to the production units. The work performed includes, road maintenance, pit dewatering, rainfall runoff management and waste dump preparation and maintenance.

Haul road design and traffic management

For the mobile mining fleet, a set of designs for haul road construction were designed by Snowden and CPC. Haulage is the largest mining activity cost and the design, construction and maintenance of haul roads is considered in the haulage costs.

Light vehicle traffic is restricted on the mining lease to only production personnel and selected technical personnel being able to drive in pit and mix with the heavy traffic. This is not the case for the dual use road connecting the mining lease and the plant lease, where public vehicles will interact with the mobile mining fleet. Traffic control via windrows and signage, with public education campaigns will be practised to minimise the risk to the public on the dual use roads.

Mine truck travel speeds are provided in Table 6.7.

Table 6.7 Pit truck speeds

Segment type	Maximum segment speed (km/hr)		
	Loaded	Empty	Rolling resistance
In-pit flat	30	30	4%
In-pit ramp	15	35	3%
Ex-pit downhill	15	35	3%
Ex-pit uphill	15	35	3%
Ex-pit flat	45	45	3%
Dump ramp	15	35	4%
Dump flat	40	40	5%
Corner	**Degrees**	**Loaded (km/hr)**	**Empty (km/hr)**
Sharp	35	15	20
Shallow	20	25	30

Major two-way, all-vehicle roads where heavy traffic occurs have PVC pipe demarcation on both sides of the road, with reflectors to increase safely at night. National roads rules are followed on site and all intersections use standard road signage.

6.4 Pit optimisation

6.4.1 Mining model

The mining model was based on the Datamine resource model named “gan1608v1.dm”.

Sulphidic sulphur

Sulphur was only estimated for some resource drillholes corresponding with higher grades of gold, and an estimate of sulphur was of low confidence in the resource model. In the absence of this resource model sulphur field, the mining model estimate of sulphidic sulphur was estimated as a field by remodelling based on regressions to other elements (model grades not used) as provided by DESPL.

The algorithms were:

- BASE=As+Cu+Zn+Pb
- SS=MAX(0,(1.145+0.602*Au-1.264*BASE));

Where:

- As is arsenic
- Cu is copper
- Zn is zinc
- Pb is lead
- Au is gold
- SS is sulphidic sulphur.

Dilution and ore loss

The approach to the application of dilution to the deposit followed a "skin" dilution approach to mimic the additional waste required to effectively mine the orebody. Given the relatively high grade of the deposit it becomes important to maximise recovery, even to the need to take additional waste. The approach applied the following steps:

1) A 25 cm "skin" of dilution was added around the ore blocks, in all directions. This dilution carried the grade of the waste (at a minimum grade of 0.12 g/t), which was an average grade for each rock-type.
2) Sub-cells were then aggregated into 5 x 5 x 5 parent blocks, summarised by Total block tonnes, total "ore" tonnes, and total "ore" contained metal.
3) Each parent block then has a calculated PORE (percentage of ore within the block), and Au grade (contained metal divided by ore tonnes).

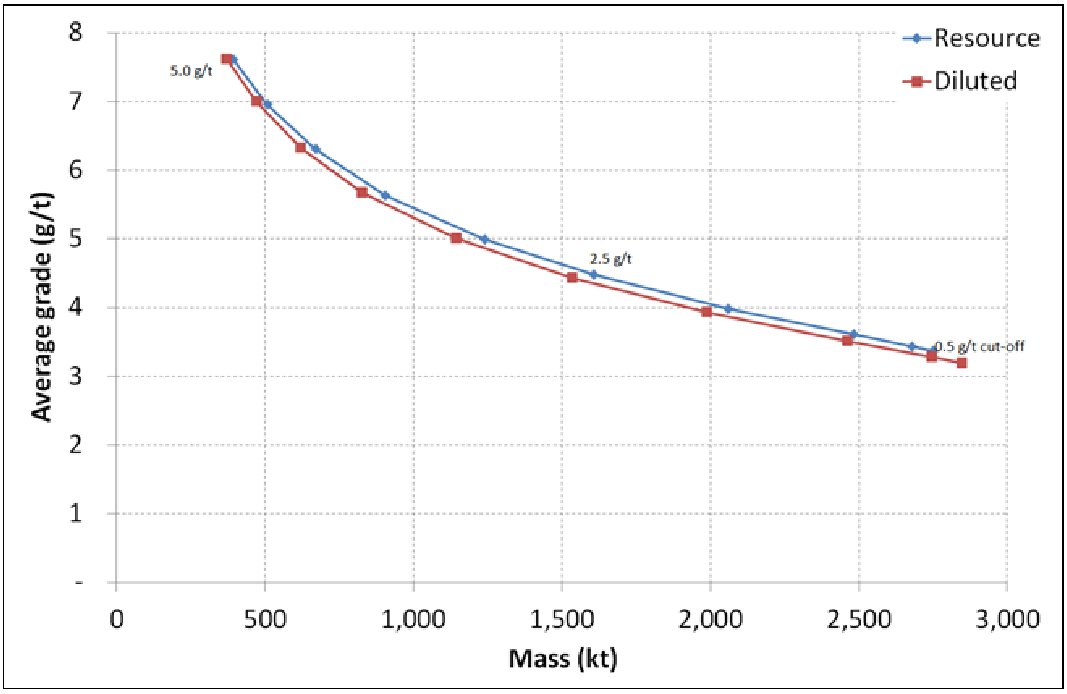
A report of the raw resource model and the diluted model at a 1.0 g/t cut-off grade is shown in Table 6.8. The effective dilution applied is 5%, and 98% (or near to) of the contained metal is recovered. Because the recovery of gold is dependent on the sulphidic sulphur concentration, this data needed to be available in the model for optimisation.

Table 6.8 Dilution analysis (at 1 g/t cut-off)

Model	Resource category and rock type	Mass (kt)	Contained gold (koz)	Au (ppm)	As (%)	Pb (%)	SS (%)	Zn (%)
Resource model	**Measured**	**2,240**	**266**	**3.70**	**0.37**	**0.0032**	**2.89**	**0.006**
	Oxide	568	52	2.84	0.16	0.0036	2.63	0.0064
	Fresh	1,671	214	3.99	0.43	0.0030	2.98	0.0059
	Indicated	**438**	**29**	**2.09**	**0.20**	**0.0029**	**2.14**	**0.0056**
	Oxide	117	7	1.96	0.12	0.0030	2.15	0.0053
	Fresh	321	22	2.14	0.23	0.0028	2.13	0.0057
	TOTAL	**2,678**	**296**	**3.44**	**0.34**	**0.0031**	**2.77**	**0.0059**
Diluted model	**Measured**	**2,286**	**261**	**3.55**	**0.35**	**0.0031**	**2.82**	**0.0060**
	Oxide	568	50	2.76	0.16	0.0035	2.59	0.0064
	Fresh	1,718	211	3.82	0.41	0.003	2.90	0.0059
	Indicated	**460**	**28**	**1.92**	**0.19**	**0.0027**	**2.05**	**0.0057**
	Oxide	123	7	1.78	0.12	0.0028	2.05	0.0055
	Fresh	337	21	1.98	0.21	0.0027	2.05	0.0058
	TOTAL	**2,747**	**290**	**3.28**	**0.32**	**0.0031**	**2.69**	**0.0060**
Difference		**3%**	**-2%**	**-5%**	**-6%**	**0%**	**-3%**	**2%**

The effect of dilution is on a local basis, i.e. dilution is higher in narrower areas, and lower in wider areas. In this orebody, the higher-grade areas tend to be wider than lower-grade areas, and hence dilution stays constant or reduces as the grade increases. This is shown in Figure 6.6.

Figure 6.6 Grade-tonnage curve comparison of resource and diluted models



Then mining model was then treated by flagging blocks that could not be mined with a lease offset of 15 m applied for pit optimisation to allow for design and provide 7.5 m buffer.

6.4.2 Parameters and modifying factors

The pit optimisations were based upon the assumption of separate streams for processing of the oxide and fresh ore.

Resource classification

Only Measured and Indicated resources were used in the base case pit optimisation. A scenario was run testing the inclusion of Inferred resources.

Starting surface

The initial surface for the optimisation was the original topography (TOPO1-31-8_snotr.dm) which was coded into the resource model.

Boundaries

Exclusion areas were coded into the mining mode for the following areas:

- Temple
- Lease.

Geotechnical constraints

Table 6.9 summarises the slope angles used in the pit optimisation. These are based on the geotechnical design recommendations in Table 6.5 and include a 5° reduction to account for ramps and slope estimation errors in Whittle.

Table 6.9 Pit optimisation slope angles by weathering

Sector	Wall	Batter angles (°)		Maximum inter-ramp angle (°)	
		Weathered	Fresh	0 to 50 m	50 to 100 m
Northwest	Footwall	45	60	46	51
	Hangingwall	45	80	48	56
	End wall	45	75	46	56
Southeast	Footwall	45	55	46	51
	Hangingwall	45	80	48	56
	End wall	45	75	46	56

Processing rate

A constant processing rate of 300 kdt/a was used in the optimisation.

Metallurgical recoveries

Metallurgical recoveries were supplied by DESPL and applied by oxidation state and sulphidic sulphur grade. For the fresh ore, a yield of flotation concentrate is considered using the formula:

- Yield (%) = SS*0.96/(22-SS*0.04) x 100.

The gold recovery from oxide ore is fixed at 90% and sulphide ore recovery is:

- Gold recovery (%) = [1-(YLD*4.5+(1-YLD)*0.27)/Au] x 100;

where, FR is the fresh ore tonnage, YLD is the yield of flotation concentrate and Au is the gold grade.

This fresh ore recovery algorithm returned an average gold recovery for fresh ore of approximately 79%.

Operating costs

A summary of operating costs is provided in Table 6.10.

Table 6.10 Operating cost assumptions

Item	Value
Mining	
Waste mining ($/t rock)	1.50
Depth increment ($/t/m below 500 mRL)	0.005
Processing	
Incremental ore mining ($/t rock)	0.20
Oxide process cost ($/t ore)	18.20
Fresh process cost ($/t ore)	3.25*SS(%)+14.43
Administration	4.00
Selling	
Refining and transport ($/oz)	2.87
Royalty (% in situ, $/oz)	5%, $67.5/oz

Sales and discount rate

A gold price of $1,250/oz was applied. A discount rate of 5% was applied.

Cut-off grade

Cut-off grades were calculated to be approximately 0.8g/t Au for oxide and approximately 0.9 g/t Au for fresh ore.

6.4.3 Results

Pit optimisation was completed using GEOVIA's Whittle Four-X™ software. Optimisations were completed for a range of revenue factors. Cash flow results are based on operating costs with no capital, bank interest or taxes included. A summary of the selected pit shell 17 is shown in Table 6.11.

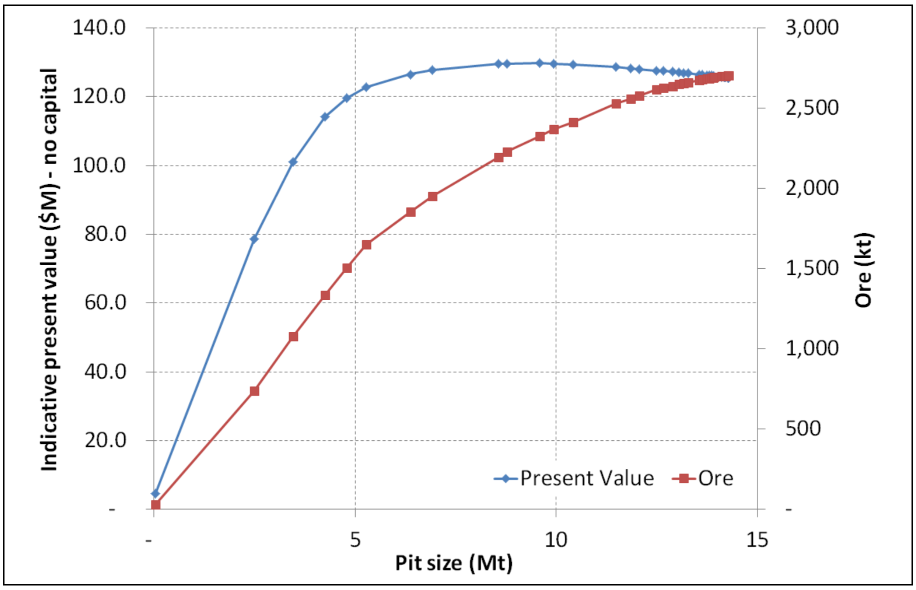
Table 6.11 Selected pit shell for Ganajur pit 17

Item	Value
Revenue factor	1
Physicals	
Pit size (kt)	12,490
Waste (kt)	9,875
Strip ratio (w:o)	3.78
Ore – total (kt)	2,615
Au grade – total (g/t)	3.35
Contained gold – total (koz)	282
Recovered metal – total (koz)	228
Ore – oxide (kt)	728
Au grade – oxide (g/t)	2.50
Ore – sulphide (kt)	1,886
Au grade – sulphide (g/t)	3.68
Economics	
Revenue ($ M)	285.0
Selling cost ($ M)	15.4
Oxide processing cost ($ M)	16.8
Sulphide processing cost ($ M)	53.5
Mining cost ($ M)	20.9
Unit cost ($/oz)	467
Operating cash flow ($ M)	178.4

Based on the Measured and Indicated Resources, there is only minor upside associated with mining cost, processing cost and price (approximately 5 koz in total for a gold price rise of 30%), mainly on the north of the pit.

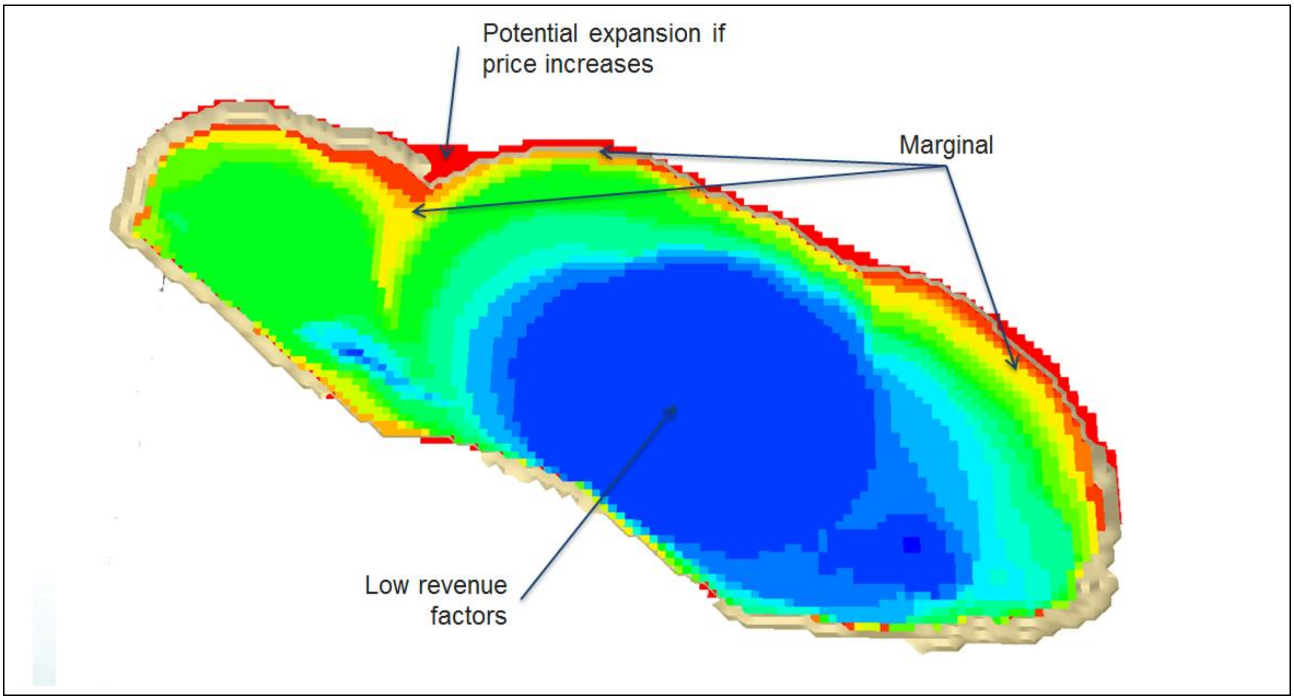
A graph of the pit size and present value is provided in Figure 6.7.

Figure 6.7 Graph of present value without capital and pit size



The pit shell development is shown in Figure 6.8 with shell 17 coloured in brown.

Figure 6.8 Pit shell development



6.5 Pit design

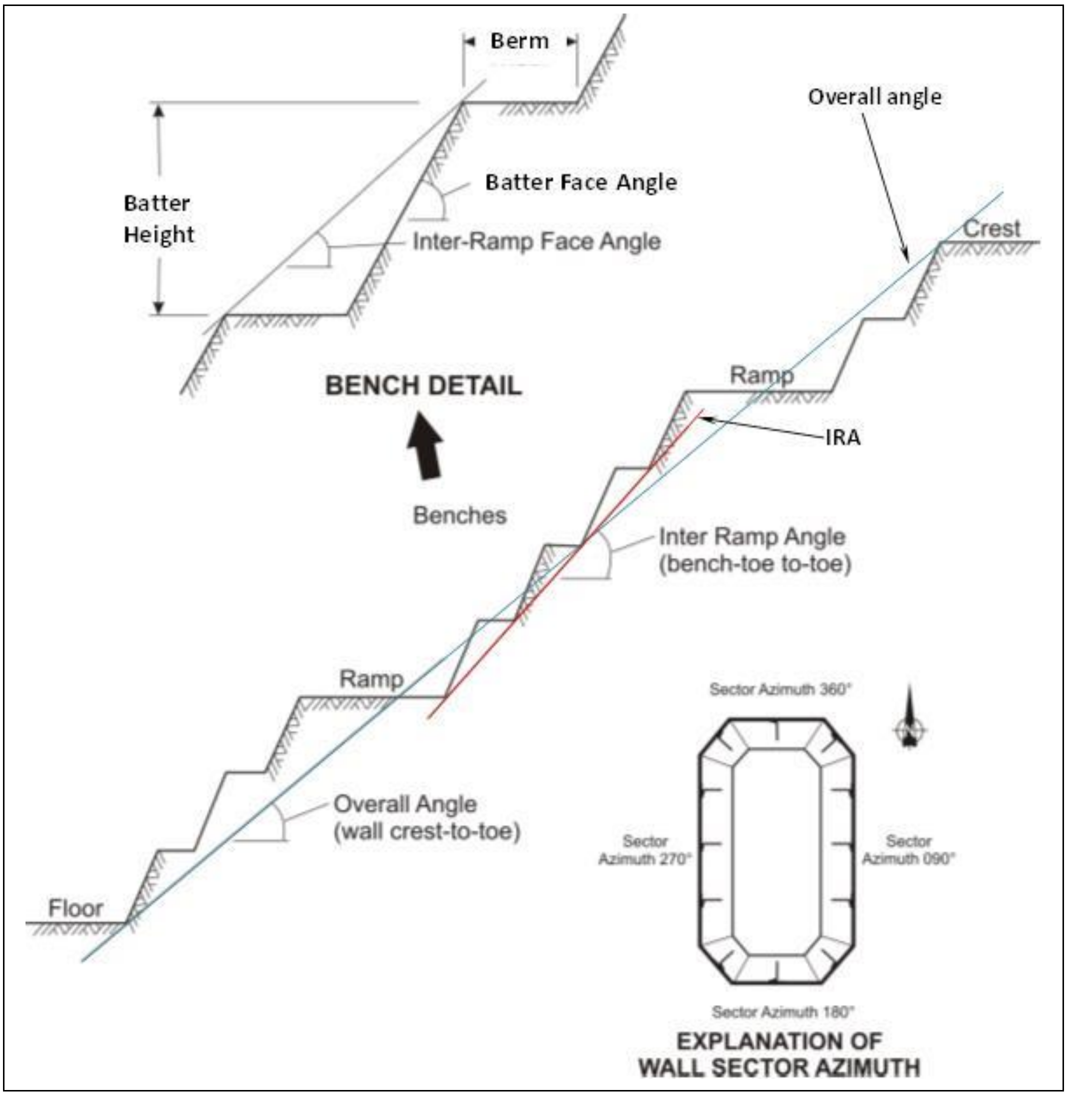
6.5.1 Parameters

The terminology used to describe the geometric arrangement of the pit wall is illustrated in Figure 6.9. The terms relevant to open pit slope design as used in this study are as follows:

- Batter slope – the sections of rock mass between catch berms within pit walls, usually excavated to a specific inclination/angle from the horizontal.
- Berm – the flat interval separating successive batter slopes. The purpose of the berm is to catch any loose material or local scale rock mass failures and to provide access for inspection and remediation, thus reducing the risk to the workforce at the base of the pit.

- Inter-ramp angle (IRA) – the angle from the toe of one batter to the toe of the next batter, exclusive of any ramp system.
- Overall slope angle (OSA) – the angle between the lowest toe and the highest crest, inclusive of any haul roads.

Figure 6.9 Geotechnical labels



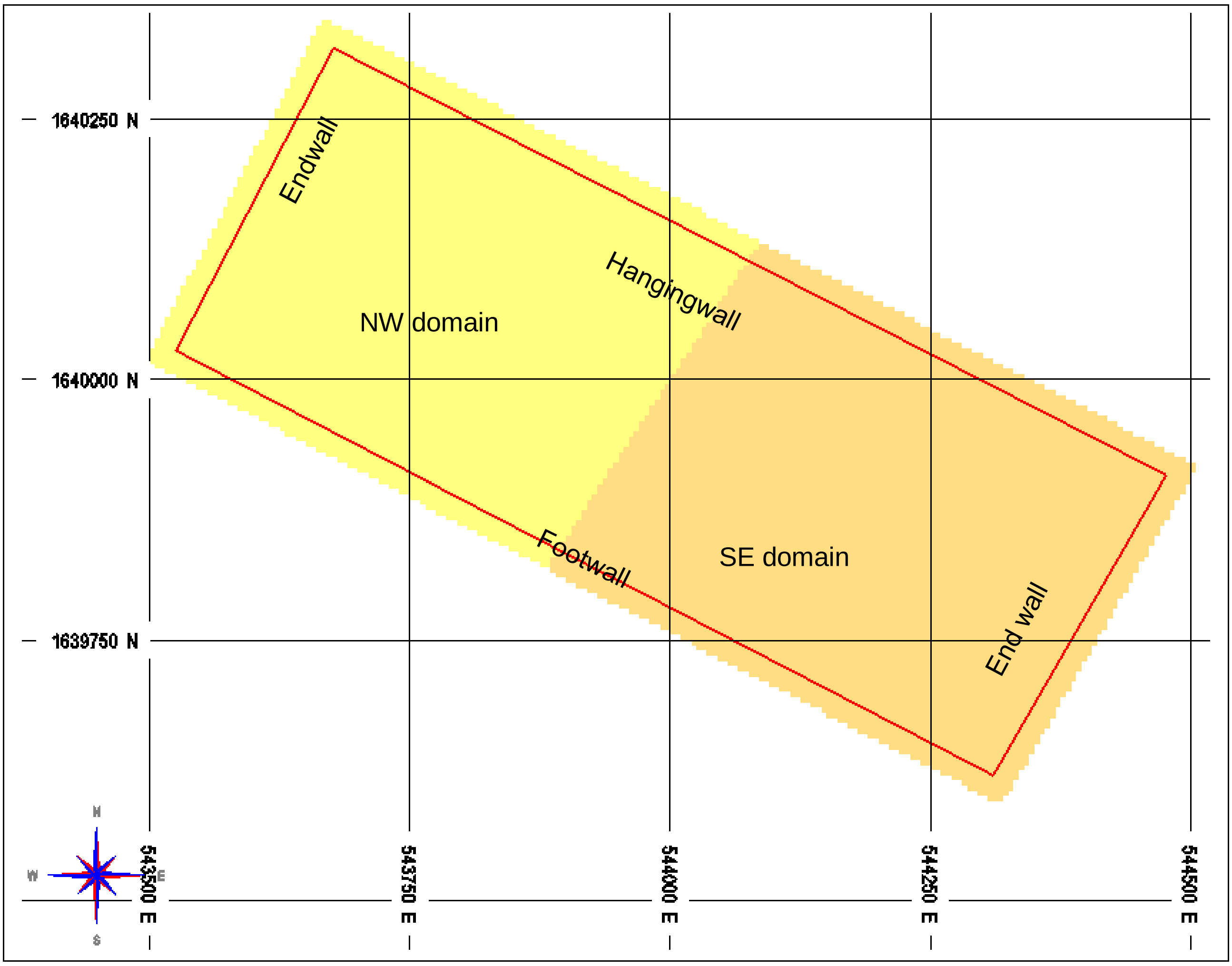
Snowden used the design criteria listed in Table 6.12 for the Ganajur pit designs. Ramp parameters are in Table 6.21. The minimum mining width was 20 m.

Table 6.12 Pit design parameters by weathering

Parameter	Oxide	Fresh, footwall, NW domain	Fresh, footwall, SE domain	Fresh, end walls, all domains	Fresh, hangingwall, all domains
Batter angle (°)	45	60	55	75	80
Batter height (vertical m)	10	10	10	10	10
Batter interval (vertical m)	20	20	20	20	20
Berm width (m)	5	5	5	5	5
Inter-ramp slope (toe to toe, no ramp) (°)	33.7	42.9	39.8	52.5	55.9
Maximum overall slope (crest to toe)	-	40.0	40.0	-	-

Figure 6.10 shows the location of the geotechnical design slope areas in relation to the lease boundary (red line).

Figure 6.10 Geotechnical design slope areas

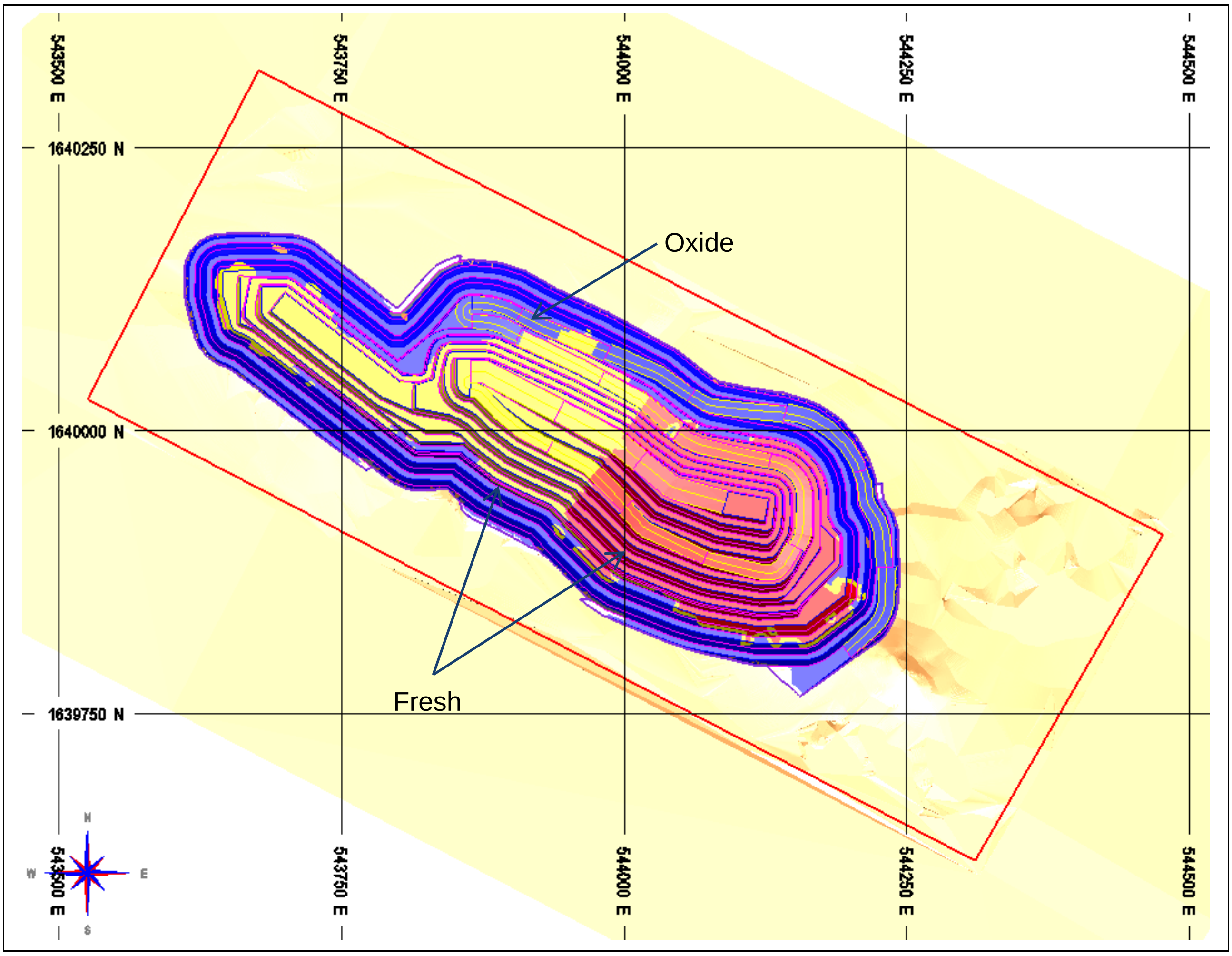


No major or minor faults were identified in the vicinity of the pit.

6.5.2 Ultimate design

The ultimate pit design is shown in Figure 6.11. It is based on pit shell 11 from the base optimisation with some north-east areas of pit 17 included. The oxide layer extends from 20 m to 30 m depth.

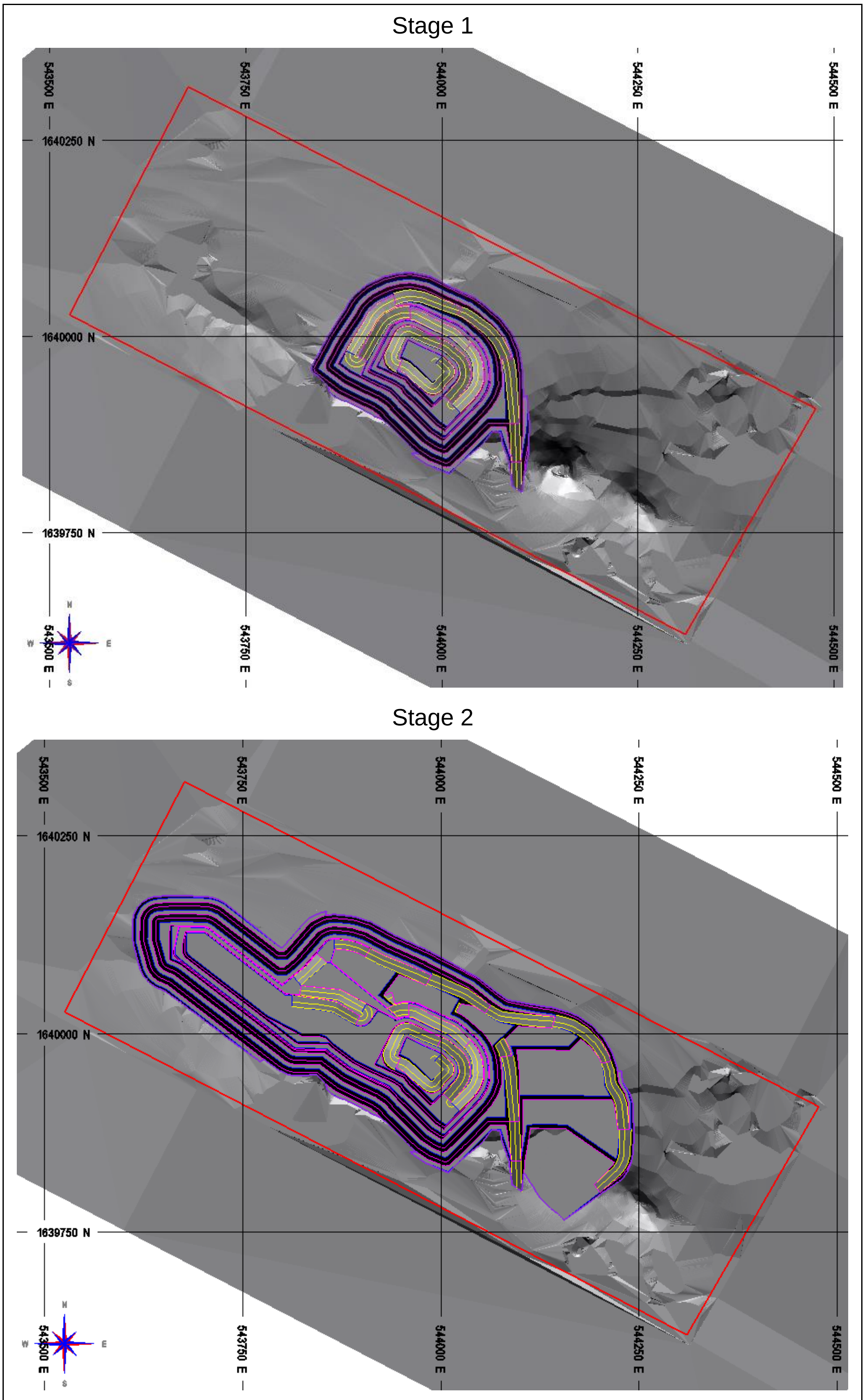
Figure 6.11 Ultimate pit design (plan)



6.5.3 Stage designs

Figure 6.12 shows the stage designs. Stage 1 is based on pit shell 3 from the base optimisation. Stage 2 is a subset of Stage 3 and was designed to allow oxide mining to be delayed, if required.

Figure 6.12 Stage designs



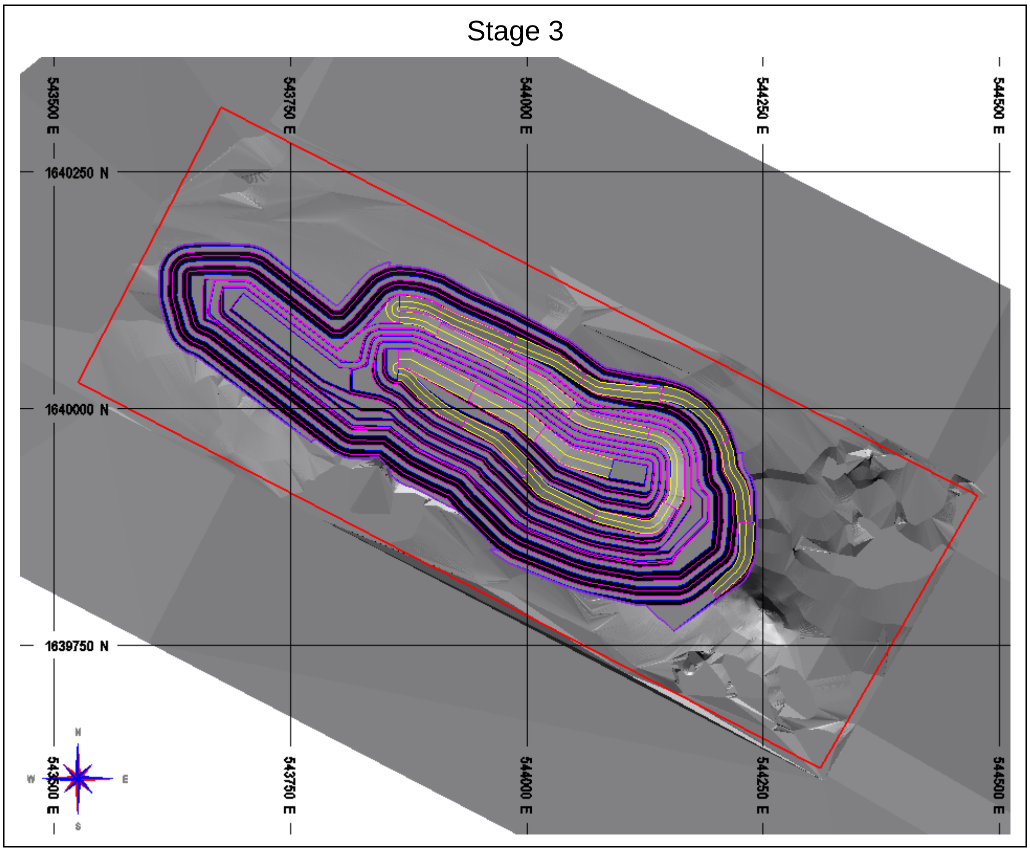


Figure 6.13 shows the plan layout of the pit designs, orebody outline (dashed line) and gold grade (coloured blocks) at the 500 mRL. The initial stage is located in an area of high grade.

Figure 6.13 Pit designs with gold grade (plan 500 mRL)

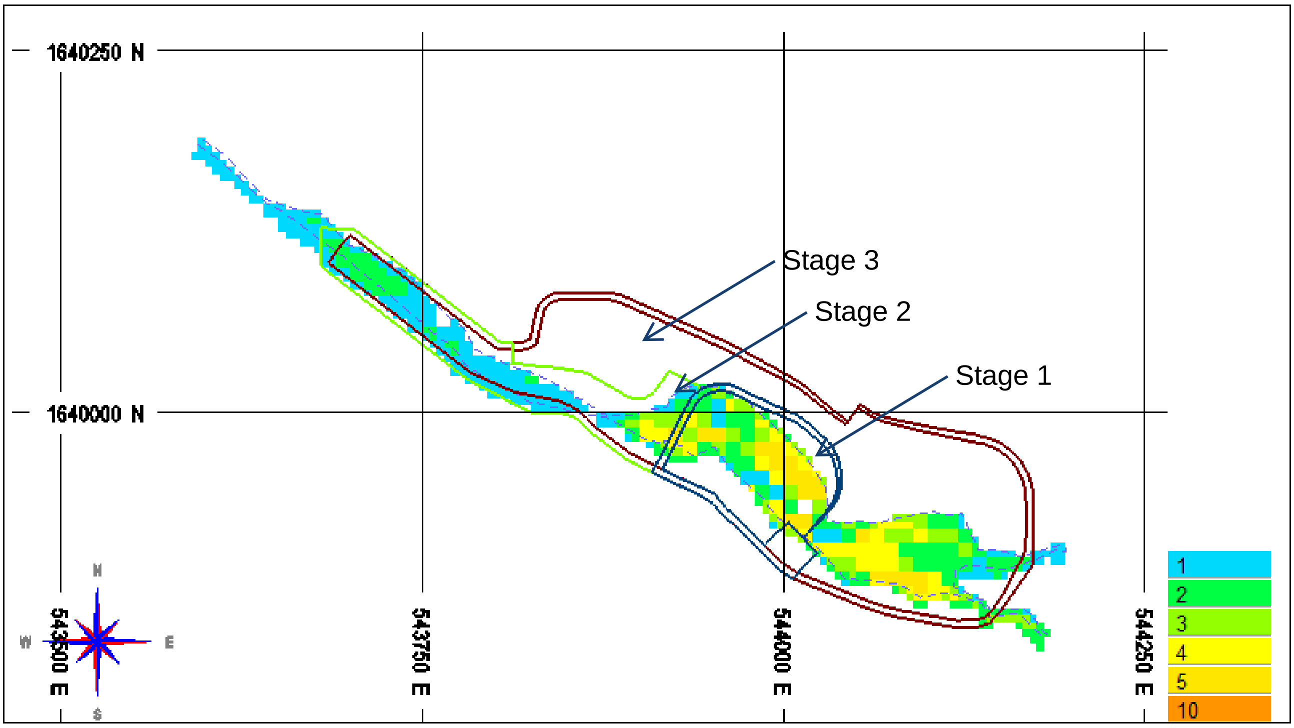
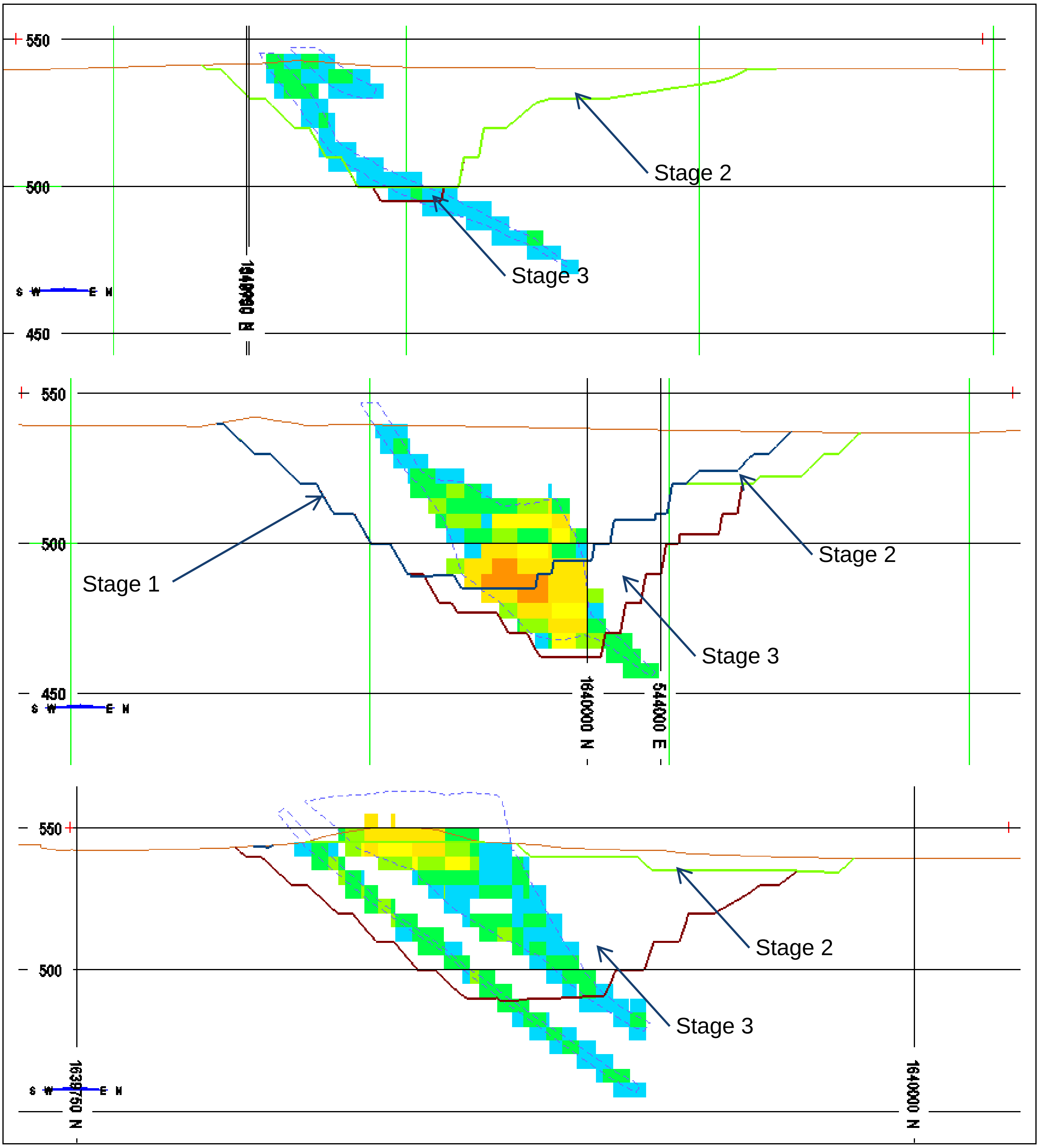


Figure 6.14 show the pit designs in section through the stages at 200 m intervals.

Figure 6.14 Pit designs with gold grade (sections on azimuth 32.3°)



6.6 Mining inventory

Table 6.13 summarises the pit design inventory by stage.

Table 6.13 Pit design summary inventory

Stage	Surface area (ha)	Volume (Mbcm)	Ore (Mdmt)	Waste (Mdmt)	Pit bottom (mRL)	Maximum depth (m)
1	5.4	1.0	0.6	2.4	485	55
2	8.5	1.3	0.4	3.1	500	40
3	0.3	1.8	1.5	3.7	455	85
Total	**14.2**	**4.1**	**2.5**	**9.2**	**455**	**85**

Table 6.14 summarises the total pit design ore inventory by oxidation state and Table 6.15 shows waste weathering type.

Table 6.14 Pit design ore inventory by oxidation state

Item	Oxide	Fresh	Total
Ore (Mdmt)	0.7	1.8	2.5
Gold grade (g/t)	2.6	3.7	3.4
Sulphidic sulphur grade (%)	0.2	2.8	2.1
Arsenic grade (%)	0.15	0.40	0.33
Lead grade (%)	0.003	0.003	0.003
Zinc grade (%)	0.006	0.006	0.006
Copper grade (%)	0.24	0.16	0.18

Table 6.15 Pit design waste inventory by weathering type

Weathering description	Code	Kt
Soil	1	547
Completely weathered	2	954
Highly weathered	3	1,381
Moderately weathered	4	1,433
Slightly/not weathered	5	4,919
Waste (total)		**9,234**

At an economic cut-off of 0.8 g/t Au, a total of 2.5 Mdmt at 3.4 g/t was included in the pit design. Table 6.16 summarises the pit design ore inventory by resource category.

Table 6.16 Pit design ore inventory by resource category

Item	Measured	Indicated	Inferred	Total
Ore (Mdmt)	2.1	0.4	-	2.5
Gold grade (g/t)	3.6	2.0	-	3.4
Sulphidic sulphur grade (%)	2.2	1.5	-	2.1
Arsenic grade (%)	0.36	0.18	-	0.33
Lead grade (%)	0.003	0.003	-	0.003
Zinc grade (%)	0.006	0.006	-	0.006
Copper grade (%)	0.16	0.28	-	0.18

Table 6.17 provides a comparison between the average of pit shells 11 and 17 and the ultimate pit design. The differences are within acceptable limits.

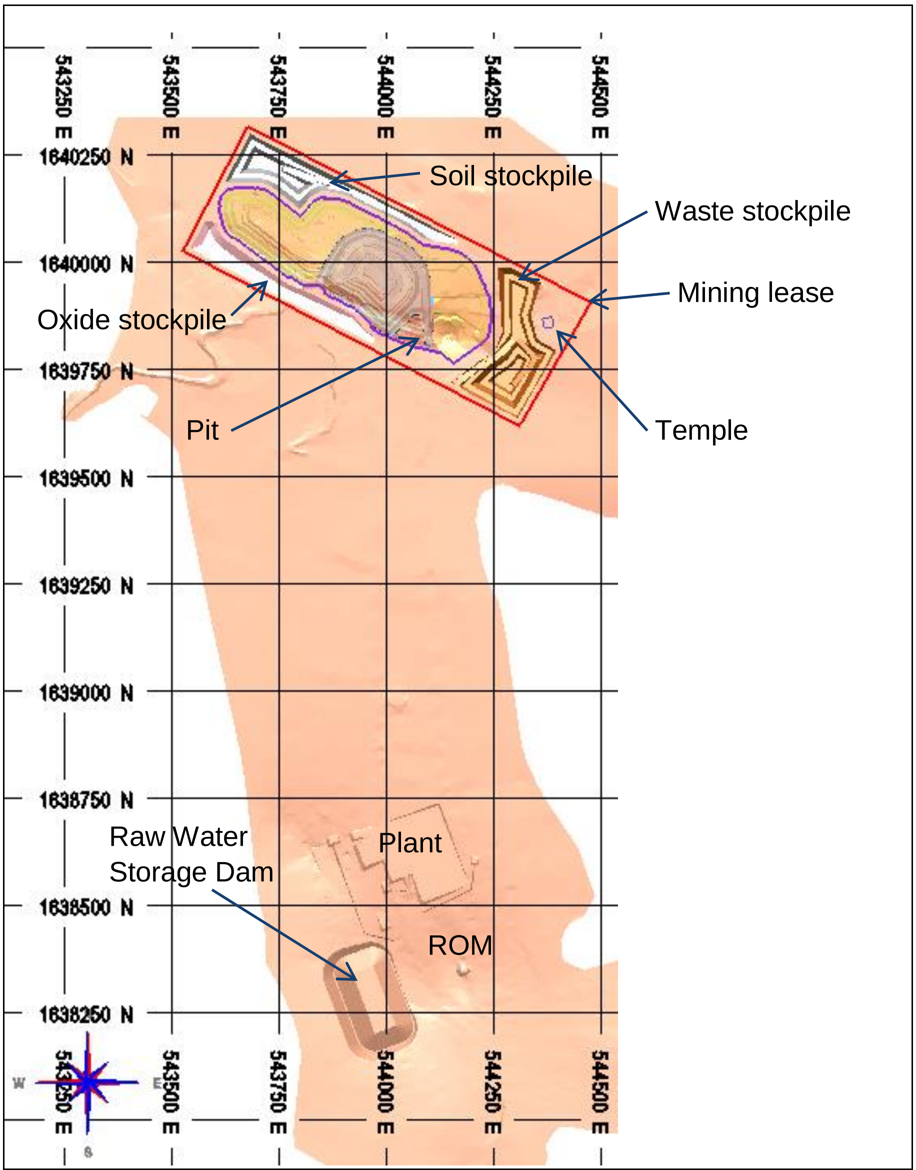
Table 6.17 Pit design ore inventory by resource category

Item	Pit shell 11	Pit shell 17	Pit design	Difference	Percentage
Total (Mdmt)	9.6	12.5	11.7	0.6	6%
Waste (Mdmt)	7.3	9.9	9.2	0.6	7%
Ore (Mdmt)	2.3	2.6	2.5	0.1	2%
Product (kdmt)	215	228	221	-0.8	0%
Undiscounted value ($M)	175.8	178.4	173.2	-3.9	-2%

6.7 Site layout

Figure 6.15 shows the overall site layout including the mining lease and the plant area.

Figure 6.15 Site layout



6.7.1 Clearing areas

Figure 6.16 shows the areas that require clearing over the life of the mine.

Figure 6.16 Clearing areas

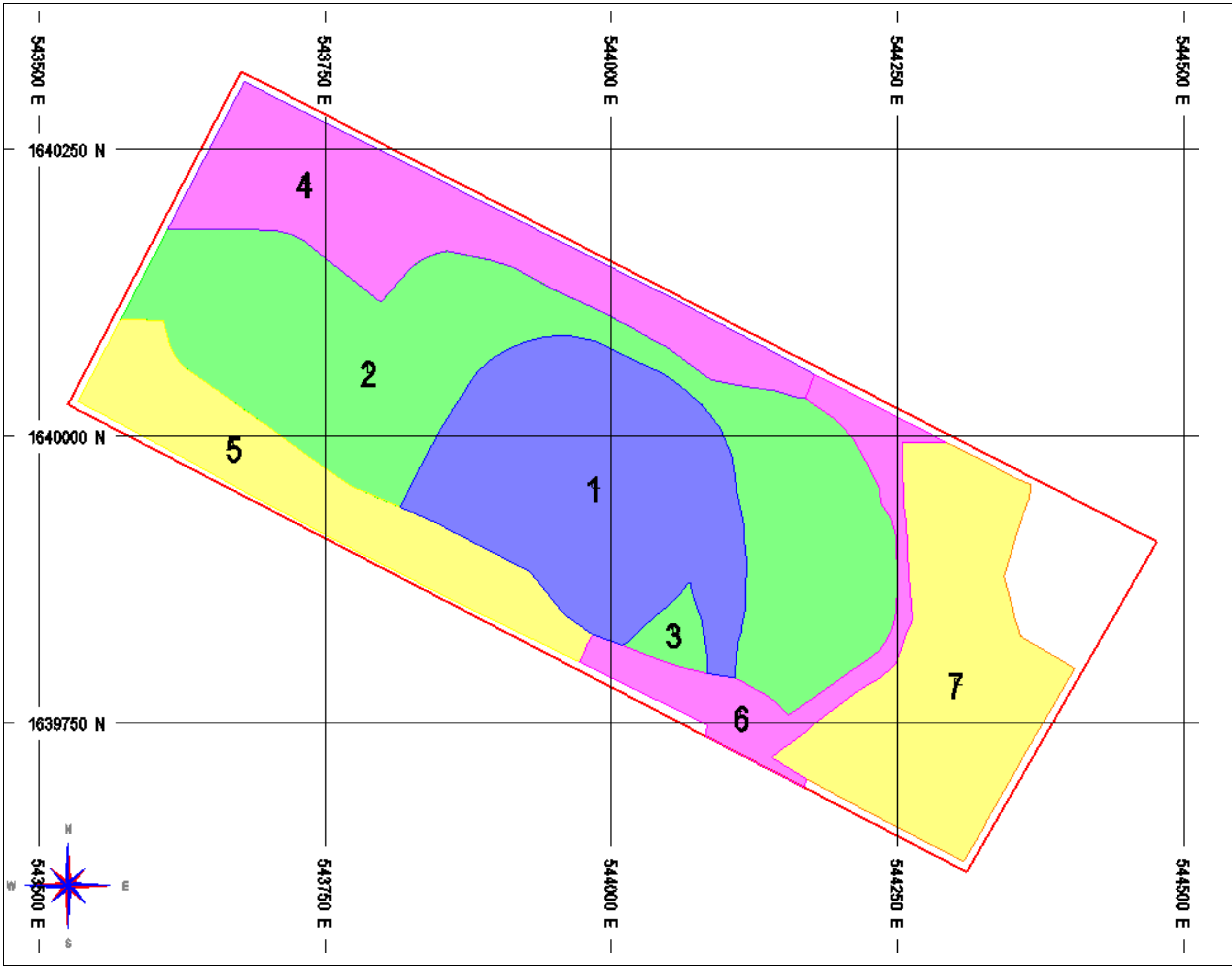


Table 6.18 summarises the clearing areas and the soil volumes recovered from these areas to a depth of 140 mm.

Table 6.18 Clearing areas and soil recovery volumes

Area	Description	Area (ha)	Total soil volume recovered (bcm)
1	Stage 1	5.4	76
2	Stage 2	8.5	119
3	Stage 3	0.3	4
4	Soil stockpile	3.8	-
5	Oxide stockpile	2.8	39
6	Roads	1.6	22
7	Waste stockpile	4.4	62
Total		**26.8**	**322**

A 40% swell factor was used to calculate load and haul volumes with a subsequent 20% shrinkage factor for stockpile volumes due to compaction (i.e. 100 m^3 of soil will swell to 140 m^3 for load and haul which will shrink to 112 m^3 at the stockpiled location).

6.7.2 Soil stockpiles

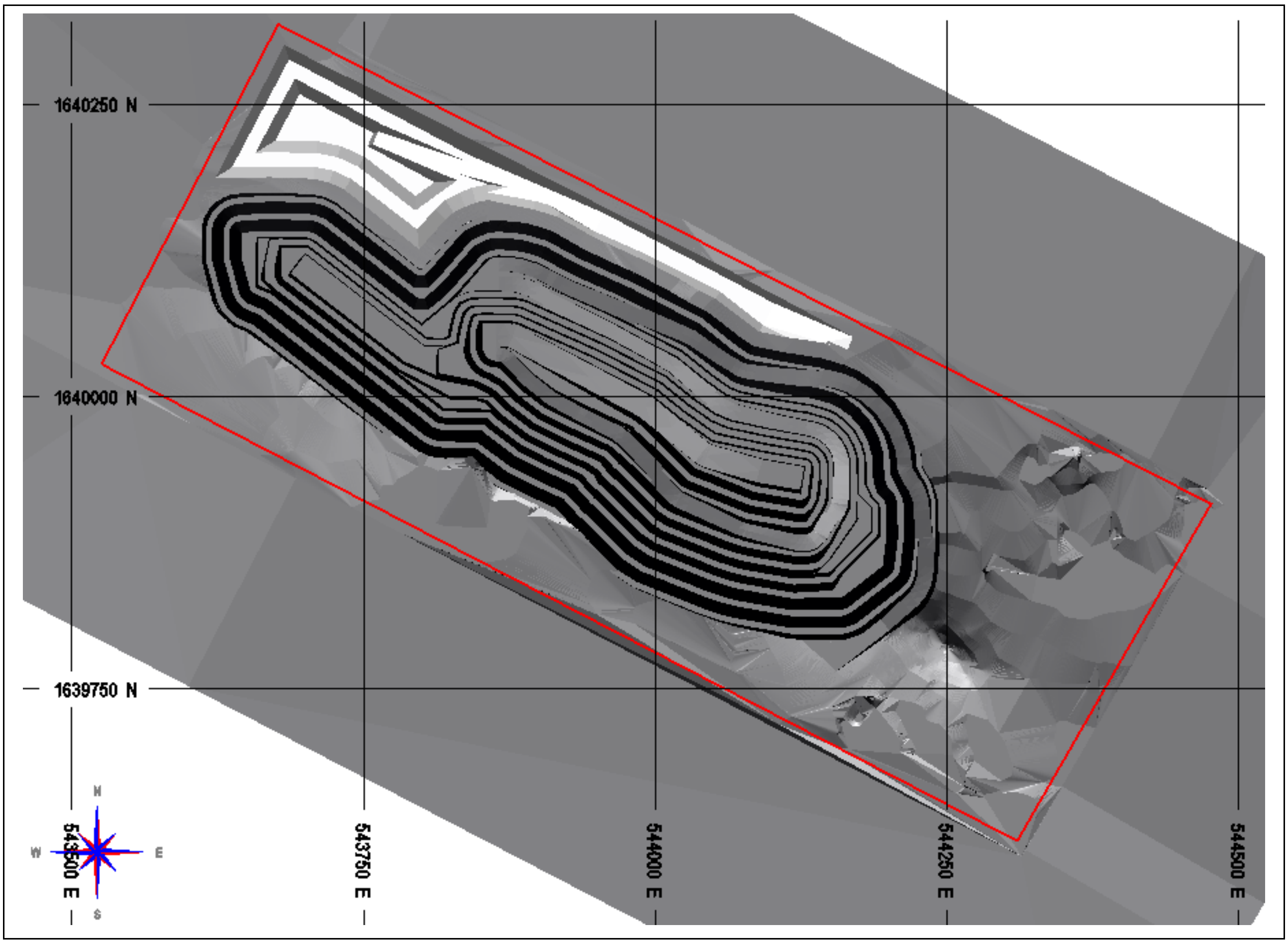
Snowden used the design criteria listed in Table 6.19 for the soil stockpiles.

Table 6.19 Soil stockpile design parameters

Parameter	Value
Maximum height (m)	15
Batter height (m)	7.5
Batter angle (°)	37
Berm width (m)	12

Figure 6.17 shows the soil stockpile location. It has a capacity of 210 klcm.

Figure 6.17 Soil stockpile location



6.7.3 Waste rock characterisation

The waste rock has undergone characterisation to ensure that waste does not contain detrimental characteristics which are likely to increase the cost of storage. The halo waste rock material around orebody has elevated arsenic levels from arsenopyrites – as does the non-halo material which has some soluble arsenic. However according to TCLP testing (Indian Standards) the waste rock classifies as non-hazardous and can be sold. There is a buffer capacity of 470 kt for waste and this stockpile will be continually built and depleted during the course the operation with the waste rock sold and moved off site for civil usage.

Seepage mitigation for any PAF rock may include a basic 300 mm in-situ compacted clay layer, with only with stormwater control back into the mine water circuit – while the waste rock material is still on the mine property.

6.7.4 Waste stockpile

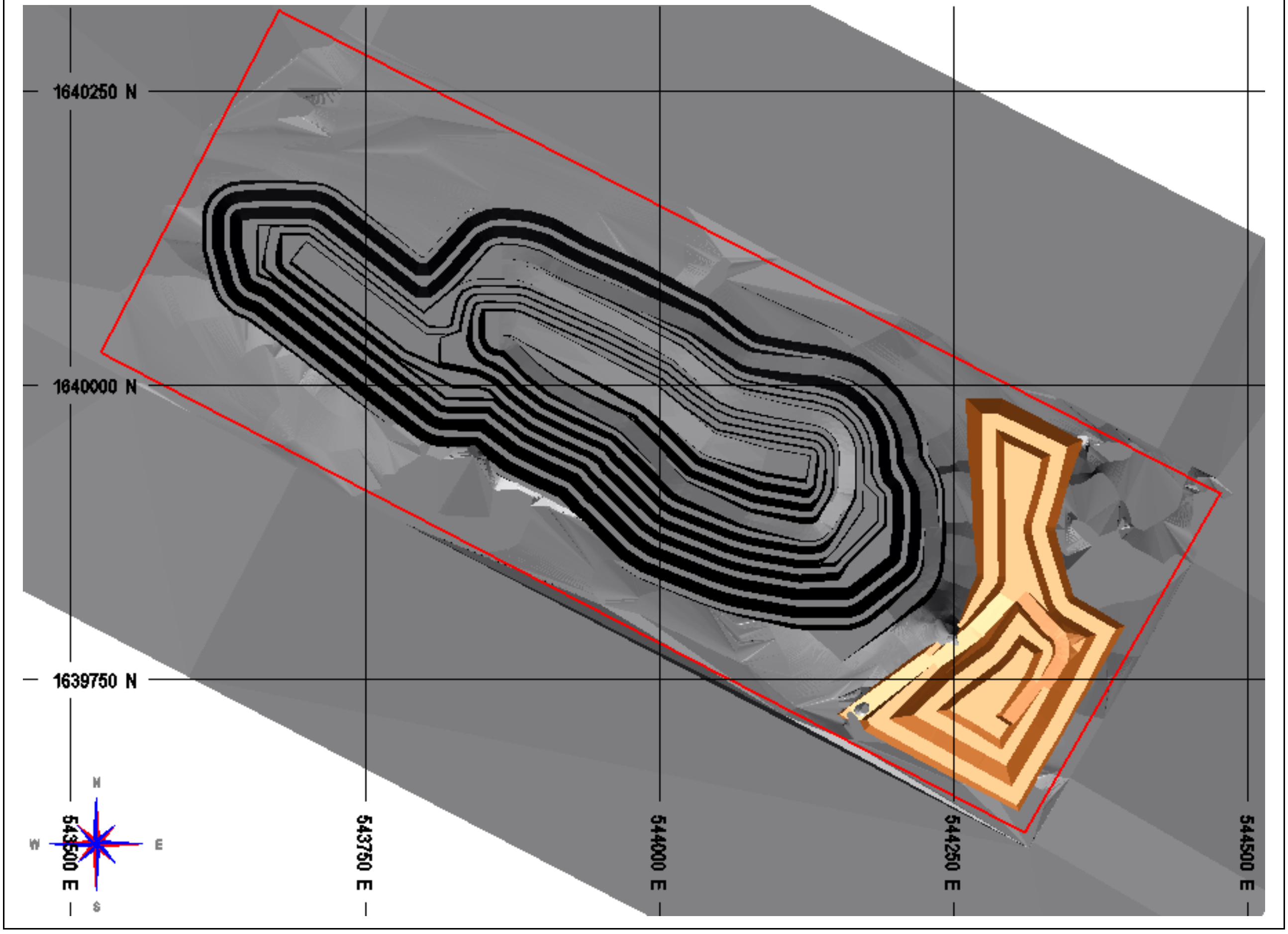
Snowden used the design criteria listed in Table 6.20 for the waste stockpile designs. Ramp parameters are in Table 6.21.

Table 6.20 Waste stockpile design parameters

Parameter	Value
Batter angle (°)	37
Lift height (m)	5
Berm interval (vertical m)	10
Berm width (m)	12

Figure 6.18 shows the waste stockpile location. It has a capacity of 300 klcm or about 480 kdmt.

Figure 6.18 Waste stockpile location



6.7.5 Stockpiles

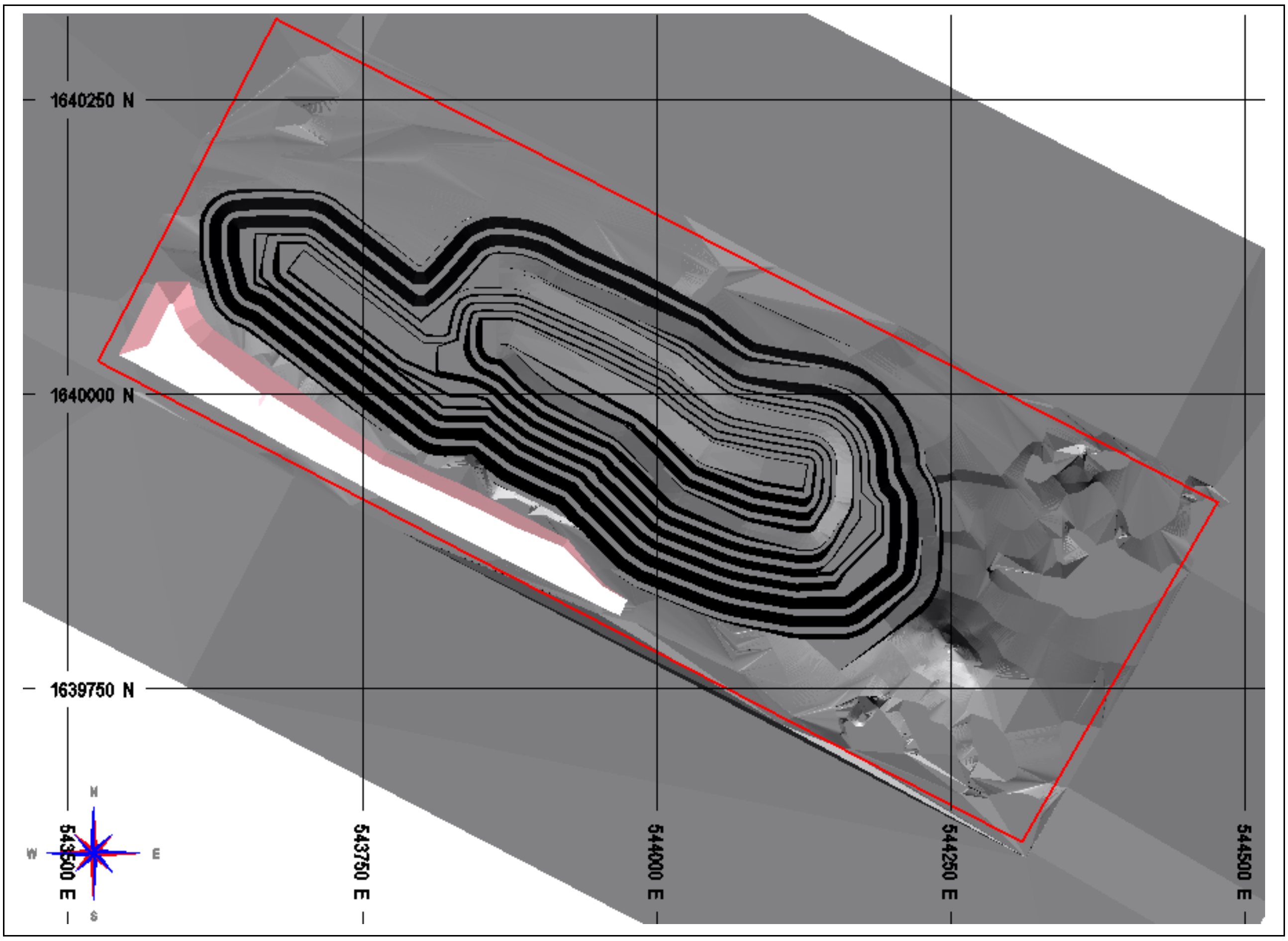
Run of mine

The ROM is located approximately 1.2 km to the south-southeast of the pit. Refer to the processing section of the report for more detail.

Long term

Figure 6.19 shows the long term low grade oxide stockpile location which has a capacity of 150 klcm or approximately 240 kdmt.

Figure 6.19 Long term oxide stockpile locations



6.7.6 Haul roads

The haul ramp and road design parameters in Table 6.21 are based on the largest equipment size proposed for use onsite with running width allowances of 3 for dual lanes and 1.5 for single lanes. If a different equipment size is used then these parameters will need to be adjusted and modifications made to all designs.

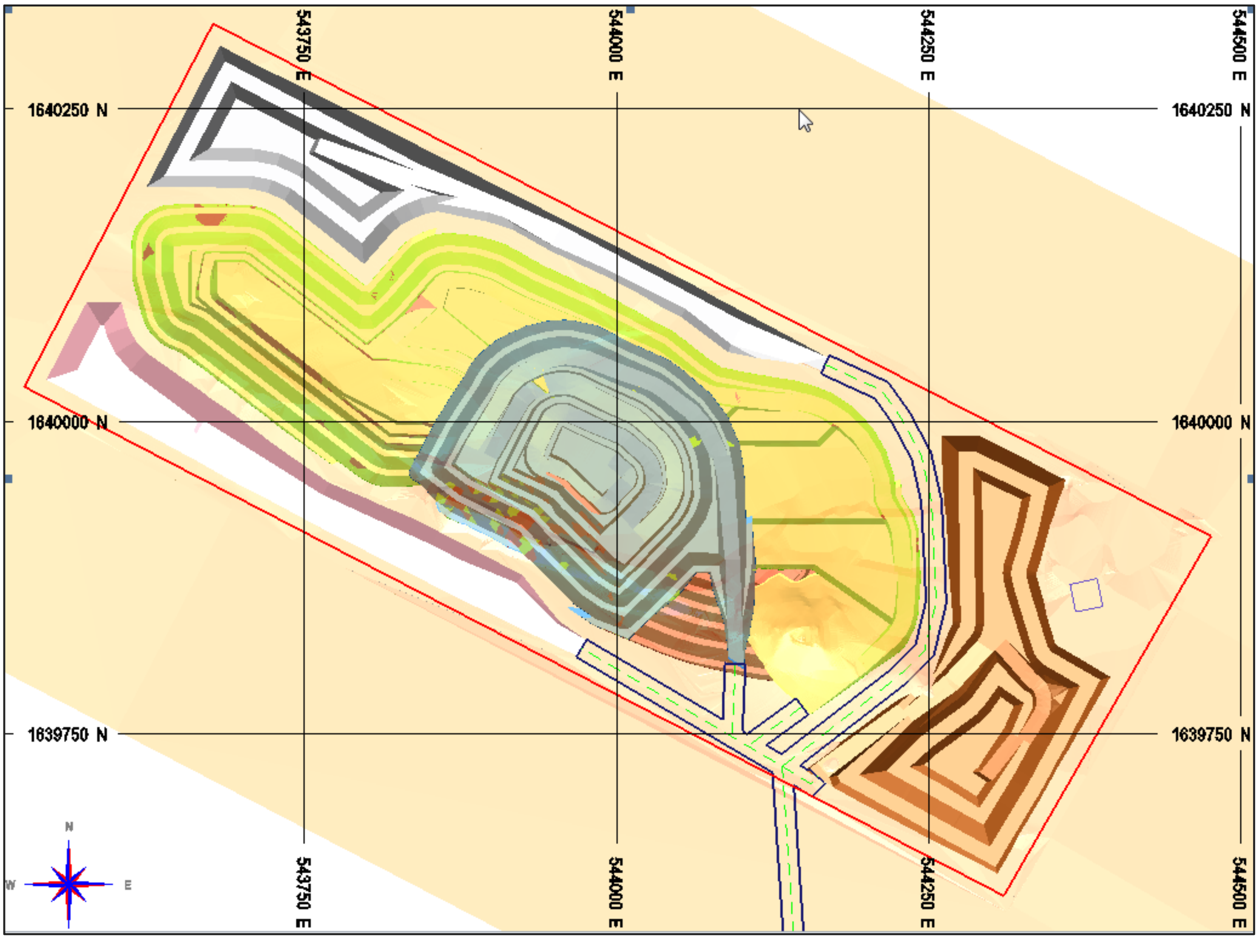
Table 6.21 Haul ramp and road design parameters

Parameter	Dual ramp	Single ramp	Dual road	Single road
Bund height (m)	0.9	0.9	0.9	0.9
Bund width – top (m)	1.0	1.0	1.0 (x2)	1.0 (x2)
Bund width – bottom (m)	3.4	3.4	3.4	3.4
Bund angle (°)	37	37	37	37
V-drain angle – road side (°)	26.6	26.6	26.6	26.6
V-drain angle – bund side (°)	45	45	45 if cut, 37 if fill	45 if cut, 37 if fill
V-drain width (m)	1.0	1.0	1.0	1.0
Running width (m)	7.5	3.8	7.5	3.8
Cross fall (%)	3	3	3	3
Pit edge clearance (m)	1.0	1.0	-	-
Maximum gradient (1:X)	10	10	10	10
Total width (m)	**11.9**	**8.2**	**16.3**	**12.6**

The maintenance of all operation roads will be the responsibility of the mining contractor.

Figure 6.20 shows the proposed ex-pit road network.

Figure 6.20 Intra-pit road network with staged pit designs with stockpiles



6.7.7 Dewatering

The current view is that spoon drains will be dug as required to channel water to the creek. All pit dewatering will be the responsibility of the mining contractor. In addition to the drains, some pit perimeter bunds will be used to control water flow.

The environmental section of this report provides likely water inflows from groundwater and rainfall. Their future studies planned to see if the pit water could be suitable for plant makeup water.

All future dewatering designs need to be validated by a hydrologist to ensure that surface water flows will be controlled within the desired factor of safety. Some drains may need to be armoured with rock or matting to ensure excessive erosion does not occur.

6.7.8 Mining infrastructure

Mining infrastructure will be supplied by the contractor and consideration for the site for the mining contractor is provided in the infrastructure section.

6.8 Production scheduling

Snowden completed a quarterly mining schedule in its Evaluator optimisation software. Evaluator is based on a Mixed Integer Programming formulation which seeks to maximise net present value (NPV) for a given inventory, economics and set of constraints.

6.8.1 Parameters

Based on the quarterly time scale, quantities were aggregated to the bench level by stage and material type.

All benches within a stage were dependent on the bench above being mined out. In addition, benches from subsequent stages were prevented from mining below the current stage

The following constraints were applied for mine scheduling:

- Mining:
  - Maximum of three 10 m benches per year of vertical advance
  - Maximum mining rate of 2,744 kt/a
  - Long term stockpile size limited to 500 kt.
- Processing:
  - 300 kt/a throughput for oxide circuit and sulphide (fresh) circuit.
  - Short ramp-up periods at start of production, and then start of fresh production.
  - Throughput at 38 dt/h or 912 t/d, or approximately 300 kt/a.
  - Maximum SS grade in the ore feed is 2.8%. This provides balance between the feed and the ultra-fine grinding circuits.
  - No simultaneous processing of oxide and sulphides.

Because there is no mixing of the plant feed by oxidation state, it is envisaged that all oxide would be processed until there is sufficient sulphide and any residual oxide accumulated in sulphide mining could be processed from stockpiles at the end of mining.

This means that during the initial processing of oxide, sulphide will be stockpiled until there is sufficient sulphide to keep the plant full. Then this sulphide will be rehandled off the stockpile for processing, so that the sulphidic sulphur in the plant feed does not exceed 2.8% SS. A tonnage of oxide will be accumulated on the mining lease whilst the sulphide is being mined.

6.8.2 Mining schedule

The total ex-pit movement is shown by stage in Figure 6.21. The mining rate is constrained by the maximum allowed during quarters 4 to 6 and 9 to 11. Until the 510 mRL in stage 2, stage 2 and 3 are effectively mined together. The total life of mining is approximately 7.6 years.

Figure 6.21 Total ex-pit movement schedule by pit stage

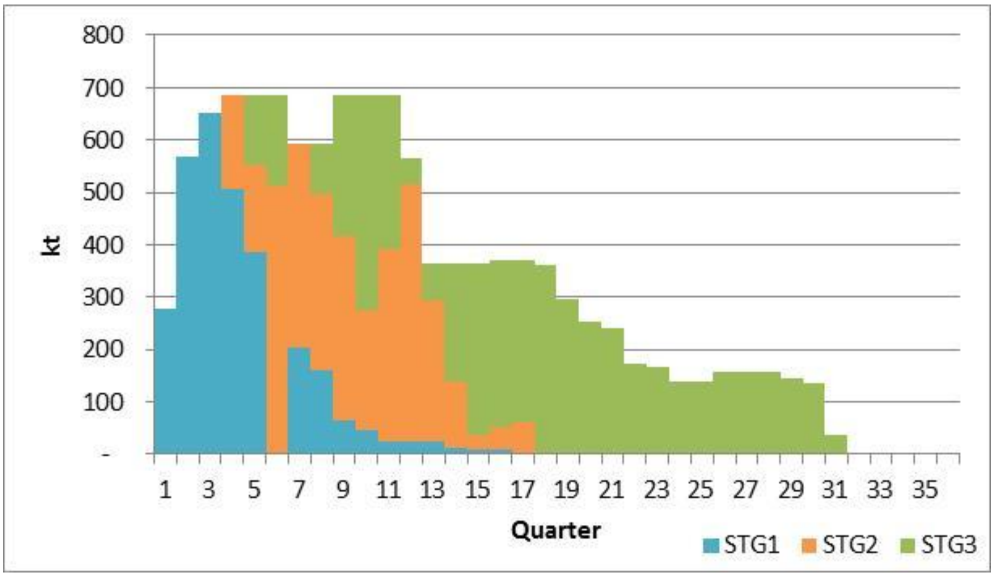


Figure 6.22 shows the total ex-pit movement by rock type. Initially the oxide is stockpiled for two quarters and processing commences in quarter 3, and then mined to satisfy an early oxide feed until continuous amounts of sulphide are available from the pit.

Figure 6.22 Total ex-pit movement schedule by rock type

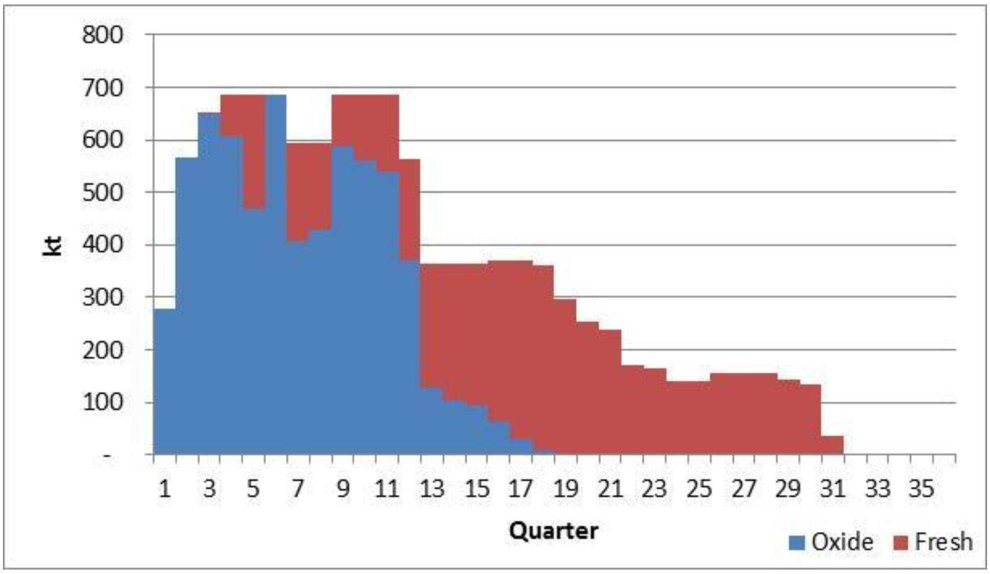


Figure 6.23 shows the total ex-pit movement by material type. After the two-quarter pre-production period (quarters 1 to 2) ore movement from the pit generally exceeds the feed requirement of 75 kt/q. This is to ensure that the feed is from a single rock type and meets the sulphidic sulphur constraint.

Figure 6.23 Total ex-pit movement schedule by material type

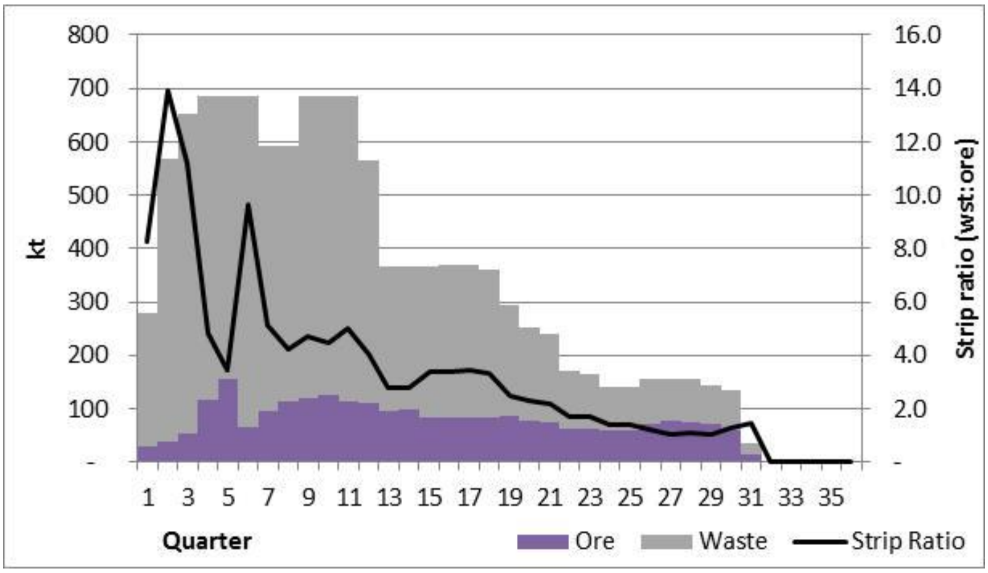


Figure 6.24 shows the ex-pit waste movement by weathering type. Until the back half of the schedule the majority of weathering types present in a quarter. Due to the limited waste stockpiling space, all these weathering types will need to be sold.

Figure 6.24 Ex-pit waste movement schedule by weathering type

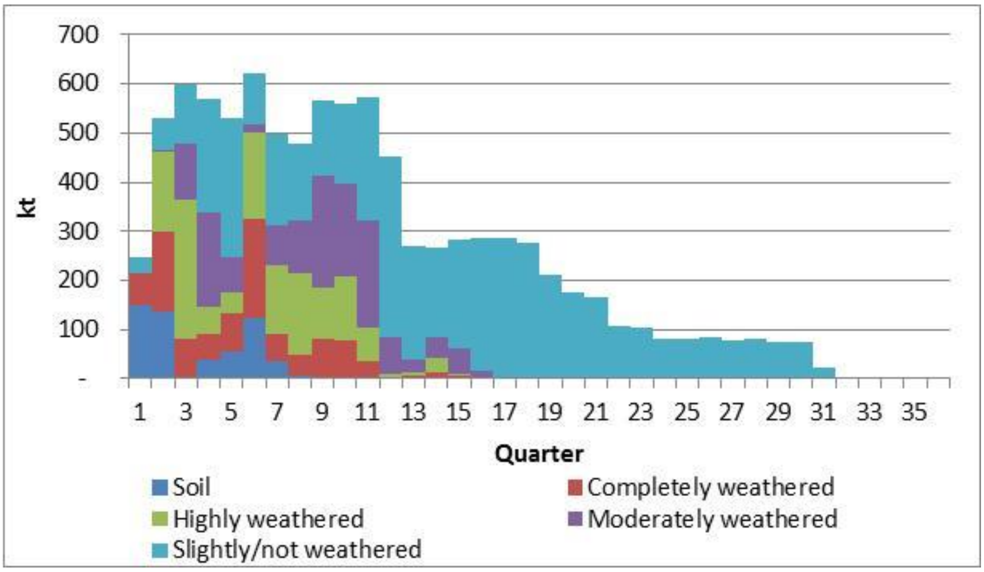


Figure 6.25 shows the long-term stockpile balance by rock type. The total stockpile size never exceeds the constraint of 500 kt.

The majority of the long-term stockpile is oxide which is stockpiled while sulphidic ore (fresh) is mined and processed (quarters 7 to 31). The oxide stockpile is processed from quarter 31 until it is entirely reclaimed.

For the six-finger ROM design, the front four fingers would be the active ROM stockpiles and the back two fingers will be high grade and a low grade strategic fresh stockpiles. The smaller fresh stockpile is primarily required to allow blending of sulphidic sulphur and will be stockpiled on the rear ROM stockpiles.

Figure 6.25 Long-term stockpile – balance

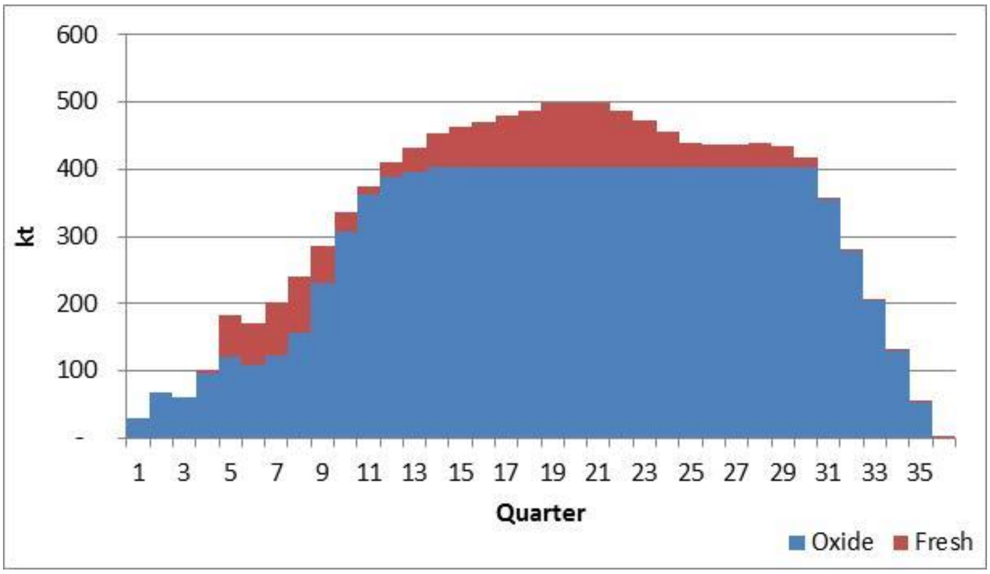


Figure 6.26 shows the long-term stockpile gold grades. The peaks in fresh gold grade correspond to mining at depth in stages 1 and 3.

Figure 6.26 Long-term stockpile – gold grade

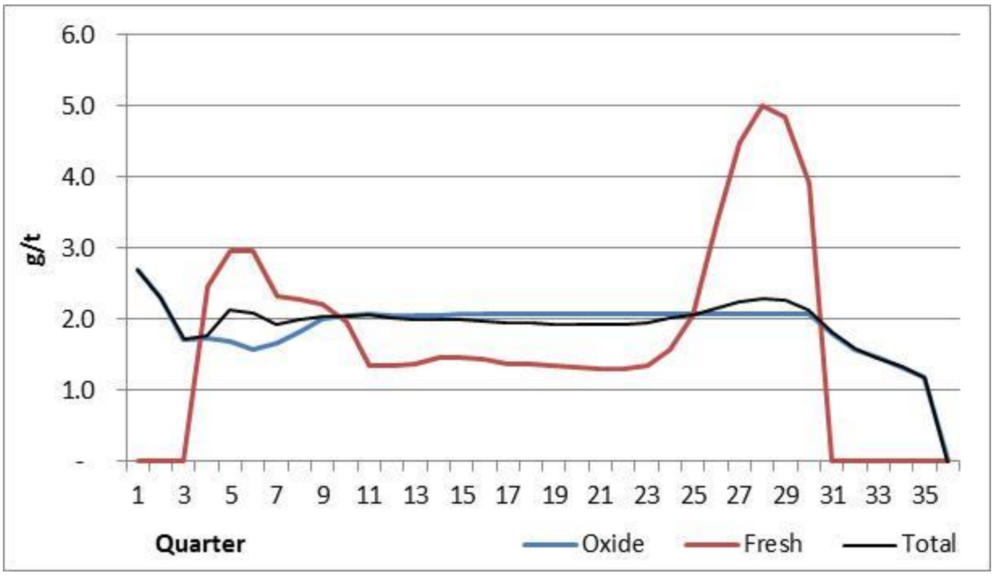
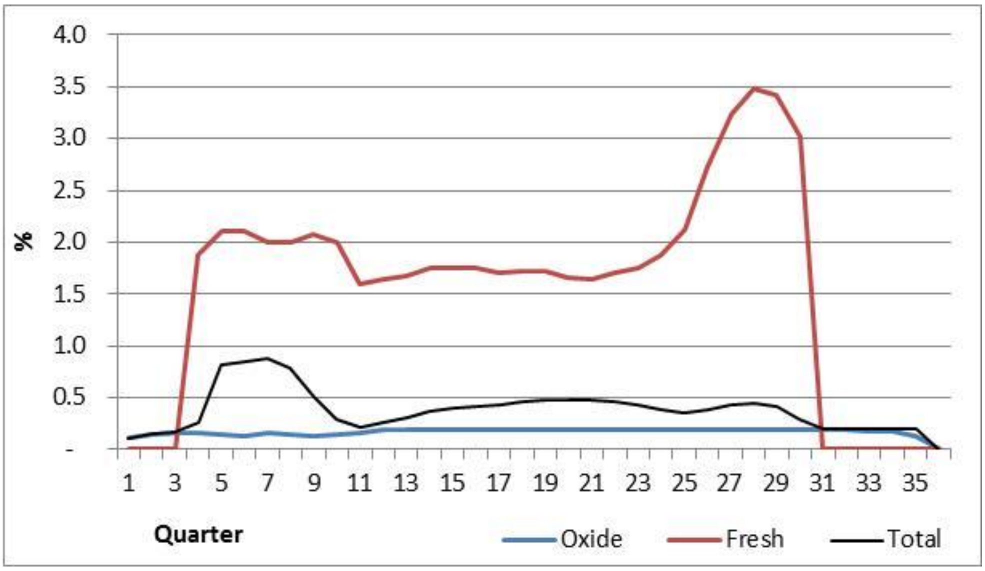


Figure 6.27 shows the long-term stockpile sulphidic sulphur grades. The peaks for fresh ore are associated with high gold grades.

Figure 6.27 Long-term stockpile – sulphidic sulphur grade



6.8.3 Processing schedule

The process feed schedule is shown in Figure 6.28. Initial processing is oxide material for a year. At this time (quarter 7), the sulphide circuit is commissioned, leading to a dip in production. When fresh ore is exhausted in quarter 31, sulphide production ceases and oxide processing recommences on the remaining stockpiles.

Figure 6.28 Process feed by rock type

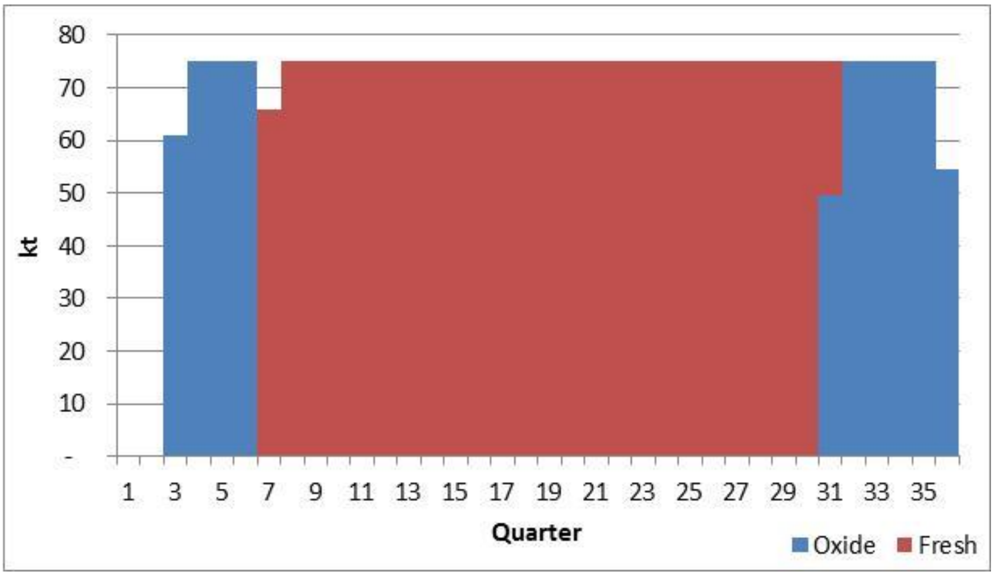
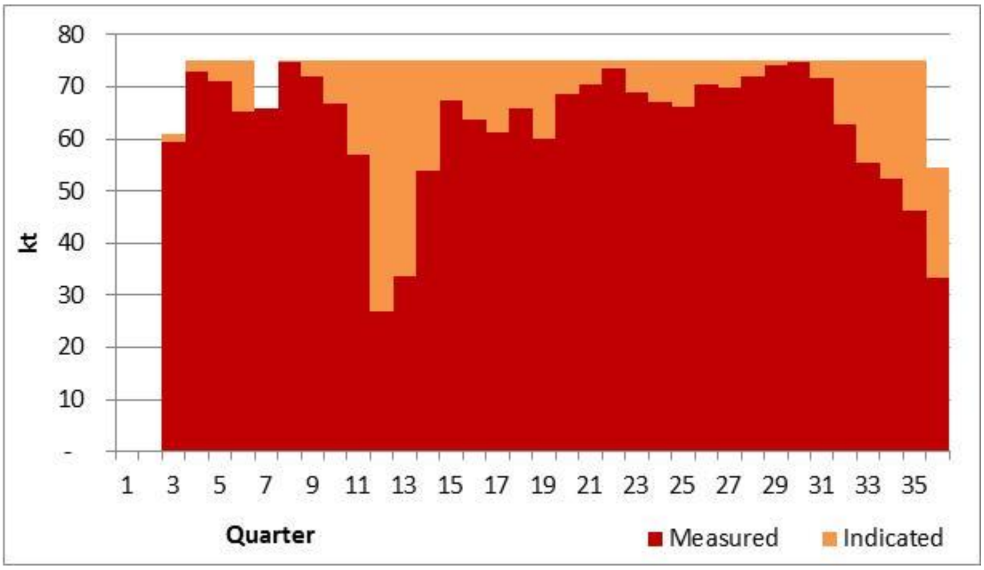


Figure 6.29 shows the process feed schedule by resource category. Feed is primarily from Measured Resources with only quarters 12 and 13 having more than half the feed from Indicated Resources.

Figure 6.29 Process feed by resource category



During fresh production, the schedule is mainly constrained by the 2.8% sulphidic sulphur grade limit, which controls the consistency of the concentrate production. The processed sulphidic sulphur grades and concentrate tonnes produced in the sulphide plant are provided in Figure 6.30 and Figure 6.31.

Figure 6.30 Processed sulphidic sulphur grade

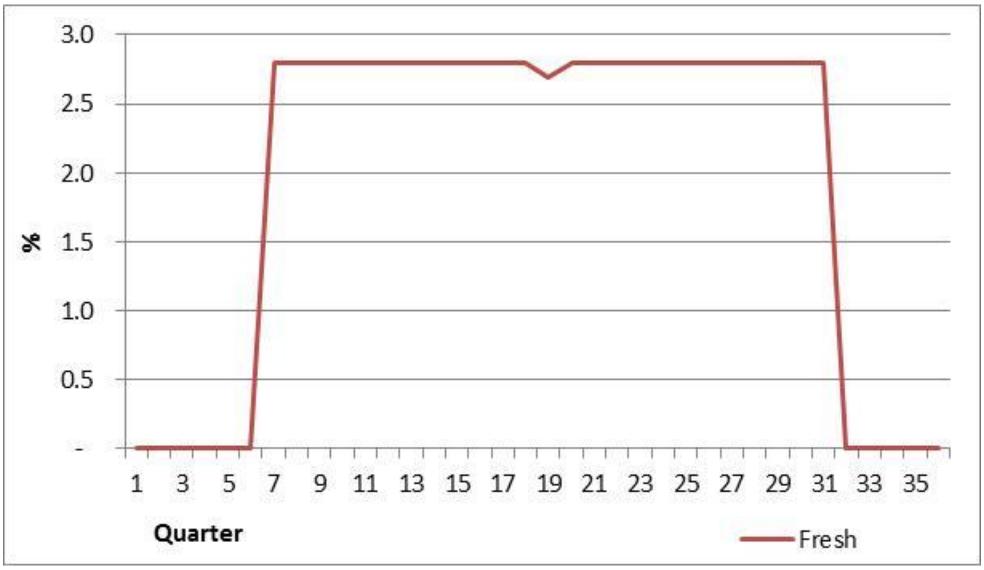


Figure 6.31 Concentrate tonnes produced

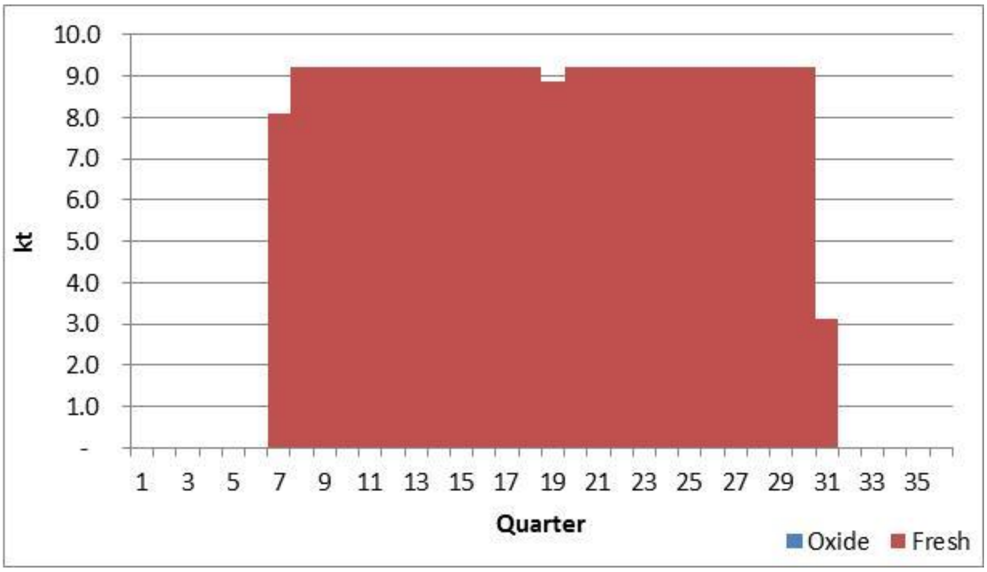
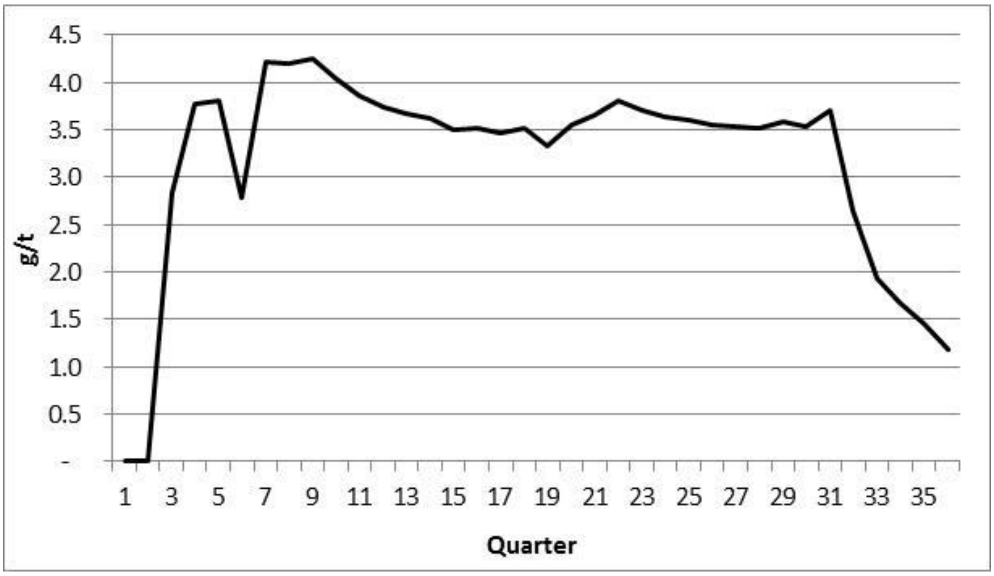


Figure 6.32 shows the process feed gold grade. As the schedule is largely controlled by the sulphidic sulphur and due to their correlation, the gold grade is relatively consistent at about 3.6 g/t (slightly above the average). To increase early cash flow, some quarters near the start of the sulphide circuit are higher and higher grade oxide is processed preferentially at the start of the schedule with lower grades pushed to the end of the schedule.

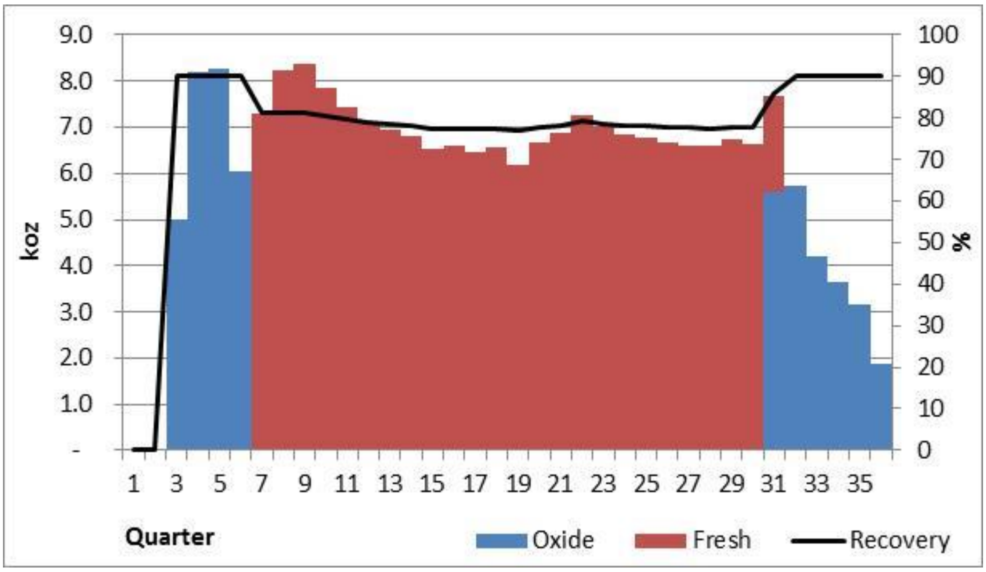
Figure 6.32 Process gold grade



6.8.4 Gold production schedule

The gold production schedule shown in Figure 6.33 is consistent at about 7 koz/qtr. When sulphide processing is complete, the production rate for the remaining oxide stockpile rapidly drops off as the gold grade decreases. Recoveries for the oxide material are constant 90% while fresh material varies around a 79% recovery.

Figure 6.33 Recovered gold schedule



6.9 Mine requirements

6.9.1 Operating philosophy

All mining activities will be executed by a contractor. A small owners team will oversee the contractors.

6.9.2 Roster

The mining operations are scheduled to work throughout the year, less public holidays and unscheduled delays such as high rainfall events which may cause mining operations to be temporarily suspended.

The mining operation operates seven days a week, in two 12-hour working shifts with the equipment services scheduled as required. The crushing plant is scheduled to operate continuously except for planned maintenance periods.

6.9.3 Equipment

Final equipment numbers will be dependent on the contractor selected for mining. The equipment requirements are based on responses from five contractor quotes.

Drilling

Depending on the contractor selected, for production drills will peak at three units. Due to the work areas available, Snowden recommends that the number of drills be minimised.

Loading

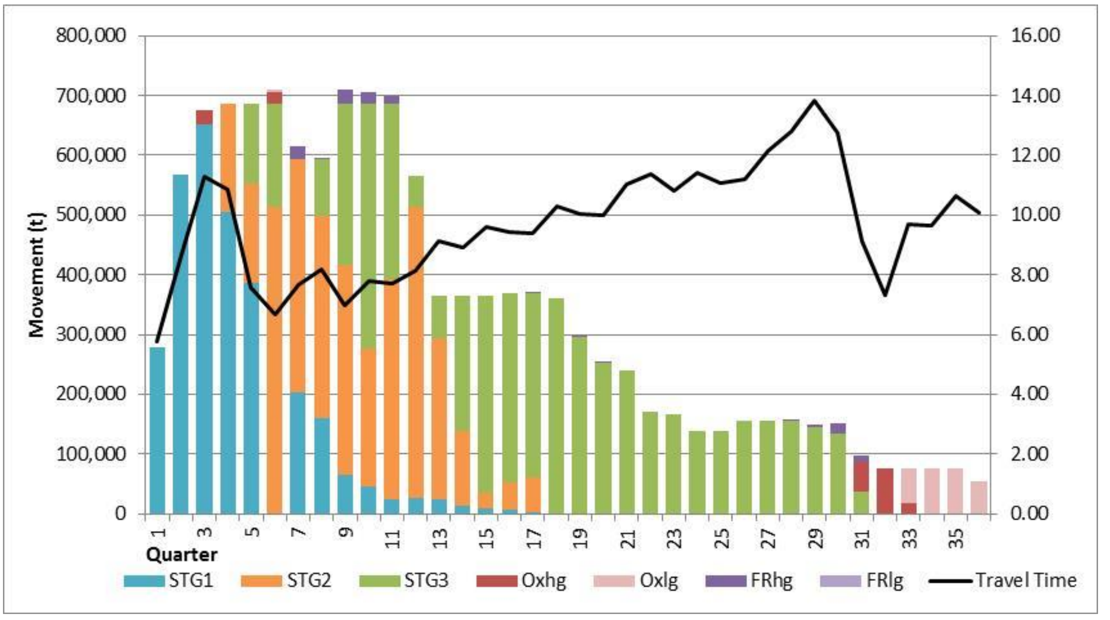
Depending on the contractor selected, excavators will peak at three to five units. Contractors capable of moving the required material with fewer excavators should be preferenced due the work areas available.

Hauling

Figure 6.34 summarises the estimated travel times and movement by stage. The early peak in travel times is due to hauling waste material to build the ROM. The later peak is a result of the increase in the depth of mining.

Due to the high movement and travel times in quarters 3 and 4, truck numbers are expected to peak early in the mine life. Based on contractor submissions this will be about 15 trucks.

Figure 6.34 Travel times and movement by stage



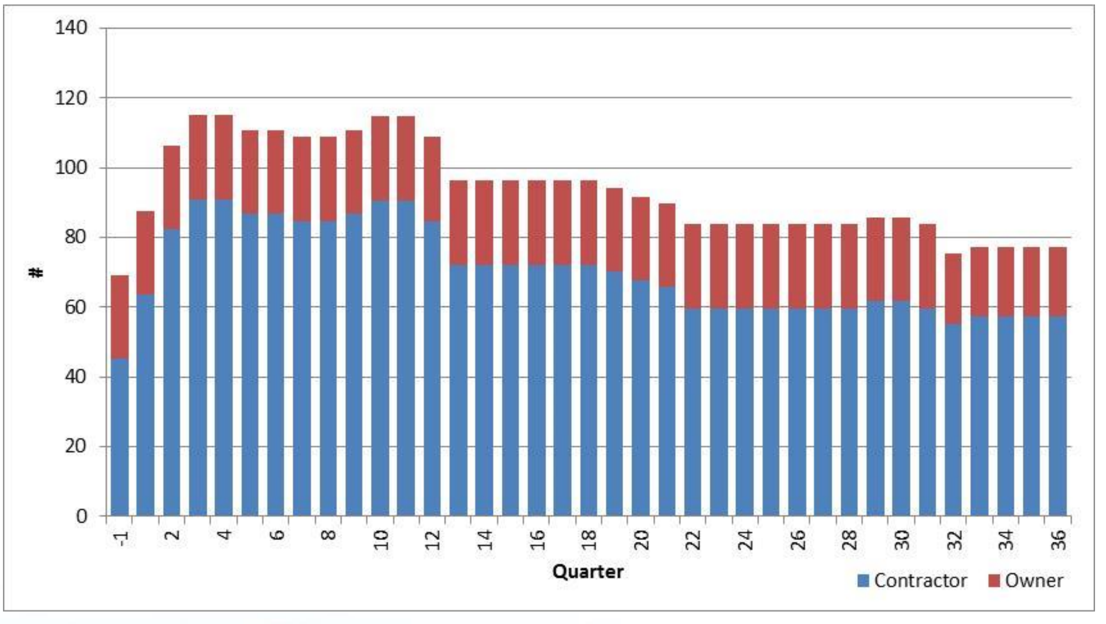
Ancillary

There is provision for wheel loaders, track dozers, graders, rock breakers and water trucks for dust suppression in the contractor submissions although unit numbers are not quoted.

6.9.4 Manning

Figure 6.35 summarises the total manning requirements for the mining operation which is estimated to peak at 115 in quarters 3, 4, 10 and 11. Total manning is provided in Figure 6.35.

Figure 6.35 Manning requirements



Contractor

The mining contractors will total about 90 and comprise the following roles:

- Mine manager
- Assistant mine manager
- Mine foremen
- Supervisors
- Operators (drills, diggers, trucks and ancillary)
- Blast crew
- General helpers
- Maintenance
- Store
- Office staff.

Owners

The owners team will total 24 full-time employees and comprise the following roles:

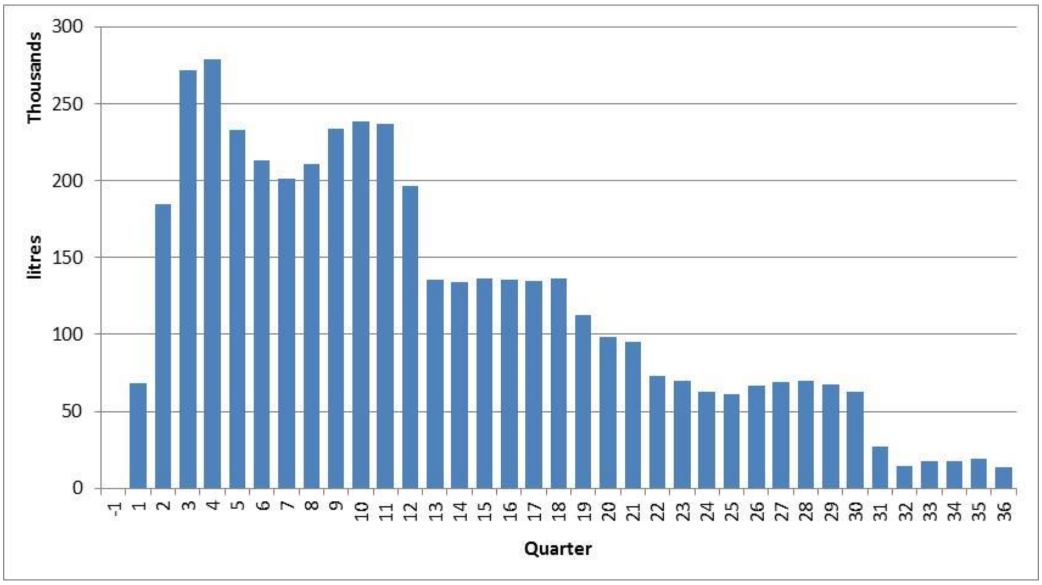
- Mine management
- Production supervision
- Mine engineers
- Geologists
- Field technicians
- Environmental
- Survey
- Geotechnical engineer
- OH&S.

6.9.5 Consumables

Fuel

An estimate of fuel usage is provided in Figure 6.36. Requirements peak at 280 kL in quarter 4 which equates to about 21.5 kL/week.

Figure 6.36 Fuel requirements



Power

Electrical power is derived from the Karnataka Power Transmission Corporation Limited.

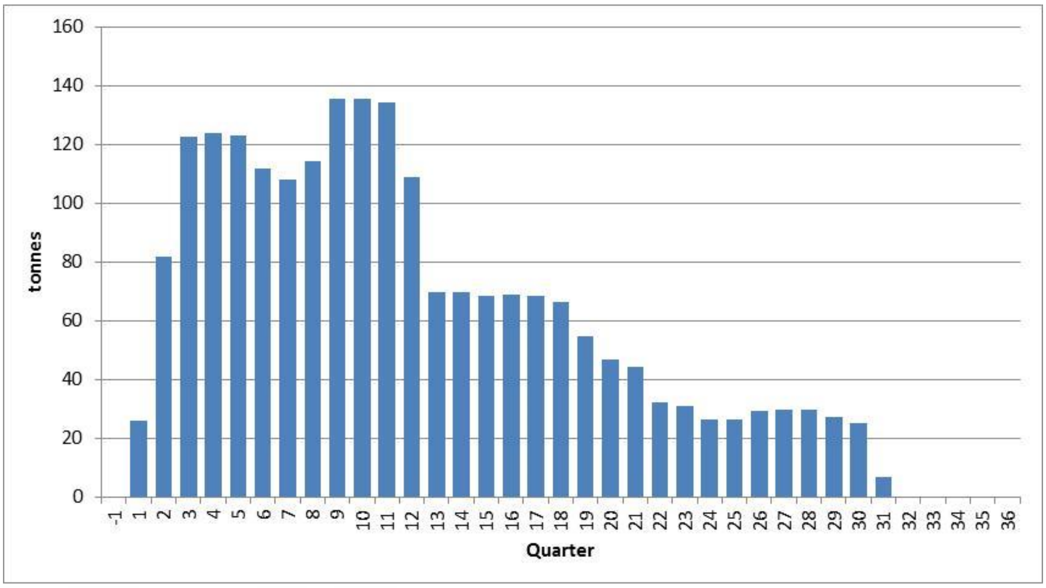
Tyres

Average monthly usage of tyres is 30 per month based on similar operations and taking into account that the trucks will be multi-axle having eight tyres per truck.

Explosives

An estimate of explosives usage is provided in Figure 6.37. Requirements peak at about 135 t in quarters 9 to 11. This peak equates to about 10.5 t/week.

Figure 6.37 Explosive requirements



Samples

Average monthly grade control assay samples produced is 3,700 samples, based on a 1 m sample interval for each assay and the sampling of blast holes.

6.10 Ore Reserve estimates

Ore Reserve estimates are currently reported for the mining plan at the Ganajur property, comprising in-situ reserves for the Ganajur deposit.

Notes: Tonnes and ounces have been rounded and this may have resulted in minor discrepancies.

6.10.1 Disclosure

Ore Reserves reported are based on this Feasibility Study, supervised by Frank Blanchfield and John Fodor who are Competent Persons for this report.

6.10.2 Known issues that materially affect Ore Reserves

Snowden is unaware of any issues that materially affect the Ore Reserves in a detrimental sense.

Snowden estimated gold Mineral Resources and Ore Reserve estimates for DESPL's Ganajur gold deposit. Snowden identified a maiden mining inventory based on the new Mineral Resource estimates from October 2016. The Ganajur Ore Reserve estimates were classified as Proved and Probable Ore Reserves using the guidelines of the JORC Code 2012.

While exercising all reasonable due diligence in checking and confirming the data validity, Snowden has relied largely on the data as supplied by DESPL to estimate and classify the Ore Reserve. As such, Snowden accepts responsibility for the geotechnical design configuration, pit design, production schedule, direct mining costs and the reserve estimate and classification, while DESPL has assumed responsibility for the accuracy and quality of the metallurgical data.

The key Modifying Factors used to estimate the Ore Reserve are based on the experience of Snowden and DESPL employees in this type of deposit and style of mineralisation. Table 6.22 summarises the status of material aspects of the April 2017 Ganajur Ore Reserve estimate, against the items listed in the table as the Competent Person's assessment of Ore Reserve estimation for the Ganajur deposit.

The information in this report that relates to Ganajur Ore Reserve is based on information reviewed or work undertaken by Mr Frank Blanchfield, FAusIMM, an employee of Snowden. Mr Blanchfield has sufficient experience relevant to the style of mineralisation and type of deposit under consideration and to the preparation of mining studies to qualify as a Competent Person as defined by the JORC Code 2012.

The scientific and technical information in this report that relates to process metallurgy is based on information reviewed by Mr John Fodor, FAusIMM, who is self employed as a metallurgical consultant and contracted to DESPL. Mr Fodor has sufficient experience that is relevant to the style of mineralisation and type of deposit under consideration and to the activity being undertaken to qualify as a Competent Person as defined by the JORC Code 2012.

Table 6.22 JORC 2012 Table 1 – Section 4

Item	Comment
Mineral Resource for conversion to Ore Reserve	Snowden prepared the updated Ganajur Mineral Resource estimate in October 2016. The Mineral Resource estimate was classified using the guidelines of the JORC Code 2012 and a summary is provided below. No planned dilution was applied to these estimates. The Ganajur Mineral Resources comprise the Ganajur, Mineral Resources and are inclusive of Ore Reserves.

Classification	Deposit	Tonnes (kt)	Au (g/t)
Measured	Oxide	580	2.82
	Sulphide	1,690	3.96
	Total Measured	2,300	3.67
Indicated	Oxide	130	1.85
	Sulphide	330	2.13
	Total Indicated	450	2.05
Measured + Indicated	**Total Measured and Indicated**	**2,700**	**3.40**
Inferred	Oxide	110	2.30
	Sulphide	110	2.29
	Total Inferred	210	2.30

Note: Small discrepancies may occur due to rounding

Item	Comment
Site visits	A visit to the Ganajur project site was undertaken by Mr Frank Blanchfield and Mr John Fodor in July 2016. Mr Frank Blanchfield and Mr John Fodor are the Ore Reserves Competent Persons for the current Ore Reserve estimate.
Study status	The current study status of the project is of a feasibility level. This Feasibility Study follows previous study work carried out by SRK (India) Pvt. Ltd in 2012, which delivered a Mineral Resource estimate and preliminary economic assessment (PEA) of the economic potential of the Ganajur Main Gold deposit, Karnataka, India. The 2017 Feasibility Study establishes the viability of oxide and sulphide ore extraction through the building of a minerals processing plant to produce dore gold.
Cut-off parameters	A nominal cut-off grade of 0.80 g/t Au was applied to oxides and 0.89 g/t for sulphides when developing the Ore Reserve estimate. There is very little material with gold grades lower than 1.0 g/t available from the resource model.
Mining factors and assumptions	To identify the Ganajur Ore Reserve, a process of Whittle pit optimisation, staged pit design, production scheduling and mine cost modelling was undertaken by Snowden. The mining method is conventional open pit drill and blast, load and haul on a 2.5 m mining flitch with a 10 m high blasting bench, reflective of semi-selective mining. The maximum excavator bucket size of 2.0 m^3 is matched to this selectivity. A stripping ratio of approximately 3.7 was identified. Overall, block dilution has reduced the recovered ounces by approximately 2% and marginally increased the ore tonnage processed by 5%.
Metallurgical factors and assumptions	The Ganajur Gold Mine is developed on the basis of treating oxide ore and sulphide ore separately. The oxide ore portion of the reserves will be processed via conventional cyanidation leach and carbon in pulp (CIP). The sulphide ore portion of the ore reserves will be processed via flotation for the recovery of gold and sulphides into a low mass sulphide concentrate. After flotation, the sulphide concentrate will then be ultrafine ground to approximately 10 microns whereby the gold can be extracted and recovered by conventional carbon-in-leach (CIL). Final gold recovery from carbon strip solution and gravity concentrate leach solution is by electrowinning onto stainless steel cathodes. The mineralisation modelling and metallurgical testwork indicate that conventional CIP extraction from oxide ores and CIL from leach fresh sulphide ores can be used to produce gold as dore. All the oxide ore unit processes included in the design are standard and common to many current gold operations, including: • Crushing • Grinding and classification • Gravity concentration (Knelson centrifugal concentrator) • Carbon adsorption (CIP) • Electrowinning • Smelting • Tailings disposal and effluent reclaim • Cyanide detoxification.

Item	Comment
	All the sulphide ore unit processes included in the design are standard and common to many current gold operations, including: • Crushing • Grinding and classification • Gravity concentration (Knelson centrifugal concentrator) • Flotation/Ultra-Fine Grinding on sulphide concentrates • Carbon adsorption (CIL) • Carbon desorption • Electrowinning • Smelting • Cyanide detoxification/arsenic removal • Tailings disposal and effluent reclaim. DESPL applied industry standard methods to prepare this estimate by developing the following components: • Metallurgical testwork • Process design criteria • Process design and flow diagrams • Engineering design criteria • Mechanical and electrical equipment lists • Process plant layout • Operating cost estimates • Capital cost estimates. The metallurgical factors for estimating the gold recovery estimates for oxide and sulphide processing were developed by DESPL. The sulphide ore gold recovery was estimated by the following geo-metallurgy methodology: • The sulphide sulphur (SS%) content of the sulphide ore was estimated by regression analyses, with gold and arsenic as the significant predictors • The mass recovery of sulphide concentrates is based on a flotation recovery of SS at 96% at a 22% SS concentrate grade • The CIL sulphide residue is estimated at 4.5 g/t gold and the flotation tailings estimated at 0.27 g/t gold • The gold recovery estimates were estimated by subtracting the final residue estimates (weighted by flotation mass recovery) from the gold ore reserve grades. This fresh ore recovery algorithm returned an average gold recovery for fresh ore of approximately 79%. The oxide recovery was fixed at 90%. It is the Competent Person's opinion that the plant production numbers are accurate and correct. It is reasonable to assume that the results obtained and design criteria and process flowsheet adapted for the project are reasonable and adequate for a feasibility study level of accuracy.
Environmental	Rock characterisation was completed in South Africa by Prime Resources and potentially acid-forming acid rock drainage items were not significant. The waste will be sold, likely back to the mining contractor for civil uses in their other operations. Waste dump size is only considered to be a holding stockpile with most of the waste transported from site in a short-term timeframe. The mining lease application was approved on 24 July 2015, via Letter No. 4/113/2010-M.IV by the Ministry of Mines, approving the grant of mining lease over an area of 72 acres for a period of 50 years for the Ganajur Gold Project, DESPL. The approval is as per Section 10 A(2)(b) of the MMDR Act 2015. The Grant order/Letter of Intent from the Karnataka State Government is awaited. The State Government will issue mining lease grant order (as per section 5(2)(b)(ii) of the MMDR amendment Act 2015). The mining lease application allows provision for tailings dams and waste dumps.
Infrastructure	DESPL is negotiating with the local authority for power purchase from the electricity grid. DESPL has indicated the plant build will be a EPCM execution with DESPL providing oversight over the EPCM Engineer. Accommodation will be in surrounding communities and mostly at the city of Haveri that has a population of 80,000 persons.

<table>
<tr><th>Item</th><th>Comment</th></tr>
<tr><td>Cost and revenue factors</td><td>Process costs were used were developed by DESPL in 2017 for oxide and sulphide as:
<table>
<tr><th>Material treated</th><th>Average gold grade (g/t)</th><th>Process operating cost (US$/t)</th></tr>
<tr><td>Oxide</td><td>2.6</td><td>8.36</td></tr>
<tr><td>Fresh</td><td>3.7</td><td>23.53</td></tr>
</table>
Mining costs were supplied by DESPL and developed from the existing contract.
The all-up mining operating cost was estimated to be US$1.83/t mined.
The mining capital cost was absorbed by contract mining.
DESPL provided other capital costs, that were estimated by CPC Engineers and others as follows:
• Owners costs: US$8.2 million
• Process capital costs: US$35.6 million
• Tailings storage facility: US$2.5 million
• Closure costs: US$0.5 million
• Sustaining costs: US$3.1 million per annum.
Closure costs are included in the valuation model.
All costs were supplied in US$.
Refining costs of US$0.075/t ore feed and royalties of 5.4% were applied to all gold produced.</td></tr>
<tr><td>Revenue factors</td><td>DESPL supplied a gold price of US$1,255/oz. This was applied as real and flat forward in the financial model.</td></tr>
<tr><td>Market assessment</td><td>DESPL supplied a gold price of US$1,250/oz.
DESPL has completed comprehensive market studies, including likely refiners.
Gold is freely traded and the price is set by the LME.</td></tr>
<tr><td>Economic</td><td>The discount rate in the DESPL financial model was set at 5%.
A financial sensitivity study was undertaken to evaluate capital expenditure, operating costs and gold price. The project was found to be most sensitive to changes in gold price.
The key performance indicators (KPIs) after taxation from the DESPL model are summarised below:
<table>
<tr><th>KPI (after taxation</th><th>Units</th><th>Value</th></tr>
<tr><td>All in cash cost (including royalty)</td><td>US$/oz produced</td><td>423</td></tr>
<tr><td>IRR ungeared</td><td>%</td><td>29.6</td></tr>
<tr><td>NPV (at 5%)</td><td>US$ M</td><td>61.4</td></tr>
<tr><td>Net cashflow</td><td>US$ M</td><td>93.1</td></tr>
<tr><td>Initial capital cost[a]</td><td>US$ M</td><td>46.6</td></tr>
</table>
[a] Excludes working capital</td></tr>
<tr><td>Social</td><td>A socio-economic study was prepared by DESPL. The commentary provides a summary of the socio-economic characteristics of the area at a household level. DESPL has a full-time Community Relations Officer engaged in maintaining open communications with the local communities. DESPL has advised that there are no community or social encumbrances that could obstruct the provision of a MLA from the Indian government.</td></tr>
<tr><td>Classification</td><td>The Ore Reserve is classified as Proved and Probable in accordance with the JORC Code 2012, corresponding to the Mineral Resource classification of Measured and Indicated for ore sources from in situ inside the mine pit design.
No Inferred Resources is included in the Ore Reserve estimate.
The following items will continue to be studied in execution:
• The sulphidic sulphur grades that were used for the sulphide recovery
• Binding mining cost contracts (subject to waste sale)
• A flowsheet for eventual production reconciliation
• Assessment of water usage with revision to pit water discharge options
• Further characterisation studies of the rock mass.
Any changes to the above items are not considered to reduce the cashflow on the project to a point where the project would not return a profit.</td></tr>
<tr><td>Audits or reviews</td><td>Snowden has completed an internal peer review of the Ore Reserve estimate.</td></tr>
<tr><td>Relative accuracy/ confidence</td><td>It is Snowden’s opinion that the Feasibility Study is of an accuracy of 10% to 15% for the capital and operating costs estimated for the Project. This is consistent with the accuracy recommendations made by The Australasian Institute of Mining and Metallurgy for a feasibility study.</td></tr>
</table>

6.11 Ganajur deposits Ore Reserve reporting

Proved and Probable Ore Reserves were reported using the 2012 edition of the "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves" (JORC Code 2012).

The Ore Reserve estimate for the Ganajur deposits, as at the end of April 2017, is provided in Table 6.23. Note that tonnes and ounces have been rounded and this may have resulted in minor discrepancies.

Table 6.23 Ganajur in-situ Ore Reserve estimate as at April 2017

Classification	Weathering	Tonnes (kt)	Au (g/t)
Proved	Oxide	568	2.76
	Sulphide	1567	3.94
Total Proved		**2135**	**3.63**
Probable	Oxide	122	1.78
	Sulphide	250	2.08
Total Probable		**372**	**1.98**
TOTAL		**2,506**	**3.38**

6.12 Conclusions and recommendations

The conclusions and recommendations are made for ongoing work, with items that may need further definition and resolution during early mine production.

6.12.1 Drilling and blasting

For final pit walls, the geotechnical study recommended that perimeter blasting on walls and batters will be required. This is to preserve the integrity of the final pit walls and promote wall stability. There may be additions to the drilling and blasting designs that were recommended on 10 March 2017 by Deepak Vidyarthi. Designs to augment the standard ore and waste drill and blast design patterns may consider perimeter blasting in the form of peripheral drill and blast designs that are:

- Modified production blasts, that reduce the local powder factor close to walls, thereby reducing damage to the fabric of the wall
- Pre-split blasts, where a tensile failure in the rock mass causes a crack on the mining limit and this crack is used to vent explosive energy away from the wall.

Both of these peripheral drill and blast methods will need to be trialled when mining reaches the competent rock mass, 30 m below the surface. Based on the effectiveness of standard ore and waste drill and blast outcomes compared to the peripheral drill and blast outcomes, a decision may be made to implement peripheral drill and blast designs.

It is not likely that the peripheral drill and blast modified designs will significantly increase the mining costs.

6.12.2 Trafficability studies

Trafficability was not assessed. Snowden recommends that this be included in future studies so that truck bearing capacity and sheeting requirements can be determined.

6.12.3 Pit dewatering studies

All future dewatering designs need to be validated by a hydrologist. The outcome of water inflows into the pit should be provided to the mining contractor for accurate costing. The current design has the water that collects in the mine pit from rainfall being pumped out from the pits, using centrifugal pumps. This water would be used for dust suppression or return to the mine raw water dam and not be discharged without water quality testing. If execution studies result in pit water being utilised in the plant, then this will change the contractor pricing estimates.

6.12.4 Binding mining contract and waste pricing

Currently the mining contractors have estimated non-binding contract mining costs. The scope of mining works including dewatering should be finalised and a single mining contractor should be engaged in a legally binding mining contract between DESPL and the chosen mining contractor so that mining costs can be finalised. The mining contractor will also need to commit to a final purchase price of the waste rock in their pricing.

6.12.5 Equipment utilisation

The production schedule should be discussed with the mining contractors to see if the production schedule is achieved, including a spike in explosive requirements in quarter 3.

7 PROCESS PLANT AND DESCRIPTION

7.1 Introduction

7.1.1 Process plant

The process plant will treat a nominal 300,000 tonnes per annum (t/a) of gold-bearing ore with a crushing availability of 70% on a single shift and an overall plant availability of 91.3%.

Ore will be campaigned based on lithology as either sulphide or oxide material and will be scheduled by the mining engineers.

Figure 7.1 shows the simplified flowsheet for the treatment process and the variations between the sulphide and oxide ore streams.

Both sulphide and oxide ores will be processed through the same crushing, milling, gravity and elution circuits; however, there are some processing differences around flotation, carbon-in-leach (CIL) and tailings.

7.1.2 Sulphide ore processing

The ore will be delivered to the ROM pad using mine haulage trucks and stockpiled into fingers. A FEL will pull ore from the various fingers as required. The jaw crusher will be fed from the FEL via a 30 m^3 ROM bin.

The jaw crusher will crush the rock to a P_{80} of 100 mm. Further size reduction will be accomplished using two cone crushers and a product screen. The crushing circuit selected is a modular design to simplify the installation process and reduce construction costs.

The crushed ore will feed the grinding circuit which is a conventional ball milling circuit in closed circuit with cyclones. The cyclone overflow will have a P_{80} value of 75 µm suitable for gravity flotation. Cyclone underflow will recycle back to the ball mill with a portion split off to feed the gravity concentration circuit.

The gravity concentration circuit consisting of a centrifugal concentrator and intensive leach reactor will treat a percentage of the cyclone underflow. The pregnant solution from the intensive leach reactor will be transferred to the gold room for electrowinning.

The cyclone overflow, via a trash screen will be fed to the rougher flotation circuit where the concentrate will feed the regrind circuit. The rougher flotation tailings will flow through a bank of scavenger flotation cells with the scavenger flotation concentrate recycled back to the ball mill feed and the tailings discharged into the tailings thickener.

The flotation concentrate will be reground in an ultrafine grinding (UFG) mill to a P_{80} of 10 µm which will operate in closed circuit with cyclones. Lead nitrate will be added to the mill feed to aid in gold dissolution in the leach circuit. The cyclone overflow will feed the flotation concentrate thickener while the cyclone underflow will be returned to the UFG mill.

The reground flotation concentrate will be thickened to 50% solids and agitated in a tank prior to six stages of leaching/adsorption in the CIL circuit. Loaded carbon will be removed periodically and replaced with regenerated and/or fresh carbon. The loaded carbon will be transferred to the elution circuit for gold recovery and doré production.

The tailings from the CIL circuit will feed the cyanide detox circuit which consists of two agitated tanks. Sodium metabisulphite (SMBS) and oxygen will be added in the first tank to convert the cyanide (CN^-) species to cyanate (CNO^-) which is relatively stable and with time will hydrolyse to ammonium and carbonate. Hydrated lime slurry will be added for pH control and the addition of copper sulphate will provide a catalyst for the reaction.

Discharge from the cyanide detox tank will flow into a single agitated tank for arsenic precipitation. Sulphuric acid (H_2SO_4) will be added to decrease the pH to approximately pH 6 and ferric sulphate ($Fe_2(SO_4)_3$ will be added to aid in the precipitation of arsenic from solution.

The arsenic precipitation circuit discharge will report to the tailings thickener where it will combine with the rougher flotation tailings and be thickened to 55% solids before being pumped to the tailings storage facility (TSF). Water will be decanted from the TSF for reuse in the process via the process water tank.

7.1.3 Oxide ore processing

Oxide ore will be processed through the same crushing, grinding and gravity circuit as the sulphide ore.

Cyclone overflow from the milling circuit will bypass the flotation circuit and instead will feed into the tailings thickener which will be used as a pre-leach thickener when processing oxide ore.

Thickened oxide ore will be leached in three large tanks prior to flowing into the six carbon adsorption tanks which are also used for processing the sulphide ore. The three larger leach tanks are required to maintain the residence time in the leach/adsorption circuit for the higher flowrate of the oxide ore stream.

Loaded carbon from the adsorption tanks are processed through the same elution circuit and gold room as the sulphide ore.

The leached tailings from the last adsorption tank flow through the carbon safety screen and are pumped into the tailings underflow hopper and out to the TSF.

7.2 Process plant

This section describes in detail the processing facilities designed for the Ganajur Gold Project. The supporting documents for the design include:

- Process design criteria (Appendix 7A)
- Mass balance (Appendix 7B)
- Process flow diagrams (Appendix 7C)
- Mechanical equipment list (Appendix 7D)
- Process plant layouts (Appendix 7E)
- Instrument list (Appendix 7F).

Ore will be campaigned based on lithology as either sulphide or oxide material and will be scheduled by the mining engineers.

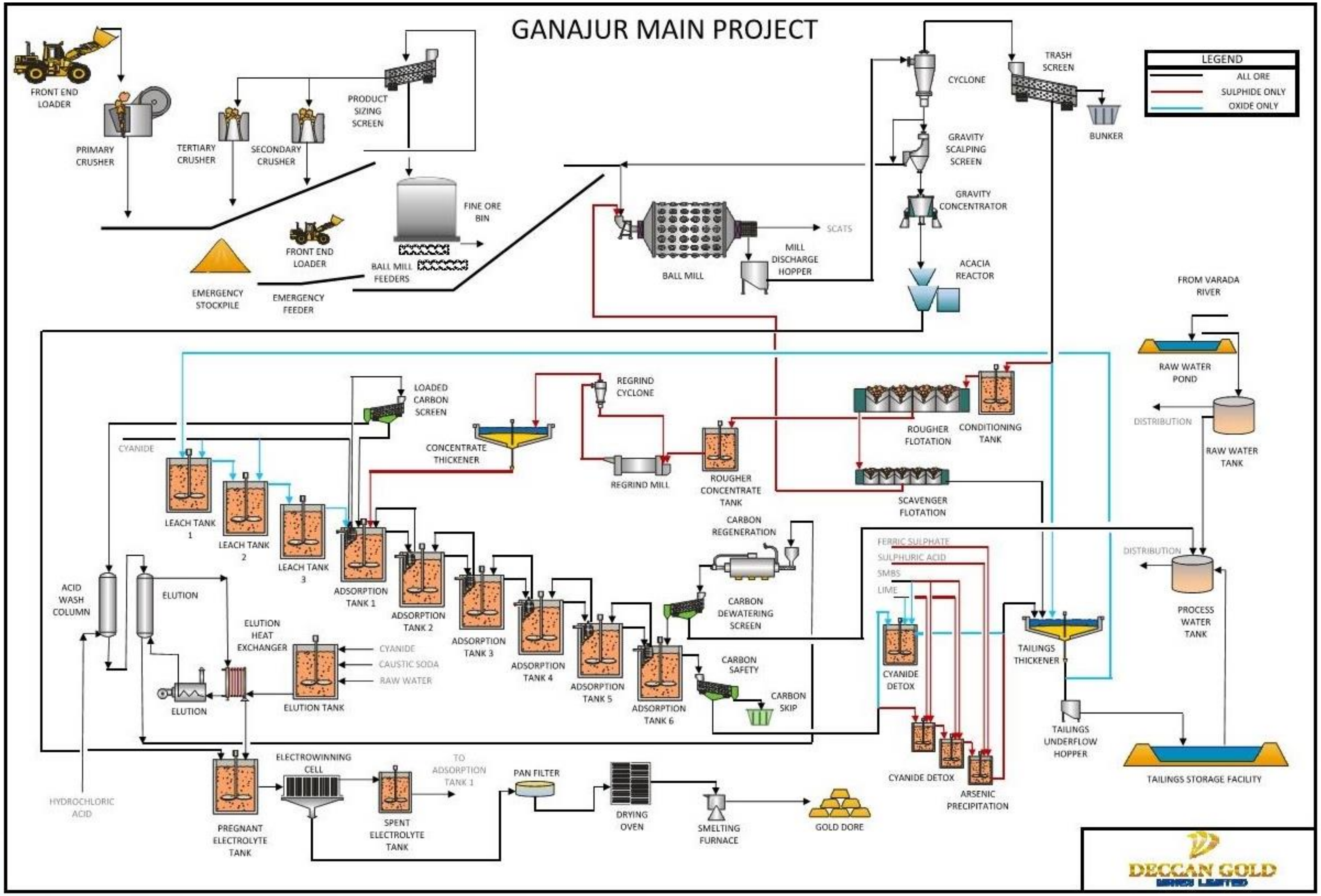
Figure 7.1 shows the simplified flowsheet for the treatment process and the variations between the sulphide and oxide ore streams.

The Ganajur Gold Project consists of a 300,000 t/a processing facility to treat both sulphide and oxide ores on a campaign basis utilising the following unit operations:

- Three-stage crushing and screening circuit producing a final crushed product with a P_{80} of 11 mm
- Ball milling and classification circuit, with a final grind size of P_{80} of 75 µm
- Gravity recovery circuit, using a centrifugal concentrator and intensive leach reactor to produce a pregnant solution for electrowinning in the gold room
- Four 8 m^3 rougher flotation cells to produce a flotation concentrate for leaching and five 8 m^3 scavenger flotation cells to capture any remaining gold, with the scavenger concentrate recycled back into the ball mill feed for processing of the sulphide ore only

- UFG mill and classification circuit to produce a concentrate with a final grind size of P_{80} 10 µm for leaching of the sulphide ore only
- Three 280 m^3 leach tanks with a residence time of 17 hours for oxide ore only
- CIL circuit, consisting of six 60 m^3 leach/adsorption tanks, with 48 hours' total residence time (sulphide ore) and seven hours (oxide ore)
- One-tonne Anglo-American Research Laboratories (AARL) stripping circuit, with separate acid wash and elution columns designed for one elution cycle per day
- Gold room for electrowinning and smelting of the elution circuit and intensive leach reactor pregnant solutions and a furnace for smelting twice per week.

Figure 7.1 Simplified process flowsheet



7.2.1 Crushing, screening and storage

The crushing circuit will reduce the ROM ore size from a nominal top size of 600 mm to a product size P_{80} of 11 mm in preparation for the grinding process.

A modular crushing plant was selected for this project to reduce capital costs, and to allow for flexibility with regards to future expansions. The modular design also simplifies on-site construction as each module is largely preassembled before shipping in sea containers from the manufacturer's factory. The compact construction allows for a smaller site footprint while maintaining access to the equipment for maintenance.

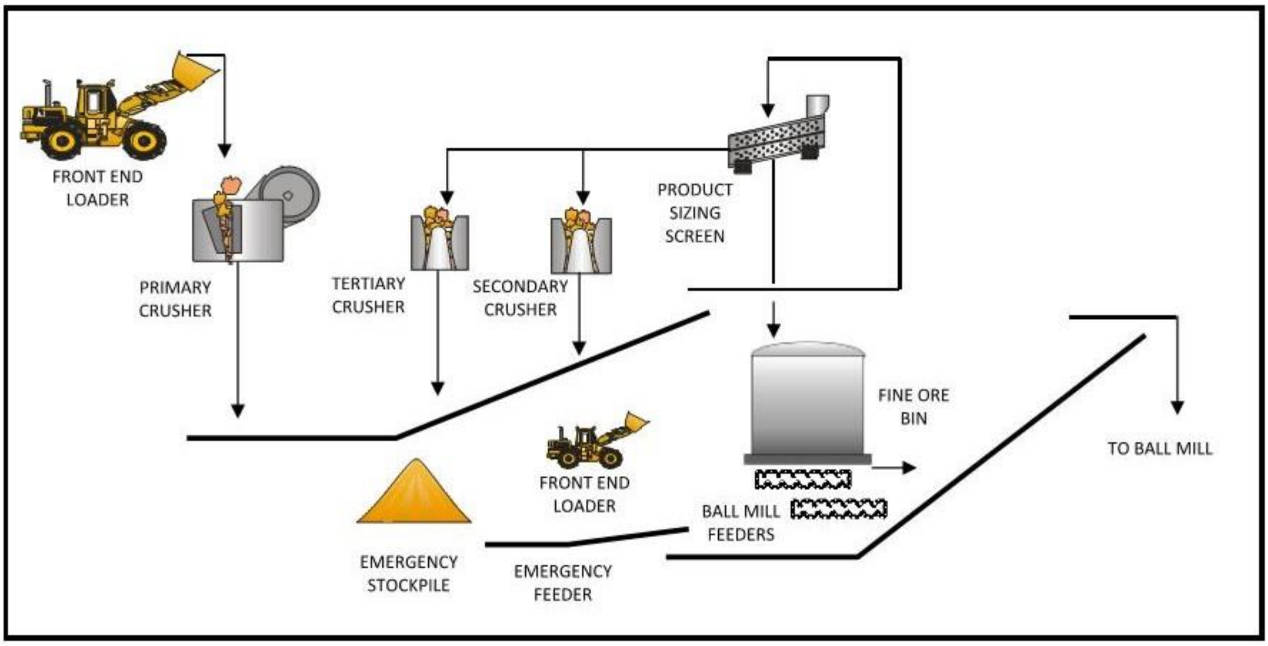
The modular plant specified includes a jaw crusher module, two cone crusher modules for secondary and tertiary crushing, a double deck scalping screen module and internal conveyors to complete the circuit.

The crushing and screening circuit is designed to process a throughput of 300,000 t/a at an overall utilisation of 70%, crushing on a single 12-hour shift only to minimise noise/dust impacts to the surrounding community overnight.

Figure 7.2 outlines the crushing, screening and storage process which is common to both sulphide and oxide ore processing.

Process flow diagram 7056-PF-310-001 in Appendix 7C outlines the crushing circuit and should be referenced in conjunction with reading this section.

Figure 7.2 Crushing, screening and storage



Process description

ROM fresh ore (-600 mm) will be delivered to the ROM pad and placed in "fingers" to accommodate blending of the various ore types and gold grades prior to feeding into the process plant.

The ore will be fed by FEL from the individual stockpile fingers to a ROM bin equipped with a stationary grizzly. The grizzly apertures will be 600 mm in size to protect the jaw crusher from oversize material. Oversize material will be removed from the static grizzly by the ROM loader and then stockpiled before being broken by a mobile rock breaker.

The capacity of the ROM bin will be 30 m^3 which is sufficient for maintaining a steady supply to the jaw crusher.

Ore will be withdrawn at a controlled rate from the ROM bin by a vibrating grizzly feeder, with a grizzly aperture of 89 mm. Oversize will be fed into the primary jaw crusher, where it will be crushed. The expected closed side setting (CSS) of the jaw crusher is 100 mm.

Undersize from the grizzly vibrating feeder and the primary crushed ore will be discharged onto the crusher discharge conveyor and transferred to the screen feed conveyor. Crushed ore from the secondary and tertiary crushers will also discharge onto the crusher discharge conveyor and be transferred onto the screen feed conveyor.

The screen feed conveyor will discharge onto a double deck vibrating product sizing screen, fitted with 32 mm aperture top deck and a 16 mm aperture bottom deck. Screen undersize will have a P_{80} of 11 mm.

Screen oversize from the top deck will be conveyed to the secondary crusher feed bin while the screen bottom deck oversize will be conveyed to the tertiary crusher feed bin. Both crushers will have material withdrawn from the respective bins at a controlled rate by a vibrating feeder to choke feed the crusher. Each cone crusher module includes a 15 m^3 feed bin, the cone crusher and vibrating feeder along with walkways, access ladders, steel supports and other ancillary items.

Screen bottom deck undersize will be conveyed to the fine ore bin which will have side wall slots to allow the stockpiling by FEL of excess crushed material. The fine ore bin is enclosed to minimise dust emissions and has a capacity of 24 hours of ore storage. The fine ore bin will feed two belt feeders which will reclaim ore at a controlled rate and will discharge onto the fixed speed ball mill feed conveyor.

Stockpiled crushed material will be reclaimed by a FEL which will load the material onto an emergency feeder where it will transfer back onto the ball mill feed conveyor.

A self-cleaning type tramp metal magnet will be installed at crusher discharge conveyor to remove any tramp metal. Fixed tramp metal magnets will be installed on both the crusher feed bin conveyors to protect the secondary and tertiary crushers.

Baghouse dust collectors with ducting at chutes and transfers will be installed in the crushing area to collect any dust and return it to the conveyor belt.

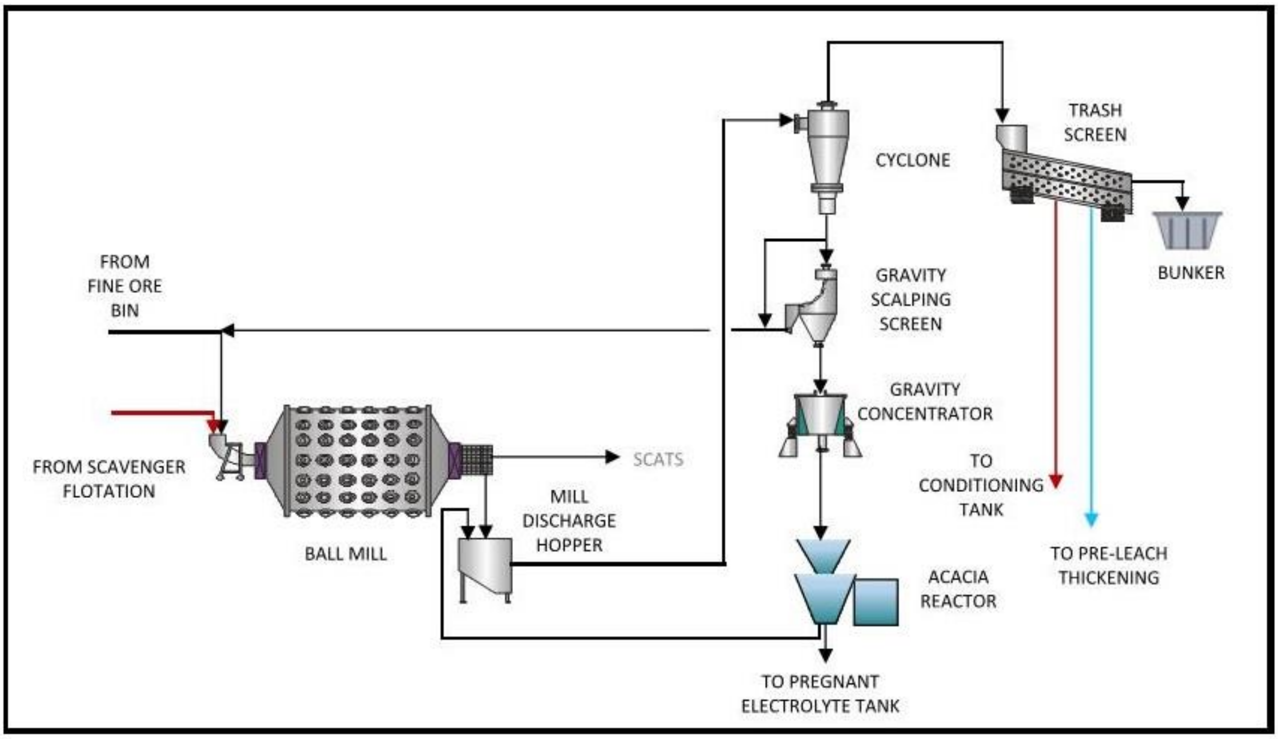
7.2.2 Grinding and classification

The grinding circuit will reduce the crushed ore size to a final product size with P_{80} of 75 µm suitable for gold recovery by gravity concentration and flotation. The grinding circuit consists of a ball mill, cyclone cluster and associated conveyors and pumps.

Figure 7.3 outlines the crushing, screening and storage process which is the same for processing of both sulphide and oxide ore.

Process flow diagram 7056-PF-320-002 in Appendix 7C outlines the grinding circuit and should be referenced in conjunction with reading this section.

Figure 7.3 Grinding, classification and gravity circuit



Process description

Ore from the fine ore bin will be drawn from the bin via two belt feeders and transported to the ball mill on the ball mill feed conveyor. The mill feed belt will be equipped with a weightometer which ensures a consistent mill feed rate is maintained.

In addition to the crushed ore feed, cyclone underflow is also recirculated back into the mill feed and process water will be added to maintain the mill discharge slurry density at 70% solids.

A single stage closed circuit ball mill has been selected for the grinding process. The ball mill has a diameter of 3.4 m, and effective grinding length of 6.0 m fitted with a 1,000 kW motor and drive. The grinding circuit availability is assumed to be 91.3% based on client supplied data which is typically industry standard for design.

The mill discharge slurry will pass through a 12 mm aperture trommel into the mill discharge pump hopper. Trommel oversize will be collected in a skip for disposal or re-treatment. The mill discharge slurry will be pumped to a cyclone cluster for classification.

The cyclone cluster will operate with a design circulating load of 250% and will consists of one operating 500 mm cyclone and one standby.

The cyclone overflow, or final product, will have a grind size P_{80} of 75 µm. The cyclone underflow will be split in a distribution box with 50% of the stream being returned to the mill feed chute and the other 50% becoming the feed to the gravity concentration circuit.

The cyclone overflow will be discharged to a trash screen with 800 µm aperture slotted panels. This vibrating screen will remove any misplaced oversize material as well as bits of plastic, organic material and other trash. The screen oversize will be deposited in a bin for periodic collection and disposal. The trash screen underflow will discharge into the flotation conditioning tank when processing sulphide ores or the tailings thickener for pre-leach thickening when processing oxide ore.

7.2.3 Gravity circuit

The gravity concentration circuit will produce a concentrate containing coarse gold from the grinding circuit that is intensively cyanide leached to produce a pregnant gold solution that will be refined in the gold room.

Figure 7.3 outlines the gravity concentration circuit which is common to both sulphide and oxide ore processing.

Process flow diagram 7056-PF-320-002 in Appendix 7C outlines the gravity circuit and should be referenced in conjunction with reading this section.

Process description

The cyclone underflow stream will flow through a splitter box with 50% of the flow used as feed for the gravity concentration circuit. The cyclone underflow split stream, or gravity feed will initially be passed over a vibrating screen in order to remove oversize and grit particles which are greater than 2.4 mm in size. The screen oversize material will be returned to the grinding circuit via the gravity oversize and tailings collection box.

The screen undersize will be the feed to a centrifugal gravity concentrator. The gravity concentrator will recover coarse particles of gold and high-density gold bearing sulphide mineral particles. Gravity tailings will be discharged into the oversize and tailings collection box and returned to the grinding circuit.

The gravity circuit will operate continuously with the gravity concentrator flushed once an hour to recover the concentrate collected in the unit. The concentrate will be discharged into a secured hopper which will feed the intensive leach reactor.

An intensive cyanide leach reactor will be used for processing the gravity-recovered gold. Intensive cyanidation of gravity concentrates has become the standard route for processing gravity recovered concentrates due to the higher (>90%) recoveries (compared to traditional tabling at 50% to 70%) and the reduced labour and security requirements associated with tabling.

The intensive gravity leach will occur once per day on the entire gravity concentrate collected over the previous 24 hours and contained within the secured feed hopper. There are six major process steps that will occur within the intensive leach reactor:

- Transfer of the gravity concentrate from the day storage hopper
- Prewashing the gravity concentrate to remove fines
- Mixing of the leach reagents
- Leaching
- Recovery and transfer of the pregnant solution and washing of the residue
- Discharging of the leach residue back into the milling circuit.

Leach solution from the intensive leach reactor is pumped to the pregnant solution tank in the goldroom where the gold will be recovered via the electrowinning circuit.

7.2.4 Flotation and regrind

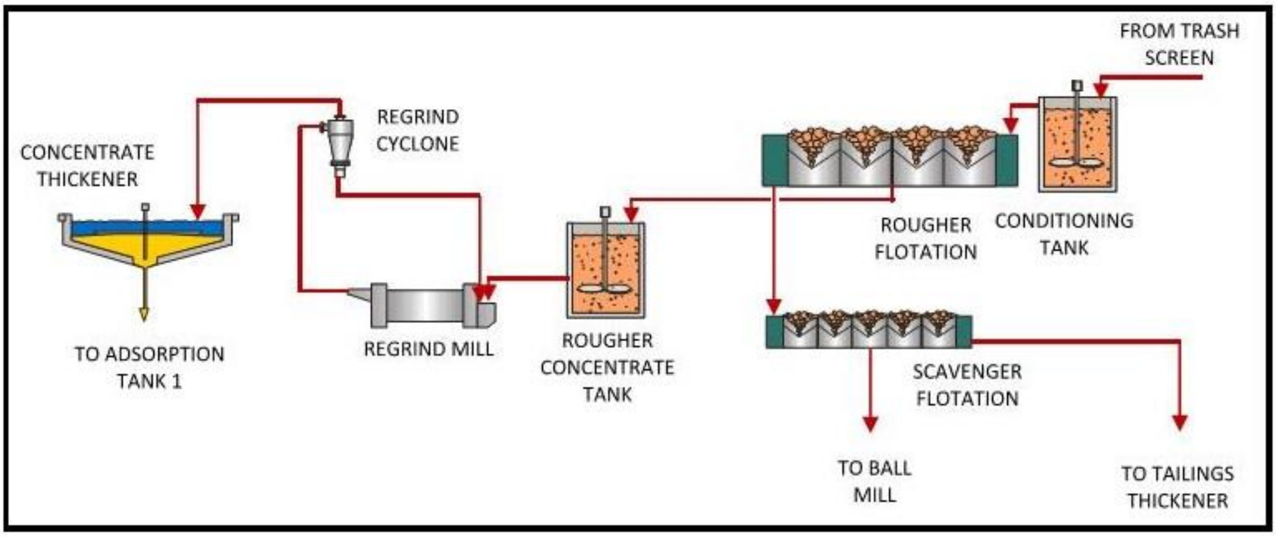
The milled ore will be subjected to two stages of flotation. A bank of rougher cells will produce a gold concentrate that will feed the leach circuit and a bank of scavenger cells that will capture any remaining gold and recycle it back into the ball mill feed.

Conventional forced air flotation cells will be used and consist of four rougher cells followed by five scavenger cells.

Figure 7.4 outlines the flotation circuit which is used only while processing sulphide ore.

Process flow diagram 7056-PF-330-003 in Appendix 7C outlines the flotation circuit and should be referenced in conjunction with reading this section.

Figure 7.4 Flotation



Process description

The flotation circuit will be utilised when processing sulphide ore only.

The feed to the flotation circuit will be the cyclone overflow from the milling circuit, after passing through the trash screen. The slurry will be conditioned in the conditioning tank that will be equipped with an agitator and has been sized for a retention time, or conditioning period of ten minutes.

Hydrated lime slurry will be added to the conditioning tank for pH adjustment of the slurry, if required. Potassium amyl xanthate (PAX) will be used as the collector reagent together with supplementary amounts of the frother reagent, Interfroth 50, both will also be added to the conditioning tank.

The conditioned slurry will overflow into the rougher flotation bank of cells. The rougher bank consists of four cells each that are 8 m^3 in volume to provide the required retention time in the rougher circuit. Forced air injection will facilitate the flotation process with a mass pull of approximately 13.4% assumed based on existing testwork.

The rougher concentrate will be discharged into the regrind mill discharge hopper and will be combined with the regrind mill discharge, diluted with process water to 20% solids and will feed the regrind cyclone.

The regrind cyclone manifold will have two operating 250 mm cyclones, one standby and one blanked off for future use to separate the reground flotation concentrate into a cyclone overflow product with a particle size of P_{80} of 10 µm at approximately 5% solids by weight. The cyclone overflow will feed an 8 m diameter concentrate thickener to produce an underflow of 50% solids to feed the adsorption circuit. The thickener overflow will be returned to the process water tank.

The regrind cyclone underflow at approximately 70% solids will be diluted in the regrind mill feed tank to 45% solids to feed the UFG which is an IsaMill™. The UFG is required to liberate any encapsulated gold prior to the leaching circuit and will produce a final concentrate with a required grind size P_{80} of 10 µm. Lead nitrate will be added to the regrind mill feed tank to reduce gold passivation in the leach circuit.

The rougher flotation tailings will feed the scavenger flotation cells to recover any remaining gold. The concentrate from the scavenger cells will be pumped back into the ball mill feed.

The tailings from the scavenger circuit will be fed to the tailings thickener.

7.2.5 Leaching and adsorption

The adsorption circuit will be utilised for processing both sulphide and oxide ore, however, three additional leach tanks will be required to maintain the overall residence time with the increased flow of the oxide ore.

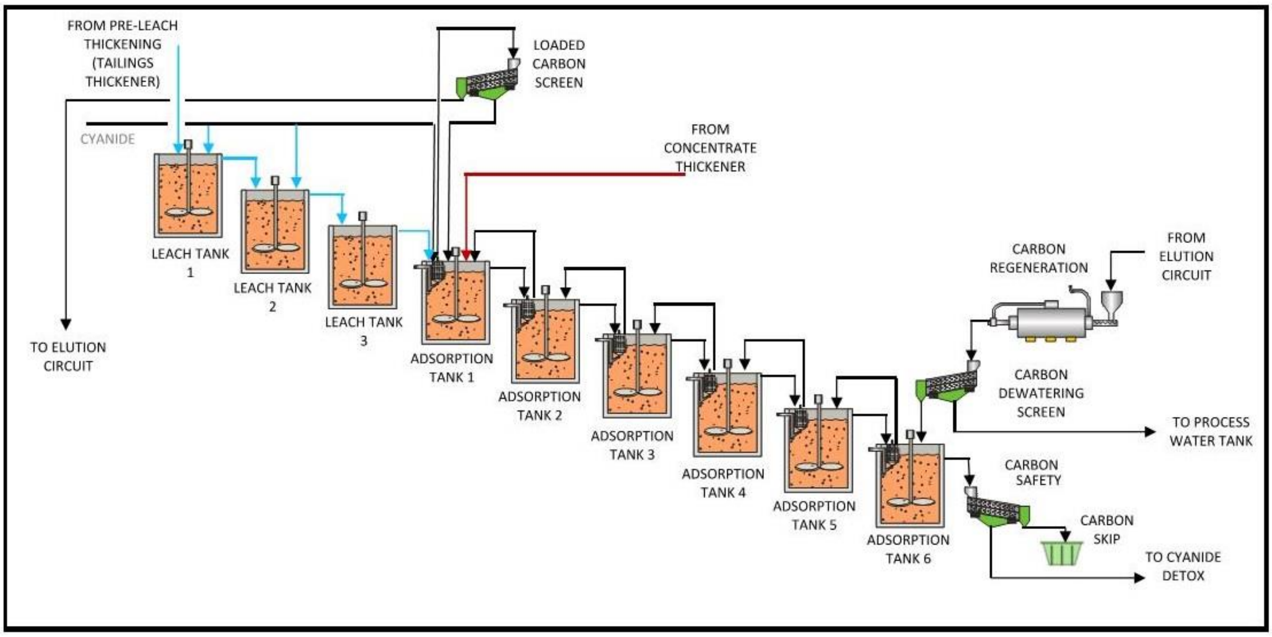
Underflow from the concentrate thickener (sulphide ore) and pre-leached thickened slurry from the tailings thickener (oxide ore) will be leached/adsorbed onto activated carbon in this circuit.

Loaded carbon will be removed periodically for elution to recover the collected gold, before being regenerated, with the regenerated carbon returned to the last CIL tank.

Figure 7.5 outlines the leaching and adsorption circuit, the first three leach tanks of which are only used when processing oxide ore.

Process flow diagram 7056-PF-340-004 in Appendix 7C outlines the leaching and adsorption circuit and should be referenced in conjunction with reading this section.

Figure 7.5 Leaching and adsorption



Process description

Thickened slurry from either the concentrate thickener (sulphide ore) or the tailings thickener (oxide ore) will be fed into the leach/adsorption circuit. The sulphide ore will enter into the first of six 60 m^3 agitated adsorption tanks all connected in series by a launder and by-pass system, whereas the oxide ore will feed into the first of three 280 m^3 leach tanks followed by the six adsorption tanks.

For sulphide ore processing, the six adsorption tanks are identical in size having a combined retention time of 48 hours, in excess of the 24-hour leach time as indicated by testwork. Larger tanks were installed to accommodate the required tank internals. The 60 m^3 adsorption tanks will be suitable for a plant expansion of up to 600,000 t/a.

For oxide ore processing, the three leach tanks have a residence time of 17 hours with the six adsorption tanks making up the additional 7 hours required for an overall residence time of 24 hours in the circuit.

To avoid short circuiting through the tanks, the slurry will enter opposite to the submerged outflow position. Slurry leaves the tank by an overflow launder. Each of the tanks will also have launders and isolation gates arranged such that a tank can be bypassed should maintenance be required.

Each adsorption tank will be equipped with a mechanical wiper inter-stage screen and a recessed impellor slurry pump. The inter-stage screen will allow the carbon to be retained in the respective CIL tank while permitting the pulp to flow through the screen to the next CIL tank in the circuit. Air lifts are used to transfer carbon forward within the adsorption circuit. A recessed impellor slurry pump will be used to remove loaded carbon from adsorption tank 1 and pump to the loaded carbon screen.

Barren (and regenerated) activated carbon is added to the last adsorption tank. Carbon advances counter-current to the pulp flow. As the pulp flows downstream through the adsorption tanks it contacts carbon, which is progressively lower in soluble gold, enabling adsorption of gold to near-completion. Conversely, as the carbon advances upstream, it contacts slurry containing increasingly higher values of gold in solution enabling a higher loading of gold on the carbon.

Loaded carbon will be pumped via the recessed impeller pump from the first CIL tank to the loaded carbon screen. The loaded carbon screen will be a vibrating screen equipped with spray water nozzles to wash slurry off the loaded carbon. The loaded carbon will then be discharged directly from the screen overflow into the elution column. The screen underflow will contain slurry and wash water and will return to the second adsorption tank.

Pulp discharged from the last adsorption tank will flow to the carbon safety screen. Any carbon which reports to the carbon safety screen will be collected in a bin and returned to the CIL circuit. The carbon safety screen undersize will be collected in the leach tails hopper and pumped to the cyanide detox circuit.

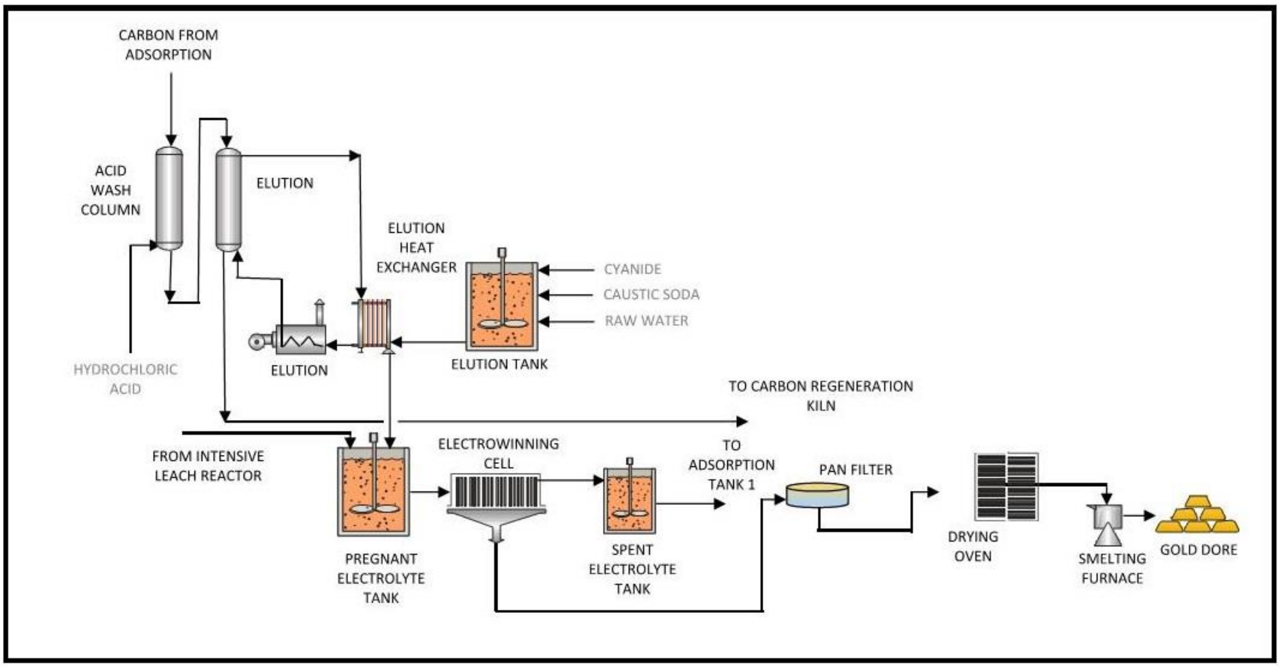
7.2.6 Elution

Loaded carbon from the CIL circuit is transferred to the elution circuit to start each elution cycle (once per day). The carbon is acid washed prior to desorbing the gold back into solution, electrowinning and refining into gold doré.

Figure 7.6 outlines the elution circuit which is common to both sulphide and oxide ore processing.

Process flow diagram 7056-PF-350-005 in Appendix 7C outlines the elution and carbon regeneration circuit and should be referenced in conjunction with reading this section.

Figure 7.6 Elution and gold room



Process description

Loaded carbon from the screen underflow is directly transferred into the acid wash column where it is held until the beginning of an elution cycle, a batch process that occurs once per day, six days a week. The elution circuit is an AARL circuit designed to strip 1.0 t of carbon per day.

Stripping of the loaded carbon can be broken down into six steps: acid wash, water rinse, carbon transfer, pre-soak, elution, and cooling.

A dilute hydrochloric acid solution (3% v/v) is pumped into the acid wash column to dissolve inorganic foulants such as calcium carbonate, magnesium and sodium salts, fine ore minerals such as silica and fine iron particles from the loaded carbon. Organic foulants such as oils and fats are unaffected by the acid and are removed from the carbon by thermal reactivation after the elution cycle.

The acid wash cycle takes approximately 20 minutes and is followed by an acid soak which is a 30-minute period where the acid solution sits in the column allowing time to dissolve the inorganic foulants.

The water rinse follows the acid soak and involves flushing four bed volumes of water through the column. The water flushes away mineral impurities that have been freed by the acid wash. The wash also rinses the carbon of acid and raises (neutralises) the pH in the column. If this were not done, the remaining acid would react with the eluate, releasing large quantities of toxic cyanide gas. The acid and rinse water are sent to the tailings hopper for disposal to the TSF.

Following the water rinse, the carbon is transferred into the elution column for the remaining stripping stages starting with the pre-soak step.

The pre-soak step involves treating the carbon with a caustic-cyanide solution at a high temperature and pressure. Sodium cyanide is required for solubilising the gold cyanide complex, while caustic (sodium hydroxide) is added to maintain a high pH to minimise the evolution of toxic hydrogen cyanide gas. Upon the completion of this stage, the gold cyanide complex is still adsorped to the carbon (i.e. no gold is removed into solution during this stage) but the attraction of the carbon is ‘weakened’, allowing it to be easily desorbed in the following elution stages.

High temperatures are used and hence high pressure is maintained to prevent the solution from boiling. After it has been through the column, the pre-treatment solution reports to the pregnant solution tank.

After the soak period, fresh potable water is recycled through the carbon bed after passing through two sets of heat exchangers to bring the solution temperature to the required level. During this process, most of the gold on the carbon will desorb into this solution which will be cooled utilising the same heat exchangers before being collected in the pregnant solution tank prior to electrowinning.

The last stage of the elution process is run without the elution heater to drop the overall circuit temperature so the carbon can be transferred safely.

7.2.7 Carbon regeneration

The carbon regeneration circuit is used to reactivate the adsorption sites on the carbon surface to ensure the CIL circuit continues to operate effectively. A horizontal rotating kiln (kiln) is used to remove organic foulants from all barren carbon prior to reintroducing it into the CIL circuit.

Figure 7.5 outlines the carbon regeneration process and shows how it integrates with the adsorption circuit which is common to both sulphide and oxide ore processing.

Process flow diagram 7056-PF-350-005 in Appendix 7C shows the carbon regeneration circuit and should be referenced in conjunction with reading this section.

Process description

On completion of the elution cycle, the barren carbon is transferred to the kiln feed hopper where residual water and fine carbon particles can be drained from the carbon prior to being fed into the kiln.

Thermal reactivation is used to remove organic foulants by subjecting the carbon to temperatures in the order of 650°C to 750°C in a steam environment. The high temperature burns off some of the organic matter whilst reaction with the steam removes the rest. Steam also serves to keep the reactivation system oxygen free (to prevent the carbon burning) and is involved in the chemical formation of active sites within the carbon.

After reactivation, the carbon is quenched in water and screened to remove any fine carbon which is directed to the tailings hopper, with the sized carbon dropping directly into the last adsorption tank.

7.2.8 Gold room

The high-grade elution solution leaving the elution column is cooled and then pumped to the gold room where it is combined with the pregnant solution from the intensive leach reactor and the gold is recovered by the process of electrowinning. An electric current is passed through the pregnant solution causing gold to plate out onto stainless steel cathodes. Once the cathodes are laden with the required amount of gold they are cleaned using high pressure water to remove the gold, which is collected, filtered to remove excess moisture then dried and smelted to produce gold doré.

Figure 7.6 outlines the gold room circuit which is utilised for both sulphide and oxide ore processing.

Process flow diagram 7056-PF-350-005 in Appendix 7C outlines the gold room processes and should be referenced in conjunction with reading this section.

Process description

The high-grade elution solution leaving elution column is cooled and then pumped to the gold room, where it is stored in a pregnant solution tank. The pregnant solution from the intensive leach reactor is also collected in the pregnant solution tank with the two solutions processed together.

The combined pregnant solution is passed through two parallel electrowinning cells in closed circuit with the pregnant eluate tank. The gold will be deposited onto stainless steel cathodes as the solution passes through the cell, with a design electrowinning time of 12 hours per batch. When the solution gold tenor is low enough, electrowinning is stopped and the depleted electrolyte is collected in the spent electrolyte tank and slowly fed back into the leach circuit.

As necessary, the electrowinning cathodes are pressure washed with water to remove the gold sludge. Gold sludge removed from the cathodes and any slimes collected from the bottom of the electrowinning cells (approximately twice a week) are washed and filtered before being transferred to a drying oven. The oven is refractory lined and operates at approximately 650°C. Dried sludge will be mixed with flux (a mixture of borax, soda ash, silica and nitre), bagged and stored until required for smelting.

Twice a week the dried the electrowinning cell sludge will be smelted in an LPG fired tilting furnace. When the charge is molten, the furnace is tilted to first pour off the slag, then the gold into a series of 500-ounce graphite moulds set up in a cascade configuration. The gold due to its higher specific gravity, displaces the molten slag from the first moulds pushing the slag down the line of moulds.

Once cooled the bars are removed from the moulds, quenched in water and cleaned to remove any remaining slag, before being weighed, stamped with a unique identifier and stored pending transfer. The slag is checked for any residual gold and then smashed up and disposed back into the grinding mill, returning any remaining gold to the process.

The smelted product is unsuitable for direct sale and is therefore sent to a refinery for further processing to produce >99.6% pure gold bullion.

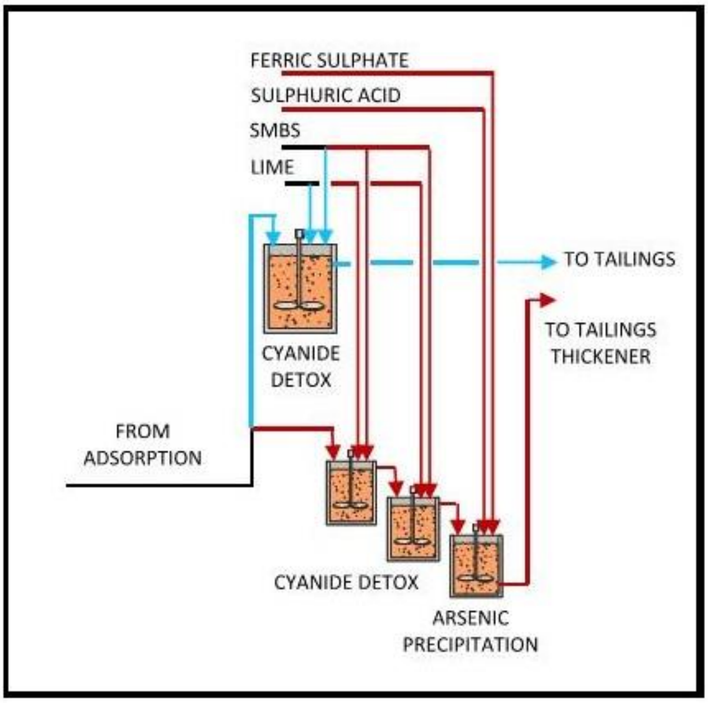
7.2.9 Cyanide detox

The cyanide detoxification circuit will remove excess cyanide from the leach tailings prior to being discharged into the TSF. Excess cyanide could cause issues with any wildlife in the area of the TSF and if returned to the process plant it has a negative impact on flotation. The cyanide will be oxidised through the addition of sodium metabisulphite ($Na_2S_2O_5$) and oxygen.

Figure 7.7 shows the differences between processing sulphide and oxide ores through the cyanide destruction circuit. The sulphide ore is processed through two tanks in series as the residual cyanide in the slurry is higher than that of the oxide ore. The oxide stream is processed in a single larger tank to achieve the one hour residence time to remove all residual cyanide.

Process flow diagram 7056-PF-370-006 in Appendix 7C outlines the cyanide destruct process and should be referenced in conjunction with reading this section.

Figure 7.7 Cyanide detox and arsenic precipitation



Process description

The screened tailings from the CIL circuit are pumped to the first of two cyanide detoxification tanks (sulphide ore) or a single tank (oxide ore) where residual free and weak acid dissoluble (WAD) cyanide species are oxidised by the addition of (sodium metabisulphite) Na_2SO_3 and oxygen in the presence of copper in solution as the catalyst.

The target cyanide concentration in the neutralised liquor is 10 ppm CN^- to minimise the cyanide discharged into the TSF and to negate any impact of cyanide on the flotation circuit, when water from the TSF is returned to the process water tank. SMBS is added to provide the sulphite ion SO_3^{2-} which combines with the liquid oxygen (O_2) in the cyanide detox tank utilising copper as the catalyst to react with the cyanide species via the following overall reaction to form cyanate (CNO^-).

$$CN^- + SO_3^{2-} + O_2 \rightarrow CNO^- + SO_4^{2-}$$

Hydrated lime slurry is also added to maintain a final slurry pH between pH 8.0 and 9.0, to neutralise the sulphuric acid (H_2SO_4) generated during the cyanide detoxification reaction and to oxidise any released metal species. The oxidation reduction potential of the slurry is monitored by an online probe with slurry pH also monitored.

The two cyanide detox tanks have a design residence time of 2 hours. Cyanide levels are monitored by an online cyanide analyser.

For sulphide ore processing, slurry from the second cyanide detox tank will overflow into the arsenic precipitation tank, while for oxide ore processing, the cyanide detox tank will overflow into the tailings underflow hopper.

7.2.10 Arsenic precipitation

The arsenic precipitation circuit will precipitate soluble arsenic from solution by utilising ferric sulphate. Sulphuric acid will be used to drop the pH to approximately 6 after the cyanide destruction circuit.

Figure 7.7 outlines the cyanide precipitation process which is utilised solely when processing sulphide ore.

Process flow diagram 7056-PF-370-006 in Appendix 7C outlines the arsenic precipitation circuit and should be referenced in conjunction with reading this section.

Process description

Arsenic bearing ore in the Ganajur Main Gold deposit results in soluble arsenic leaching in the CIL circuit. While there are no specific regulations for the discharge limits of arsenic in solution in TSFs, any discharge or seepage from the dam into the groundwater would cause catastrophic events to the surrounding community and farmlands. Arsenic discharge into the groundwater is regulated and therefore reduction in the soluble arsenic must occur prior to discharge into the TSF.

A target of 1 mg/L in solution at the discharge into the TSF has been specified utilising arsenic adsorption on ferrihydrate which is the Environmental Protection Agency's (USA) Best Demonstrated Available Technology (BDAT) for removing arsenic from process and wastewater and is the most widely used arsenic removal technology in the world.

Arsenic can be removed by two primary mechanisms: adsorption and coprecipitation depending on arsenic valence (III or V) and solution pH. The chemistry for adsorption and coprecipitation of arsenic with ferric ions is complex and not discussed here, however testwork on the Ganajur leached tailings indicate that the addition of ferric sulphate at a pH of 6 reduce the arsenic in solution to less than 0.5 mg/L.

A pH of 6 promotes iron oxidation and the ferric ion hydrolyses and precipitates spontaneously. The precipitation process is operated such that the ratio of Fe:As is >3. The entire process occurs in a single agitated tank with a retention time of 30 minutes.

The slurry discharge which includes the iron-arsenic precipitate will overflow the tank and be pumped into the tailings thickener where it will be combined with the flotation tailings (arsenic free).

7.2.11 Tailings

Pulp discharged from the arsenic precipitation circuit (sulphide ore) will be combined with flotation tailings in the tailings thickener, thickened and pumped from the tailings underflow hopper to the TSF. The discharge tailings stream will be approximately 55% solids.

Cyanide detox overflow (oxide ore) will feed the tailings discharge hopper as it has been thickened prior to leaching utilising the tailings thickener. The discharge slurry (oxide ore) will be at approximately 50% solids.

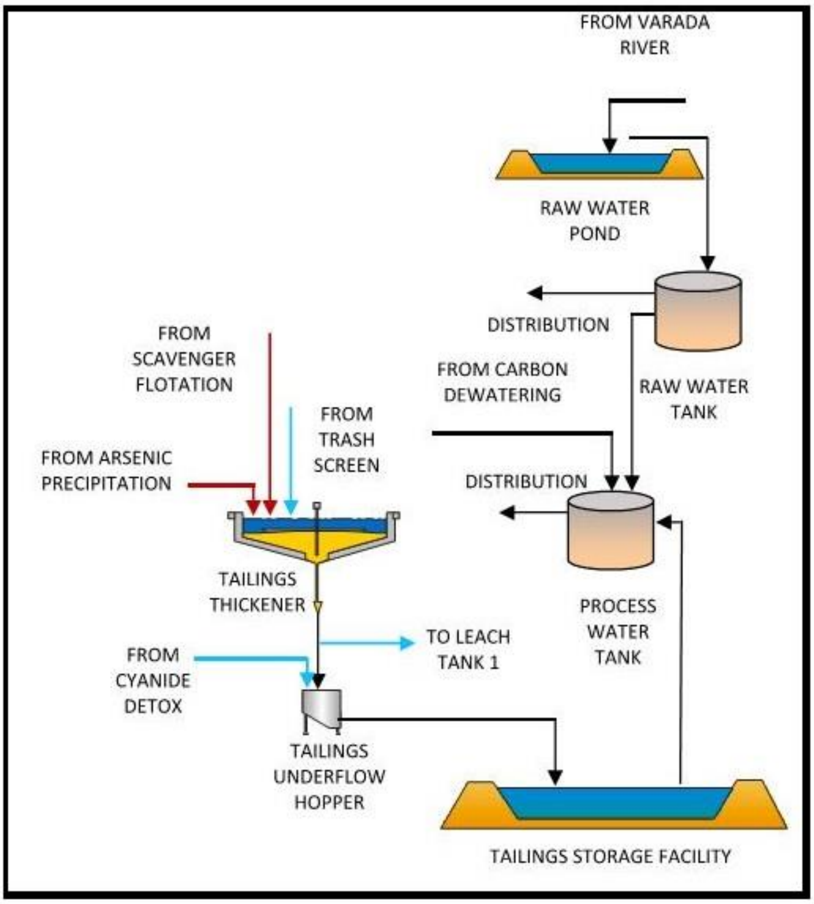
Acid wash liquor will be mixed into the tailings slurry in the discharge hopper on a batch basis from the elution circuit for discharge into the TSF. A final tailings solids concentration of 55% solids has been utilised to maximise water recovery.

Decant solution from the TSF is returned to the process water pond for reuse in the plant.

Figure 7.8 outlines the tailings discharge and reclaim water processes for both sulphide and oxide ore.

Process flow diagram 7053-PF-370-006 in Appendix 7C outlines the tailings process.

Figure 7.8 Tailings



7.2.12 Reagents

Chemical reagents will be added to the process stream to facilitate the flotation (sulphide ore processing only) and CIL processes. Additionally, they will be added to remove excess cyanide and soluble arsenic (sulphide processing only). The preparation of the various reagents will require the following equipment:

- Bulk handling system
- Mix and holding/buffer tanks
- Transfer and dosing pumps
- Flocculant preparation facility.

PAX (collector), Interfroth 50 (frother) and copper sulphate ($CuSO_4$) (activator) will be added to the flotation circuit to modify the mineral particle surfaces to enhance the floatability of the gold particles into the concentrate. PAX and $CuSO_4$ will be supplied as bulk solids and mixed on site while Interfroth 50 will come in an intermediate bulk container (IBC) and dosed at full strength.

Cyanide, lead nitrate and hydrated lime slurry will be added into the CIL circuit for leaching of the gold at a pH of 10. Lead nitrate will minimise the passivation of gold allowing for higher overall recoveries in the CIL circuit.

Hydrochloric acid, cyanide and caustic will be used in the gold room for washing and recovering gold adsorbed onto the carbon in the CIL circuit.

Fresh water will be used in the making up or the dilution of the various reagents that will be supplied in powder/solids form, or which require dilution prior to the addition to the slurry. These solutions will be added to the points of addition of the various circuits and streams using dosing pumps.

The solid reagents will generally be made up to a solution of 20% strength in a mix tank except for ferric sulphate (10% w/w) and flocculant (0.25% w/v with further dilution during dosing to 0.025% w/v). Where large consumption of reagents are used, the mixed solution will be transferred to a holding tank so that dosing and mixing can occur concurrently.

Reagents will a small flow will have a buffer tank which consists of a small pipe stand which will contain sufficient solution to use while another batch is being mixed. The dosing pump will feed off both the mixing and buffer tanks with valves to control where the solution originates.

Hydrated lime slurry will be slaked in an agitated tank then transferred to a storage tank before being pumped to the points of addition using a ring main system. The valves will be controlled by pH monitors, which will control the amount of lime slurry added.

Solid reagents will arrive on site in 1,000 kg bulk bags except for flocculant which will be shipped in 25 kg bags. Hydrochloric acid, sulphuric acid and Interfroth 50 will be shipped in 1000 L IBCs.

Oxygen used in the CIL and cyanide detox circuits will be delivered in liquid form and discharged directly from the bulk tanker into the liquid oxygen storage system which will be leased from the oxygen supply company. The benefit to leasing the equipment is that upfront capital costs are reduced and all maintenance and monitoring will be conducted by the oxygen supplier.

Flux reagents for use in the smelting process will be delivered in bags that are approximately 25 kg and will be stored and handled directly in the gold room.

Carbon for use in the CIL circuit will be delivered regularly in one-tonne bulk bags, while grinding balls used within the grinding mills will be delivered in 200 L drums.

LPG will be delivered by bulk tanker and discharged directly into the LPG storage tank.

Process flow diagrams 7056-PF-380-007 and 7056-PF-300-008 in Appendix 7C outline the main reagents used in the process.

7.2.13 Water

There will be two water supplies to meet the requirements of the process plant. A basic outline of the site water system is shown in Figure 7.9.

The raw water dam will be supplied from the Varada River during the monsoon season and contain sufficient raw water for the yearly operation. The pond has a live capacity of 300,000 m^3 with a buffer of two months' supply. While the monsoon season generally lasts for four months the supply pumps have been sized to fill the pond over a four-month period during day shift hours.

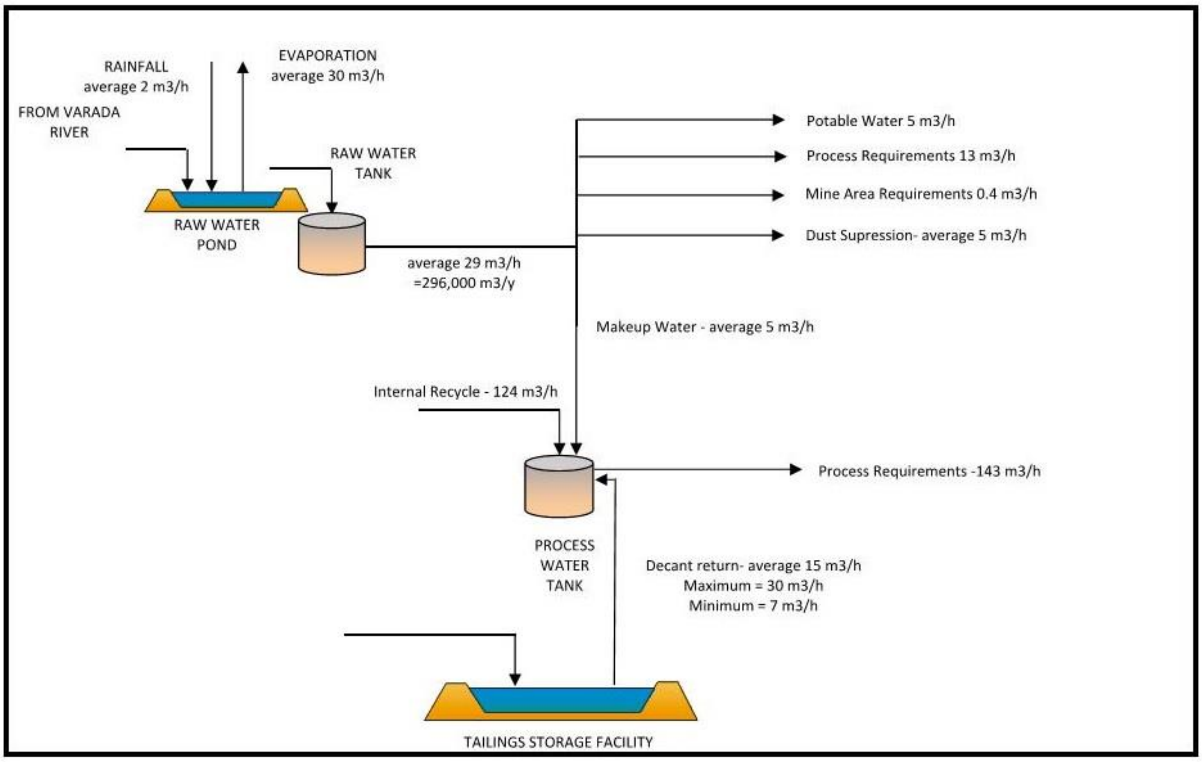
Raw water from the raw water dam will be pumped to the raw water tank at the process site. Raw water will be utilised throughout the site and will namely feed the potable water treatment plant (5 m^3/h), gravity, elution circuits and other process raw water requirements (13 m^3/h), mining area requirements (10 m^3/d) and dust suppression (average of 5 m^3/h). Additional process water make-up (5 m^3/h average) will also come from the raw water supply when there is insufficient decant return water from the TSF.

A firewater reserve will be contained in a dedicated portion of the raw water tank.

The process water tank will provide water as required in all areas of the process where raw water is not specifically required. The process water tank has been sized to contain 24 hours of process water and will receive recycle water from the thickener overflows and dewatering within the process plant, decant water from the TSF and any additional process water make up required from the raw water tank.

The potable water plant is a 5 m^3 reverse osmosis facility and is sized to sufficiently supply water to the elution circuit and for ablutions on site. The potable water storage tank has sufficient capacity for the short periods of time that a higher flow is required for the elution circuit, filling up slowly between elution batch operations.

Figure 7.9 Average site water balance



7.2.14 Air

Air supplied to the plant is via duty/standby 250 m^3/h air compressors fitted with dryers. The dried air is feed to a 2 m^3 pressure vessel (air receiver) before distribution throughout the site. The crushing area and grinding/flotation areas will each have a dedicated 1 m^3 air receiver.

Flotation air is supplied via 3000 m^3/h air blowers directly to the flotation circuit.

7.2.15 Process design criteria

The process design criteria (PDC) forms the basis for the mass balance, PFDs, and cost estimates. Table 7.1 provides a summary of key data from the 300,000 t/a base case PDC. The PDC is provided in Appendix 7A.

Table 7.1 Process design criteria key data

Description	Sulphide ore	Oxide ore
Feed tonnes	300,000 t/a	300,000 t/a
ROM ore P_{100}	600 mm	600 mm
Annual operating hours – crushing	3,066 h/a	3,066 h/a
Annual operating hours – grinding/CIL	8,000 h/a	8,000 h/a
Feed grade – design	3.70 g/t	2.02 g/t
Rougher flotation mass pull	13.4%	-
Scavenger flotation mass pull	2.5%	-
Recovery		
Gravity circuit	5%	10%
Overall	79.3%	91.2%
Adsorption residence time*	48 h	24 h (including leach)
Type of stripping system	AARL	AARL
Strips per day	1	1

7.2.16 Mass balance

The mass balance contains mass and volumetric flow rates, specific gravities, slurry solids density, solids and liquid concentrations, and gold. The mass balance reports only steady state streams, with intermittent streams not included.

The process design has been based on processing 100% sulphide ore and as such the mass balance reflects this.

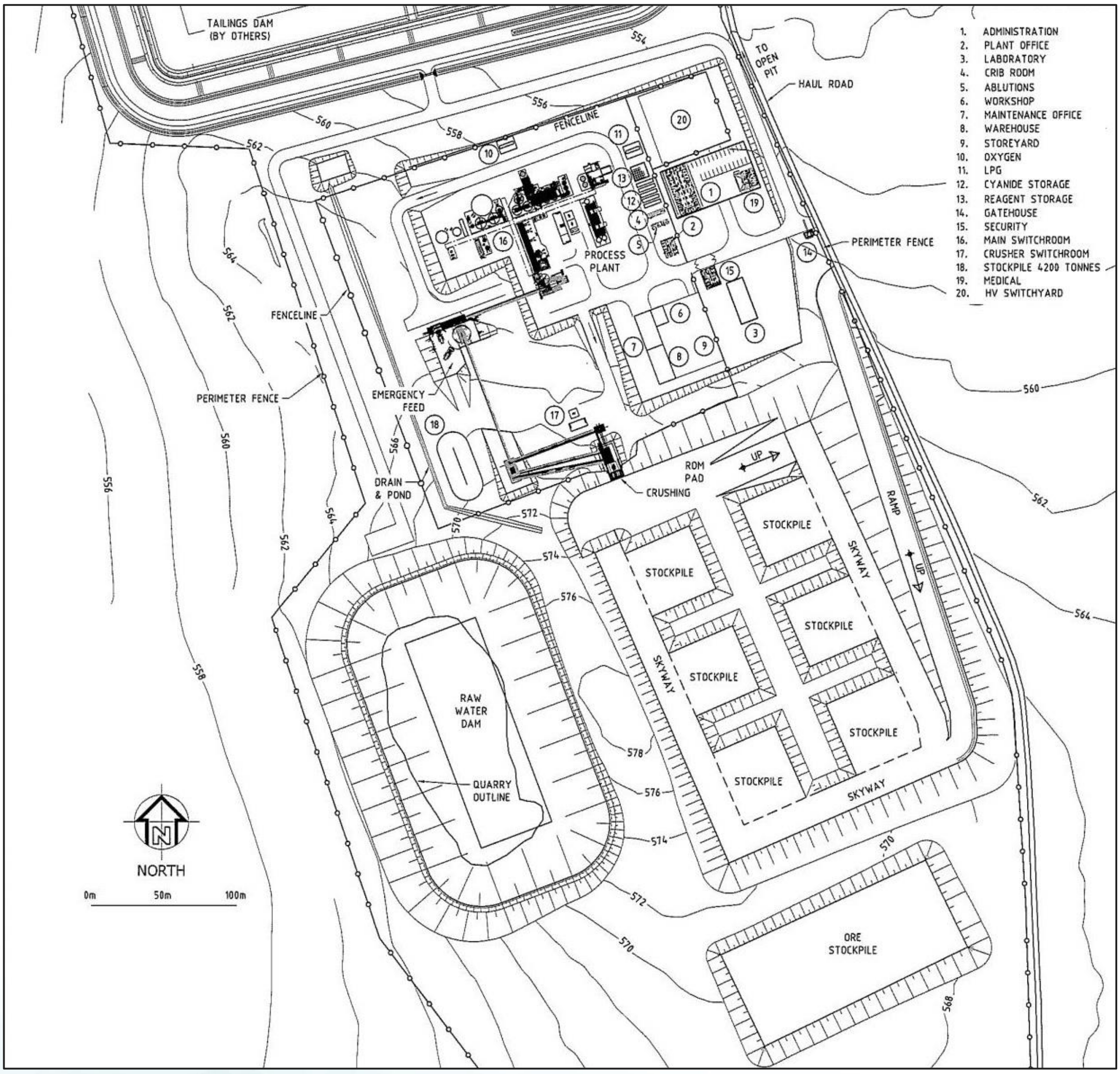
The mass balance output is presented in Appendix 7B and should be read with the process flow diagrams in Appendix 7C.

7.2.17 Process plant layout

The process plant area can be seen in Figure 7.10 with additional details and elevations in Appendix 7E.

The gold process plant utilises a ROM pad with six ore stockpiles that can be blended when feeding the crushing plant. An additional unlined area is allocated for stockpiling oxide ore to the south of the ROM pad. The raw water dam located to the west of the ROM pad.

Figure 7.10 Process plant layout



The overall site location is gently sloping to the northeast and utilises the natural contours to minimise site earthworks where possible.

Access to the process plant is through a manned security office with card readers. Personnel working with in the process plant and offices, maintenance office, warehouse and workshops will have access to the plant site area.

Deliveries to the process plant (oxygen, LPG, reagents, consumables, maintenance supplies, etc.) will be through the security office and along a road to the east side of the site. Oxygen will be stored on the north edge of the process plant site while LPG and reagents will be stored in an area to the northeast.

The workshop, warehouse, maintenance office and store-yard are located in the southeast corner of the site.

Two switch-rooms, one for the crushing plant and one for the main process area, are located within the process plant area.

7.2.18 Process control philosophy

The Client provided the basic premise of operational control and CPC used this as the basis for the overall design. The basic features of the control scheme are summarised in this section.

The instrument list can be found in Appendix 7F.

General

Valves

Valves are typically equipped with the following instrumentation:

- On/off slurry valves have double acting solenoid pilot valves with both open and closed position feedback
- On/off water valves have single acting solenoid pilot valves with position feedback
- Control valves have an integral positioner without any position feedback to the plant control system (PCS)
- Manual valves do not have any position feedback unless specifically required.

Flotation cells

The flotation cells include the following vendor supplied instruments:

- A single air mass flow meter for each group of cells
- A single air flow modulating valve for each group of cells
- A manual valve for air flow trim at each cell air inlet
- A dart valve station generally with two air actuated modulating valves for each group of cells located at the discharge of the group
- Float, target plate, and level transmitter for each group of cells.

Slurry pumps

Duty only (single) slurry pumps typically have:

- A manual drain valve on the suction
- A manual flushing point on the suction, located before the suction isolation valve
- A manual flushing/sampling point on the discharge
- Gland seal water as described in below.

Duty and standby slurry pumps have:

- A manual isolation valve on the suction
- A manual flushing point on the suction, located before the suction isolation valve
- A manual flushing point on the discharge, located before the discharge isolation valve
- A manual drain valve at the lowest point on suction or discharge
- A manual drain valve on the discharge, where applicable
- A manual isolation valve on the discharge
- Gland seal water as described below.

Gland seal water

Gland water supply lines will include:

- A manual isolation valve
- A flow switch
- A check valve
- A MARIC combined pressure regulator-flow restrictor
- A pressure gauge.

Sump pumps

Sump pumps have high level contact-type switches.

Typically, the pump starts automatically on high sump level and continues running for a pre-set time after the high level switches off.

Level switch and sump pump monitoring

The control system monitors sump level switch status and associated sump pump operation and creates alarms as follows:

- Sump level switch remains high for >5 minutes at any time
- Sump level switch remains high for >5 minutes and sump pump has run for this duration
- Sump pump switches on more than 12 times in an hour.

Slurry hoppers

All slurry hoppers have level measurement using an appropriate instrument. Typically, slurry hopper level is controlled by:

- PID feedback loop to the associated pump VSD; or
- PID feedback loop to hopper process water addition.

Water pumps

All water pumps have:

- Manual isolation valves on suction and discharge
- A check valve on the discharge
- A manual drain valve at the lowest point on suction or discharge
- A flanged or threaded stub on the discharge with a suitable isolation valve, for connection of a manual pressure gauge.

Water tanks and ponds

The raw and process water ponds have level measurement by immersed pressure transmitters.

Water tank level is measured by ultrasonic or immersed pressure transmitter.

Thickeners

Each thickener includes a local control panel (LCP) providing local indications, pushbuttons, motor LCS and signal marshalling. All indications and data are communicated to the main PCS. Control of the thickener is by the main PCS, not a local PLC.

Each thickener has a recirculate mode. Two manual valves are provided on the thickener underflow pump discharge for diverting thickener underflow to the thickener feed box.

Crushing, screening and storage

Primary crusher feed rate

The primary crusher feed rate is measured using a weightometer on the primary crusher discharge conveyor and controlled by a proportional-integral-derivative (PID) feedback loop to the primary crusher feeder VSD. The primary crusher feed rate set point is a human/machine interface (HMI) operator input.

Secondary crusher feed rate

The level in the secondary crusher feed cavity will be measured and displayed in the PCS. The secondary crusher will be choke fed to maintain a level in the feed cavity by a PID feedback loop to the crusher feeder variable speed drive (VSD).

Tertiary crusher feed rate

The level in the tertiary crusher feed cavity will be measured and displayed in the PCS. The tertiary crusher will be choke fed to maintain a level in the feed cavity by a PID feedback loop to the crusher feeder VSD.

Grinding and classification

Ball mill feed rate

The ball mill feed rate is measured using a weightometer on the ball mill feed conveyor and controlled by a PID feedback loop to the ball mill feeder VSDs.

The ball mill feed rate set point is an HMI operator input. One or more ball mill feeders may be selected to operate and the operator sets the percentage that each feeder will output, based on a total output of 100%. For example, feeder 1 could be set to 20% and feeder 2 set to 80% to meet the required throughput set point.

Cyclone feed density

Cyclone feed density is measured at the cyclone feed pump discharge and controlled by a PID feedback loop on the mill discharge hopper process water addition. Process water flow is measured using a flow meter and controlled using a modulating valve. Cyclone feed density is an HMI operator input.

Flotation and regrind

Rougher flotation air

Rougher flotation air flow is measured by a mass flow meter, and controlled to the HMI input set point using a single modulating air control valve.

Scavenger flotation air

Scavenger flotation air flow is measured by a mass flow meter, and controlled to the HMI input set point using a single modulating air control valve.

Rougher flotation level

Rougher slurry level is measured by a laser and target plate assembly, and controlled by a PID feedback loop to two modulating dart valves in the discharge box.

Scavenger flotation level

Scavenger slurry level is measured by a laser and target plate assembly, and controlled by a PID feedback loop to two modulating dart valves in the discharge box.

Concentrate thickener

The PCS controls the following variables:

- Bed pressure and thickener underflow density by VSD control of the thickener underflow pump speed
- Bed level by VSD control of the flocculant dosing pump
- Thickener rake height depending on rake torque.

Regrind mill

The regrind mill will be controlled by a vendor control panel.

Leaching and adsorption

Manual operation.

Gold recovery

Manual operation.

Tailings

Tailings thickener

The PCS controls the following variables:

- Bed pressure and thickener underflow density by VSD control of the thickener underflow pump speed
- Bed level by VSD control of the flocculant dosing pump
- Thickener rake height depending on rake torque.

Reagents

Flotation reagent dosing flow

Reagent addition to the various flotation dosing points is controlled using variable speed dosing pumps with HMI operator input set points.

CIL reagent dosing flow

Reagent addition to the various CIL dosing points is controlled using variable speed dosing pumps with HMI operator input set points.

Detox reagent dosing flow

Reagent addition to the various detox dosing points is controlled using variable speed dosing pumps with HMI operator input set points.

Lime dosing flow

Lime slurry will be circulated around the CIL and detoxification areas via a ring main. Dosing will be controlled via on/off pinch valves set on timers from the PCS.

Elution reagent dosing flow

Elution reagent pumps will be stopped and started manually as part of the acid wash and elution sequence.

Reagent make-up

All reagent make-up activities except flocculant will be manual. Storage tank levels will be measured and indicated in the PCS.

The flocculant mixing plant will be supplied as a vendor package and controlled from a local control panel. Initiation of a flocculant mixing sequence will be a manual activity.

7.3 Process risks and opportunities

7.3.1 Risks

A risk review session was undertaken as part of the feasibility level of engineering. Extreme, high and moderate risks were highlighted and discussed. The process risks that were outlined in this session are briefly discussed below, along with control measures to mitigate risks where possible.

Table 7.2 Process area risks and mitigating controls

Risk	Current controls	Future controls
Production targets may not be achieved or equipment may be damaged if there is inadequate training for process and maintenance personnel.	Awareness (strategy and procedural development are execution phase activities).	Execution phase will include development of training strategy, employment of skilled operators, education programs for locals.
Reagent incident due to the use of corrosive and toxic reagents, the most hazardous of which is cyanide. Low pH can cause the evolution of HCN gas which can be fatal.	Process control within the design of the plant, cyanide monitors, alarms, cyanide management and procedural development are execution phase activities.	Cyanide management plan to be developed in execution phase, personal protective equipment (PPE) and HCN monitoring.
Personnel injuries due to some high-risk activities that occur for process and maintenance personnel including working at heights, confined space, electrical isolation procedures, chemical exposure.	Awareness (strategy and procedural development are execution phase activities).	Execution phase will include development of training strategy, employment of skilled operators, education programs for locals.
Water supply is based on harvesting water from Varada River over four months and storing sufficient water for the year. Lower rainfall or higher evaporation may impact the supply of water.	Contingency (two months) within the raw water pond design, option to collect water over a four-month period from the river.	Explore the potential for accessing deep aquifer supply water, investigate bore water access at the river, investigate pit dewatering flows and reuse possibilities.
Potential for a traffic accident when transporting corrosive and toxic reagents required in the process plant.	In country suppliers with industry expertise. Logistic and safety plans are execution phase activities.	Investigate local supply options and develop logistic and safety plans during execution phase.

Risk	Current controls	Future controls
Potential for single vehicle accident on the project site, vehicle colliding with plant equipment or infrastructure, heavy vehicles in use on shared roadways.	Fenced off operations area with controlled access, signage, road design, Traffic Management System and safety procedures are execution phase activities.	Traffic Management System and safety procedures to be developed during execution phase.
Excessive noise or dust from the process plant may cause issue with the local farming communities.	High community engagement during the design/planning phases of the project, good management team, bins and dust collection in the design, crushing on day shift only, plant location utilises geographic features to buffer noise/dust.	Development of an Environmental Management Plan during the execution phase, community engagement throughout the life of mine, monitoring of noise and dust during operations.
Quality of power supply may be an issue causing equipment to trip, reduced plant availability, unable to meet production targets, equipment damage.	Quality supplier in area, other similar operations report no issue of this type.	Ongoing communication with power supplier.
Long term stability of arsenic in the TSF.	Ongoing testwork to understand the potential of arsenic precipitants re-dissolving in the TSF.	Modifications to process design measures should they be required once testwork is complete.

7.3.2 Opportunities

There are several opportunities to simplify the process design to reduce upfront capital costs through either changing selection of equipment or by staging construction of unit operations. The opportunities are detailed below:

- The gravity circuit could be installed after the plant has started up without potentially much reduction in overall recovery based on the increased leaching time in the CIL circuit. Additionally, it may be possible for tabling of the gravity concentrate rather than including an intensive leach reactor and direct smelting of the concentrate dependent on security and other operating constraints.
- Inclusion of the flotation scavenger cells is based solely on other operations and not testwork for the Ganajur Gold Project. It may therefore be possible to remove this circuit without significantly impacting the overall gold recovery. Further testwork on the flotation rougher tailings should be completed to investigate the viability of this circuit.
- Currently an IsaMill™ has been installed for regrinding the flotation concentrate. It may be possible to install a similar piece of equipment for achieving a P_{80} grind of 10 µm that is not proprietary technology and therefore less costly.
- There is opportunity to reduce the complexity and installation cost of the elution circuit by combining acid washing and elution in a single column.
- The reagent make-up systems could be simplified by sourcing some of the reagents with lower consumption (PAX, copper sulphate, lead nitrate, caustic) in liquid form or in smaller 25 kg bulk bags. This may increase slightly the cost of the supply, however, would remove bulk unloading systems, mixing tanks and agitators.

8 SURFACE GEOTECHNICAL AND TAILINGS DISPOSAL

The following chapter was provided by Prime Resources environmental consultants.

8.1 Tailings disposal

8.1.1 Introduction

Deccan Exploration Services Private Limited (DESPL) aim to complete a Feasibility Study (FS) of the Ganajur Gold Project gold mine located in the state of Karnataka, India.

Snowden was appointed by DESPL as the principal project consultant. Prime Resources was approached by Snowden to undertake the Feasibility Level Design Aspect of the tailings storage facility (TSF) and its associated infrastructure.

A description of the design process, the resultant design work and capital estimates to the appropriate level are documented in this chapter.

8.1.2 Terms of reference

The terms of reference laid out to Prime Resources regarding the Ganajur Gold Project TSF are as follows:

- The development of design criteria and information requirements for the FS design of the TSF
- Geotechnical investigation
- Selection and evaluation of potential candidate site/s for the TSF and deposition methodology
- Design of the selected TSF including capacity analysis, earthworks components and slope stability
- Development of a water management strategy and design of water management infrastructure
- Capital cost estimation, accurate to 15%
- Feasibility level design report and supporting documentation in terms of the JORC standards and requirements.

8.1.3 Battery limits

The battery limits for the Prime Resources feasibility level TSF design are as follows:

- Downstream from the intersection of the tailings slurry delivery pipeline with the starter wall
- Upstream of the first flange prior to the return water pump station
- Any electric, electronic, motorised or mechanical equipment including instrumentation are considered as outside the TSF scope but included in the infrastructure scope.

8.1.4 Design criteria

Various technical investigations, discussions between the various project consultants and specialists, technical, social and environmental considerations, legislative requirements and constraints including ongoing laboratory testing regimes have been undertaken in developing the design criteria for the TSF.

The design criteria are summarised in Table 8.1 below.

Table 8.1 TSF design criteria

Item	Description	Specification	Source
1	Life of mine	8.4 years	Snowden
2	Total ore tonnage	2,520,000 t	Snowden
3	Annual throughput	300,000 t/a	Snowden
4	Tailings specific gravity	3.12	Laboratory testing
5	Tailings placement dry density	1.55 t/m^3	Laboratory testing
6	Slurry – solids percentage by mass	55%	DESPL metallurgist
7	Slurry density	1.94 t/m^3	Laboratory testing
8	TSF volume requirement	1,626,000 m^3	Calculation
9	Maximum rate of rise	2 m/a	Prime Resources
10	Maximum height	30 m	DESPL
11	Overall side slope	1V:3H	Prime Resources
12	Lining requirement	Clay layer and HDPE geomembrane	Prime Resources
13	Deposition method	Slurry, spigot pipeline	Prime Resources
14	TSF construction method	Self-raising, upstream	Prime Resources
15	Decant method	Vertical gravity penstock	Prime Resources
16	Seepage management	Filter drains	Prime Resources
17	Return water management	Return water dam	Prime Resources
18	Stormwater management	Stormwater dam (event dam)	Prime Resources

8.1.5 Available information

The Ganajur Gold Mine FS involved several specialist studies and investigations which has been made available on a shared information platform. This includes among others, geographical, topographical, climatic and geotechnical information. The information listed below was considered key to the TSF design process:

Climate and meteorology

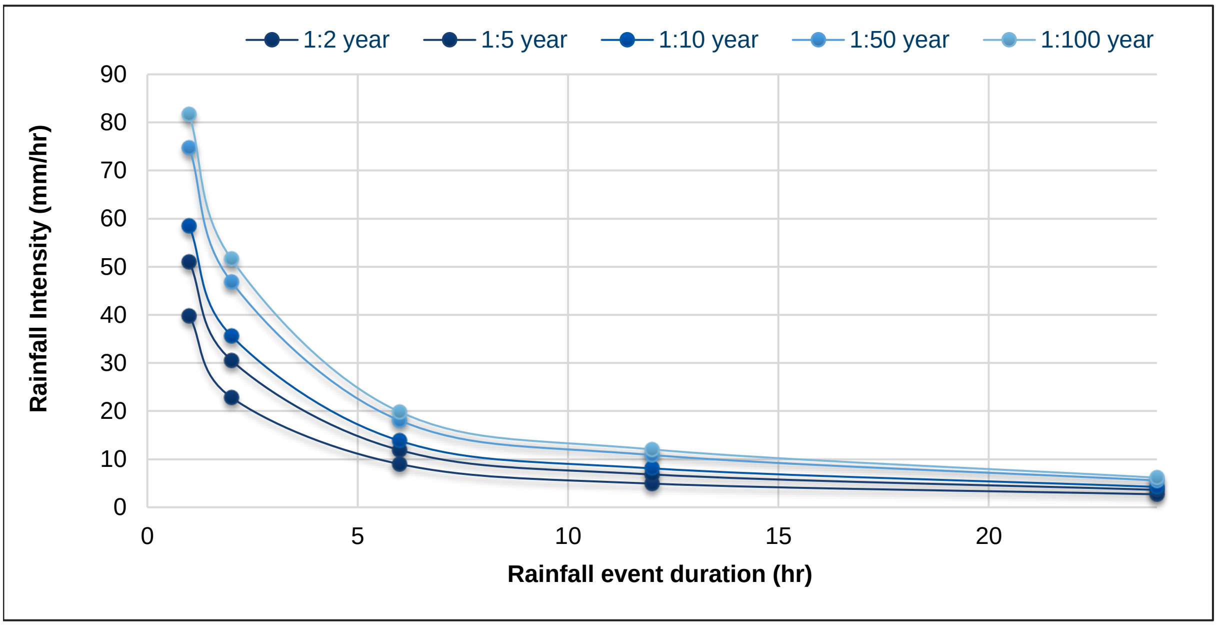
The meteorological and climatological data for the project area was provided by E I Technologies Pvt. Ltd (EIT). This data includes historic rainfall and evaporation data, including statistical data regarding return periods, rainfall intensities and storm durations. The average rainfall and evaporation for the Haveri District were reported as shown in Table 8.2.

Table 8.2 Average rainfall and evaporation for the Haveri District

Month	Rainfall (mm)	Evaporation (mm)
October	77	114
November	43	110
December	3	102
January	5	110
February	1	134
March	14	161
April	57	171
May	73	166
June	106	114
July	141	89
August	116	92
September	101	99
Annual	**738**	**1,461**

Intensity Duration Frequency (IDF) curves were produced by EIT which was used in determining the sizes of water management components such as diversion channels and containment dams. The design of conveyance channels was based on a high intensity, short duration storms (1:50-year, one hour), while the containment structures were based on lower intensity, longer duration events (1:100-year, 24-hour). Figure 8.1 shows the IDF curves used in the design and sizing of stormwater management infrastructure.

Figure 8.1 IDF curves for the Haveri District



Source: EIT

Further to the available average rainfall data, the Haveri District monthly rainfall figures for a total of 103 years (1901 to 2014) have been supplied by EIT.

A meteorology study was undertaken during the summer season, whereby temperature, humidity and wind direction was recorded. The temperature ranged from a minimum of 18°C to a maximum of 41°C, with a minimum and maximum humidity of 11% to 76% respectively.

The wind direction was recorded from which the wind pattern was determined for eight-hour intervals of the day. The predominant wind direction for each of the intervals and in general is illustrated in Table 8.3.

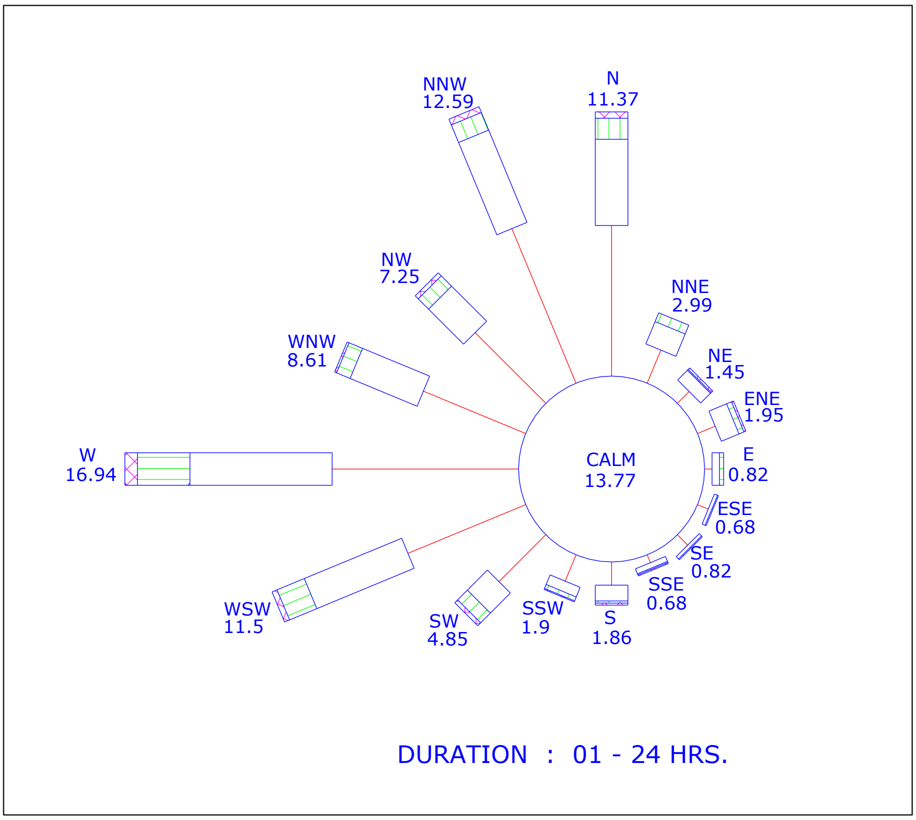
Table 8.3 Predominant wind direction

Interval (hrs)	Predominant wind direction
00:00 – 08:00	WSW-W-WNW-NW-NNW
08:00 – 16:00	WSW-W-WNW-NW-NNW-N
16:00 – 24:00	WSW-W-WNW-NW-NNW-N
00:00 – 24:00	WSW-W-WNW-NW-NNW

Source: EIT

The wind rose representing the wind direction over a 24-hour period in summer is shown in Figure 8.2.

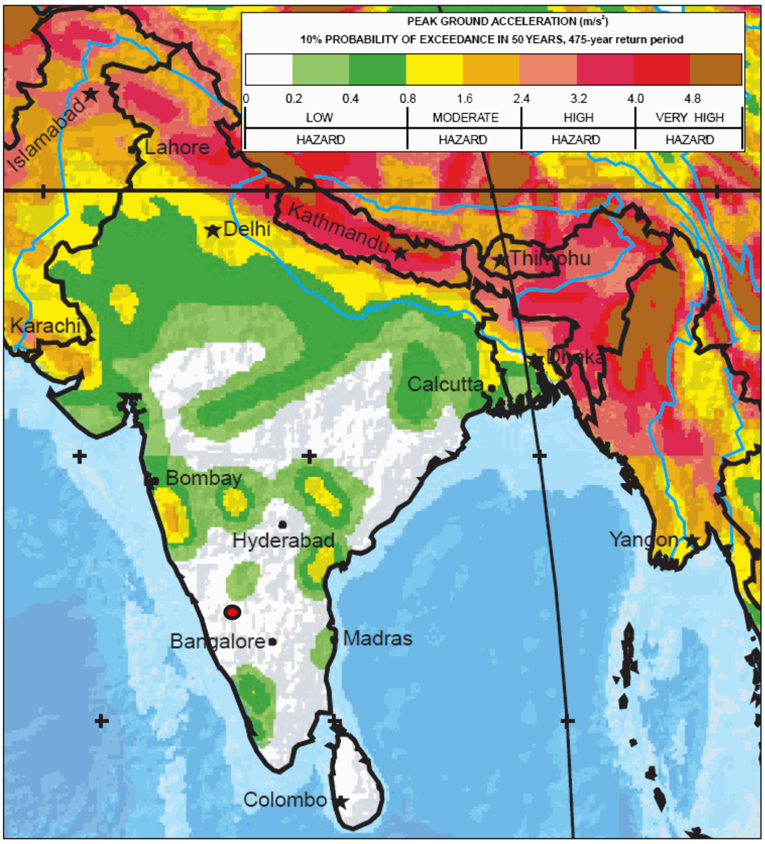
Figure 8.2 Wind rose diagram for the Ganajur Village (EIT)



Seismicity

India has been divided into seismic risk zones II, III, IV, and V indicating the level of seismic risk. Zone II represents areas that are least seismically active with the lowest risk of damage, where Zone V shows areas that are most active with the highest risk of damage to property and loss of life. The project area is situated in the Haveri district, at the centre of Karnataka State, which falls within a Zone II area. The TSF was designed with a low seismic risk factor (Building Materials and Technology promotion Council, 2016). Figure 8.3 below indicates the seismic hazard areas of India and the low hazard rating for the project site.

Figure 8.3 Seismic hazard map of India



Source: Global Seismic Hazard Assessment Program, 2017

8.1.6 Tailings geochemical characterisation

A geochemical assessment of the tailings material was undertaken by Geostratum Groundwater and Geochemistry Consulting (Pty) Ltd (Geostratum), together with Prime Resources.

The detailed geochemical analysis on the tailings material can be seen in Chapter 11 of this report. The tailings have been classified as potentially acid forming (PAF) and described in detail. The geochemical assessment also included comments on the presence of arsenic in the ore due to the residual arsenopyrites. An arsenic stabilisation step has also been included in the metallurgical process to reduce arsenic in the tailings stream.

As a result of the geochemical classification, the design of the TSF was completed to match Indian legislative requirements and to mitigate any predicated environmental impacts.

8.1.7 Tailings mechanical characterisation

Geotechnical laboratory testing was conducted on a representative tailings sample to determine its geotechnical and mechanical properties. This information was used as input parameters in design aspects of the TSF. The following is a summary of the various tests conducted, their results and analysis:

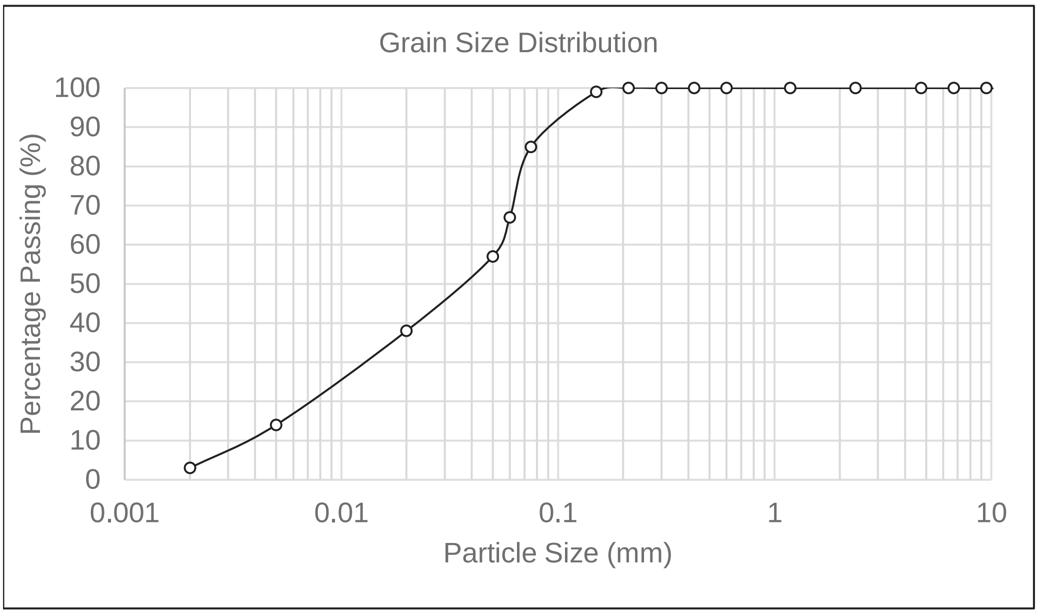
Particle size distribution and classification

The tailings material was graded, Atterberg Limits determined and classified. A preliminary indication of the deposited tailings likely behaviour can be deduced from these results. Behavioural aspects include packing structure and permeability. This is specifically important in the case of a high clay fraction. The particle density or specific gravity was also determined in aid of design calculations and further characterisation tests. Detailed in Table 8.4 is a summary the grading, classification and particle density tests conducted. Particle size distribution is shown in Figure 8.4.

Table 8.4 A summary of the grading, classification particle density tests on the Ganajur Gold Mine tailings

Test	Parameter	Detail
Particle density	g/cm^3	3.12
Particle size distribution	Sand: 0.075 mm to 2 mm	15%
	Fines: 0.002 mm to 0.075 mm	82%
	Clay size: < 0.002	3%
Atterberg Limits	Plasticity Index	2%
USCS Classification	ML or OL: Inorganic- or organic silts and silty clays of low plasticity	

Figure 8.4 Particle size distribution of the Ganajur Gold Mine tailings



The initial characterisation testing results indicate that the tailings material is well graded and is classified as organic or inorganic silty clay with low plasticity. The Atterberg Limits results show a Liquid Limit of 22%, a Plastic Limit of 20% and a Linear Shrinkage of 1.3%.

Settling, drainage and consolidation

Settlement tests were performed on the tailings to characterise the settling of the tailings once deposited and to determine an initial estimate of the placement density, which is used in preliminary calculations and further testing. The settling tests were conducted under drained and undrained conditions to simulate the various conditions regarding permeability typically found in a TSF. Undrained conditions are found against a barrier system or liner, whereas permeable conditions are found throughout the tailings material, but in varying degrees of hydraulic conductivity. The permeability was also measured during the settling/drainage tests. During each test, the dry density, void ratio and permeability was measured/calculated and recorded. The samples comprised of a slurry with 55% solids by mass. Table 8.5 (below) contains a summary of the settling and drainage tests.

Table 8.5 Summary of the settlement and drainage tests of the Ganajur Gold Mine tailings

Test	Parameter	Detail
Sedimentation test (undrained)	Final dry density (g/cm^3)	1.42
	Final void ratio	1.19
	Settling time to settle (days)	1
Settlement and bottom drainage	Final dry density (g/cm^3)	1.639
	Final void ratio	0.90
	Maximum permeability (cm/s)	1.62×10^{-4}
Immediate bottom drainage	Final dry density (g/cm^3)	1.796
	Final void ratio	0.74
	Maximum permeability (cm/s)	8.84×10^{-4}
	Settling time to settle (days)	7

The results indicate a higher density is achieved under drained (bottom) conditions, which implies improved settlement and consolidation with functional drainage or an underlying layer with greater permeability. The final dry density is unlikely to increase to 1.796 g/cm^3 as shown during the draining (bottom) of the immediately drained sample. Considering the drainage conditions and final dry densities, a target dry density of 1.55 g/cm^3 and void ratio of 1.013 was selected as the design density and for the preparation of the triaxial cell test sample. This estimate is considered conservative in the analysis of capacity and stability of the storage facility.

The consolidation characteristics of the tailings material is required to determine the behaviour of the material during deposition and post-closure of the facility. The consolidation testing comprised of placing a tailings sample into a oedometer and applying an increasing vertical load with top and bottom drainage allowed. The change in volume is made with each increment in applied load. The results can be used to determine the behaviour of the tailings under various loads and the required time to achieve required densities.

The load varying from 12.5 kPa to 800 kPa was applied to the tailings sample and the change in volume measured. The recorded measurements can be seen in the laboratory test certificate in Appendix 8A. Figure 8.5 (normal consolidation line) and Figure 8.6 show the void ratio and dry density vs. the effective vertical stress, respectively. As the load is increased, the void ratio decreased and the dry density increased. This shows that the density of the deposited tailings and the effective stress will vary with depth from the surface.

Figure 8.5 Consolidation test results – void ratio vs. effective vertical stress

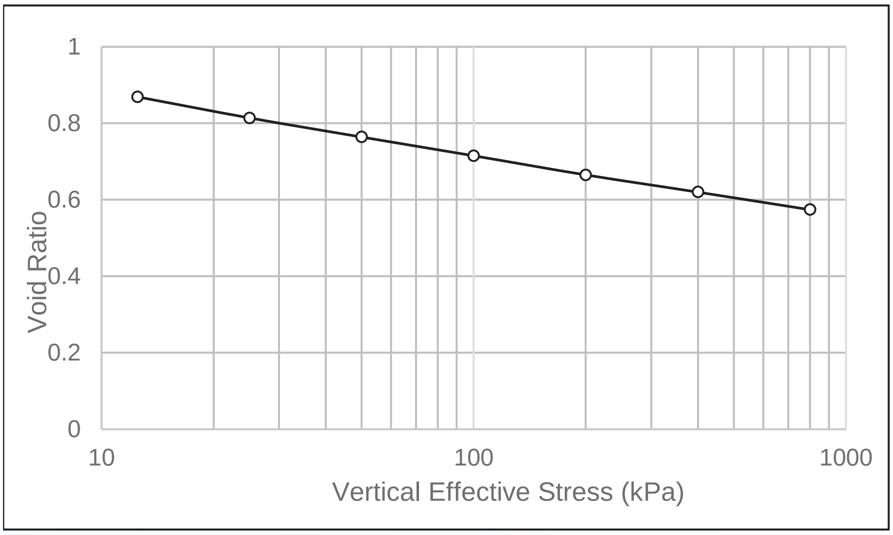
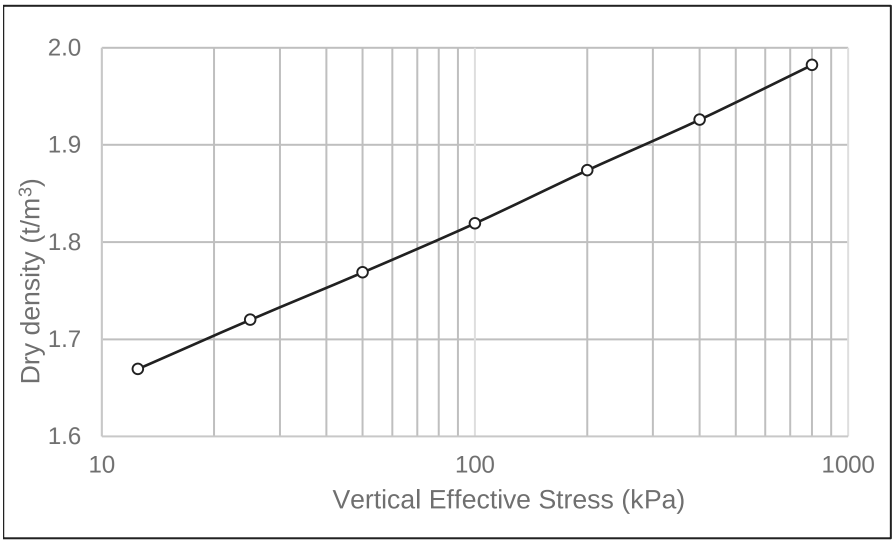


Figure 8.6 Consolidation test results – dry density vs. effective vertical stress



The deposited tailings will settle to an initial approximate density of 1.55 t/m^3. As additional tailings are deposited, the density of the underlying tailings will increase. Once the TSF has reached its final height, the effective vertical pressure at the bottom of the facility is approximated at 320 kPa. This corresponds to a density of approximately 1.90 t/m^3 at an approximate final depth of 18 m. The tailings are expected to achieve this density given that the rate of rise <2m/year is not exceeded.

Shear strength and permeability

The tailings were subjected to a consolidated undrained (CU) triaxial test during which the undrained shear strength of the material was determined. The test comprised of a saturation, consolidation and shear phase. Three specimens were tested varying applied stress and measurements taken during each test. The official laboratory certificate can be seen in Appendix 8A.

The results show an effective internal friction angle (Φ') of 36° and an effective cohesion (c') of 0 kPa. These results were as expected and indicative of typical tailings shear strength properties. The 3% clay particles had no impact on the cohesion of the tailings. The shear strength properties have been used in the final slope stability analysis of the TSF.

The triaxial apparatus was also utilised to determine the permeability of the tailings at the target dry density at an effective pressure of 200 kPa. The results indicated an average permeability of 1.04 x 10^{-7} m/s with an order of magnitude difference in the maximum and minimum permeability respectively. The permeability of the filter drainage systems will be required to adequately drain water (from the bottom) from the tailings.

8.1.8 Site selection

The project site is located on agricultural farming lands. The overall footprint and extend of the project must be kept to a minimum so as to reduce the impact on the current land use. The site selection process was governed by various factors, including:

- Required storage capacity and footprint
- Construction and development methods
- Local structural geology
- Topography
- Land ownership
- Rehabilitation requirements
- Existing and significant surface infrastructures and features.

A single area with an approximate footprint of 25 ha was allocated following a site selection study during the Prefeasibility Study and only minor adjustments were made to the final layout of the TSF. The area is situated to the north of the proposed processing plant and west of an existing local road that traverses the project site. The road will be utilised during the construction and operational phases of the project and forms the eastern boundary of the TSF site. The proposed processing plant area forms the southern boundary.

8.1.9 Technology selection

The preliminary geochemical testing on the tailings material indicated an acid producing potential (PAF) which raised concerns and called for additional kinetic testing. During the analyses of the preliminary tests it was determined that arsenic leaching from the tailings is also of a concern. An arsenic stabilisation step was also then incorporated into the metallurgical extraction process. A representative tailings sample was geochemically analysed and the stabilisation appears to be successful. The short-term leach tests showed a concentration of arsenic that would exceed the Indian Drinking Water Standards but was within the Indian General Effluent Standards.

The preliminary testing indicates that acid is likely to be produced under these conditions (sulphide S% above 2.5% and exposure to atmospheric conditions). Short-term geochemical testing has been completed and long-term leach tests are currently still ongoing to ascertain the sulphide S oxidation rate.

As a result, acidic conditions are expected to develop within the top few metres of the TSF in the presence of oxygen, with the deeper material remaining largely unoxidized and neutral. The oxidation mitigation measures include the capping of the TSF to reduce oxidation and the formation of acidic conditions.

Despite the addition of the arsenic stabilisation, a leachate with arsenic concentrations above Indian Drinking Water Standards remains. The geochemistry testing also showed that the release of arsenic is independent of the acidification process.

The short term geochemical testing implied the requirement for effective leachate collection and management and possible measures to reduce the oxidation and acidification.

The initial design was for a conventional upstream facility, typical for a gold tailings facility. The preliminary geochemical testing results and recommendations resulted in the technical trade-off and consideration of a downstream type facility which comprises of a fully constructed embankment behind which the tailings material can be deposited. The waste rock excavated from the open cast pit was identified as a potential construction material for this embankment. The disadvantages of the development of a downstream facility is such that its footprint increases with an increase in height and it needs a full water cap to be maintained over the life of the TSF up to closure. The downstream method however would allow subaqueous deposition of the tailings in a fully lined facility so as to reduce the oxidation of the sulphides and the formation of acidic conditions.

Each of the types of facilities (up- and down-stream) have various implications. The selection of the facility type required for a risk-based approach in evaluating the options and selecting a final impoundment type. The evaluation criteria included those listed in Table 8.6 (below).

Table 8.6 TSF construction technology selection criteria

Issue	Upstream TSF	Downstream TSF
Environmental permitting	Additional mitigation measures to act against the acid generation. Indian authorities will need to consider an upstream potentially acid forming (PAF) TSF.	The downstream method prevents oxidation of the PAF material as part of its design, thereby preventing acid generation and leaching of metals.
Long term/legacy issues	Longer term environmental risks (legacy), including AMD issues and dust generation. Additional mitigation and management measures are required.	Downstream facilities encapsulate the material and prevent the long-term generation of AMD, and prevent dust generation.

Issue	Upstream TSF	Downstream TSF
Reputational risk	From a best practice perspective, appropriate mitigation (liner and treatment) should be put in place to address long term concerns related to AMD/water quality.	The designed addresses the encapsulation of the tailings, thereby preventing oxidation and any long-term AMD concerns.
Footprint size	Smaller footprint (+- 22 ha).	Larger footprint (+- 32 ha).
Land requirements	Defined at construction.	Expansion during life of facility.
Liner requirement	Composite liner design (clay and HDPE) including the basin floor and the perimeter catchment paddocks.	Composite liner design (clay and HDPE) includes basin floor and upstream surface of embankment wall to encapsulate the tailings.
Embankment/starter walls (construction materials)	Embankment walls can be constructed with homogenous soil and fill from within the TSF basin.	Embankment walls are constructed from large quantities of waste rock, preferably “clean”/greywacke.
Requirement for catchment paddocks	Perimeter catchment paddocks required around the entire circumference to contain the surface water runoff from the side-slopes. After closure, this water could become acidic and contain leached metals in higher concentrations than allowed by effluent standards. The paddocks need to be lined and run-off water contained or returned to the plant or TSF, or treated prior to discharge.	If the waste rock is “clean” then paddocks are not required. If the waste rock has the potential to leach metals, the waste rock will require a liner with a catchment paddock and seepage control measures as well.
Freeboard/Walls/ Stability	Freeboard in the TSF is managed by depositing material in the perimeter day wall during the day and into the central pond at night. The TSF stability relies on the drying of the outer tailings material which provides the structural support.	Freeboard is maintained by raising the large compacted waste rock embankment wall keeping the waste rock well above the surface of the saturated tailings. The stability is reliant on the stability of the waste rock.
Central penstock/ causeway	A central penstock is required to decant water off the facility into water catchment dams below the TSF.	No penstock is required. Sump/pump inside the TSF will collect water to be pumped back to the plant.
Return water dam/ stormwater dams	Return water dam and stormwater dam will be required. Free water cannot be stored on the TSF (safety risk). Dams need to be sized to contain a 1:100 flood event.	No requirement for additional dams below the TSF – water will be pumped from the top of the TSF back to the plant water tank.
Factor of safety (FoS)	Lower FoS – the tailings walls are expected to dry and consolidate and provide the stability. Flat outer slopes required (23°).	High FoS – embankment walls are constructed from compacted waste rock, which form solid embankments with high stability.
Oxidation of sulphides in the tailings material	Oxidation of tailings material will occur on the outer surfaces of the TSF. An oxidation front develops and moves towards the centre of the facility.	Due to permanent tailings saturation, there is no oxidation of the sulphides.
Drain water quality	Initially, water quality will be similar to process water. As oxidation occurs, any leachate becomes progressively more acidic and metal rich.	Initially, interstitial water similar to process water. Water will equilibrate with soluble minerals (equilibration with siderite will give an alkaline solution with Fe^{2+}). No acidity generated, no oxidation of sulphides or Fe^{2+}.
Water storage and requirement	Less water is contained on the top of the TSF. Pond size is kept to a minimum (approximately 20% to 30% of the top area of the TSF). 70% to 80% of the top area is subject to rain and tailings reposition. Less evaporation, lower water requirement.	A full water cap on top of the TSF is required, covering 100% of the top surface area. Greater evaporation, greater water requirement.
Dust control	Drier TSF walls are more susceptible to wind erosion and dust generation if not rehabilitated and grassed.	Wet TSF surface, no dust is generated. Sidewalls are greywacke rock.
Capital cost	No waste rock embankments. Smaller starter berms built from material from within the TSF basin. Lined water dams. Lining of TSF floor and catchment paddocks. Smaller total TSF footprint.	Large rock embankments. No additional dams. Lining of TSF floor and also up the waste rock walls. Liner required between the tailings and the waste rock. Larger total footprint and more land purchase.

Issue	Upstream TSF	Downstream TSF
Operating costs	The outer walls of the impoundment will ultimately be covered in a clay cap and sandy loam material during the life of the facility. No outer waste rock wall, no compaction.	Waste rock will need to be compacted to form the buttress embankment. Operating costs include waste rock transport and placement.
Closure	At closure the small water pond will be drained from the top of the facility, the surface covered with a clay cap and thereafter a loamy layer for vegetation cover.	At closure the water will be drained from the top of the facility, the surface covered with a clay cap and thereafter a loamy layer for vegetation cover. The outer rock slope will also require shaping, capping and rehabilitation.
Post Closure	Water in the catchment paddocks will need to be treated for several years post closure until the levels of acid/sulphide and metals are reduced. Monitoring of the groundwater around the perimeter would need to continue post closure.	No long-term treatment will be required for the encapsulated material. Monitoring of the groundwater around the perimeter would need to continue post closure.

Each of the impoundment types were evaluated according to the criteria above. The following were identified as the key criteria by DESPL for the selection of the TSF construction and operation methodologies:

- Cost effective solution
- Environmentally acceptable practice
- Maximum water conservation
- Minimum land use (in an arable area).

Prime Resources issued a final recommendation for the selection of the upstream TSF design, based on the key criteria. The key design features to be included in the upstream TSF design include:

- Smaller TSF footprint that does not rely on waste rock for construction
- An HDPE lined (geomembrane) footprint over the in-situ compacted clays, to form a double barrier system to protect the groundwater against any potential contamination
- Effective, rapid and progressive sidewall rehabilitation, during the operation of the facility, to seal the lower tailings benches and to rehabilitate the sidewalls as the tailings walls rise, if high oxidation rates (to be confirmed in long term geochemical tests) are expected
- Minimise evaporation water losses, maximum water recovery to the plant
- HDPE and clay lined catchment paddocks around the TSF to catch any surface runoff from the TSF sidewalls for reuse
- The water from the decant structures and internal drains will collect in a lined facility for reuse.

The long-term kinetic leach tests currently underway will provide an indication as to the rate of oxidation of the tailings material, which can be related to the life of the facility, i.e. a high oxidation rate implies that tailings material will oxidize rapidly during operation; a slower oxidation rate implies that tailings material will oxidize at or only after closure. The oxidation of tailings would lead to the generation of acidic, metal-rich drainage from the TSF. Several factors will affect the rate of oxidation. Some of these factors are intrinsic to the material, for example the reactivity of the material, the reactive surface area of the minerals and the packing density of the tailings (void space).

Other factors include climate (temperature and humidity), the presence and abundance of oxygen and the presence of oxidizing bacteria. If the tailings material has an intrinsic tendency to oxidize quickly, controlling the ingress of oxygen into the TSF can slow down the rate of oxidation.

A high intrinsic rate of oxidation requires that progressive rehabilitation of the TSF is undertaken during operation, so that ingress of oxygen into (and therefore oxidation of) the tailings material is managed on an ongoing basis. A slower oxidation rate allows for effective rehabilitation at closure.

The upstream facility has been designed with sufficient detail so the current risks associated with the upstream disposal methodology are manageable. The final oxidization rate/long term kinetic test results are expected at the end of June 2017.

8.2 Surface geotechnical investigation

A geotechnical investigation was undertaken to characterise the surface soils over the proposed TSF area. The investigation was jointly undertaken by Prime Resources and Sarathy Geotech & Engineering Services Pvt. Ltd (SGES).

Prime Resources conducted a preliminary site visit and selected all the test pit positions. The geotechnical investigation was conducted in three stages, with the Prime Resources test pits selected for initial testing. The first test pits were also profiled and samples taken for mechanical testing.

Once the preferred position and layout of the TSF had been selected and the position of key supporting infrastructure confirmed, Prime Resources selected the number and positions of additional geotechnical test pits for the profiling and sampling and mechanical testing of the near surface soils. The coordinates of the test pits were then supplied to SGES for the site investigation.

In the final phase of testing, additional test pits were excavated over the proposed open pit mining area (mining area) as the surface soils as borrow materials; will be made available for use as the area is cleared preparation for mining.

The laboratory testing of the surface soils over the TSF area included:

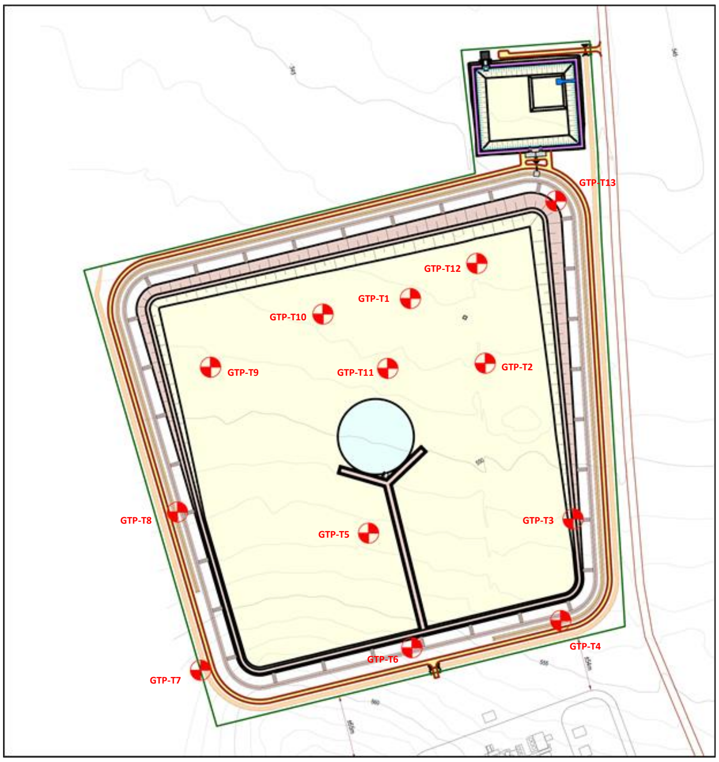
- Foundation indicators: particle size distribution, Atterberg Limits and classification per the Unified Soil Classification System (USCS)
- Free swell index
- Moisture-Density relationship: compaction density and moisture content
- Permeability
- Shear strength parameters
- Dispersion.

These tests were selected in determining the suitability of the in-situ soils for its ended use as construction material (embankments), as low permeability barrier and support for medium to heavy foundations.

8.2.1 Site investigation and fieldwork

A total of 13 test pit positions were selected over the proposed TSF footprint (see Figure 8.7 below). Four additional test pits were also positioned over the mining area in the borrow areas.

Figure 8.7 Test pit locations over the TSF area



The test pits over the TSF and open pit areas were manually excavated to an average depth of 1.5 m, profiled and logged. The respective surface soil profiles over the TSF and mining areas have general profiles as detailed in Table 8.7 and Table 8.8 below.

Table 8.7 General surface soil profile over the TSF area

From depth (m)	To depth (m)	Description
0.00	0.30	Brownish soil with plant roots and humus
0.30	0.70	Black cotton soil
0.7	1.00	Ferruginous chert
1.00	1.50	Greywacke

Table 8.8 General surface soil profile over the mining area

From depth (m)	To depth (m)	Description
0.00	0.30	Brownish black soil
0.30	0.75	Black cotton soil
1.00	1.50	Yellowish grey, completely weathered greywacke

The test pits showed variations in thickness for the profile layers with some of the black cotton soil layers containing some ferruginous chert nodules, chips of quartz and greywacke. No groundwater was observed in any of the pits. The full set of geotechnical results are included in Appendix 8B.

The key soil type of interest for the TSF components is a clayey sand or sandy clay for the construction of a low permeability barrier and embankments such as the initial starter wall. The sampling program included as a minimum a 30 kg bulk sample being taken from each test pit taking into account the soil type and test types required.

Laboratory testing

Selected laboratory tests were assigned to each of the samples. The foundation indicator free swell index tests were performed on each of the samples to ascertain the clay content and clay activity. Selected samples were then subjected to a combination of the remainder of the mechanical tests. The test work was undertaken by SGES with instruction from Prime Resources. A summary of the test results is shown in Table 8.9 below.

Table 8.9 Laboratory test results for the TSF area surface soils

Test	Result summary
Classification	Sandy clay/clayey sand with gravel
Clay size particles	8% to 20%
Specific gravity	2.62 to 2.72
Free swell index	0% to 30%, moderately expansive
Maximum dry density	1.88 t/m^3 to 1.97 t/m^3
Plasticity	Slight to medium, with isolated non-plastic soils
Permeability	2.3 x 10-4 cm/s to 8.7 x 10-6 cm/s at 100% Proctor density
Shear strength parameters	Φ’: 32° to 36°, C’: 7 kPa to 12 kPa
Dispersion	22% to 33%, slight to moderately dispersive

Similarly, laboratory tests have been conducted on the surface soil samples from the test pits over the mining area. Table 8.10 shows a summary of the key results:

Table 8.10 Laboratory test results for the mining area surface soils

Test	Result summary
Classification	Silty sand/clay with gravel
Clay size particles	8% to 48%
Specific gravity	2.64 to 2.70
Free swell index	11% to 31%, moderately expansive
Maximum dry density	1.64 t/m^3 to 1.99 t/m^3
Plasticity	Slight to medium
Permeability	1.1 x 10-3 cm/s to 8.92 x 10-5 cm/s at 100% Proctor density
Shear strength parameters	Φ’: 38° - 40°, C’: 5 kPa to 13 kPa
Dispersion	22% to 33%, slight to moderately dispersive

The mine pit geotechnical drillholes yielded core samples from 0 m to 6 m below ground level have been tested for classification, index and strength parameters. The general profile of the cores for the depth range is shown in Table 8.11.

Table 8.11 General surface soil profile of near surface cores samples over the mining area

From depth (m)	To depth (m)	Description
0.00	3.00	Brownish black soil
3.00	6.00	Yellowish brown-grey, highly- to completely weathered Greywacke

The geotechnical testwork on the core samples was undertaken by SGES with instruction from Prime Resources. A summary of the test results shown in Table 8.12 below.

Table 8.12 Laboratory test results of the near surface cores samples over the mining area

Test	Result summary
Classification	Sandy clay/clayey sand
Clay size particles	16% to 35%
Specific gravity	2.67 to 2.74
Free swell index	30% to 37%, moderately expansive
Maximum dry density	1.76 t/m^3 to 1.92 t/m^3
Plasticity	Medium
Permeability	2.3 x 10-4 cm/s to 8.7 x 10-6 cm/s
Shear strength parameters	Φ': 4° - 7°, C': 57 kPa to 98 kPa (UU triaxial) Φ': 28° - 31°, C': 13 kPa to 19 kPa (direct shear)

The triaxial tests were conducted on the black cotton clayey soils located within 3 m from the surface. The direct shear tests were used on the weathered greywacke material found below 3 m from surface.

The complete laboratory results are attached in Appendix 8B.

Analysis

The laboratory tests were analysed in relation to the soil's intended use, location and available volumes. The intended use for soils over the TSF area is primarily for the construction of a low permeability barrier by in-situ compaction. The TSF soils will potentially also be used in the construction of embankments and berms that require strength and low permeability properties. The soils excavated over the mine pit area will be imported and also used for the liner and any embankment construction if additional material is required.

Classification and clay content

The TSF test pit samples were taken across multiple layers of the soil profile resulting in a bulk sample consisting of various types of soil. The majority of the samples were classified as sandy clay or clayey sand with gravel in some samples. Isolated samples were classified as silty sands with a low clay content. Due to the classification of the majority of the samples, together with their respective clay contents, there is confidence that the near-surface cotton soils over the TSF area will be suitable as a compacted sandy clay liner.

The near surface soils over the mine pit area have similar characteristics to that of the TSF soils, but with notably higher clay content nearer to the stream areas. Some of these samples were also taken at a greater depth. The general classification of the upper 3 m is sandy clay to silty clays and even gravelly clays.

Permeability

A total of seven samples were subjected to falling permeability tests. The samples were selected based on their classification and position relative to the layout of the TSF. The measured permeability of the TSF samples ranged from 2.3×10^{-4} cm/s to 8.7×10^{-6} cm/s. A permeability of 1.0×10^{-7} cm/s is considered the maximum allowed permeability for a full clay layer, when used as a hydraulic barrier (almost achieved in some of the "black cotton soil" samples). The average measured permeability is however greater than the allowable maximum permeability, to solely rely on a clay liner.

The blending of the soils layer during the sampling program can be attributed to the higher permeability due to an increase of larger granular particles. The compaction effort could also be increased to further decrease the overall permeability. It is therefore suggested that during the detailed design phase, specific testing is undertaken where the focus is placed on the sampling of individual soil layers (black cotton soil specifically) and achieving the adequate field compaction.

Considering the clay content and plasticity of the soils, it is however suspected that the upper black cotton soils will adequately perform as a low permeability liner once suitably moisture conditioned and compacted to specification.

Shear strength

It is planned to use the TSF area in-situ soils for a low permeability barrier and the imported soils from the mining area for the construction of embankments, berms and containment earthworks structures. The TSF area in-situ soils may also be used for embankment construction is available in sufficient quantities. The in-situ soils will also be required to accommodate medium to heavy foundations such as the penstock intake structure bases.

Selective TSF soil samples were subjected to Direct Shear tests. The results showed an angle of friction ranging from 32° to 36° and cohesion values of 7 kPa to 11 kPa. These shear strength parameters have been used in the TSF slope stability analysis (Section 8.3.7). The use of the TSF in-situ soil shear strength parameters follows a conservative approach. The surface soils over the mining area show shear strength parameters of 38° to 40° (Internal friction angle) 5 kPa to 13 kPa (cohesion) which indicate similar properties to the TSF soils.

The bearing capacity of the in-situ soil have been estimated with the shear strength parameters. It is required for the base of the penstock intake structure which will be constructed of reinforced concrete. The tailings covering the penstock base will exert additional pressure on the foundation. An estimate bearing pressure of 350 kPa will be exerted on the foundation. The bearing capacity is estimated at 1155 kPa, resulting in a Factor of Safety (FoS) of 3.3. The in-situ soil strata at 1.5 m below surface is considered to have adequate bearing capacity in relation to the requirements of the TSF.

The shear strength parameters of the upper surface soils over the mine area, determined from the UU triaxial tests, show elevated cohesion values and low friction angles. The direct shear tests performed on the lower soils show balanced cohesion and friction angle values. These parameters have been incorporated into the slope study.

Dispersion and swell index

The classification and foundation indicator tests have indicated that the majority of the samples are comprised of a clay content in excess of 15%, especially the upper black cotton soils over the mine area. The soils are expected to exhibit plastic behaviour, swelling and shrinkage. The free swell index range of 0% to 37% show the soil to be low to moderately to highly expansive with no surcharge load. The pressure swell index for the soils will however be lower due to confining surcharge.

The dispersion index of the soil show slight to moderate dispersion. This is applicable to the upstream face of the TSF starter wall and the areas around the filter drains of the TSF, where saturated conditions exist with added pressure from the overlying saturated tailings. Piping or internal soil erosion could occur under these conditions. An additional liner material will be place over the compacted clay base and compacted starter walls which will eliminate the possibility any piping thought the starter wall.

Resistivity survey

Geologic mapping and a ground resistivity geophysical survey in and around the proposed TSF area was conducted to examine and understand geological structures and its parameters such as the presence of a mineral orebody, fluid content, porosity and degree of water saturation in the rock, weathering, potential faults, and shear zones. It is required to take cognisance of subsurface geological features that may influence the design, construction and operation of the TSF. The resistivity profiling was undertaken for a period of 21 days and comprised 10 profiles with various orientations and 3.36 line kilometres.

The resistivity profile survey was carried out according to the Schlumberger Method. At each station readings were taken at 5 m, 10 m and 20 m electrode spacing by keeping the potential electrode at a spacing of 2 m. This method was employed to understand the resistivity at three different depth intervals. The station interval was maintained at 5.0 m for each electrode separation. A fixed current of 50 mV was passed for each current electrode separation. A cycle of four was measured for each reading.

A total of 12 profiles were surveyed that covers the processing plant, water tank facility and TSF. The processed survey data are presented as line graphs and the 2D inversion models in the survey report (Appendix 8C).

Table 8.13 shows the details of the survey profiles taken over the TSF, plant infrastructure area:

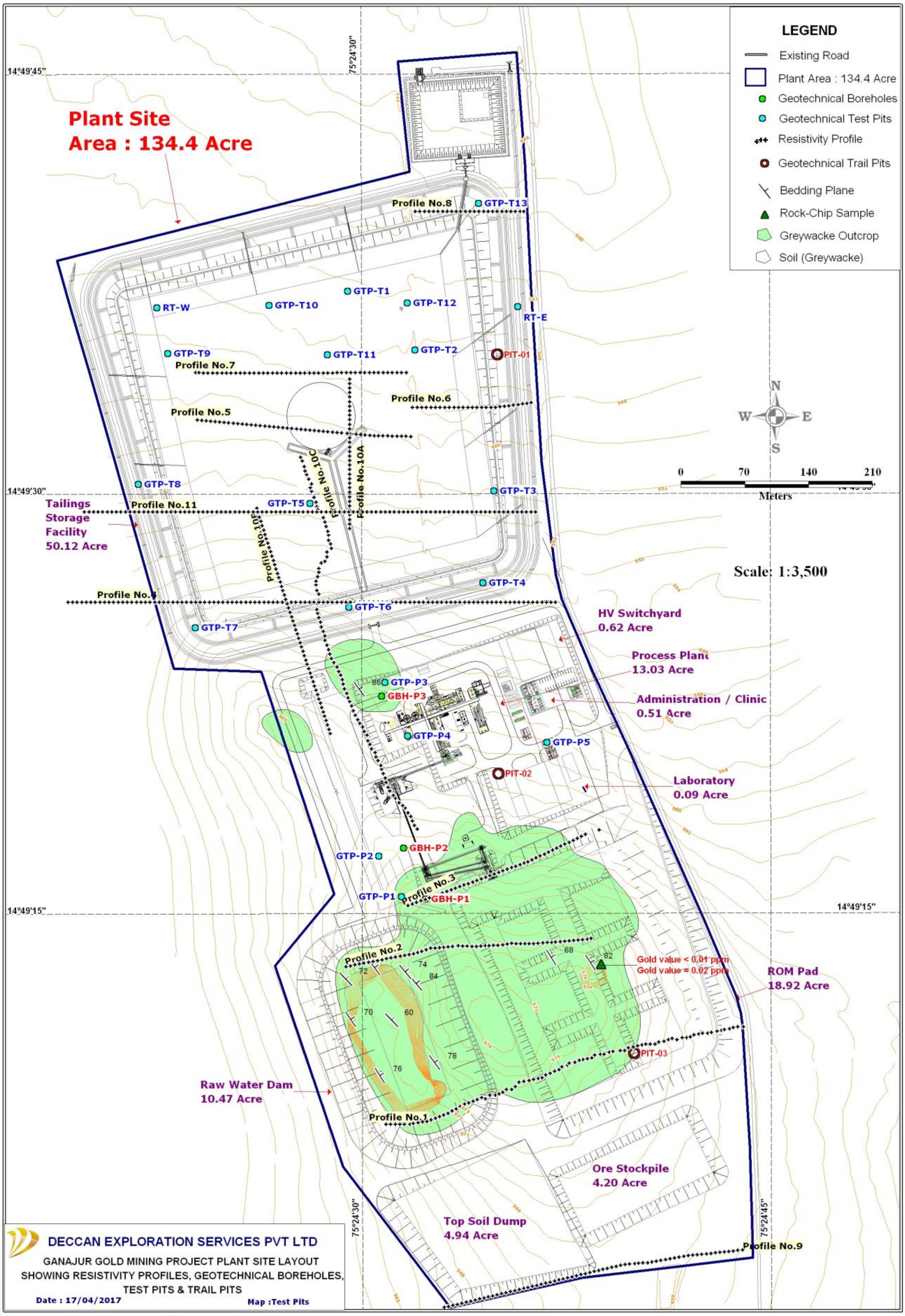
Table 8.13 Resistivity survey profile details

Profile number	Figure no.	Location	Length of profile (m)
Profile 1	6	Water storage facility	410
Profile 2	7	ROM pad	275
Profile 3	8	ROM pad	220
Profile 4	9	TSF	540
Profile 5	10	TSF	235
Profile 6	11	TSF	135
Profile 7	12	TSF	230
Profile 8	13	TSF	125
Profile 9	14	South of water storage facility	335
Profile 10A	15	TSF	165
Profile 10B	16	TSF	260
Profile 10C	17	TSF and process plant facility	430
		Total	**3,360**

Source: DESPL, 2016

Figure 8.8 below shows the location and orientation of the survey profiles.

Figure 8.8 Resistivity survey profile lines



Source: DESPL, 2016

The geologic survey and resistivity survey across the entire plant and the TSF examined the presence of concealed structural features. The survey data analysis for the plant infrastructure area indicated the presence of highly resistive features that could probably be due to unweathered chert and greywacke and does not indicate the presence of any major structures.

The data indicate that the plant and the water storage facility area is underlain by resistive material, suggesting that the rock is competent.

8.3 Tailings storage facility design

8.3.1 Layout

The footprint area and layout of the TSF has been determined by the design criteria including:

- Life of mine: 8.4 years
- Total ore tonnage: 2,520,000 t
- Annual throughput: 300,000 t/a
- TSF volume requirement: 1,626,000 m^3
- Maximum rate of rise: 2 m/a
- Maximum height: 30 m
- Overall side slope: 1V:3H.

The key design properties are summarised (Table 8.14) and the layout of the TSF is shown on Drawing 160776-001 contained in Appendix 8D.

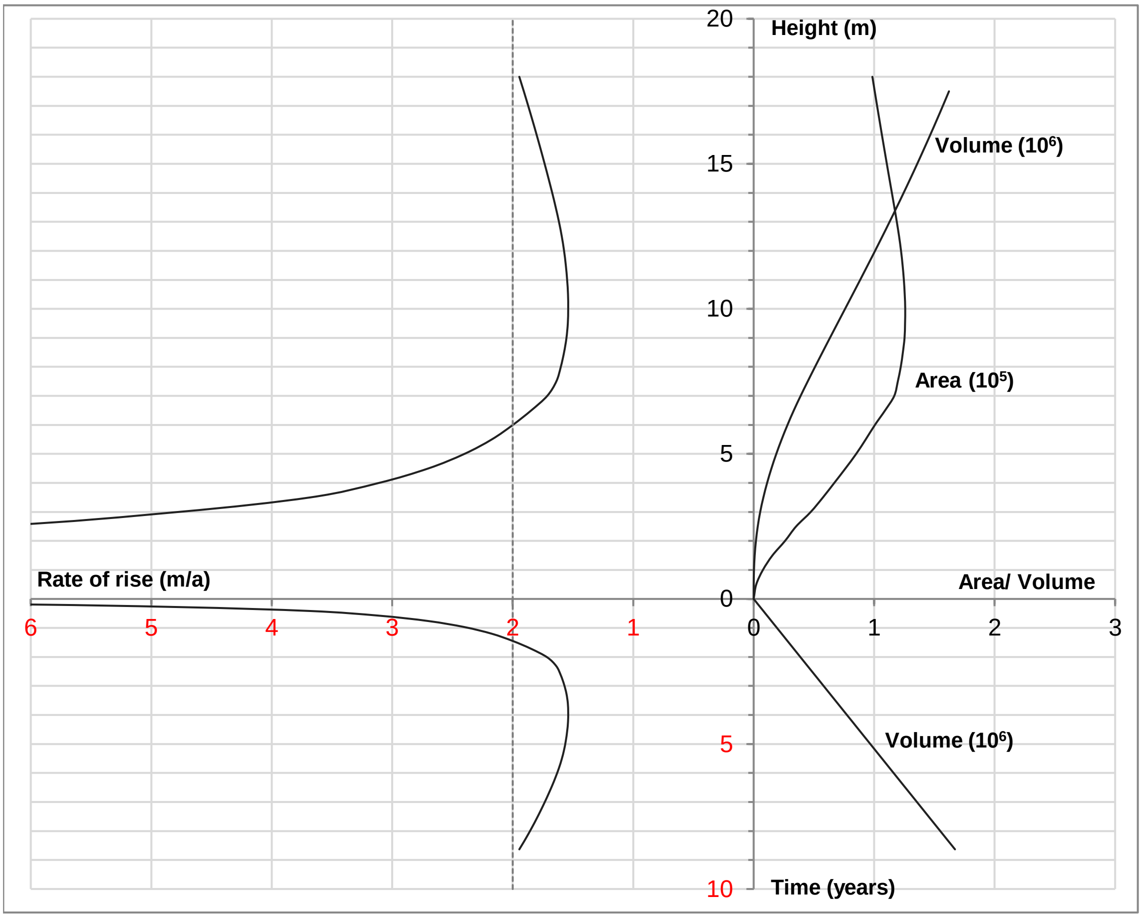
Table 8.14 TSF key design properties

Design property	Description
Starter embankment crest width	4 m
Starter embankment upstream side slope	1:2
Starter embankment downstream side slope	1:2
Starter wall maximum downstream height	8 m
Starter embankment fill volume	70,744m^3
Total TSF footprint to outer toe of starter embankment	15.7 ha
TSF basin area	14 ha
TSF overall outer side slope	1:3
TSF final elevation	564 mamsl
TSF maximum downstream	19 m
TSF final storage volume	1,670,762 m^3
TSF final storage capacity	2,589,681 t at 1.55 t/m^3
Rate of rise at starter embankment crest	1.71 m/year
Rate of rise at final height	1.95 m/year
Final tailings crest area	9.85 ha

8.3.2 Stage capacity curve

The Stage Capacity Curve (Figure 8.9) shows the relationship of the spatial properties of the TSF with time. Some of the design properties above are reflected. The minimum starter wall height was determined by the elevation when the rate of rise was below 2 m/a, after which it remains below 2 m/a for the remainder of the life of the facility.

Figure 8.9 TSF stage capacity curve



8.3.3 Side slope profile

The side slope profile of the TSF was influenced by the mechanical properties of the tailings, slope stability considerations and allowance for progressive rehabilitation. An overall side slope of 1:3 (18.4°) is selected with 1:2.3 (23.5°) side slopes, 7 m vertical lift height and 5 m wide step in bench width.

8.3.4 Depositional strategy

The selection of the method of deposition is influenced by the facility type, supernatant water management strategy, the physical and mechanical properties of the tailings material including:

- Particle size distribution of the tailings material
- Water content of slurry
- Particle segregation upon slurry deposition
- Settling, drainage, drying and consolidation characteristics.

The tailings will be delivered to the TSF via a 250 mm mild steel delivery pipeline and distributed around the perimeter of the TSF with a HDPE ring main of distribution pipe lines. The initial deposition will occur along the inner crest of the starter wall where a series of spigot pipeline will direct the tailings to the inner toe of the starter wall. The deposition of the initial tailings is directed at the filter drains situated close to starter wall, a process that is carefully controlled to avoid the blinding of the drains by the fine tailings material. Care is also taken not to cause of erosion of the granular filter materials that are already in place.

Once the filter drains have been covered and the tailings level has reached the level of the starter wall crest, the tailings will be deposited in day paddocks, where the fine tailing fraction will overflow onto the central beach. The coarse tailings fraction will be used to construct the day paddock walls and maintain the required freeboard. The deposition of tailings will move around the facility in cycles to ensure that the maximum rate of rise is never exceeded, adequate consolidation and strength gain of the tailings is allowed and that an appropriate freeboard is maintained. The day paddocks walls will be manually constructed and hand-packed by a series of suitably trained operators.

Night deposition will occur on the inside of the day paddocks and directly onto the tailings beach towards the penstock position. As the tailings flow toward the centre of the facility/position of drainage measures, particles will settle out forming the natural deposition profile or beach. Supernatant water will be collected around the penstock inlet from where it is decanted into the return water dams via the penstock system.

8.3.5 Design philosophy and preparatory works

The drawings contained in Appendix 8D show the details of the preparatory works for the TSF and associated infrastructure as discussed below.

Site clearance and preparation

The vegetation over the proposed TSF area will be removed and stockpiled if required for any timber.

Removal and storage of topsoil

From the geotechnical site investigation, the topsoil will be removed to a depth of 300 mm and stockpiled for reuse during the rehabilitation phase.

Bulk and restricted excavations

Bulk excavations include the cut required for the stormwater dam and the boxcut beneath the TSF starter embankments. The restricted excavations include drainage channels, diversion channels and trenches, cut-off keys, penstock outfall pipe trench, penstock intake bases and pipeline trenches. All excavated material will be temporarily stored and can be used in the construction of embankments, berms and walls. Diversion trenches and channels will be excavated with slide slopes of 1V:1.5H. Pipe trenches will have vertical side walls and will be backfilled thereafter.

Preparation of surfaces

Once the vegetation and topsoil has been removed, the in-situ sandy clay/black cotton soils covering the footprint of the TSF and water dams will require scarification and re-compaction before further preparatory work can be done. Due to the geochemical properties of the tailings material, the TSF and water dams will require a dual liner system comprising of a primary impermeable- and secondary low permeability liner. The in-situ soils, forming the secondary low permeability liner, will be ripped (scarified), moisture conditioned and recompacted to a minimum of 98% Proctor density to ensure the maximum allowed permeability is not exceeded. Allowance should be made for additional black cotton soils from the mining area, imported to cover areas where the cotton soils are not found within the TSF footprint. Pipe trenches, drainage trenches and channels will also require this compaction specification to prepare for the installation of pipework, concrete work, drainage and geosynthetic materials.

Construction of embankments

The material from the excavation activities will be reused in the construction of berms, embankments, catchment paddock walls and division walls. Additional material can be imported from the mine area as required for use in earthworks and construction. As the majority of the embankments will serve as an extension of the secondary low permeability liner, the material to be used in embankment construction will be moisture conditioned, and compacted in 300 mm layers to 98% Proctor Density. This applies to all subsurface fill required such as the starter embankment boxcut and cut-off key. The side slopes of the berms, embankments and walls will range between 1V:1.5H to 1V:2H

Containment barriers

Once all base preparation has been completed, which includes the 300 mm thick secondary low permeability clay layer discussed above, the primary HDPE barrier will be installed. The entire basin of the TSF and stormwater dam will be lined with the HDPE geomembrane. The TSF basin and upstream portion of the starter embankment will be lined with a 1.5 mm HDPE single side textured geomembrane. The textured side will be installed face down to assist in slip resistance along the slope of the TSF basin. In addition, the HDPE will be underlain by a geofabric (size Bidim A4 or similar approved) which will act as a cushion and protection layer for the geomembrane against the puncture due to any residual debris such as small stones (<20 mm) on the TSF basin. The filter drain trenches will also be lined with the geofabric and geomembrane.

Slurry delivery and distribution

The slurry will be delivered to the TSF via a 250 mm mild steel delivery pipeline. The distribution of the tailings around the facility will comprise of a 250 mm class PE63 PN 12.5 HDPE pressure rated pipeline. A series of 250 mm pinch and knife gate valves will be installed along the distribution line to control the deposition cycle/sequence.

Toe drain and blanket drain

The toe drain and blanket drains (20 m apart) will both consist of 500 mm deep, 3 m wide trapezoidal trenches located along the perimeter of the upstream toe of the starter embankment. A 160 mm perforated HDPE pipeline placed along the trench will be overlain by layer granular drainage material: 19 mm stone, 6 mm pea gravel and finally washed filter sand. 160 mm non-perforated pipes will lead from the perforated pipes at 150 m centres to divert any collected seepage water into the perimeter solution trenches.

Vertical penstock decant structure

The supernatant water will de decanted via a temporary and permanent vertical precast concrete intake structures. The temporary intermediate penstock will comprise of a single 510 mm diameter precast concrete penstock ring inlet, with the final permanent penstock consisting of a double 510 mm precast concrete penstock ring inlet. The penstock inlets will decant into a buried 450 mm spigot and socket precast concrete outfall pipeline leading from the base of the penstock towers to an energy dissipator downstream of the starter embankment wall. The outfall pipe will be placed in a trench and encased in reinforced concrete. The remainder of the trench will be backfilled with selected uncompacted fill.

Catchment paddocks and solution trenches

Catchment paddocks will be constructed along and around the perimeter of the starter embankments. The paddocks will be 10 m wide and approximately 50 m between division walls. The perimeter and division walls will be constructed from material excavated from within the paddocks or material imported from the mine area if required. The basin and walls of the catchment paddock will require the preparation and compaction as described above for the secondary low permeability liner.

The catchment paddocks will collect and contain any runoff from the side slopes of the TSF. A concrete lined solution trench will be constructed along the perimeter of the catchment paddock wall. Runoff from normal rainfall events will be left in the paddocks to evaporate. In the event of a 24-hour storm with a recurrence interval greater than 100 years, runoff will be allowed to overflow the catchment paddock walls and into the solution trenches.

Return water and stormwater management

The general TSF water management strategy is concerned with:

- The separation of clean and polluted water which includes runoff and seepage from the TSF
- Minimising losses and maximising return water.

All water conveyance structures such as channels and trenches have been design to accommodate flow as a result of the 1:50-year, one-hour duration storm with a depth of 74.8 mm. All essential containment facilities such as the stormwater dam is sized to contain the runoff from the TSF from a 1:100-year, 24-hour rainfall event with a depth of 147.6 mm.

Return water strategy

Supernatant water from the deposited tailings and rainfall falling on the tailings surface will collect in the supernatant pool, which is controlled by the pool wall (see drawing 160776-001). The collected water will be drained through the penstock inlets. The inlets are sized to drain the water collected from a 1:100-year, 24-hour rainfall event within a period of five days. The rate of decant from the TSF can be controlled by removing or adding the precast concrete penstock rings. Water decanted from the TSF will flow into an energy dissipator/stilling basin, from where it is diverted to a silt trap.

Any water seeping through the tailings material will collect in the toe and blanket drains. These drains will decant into the concrete lined solution trenches every 150 m. The seepage water and any water conveyed down the solution trenches will also collect in the energy dissipator/stilling basin and diverted via the silt trap.

The silt trap will comprise of a dual compartment constructed from reinforced concrete that will allow the use of one compartment while the other is being cleaned (silt removed manually). The silt trap overflow leads into the lined stormwater dam facility. An HDPE lined deep-set area within the basin of the stormwater dam will be used as the return water sump (see drawings 160776-001, 7). Under normal operating and climate conditions, water from the silt trap will collect in the return water sump from where it will be returned to the plant area by means of a floating barge pump. The location of the return water sump within the stormwater dam is due to space restrictions. The deep-set area has a small surface area that will reduce evaporation.

Stormwater strategy

In the event of large rainfall events, runoff and decant form the TSF will collect in the stormwater dam and return water sump. The floating barge pump will allow for the pumping of the return water independent of the water level within the stormwater dam. In the event of the design storm (1:00 year 24-hour rainfall event), runoff and decant water from the TSF will be collect in the stormwater dam within 5 days from the storm governed by the penstock discharge capacity. The stormwater dam is sized to contain all runoff and decant water from the TSF as a result of the 1:100 24-hour storm. Any additional water from a storm greater than the design storm that collects in the stormwater dam will be discharged via an emergency spillway. The spillway will end in an energy dissipator to avoid any erosion at the discharge point.

Clean stormwater from the areas upstream of the TSF area will be diverted around and away from the TSF through stormwater diversion trenches. This is to avoid the mixing of clean and potentially polluted water.

Design details and preparatory works

Based on the above design philosophy, Table 8.15 describes the key design and construction work details of the stormwater and return water facilities.

Table 8.15 Design and construction work details of the stormwater dam and return water sump

Design property	Description
Crest elevation	544.7 mamsl
Sump basin elevation	539.50 mamsl
Stormwater dam basin elevation	541.00 to 540.50 mamsl
Crest width	4 m
Inner and outer side slopes	1V:2.5H
Maximum wall height	1.3 m
Maximum excavation depth	±4 m
Full supply level elevation	543.90 mamsl
Storage capacity at full supply level	20,309 m^3
Cut volume required	20,685 m^3
Fill volume	1,455m^3
Total footprint	9,900 m^2

Similar to the TSF, the stormwater dam and return water sump will include the primary and secondary liners (low permeability clay and HDPE geomembrane).

8.3.6 Water balance

A monthly deterministic water balance model has been compiled for TSF. The input parameters to the water balance include the tailings characteristics, TSF layout and climate data. Table 8.16 is a summary of key inputs to the water balance.

Table 8.16 TSF water balance input parameters

Input	Value
Tailings production rate	300,000 t/a
Tailings specific gravity	3.12
Tailings in-situ dry density	1.55 t/m^3
In-situ void ratio	1.01
Slurry – solids percentage by mass	55%
Slurry – water percentage by mass	45%

Assumptions

The water balance has been simulated with the interaction between the TSF, the return water and the stormwater, the processing plant and applying conservative climatic conditions for conservative low flow conditions. Three distinct stages of the TSF were considered:

- Stage 1:
  - TSF newly commissioned
  - Maximum catchment area, HDPE covered
  - Minimal interstitial losses/retention, maximum water return.
- Stage 2:
  - The tailings surface covering entire basin (Year 5, month 3)
  - Smaller top catchment area as the lower end of the TSF has been raised inward (12.3 ha)
  - Increased interstitial losses /retention
  - Reduced water return.

- Stage 3:
  - The TSF close to the end of its life (Year 8, month 3)
  - Minimum catchment area (9.85 ha)
  - Minimum water return.

The TSF supernatant pool, the wet and dry beach sizes were conservatively determined from literature as a percentage of the total top surface as follows:

- Supernatant saturated pool: 20%
- Wet beach: 20%
- Dry beach: 60%.

The interstitial water in the voids is considered as lost to the overall water balance.

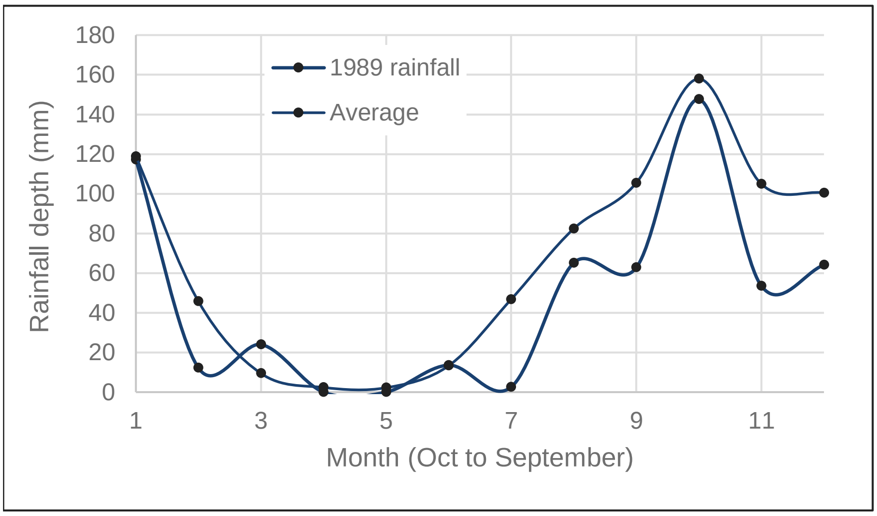
Rainfall data

The input parameters for the water balance models include rainfall and evaporation data received from EIT. The rainfall data has been statistically analysed to incorporate the seasonal variability and to compile a conservative water balance and to determine the required size of the raw water storage Dam. The statistical approach involved the final monthly rainfall by selecting the year with an annual rainfall that is closest to the 15th percentile annual rainfall of the available dataset of 103 years. The selected annual rainfall occurred in 1989 with a total rainfall of only 563 mm. The monthly rainfall data for 1989 and the 103-year average for the rainfall data is shown in the Table 8.17 and Figure 8.10 below.

Table 8.17 Rainfall data comparison for the Haveri District

Month	Average rainfall (mm)	1989 rainfall (mm)
October	119	117
November	46	12
December	9	24
January	2	0
February	2	0
March	13	14
April	47	3
May	82	65
June	105	63
July	158	148
August	105	54
September	100	64
Total	**790**	**563**

Figure 8.10 Rainfall data comparison for the Haveri District



A statistical approach was not applied to the evaporation as insufficient data was available. The following average evaporation figures (Table 8.18) reported by EIT were used.

Table 8.18 Evaporation data for the Haveri District

Month	Average evaporation data (mm)
October	114
November	110
December	102
January	110
February	134
March	161
April	171
May	166
June	114
July	89
August	92
September	99
Total	**1461**

A deterministic average monthly water balance has been developed for each of the three TSF stages described above. In line with the conservative approach methodology, Stage 1 was not considered, as the return water volumes are abnormally high due to the impermeable HDPE surface over the basin of the TSF.

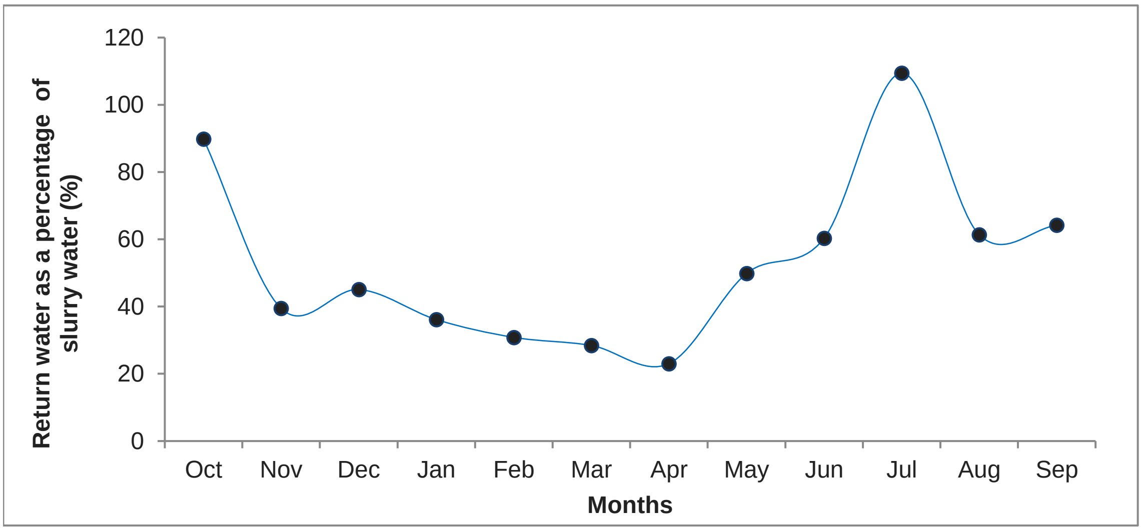
The monthly return volumes for Stages 2 and 3 are observed as being closely comparable, despite the reduction in catchment area of 2.45 ha during the last three years of production. The return volumes during the dry months increase with a decrease in catchment area (reduction in evaporation), while the wet months show a decrease in return volumes as the catchment area decreases. Stage 2 showed higher return volumes in the wet months than during Stage 3, while higher return volumes are expected toward Stage 3 in the dry months. Due to the small variance in annual return volumes, an average catchment area of 11.7 ha was used in the water balance calculations.

Table 8.19 shows the return water per month.

Table 8.19 Return water per month

Month	Return water (m^3)	Return water as a % of slurry water	Season
October	18,362	90	Wet
November	8,068	39	Dry
December	9,221	45	Dry
January	7,391	36	Dry
February	6,307	31	Dry
March	5,815	28	Dry
April	4,711	23	Dry
May	10,193	50	Wet
June	12,341	60	Wet
July	22,386	109	Wet
August	12,554	61	Wet
September	13,126	64	Wet
Annual average	10,873	53	Annual
Dry season average	6,919	34	Dry
Wet season average	14,827	72	Wet
Driest month	4,711	23	Dry
Wettest month	22,386	109	Wet

Figure 8.11 Percentage return water per month



A conservative approach was adopted in compiling a deterministic monthly average water balance. The early stages of the TSF will yield higher return water volumes during the wet season. A conservative annual average return water volume of approximately 130,480 m^3 per annum is expected from the TSF.

8.3.7 Slope stability analysis

A preliminary slope stability analysis was undertaken for various external slope profiles for the proposed Ganajur Gold Mine TSF at final height. The analysis incorporated international best practice guidelines, accepted slope stability guidelines, standards and recommendations from India, South Africa and Australia.

At the time of the slope stability analysis, some input parameters such as the shear strength properties of the embankment and tailings material were initially unknown. The preliminary analysis included a range of each input parameter based on past experience and research. The input parameters include effective internal friction angle and cohesion, including intermediate slope angles, bench widths and lift heights. This approach incorporates the inherent variability of the embankment and tailings material properties, including the geometry of the facility.

A minimum factor of safety (FoS) of 1.3 is recommended by the Indian Bureau of Mines (Government of India, Ministry of Mines, 1995). The South African Chamber of Mines has made recommendations regarding the minimum slope stability for tailings dams as follows:

- A minimum FoS of 1.3 for a regularly monitored facility
- A minimum FoS of 1.5 for a facility post-closure (i.e. an abandoned facility).

The results produced from the analysis include factors of safety for small incipient localised slope failures to full deep seated slope failures. The small localised failures could progress to larger failures and ultimately catastrophic embankment failures should it not be repaired. Changes in moisture content, large rainfall events and seismic activity can cause instability and subsequent failure. This analysis was focused on determining the lowest factors of safety for a deep-seated embankment failure.

The following input parameters were used in the slope stability analysis:

Table 8.20 Tailings and embankment material shear strength properties

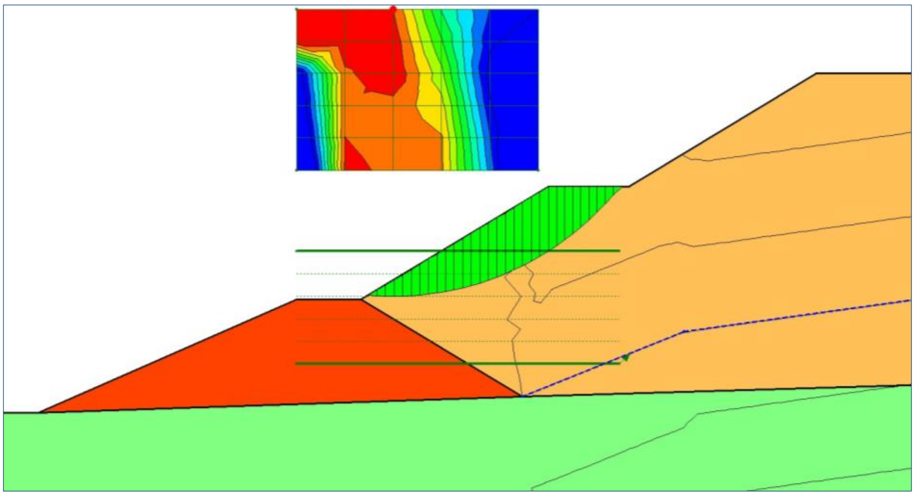
Input parameter	Value
Tailings unit weight	15.2 kN/m^3
Tailings effective friction angle	36°
Tailings effective cohesion	0 kPa
Embankment unit weight	18.2 kN/m^3
Embankment material effective friction angle	32° to 36°
Embankment material effective cohesion	7 kPa to 12 kPa

The final and suitable TSF slope geometry was selected based on the preliminary analysis and further analysis with the actual material parameters. The final TSF slope geometry was selected as follows:

- Intermediate slope angle: 23.5° (1V:2.5H)
- Bench width: 5 m
- Lift height: 7 m
- Overall slope angle: 18.4° (1V:3H).

The slope analysis on the final slope geometry and shear strength properties yielded a range of values ranging from FoS of 1.35 for small localised slope failures to a FoS of 1.66. The failure is shown in Figure 8.12.

Figure 8.12 TSF slope failure mechanism



The selected geometry is therefore considered adequate from a stability and safety perspective, and also suitable for the concurrent rehabilitation of the side slopes.

8.3.8 Rehabilitation and closure

The objectives of the rehabilitation and closure activities of the TSF are to reduce the formation of acidic conditions as a result of the geochemical nature of the tailings and protect the downstream and surrounding environment. Some of these activities will be undertaken during the operational phase of the project, whereas other will be performed during the decommissioning and closure phases.

In general, the activities are concerned with:

- Reduction of oxygen ingress into the TSF thereby mitigating the formation of acidic conditions
- Improvement of the quality of residual seepage water emanating from the facility post closure
- Reduction of wind and water erosion and transport of tailings material into downstream and surrounding water courses
- Reducing the extent, capital requirements and duration of rehabilitation activities required during the rehabilitation and closure activities
- Reduction is post-closure liabilities
- Addressing long term (legacy) and reputational risk issues
- Improvement of the post-mining environmental conditions and visual impact.

Tailings storage facility and associated infrastructure

The following rehabilitation activities will be undertaken concurrently with the deposition of tailings and operational activities associated with the TSF:

- Following the completion of each side slope and bench (vertical rise of 7 m), a clay cap will be installed on the side slope forming a low permeability liner, preventing the ingress of oxygen into the tailings material
- A loamy typically topsoil layer will be installed over the clay cap for the establishment of vegetation against the side slope and along the bench
- Construction of small paddocks along the edges of the benches to reduce the runoff velocity from the side slopes
- Installation of paddock drains to drain any collected water in the bench paddocks into the concrete lined solution trenches.

Once the final tailings have been deposited, the final bench and side-slope can be rehabilitated according to the process described above. The rehabilitation of the top surface of the TSF can then commence which includes the following activities:

- Removal of delivery and distribution pipelines, valves and associated hardware
- Draining of final supernatant water via the final penstock inlet
- Capping and sealing of the penstock inlets and outfall pipe
- Installation of a waste rock layer, commencing at the perimeter of the TSF top surface and progressing towards the centre
- Installation of clay cap/low permeability liner
- Placement of loamy soil layer for the establishment of vegetation
- Formation of paddocks or compartments to contain any precipitation on top of the TSF, allowing collected water to evaporate (evaporation is greater than rainfall in this region).

The perimeter paddocks at the base of the TSF can also be covered with a layer of loamy soil and vegetation established.

Stormwater dam and return water sump

The decommissioning and closure activities of the stormwater dam and return water sump will include:

- Removal of pipeline materials, valves, pumps and associated hardware once all water ceases to drain into the water dam
- The remainder of the materials such as HDPE geomembrane, concrete works, spillway and dissipater will remain as it will continue to be in use post closure for a period to be determined.

8.3.9 Post-closure and monitoring

Following the completion of the rehabilitation and closure program, measures will be put in place to monitor the effectiveness of the rehabilitation and closure measures. The period of monitoring and possible remedial or aftercare will be confirmed once the final tailings geochemical results are released and analysed. A minimum period of five years is recommended. The period can also be extended based on the monitoring results for the initial five years.

The requirement for the monitoring and remedial period is as a result of the geochemical properties of the tailings and long term acid production. Another objective of the monitoring assessment of the stability of the TSF post closure. The silt trap and stormwater dam will be kept in place for the duration of the monitoring period, as water may have to be collected and treated, pending water quality monitoring during the operational and post-closure phases. The activities to be undertaken during the post-closure phase will include:

- Monitoring, sampling and testing of collected surface water from the areas such as the catchment paddocks, bench paddocks, the top surface of the TSF, the stormwater dam and silt trap
- Monitoring, sampling and testing of residual seepage from the toe and blanket drain outlets
- Monitoring and measurement of the phreatic surface in the TSF
- General monitoring of effectiveness of measures implemented during the rehabilitation- and closure phases
- Groundwater monitoring through boreholes that will be installed around the perimeter of the TSF.

In the event collected water quality does not fall within the Indian discharge standards, the water will require treatment before any discharge. The treatment will be required until such time that the collected water adheres to the Indian discharge water quality standards.

8.4 Capital estimate

8.4.1 Capital cost

The Schedule of Quantities for the TSF has been complied by Prime Resources and the capital cost determined to an accuracy of ±10%. The pricing and rates have been obtained from in-county contractors in India and compared to similar rates and prices for similar sized gold operations in Southern Africa. The final costs are presented in real terms base date 1 April 2017 and do not account for inflation or escalation.

Capital costs pertaining to the construction of the TSF, stormwater dam and associated infrastructure have been defined as all costs related to construction, before the production of the first tonne of ore. The capital costs predominantly include earthworks, the supply and installation of geosynthetic and pipeline materials and the installation of concrete works. Capital costs therefore comprise:

- The cost of the mobilisation of the relevant contractor (earthworks, geosynthetics- and pipeline supplier and installer, concrete works supplier and installer)
- The cost of any infrastructure related to the earthworks, geosynthetic- and pipeline works and concrete works.

A summary of the capital costs is given in Table 8.21.

Table 8.21 Capital cost estimate for the TSF, stormwater dam and associated infrastructure

Description	Capital cost (US$)
Earthworks	$689,612
Drainage	$983,808
Pipe works	$202,258
Concrete works	$606,059
Miscellaneous items	$9,394
Subtotal	**$2,491,131**
Preliminary and general (5%)	$124,557
Contingencies and unmeasured items (5%)	$124,557
TOTAL	**$2,740,245**

The capital cost for the TSF, stormwater dam and associated infrastructure **excludes** the following components and activities:

- All electrical and mechanic components
- Pump stations, return water and slurry delivery pipelines (subject to the battery limits)
- Any electric, electronic, motorised or mechanical equipment including instrumentation.

There are no phased construction activities planned for the TSF, stormwater dam and associated infrastructure. The capital cost summary is therefore indicative of the total capital expenditure at the onset of construction.

8.4.2 Operating costs

The operating costs related to the TSF, stormwater dam and associated infrastructure are primarily related the maintenance, monitoring and remedial work where required. The following operational costs apply to the TSF:

- Continual raising of the TSF perimeter containment walls from deposited tailings
- Periodic raising of the slurry distribution line
- Operation of the penstock inlet structures (supply and install precast penstock rings)
- Maintenance of slurry pipelines and valves (periodic replacement of pipeline components)

- General maintenance including the cleaning of trenches and silt trap.

An annual operational cost of US$88,000 per annum is estimated and includes materials, pipes, penstock rings, consumables, use of machinery and labour.

Rehabilitation and closure cost

The rehabilitation activities will be undertaken partly during the operational phase and after the decommissioning of the TSF as described in Section 8.3.8.

The rehabilitation cost related to the continuous capping of the side slopes, benches with clay, topsoil and vegetation during the operational phase, including the placement of topsoil and vegetation over the catchment paddocks, is estimated at US$37,500 over the life of the facility.

The final rehabilitation and closure activities of the TSF and stormwater dam, which includes the capping of the top surface of the TSF, removal of pipelines and hardware, are expected to have a duration of one year. During this year, the decommissioning, closure and rehabilitation costs are estimated at US$181,000.

The post closure and monitoring costs will be determined once monitoring requirements have been confirmed.

8.5 Conclusions

The Ganajur Gold Project infrastructure will include a TSF, a return water dam and storm dam, and other surface water management measures.

A geochemical assessment of the tailings material was undertaken and the tailings have been classified as potentially acid forming and with residual traces of arsenic despite the stabilisation step in the process plant. Due of the geochemical classification, the selection of a suitable deposition and construction methodology required an analysis of the design criteria of both an upstream and downstream facility. A risk-based approach was used in selecting a preferred construction method, taking into account aspects such as oxidation rates and other mitigation measures, drainage and long term environmental impacts. The key criteria for the selection of the TSF construction and operation methodologies included a cost-effective solution, environmentally acceptable practice, maximum water conservation and minimum land use. A final recommendation and design option was made in favour of an upstream facility based on a smaller footprint, suitable pollution mitigation solutions and maximising water recovery.

Geo-mechanical laboratory testing of the tailings material was undertaken to determine its behaviour under conditions expected during the life of the TSF. These include density, permeability, shear strength and consolidation and were used in the design of the TSF.

A geotechnical investigation was undertaken to determine the suitability of the in-situ soils for its intended use as low a permeability liner and the construction of containment embankments. The results show that the in-situ soils have suited mechanical properties for embankment construction, but require additional compaction and permeability testing.

The TSF has been positioned adjacent to and downstream of the processing plant. The site selection was influenced by the required storage capacity and footprint, construction and development methods, local structural geology and topography, land ownership, rehabilitation requirements and existing significant surface infrastructures and features.

The TSF is designed to store a total dry tailings tonnage of 2,589,681 t over the 8.4-year life of mine requiring a volume capacity of 1.63 Mm^3 and footprint of 15.7 ha. The facility reaches maximum height of 19 m at a final rate of rise of 1.95 m per year and overall slope of 1V:3H.

The basin of the TSF will be lined with a low permeability clay layer, overlain with a geotextile and an HDPE geomembrane. Tailings will be deposited via a ring main distribution system. Drainage of supernatant water and rainfall will be via vertical penstock intake structures. Seepage water will be drained thought toe and blanket drains positioned along the basin of the TSF.

Decant and seepage water will be collected in a return water dam for reuse in the process water. Stormwater runoff from the side slopes and the top surface of the TSF will collect in the perimeter catchment paddocks from where it will drain into the stormwater or event dam.

The project site is located in a low risk seismic zone. A slope stability analysis, which incorporated tailings and soils mechanical properties and the TSF geometric profile, indicated a suitable FoS against major slope failure.

A monthly deterministic water balance model has been compiled for the TSF, return water and stormwater dams. The input parameters to the water balance include the tailings characteristics, TSF layout and climate data. The rainfall data has been statistically analysed to incorporate the seasonal variability and to compile a conservative water balance. The annual return water volume was estimated at 130,476 m^3, approximately 53% of the slurry water deposited on the TSF per annum will return to the process plant.

The side slopes of the TSF will be progressively rehabilitated with a clay and topsoil layer and vegetation. Once the TSF has been decommissioned, the top surface of the TSF will also be capped with a rock, clay and topsoil layer, and vegetated.

Capital costs associated with the construction of the TSF, stormwater dam and associated infrastructure have been defined as all costs related to construction, before the production of the first tonne of ore. The capital costs predominantly include earthworks, the supply and installation of geosynthetic and pipeline materials and the installation of concrete works and amounts to **$2,740,245**, and includes the cost of the mobilisation of the relevant contractors and a contingency of 5%.

The operating costs related to the TSF, stormwater dam and associated infrastructure are primarily related the maintenance, monitoring and remedial work which is estimated at US$88,000 per annum.

Progressive rehabilitation costs are estimated at US$37,500 per annum during the operational phase and a final rehabilitation cost of US$181,000 for the final capping of the TSF.

8.6 Recommendations

Long term kinetic testing is currently being undertaken on the tailings material to determine the long term intrinsic oxidisation rate of the tailings material. The final results (expected in June 2017) should be analysed together with other design criteria to confirm the suitability of an upstream facility.

The geotechnical investigation has shown that the surface soils comprise of medium to high clay content. The laboratory results have not conclusively shown that the soils are suitable for use as a primary low permeability liner. Additional soil testing is required during the detailed engineering stage where the focus should be placed on sampling and testing of the cotton soils found over the project site.

It is recommended that during the detailed engineering phase additional slope stability be undertaken and including seepage and the interface between the geomembrane and the in-situ soils.

The compilation of a construction and operations manual during the detailed engineering phase is required, specifically on drainage and seepage water management.

9 PROJECT INFRASTRUCTURE

9.1 Non-plant infrastructure

As part of the FS, CPC Project Design Pty Ltd (CPC) has incorporated infrastructure into the overall project design. The areas covered include:

- Roads both within the project area and access to the project site
- Office buildings
- Gatehouses
- Laboratory and sampling areas
- Workshops and maintenance facilities
- Power supply and distribution
- Communications and computer network
- Water supply and storage infrastructure
- Stream diversion channel.

9.1.1 Layout

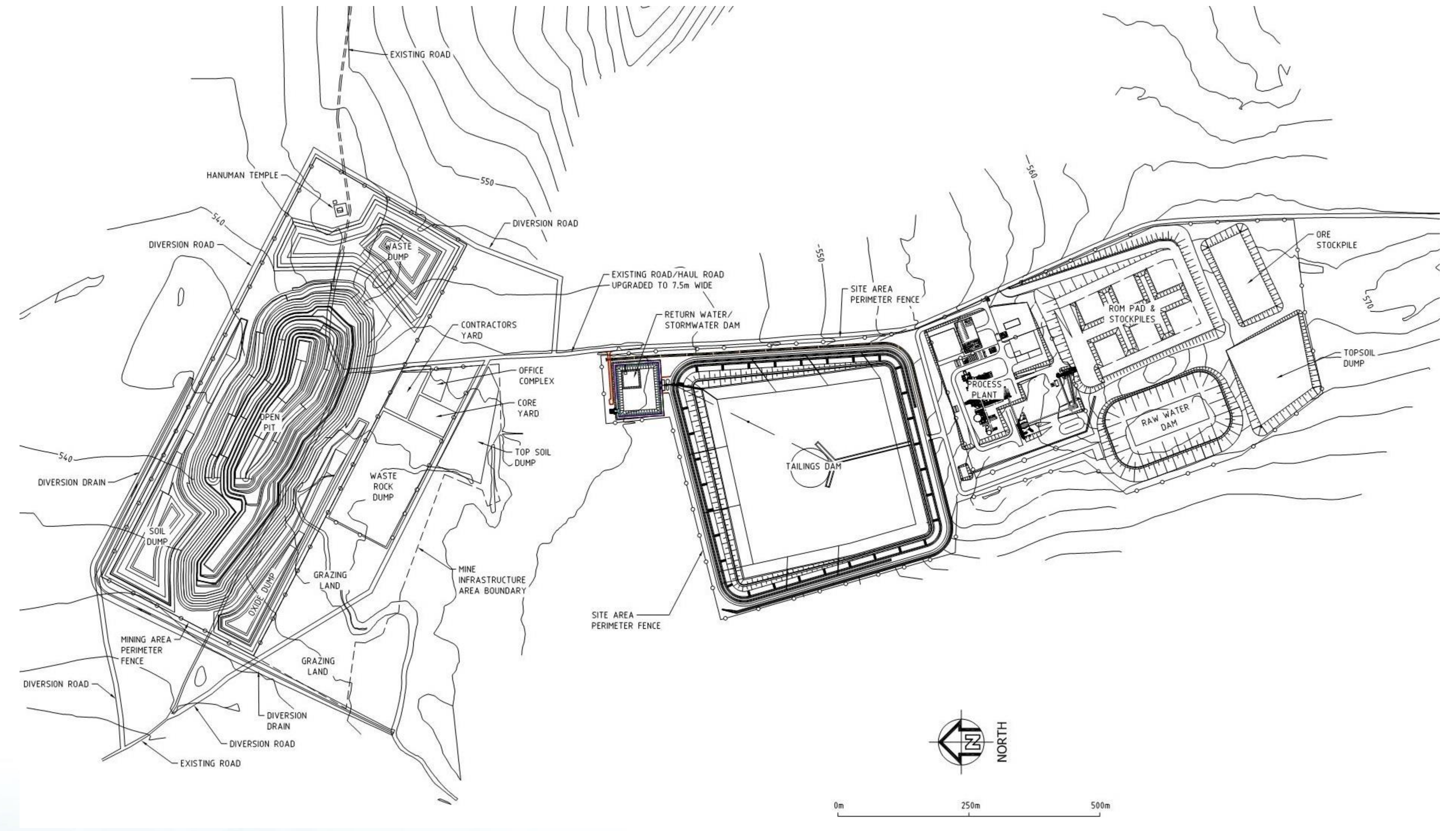
A proposed layout of the project site and associated infrastructure is shown in Figure 9.1. The layout shows the location of the raw water dam, ROM pad and ore stockpiles, processing facilities, roads and buildings within the project boundaries.

The following sections discuss in more detail the infrastructure requirements as per the layout. Further details on the processing facilities can be found in Chapter 7 (Process Plant and Description).

Security fencing around both the process plant site and mine area has been included to ensure only authorised personnel have access to the sites. Entry and exit will be through gatehouses for each location which will be monitored by security personnel. Additionally, security fencing has been included around the entire plant area to keep unauthorised people away from the ROM pad, water storage dam and TSF. Access will be to the northeast along the haul road and to the west at the gatehouse.

An area of 50 m x 50 m is allocated for the high voltage (HV) switchyard in the vicinity of the process plant. Further details of the power supply and switchyard can be found in Section 9.1.3.

Figure 9.1 Overall plant site layout



9.1.2 Project site roads

Access to the site will be via an existing public road to the east of area. The existing paved road is only 3.5 m wide and is accessed from Haveri up to Devagiri-Karajgi road.

The existing road will be upgraded to a 7.5 m wide paved road as it will be used as a haul road from the pit to the ROM pad.

A traffic management plan will be put in place to allow both mining and public use.

The existing road currently runs through the proposed mining pit. The road will be diverted to the north around the pit and reconnected to the eastern site access road. Access to the Hanuman Temple will be provided from this diversion road.

Another road will be constructed to divert the road to the south of the pit and reconnect to the eastern site access road.

Refer to the overall plant site layout 7056-300-LG-011 in Appendix 9A for routing of the proposed roads.

9.1.3 Power supply and distribution

National grid plant power supply

The Karnataka Power Transmission Corporation Limited (KPTCL) is the sole electricity transmission and distribution company in the state of Karnataka.

The electrical power for the project will be supplied from a take-off bay at a nearby 110 kV KPTCL substation located approximately 7.5 km from the site. The exact location hasn't been finalised, however there are ongoing discussions to locate the take-off closer to the site.

An 8 MVA substation located in close proximity to the site will be constructed to transform the 110 kV voltage supply to 11 kV. The substation will be constructed to meet the specifications and requirements of KPTCL.

The exact route of the 7.5 km 110 kV transmission line will need to be finalised in consultation with KPTCL and local farmers to an agreed right of way (ROW).

The requirements for the project power supply have been estimated as follows:

- 11 kV
- 5 MVA maximum demand
- Largest motor is 1000 kW limited to four times full load current (FLC) on start
- 0.8 power factor (PF) lagging
- 20 kA 3-phase fault level.

These values will need to be confirmed once negotiations with KPTCL have concluded.

Plant power supply

A short length (100 m) of 11 kV cable has been allowed in the project costing to carry electricity from the termination of the service line into the plant's 11 kV switchboard (300SB001). From 300SB001 electrical power is distributed to the following substations/equipment:

- 300RMU001 Mine services power distribution
- 310TF001 Crushing plant
- 320TF001 Milling/Flotation plant

- 320ML001 Primary ball mill
- 340TF001 Leaching plant.

These substations contain the distribution boards and motor control centres (MCCs) that service the power requirements for the respective areas.

Project electrical load

Based on the loads identified in the mechanical equipment list, power system models were developed for each of the substations. Table 9.1 outlines the maximum demand figures for each substation.

Table 9.1 Maximum power demands

Substation	Substation/ Equipment no.	Maximum demand (kVA)	Maximum demand (kW)
Mine services	300RMU001	250	214
Crushing substation	310TF001	810	713
Milling/Flotation substation	320TF001	1,114	1,084
Primary ball mill	320ML001	1,261	1,173
Leaching/Emergency substation	340TF001	1,088	931
River pumps	680MCC01	138	120
Site maximum demand*[1]		**4,625**	**4,145**

**Excludes river pumps which are powered by their own diesel generator*

[1]*Includes cable and transformer losses*

The largest electrical load within the plant is the primary ball mill (320ML001) fitted with a 1,000 kW motor. In order to reduce the demand on starting, the primary ball mill has been specified with a liquid resistance starter. This will limit the starting current draw to 4 times full load current, minimising the voltage dip on the 11 kV bus (300SB001) to 8% and short term overload on the 110/11 kV 8 MVA substation transformer to 16%.

The regrind mill is to be equipped with a soft starter designed to limit the in-rush current on starting to four times full load current. This will limit the voltage dip on the 320MCC001 415 V bus, to less than 15% on start-up of the regrind mill.

Main substation

The main substation (300SB001) has been designed to be an 11 kV switchboard rated as follows:

- Three-phase
- 1000 A
- 25 kA for one second.

The main switchboard will be fitted with the following fixed pattern circuits:

- 1 x incomer
- 1 x mine services distribution feeder
- 3 x 1.5 MVA transformer feeders
- 1 x liquid resistance motor starter (1000 kW).

Refer to the single line diagram (SLD) 7056-492-ED-001 in Appendix 9C for further details.

The incomer will be equipped with a protection relay capable of instantaneous and time graded overcurrent and earth fault protection.

The mine services distribution feeder will be equipped with a protection relay capable of instantaneous and time graded overcurrent and 2 A sensitive earth fault protection.

The three transformer feeders will be equipped with fail safe communications relays capable of monitoring transformer protection as well as protection relays capable of instantaneous and time graded overcurrent and earth fault protection.

The feeder circuit breakers will be fitted with earth switches interlocked with the circuit breaker.

Plant power distribution

Electrical power will be radially distributed around the plant at 11 kV via buried cables within the plant boundaries, reducing the opportunity for mobile equipment to clash with exposed power lines thereby increasing safety.

The main processing plant will utilise 11/0.415 kV transformers as indicated in Table 9.2.

Table 9.2 Transformer ratings

Equipment no.	Substation	Transformer rating (kVA)
310TF001	Crushing substation	1,500
320TF001	Milling/flotation substation	1,500
340TF001	Leach/emergency substation	1,500
378TF001	Tailings decant substation	50
590TF001	Mine services distribution board	500

The 11 kV power distribution system has been designed to be direct earthed utilising cable screens and a supplementary earth conductor. The supplementary earth conductor will also facilitate maintenance checks on the condition of cable screens by facilitating earth continuity testing.

The mine services feeder supplies a small ring main unit (300RMU001) at the tailings decant substation. The ring main unit has been designed to be equipped with self-powered vamp protection relays offering instantaneous and inverse time overcurrent and earth fault protection for the tailings dam decant substation and the mining contractor’s yard distribution board. The ring main unit is considered to represent the extremity of the plant area earth grid in relation to the mine area earth grid.

The mining contractor is responsible for supplying the mining contractor’s yard distribution board and earthing grid requirements.

The 1.5 MVA transformers have been designed to be equipped with pressure relief valves, oil temperature indication and winding temperature indication. These devices will be equipped with alarm and trip contacts, with the alarm signals communicated to the PLC/SCADA system for monitoring, and the trip signals communicated to the respective transformer feeders to facilitate tripping of the feeder in the event of any of the following transformer conditions:

- Pressure relief valve operation
- Oil temperature excessive
- Winding temperature excessive.

The smaller 378TF001 and 590TF001 transformers have been designed to rely on the respective 11 kV feeder overcurrent and earth fault settings for protection.

The river pumps operate for a few months each year, and as a result they are designed to be powered by a 500 kVA 415 V packaged generator (680GN001).

Emergency power is provided via an emergency MCC (340MCC002) and is equipped with a changeover switch enabling power to be sourced from 340MCC001 under normal operations and a 500 kVA packaged generator (340GN001) under loss of utility supply. The emergency generator is designed to supply only those loads identified in the mechanical equipment list as requiring emergency power such as tank agitators and critical buildings. The emergency generator has not been sized to run the entire plant.

Motor control centres

In general, plant final sub-circuits and sub-mains will be sourced from the nearest MCC. MCCs and associated fault levels are outlined in Table 9.3.

Table 9.3 MCC single line diagrams

Equipment no.	MCC	Transformer	Three-phase fault level
378MCC001	Tailings Decant MCC	378TF001	6 kA
310MCC001	Crusher MCC	310TF001	40 kA
320MCC001	Milling/flotation MCC	320TF001	40 kA
340MCC001	Leach MCC	340TF001	40 kA
340MCC002	Emergency MCC	340GN001	40 kA
680MCC001	River pumps MCC	680GN001	20 kA

Submains in the mining contractor’s yard and mine will be sourced from the mine contractor’s yard distribution board (590DB001).

Drives have been designed to be controlled locally via a local start stop station in close proximity to the drive and via PLC for SCADA control. Local remote selection will be carried out via the SCADA/ PLC system.

The river pumps have been designed to be local control only without SCADA monitoring.

Typical MCC switch-room layouts drawings are listed in Table 9.4 and can be found in Appendix 9B.

Table 9.4 Typical MCC layout drawings

Equipment no.	Substation	Drawing
550BG007	Main switch-room layout	7056-492-EG-006
550BG007	Main switch-room elevation	7056-492-EG-007
550BG006	Crusher switch-room layout	7056-492-EG-009

Cabling

The process plant will be cabled utilising a combination of PVC and XLPE insulated cables as outlined in Table 9.5.

Table 9.5 Cable insulation

Size	Voltage rating	Insulation
<16 sq mm	0.415 kV	PVC/PVC
>16 sq mm	0.415 kV	XLPE/PVC
	11 kV	XLPE/PVC

All low voltage cables will be unarmoured and run above ground in heavy duty galvanised cable ladder with an allowance of 10 m of open heavy duty galvanised conduit per drive.

11 kV distribution cables will generally be direct buried armoured cables.

24 V DC has been specified for control voltages with 8 core control cables specified for local control stations and all analogue signals run in twisted pair cables to area specific instrument marshalling boxes.

Earthing and lightning protection

Earthing will generally be direct effectively earthed at the substation with multiple earth neutral (MEN) points in each of the MCC's supplied directly via a transformer.

Each transformer will be equipped with a local earthbar for equipotential bonding purposes.

Each substation will be equipped with a main earthbar configured with a minimum of two connections to the substation earth grid in order to facilitate regular maintenance and testing.

All metal structure, reinforcing, cable ladders, mechanical equipment will be equipotentially bonded back to the station main earthbar.

All 11 kV distribution cables will be provided with supplementary earth cables to facilitate earth fault current withstand and testing of cable screens and armour for electrical continuity.

All tall structures and buildings exposed to lightning will be equipped with lightning masts designed to minimise lightning associated hazards. All tall structures will be equipped with down conductors, earth grids and grading rings.

All lightning earths shall be bonded to the main power earth to ensure uniform potential rise in the event of a lightning strike.

Lighting and small power

Allowance has been made for the following small power and lighting:

- Emergency lighting above stairways in plant areas
- Emergency lighting above substation access stairs
- Emergency lighting above building entrances
- Area floodlighting in plant areas
- Minimal area floodlighting around buildings
- Area floodlighting around transformer bays and substations
- Swivel poles for standalone area lighting
- 15 A weatherproof GPOs in plant areas
- 32 A 3 phase welding outlets in plant areas
- 32 A 3 phase welding outlet in substation vicinity
- Fire detection and warning system in each substation reporting back to SCADA via PLC.

Each substation shall be equipped with an area small light and power distribution board complete with photoelectric (PE) cell controlled lighting circuit and emergency lights test circuit.

9.1.4 Raw water supply and storage

Raw water supply

Raw water will be supplied from the Varada River with supply limited to the monsoon months of June through September when the water overflows the weir located at the Kolur-Kalasur barrage. Figure 9.2 and Figure 9.3 show the existing barrage at Kolur-Kalasur.

An intake well will be constructed at Kolur-Kalasur barrage that will allow for two submersible pumps to draw water from the river on a duty/ standby basis. A packaged generator will power the pumps. Both the generator and the pumps will be removed during the offseason. Figure 9.4 shows the design of the water extraction facility.

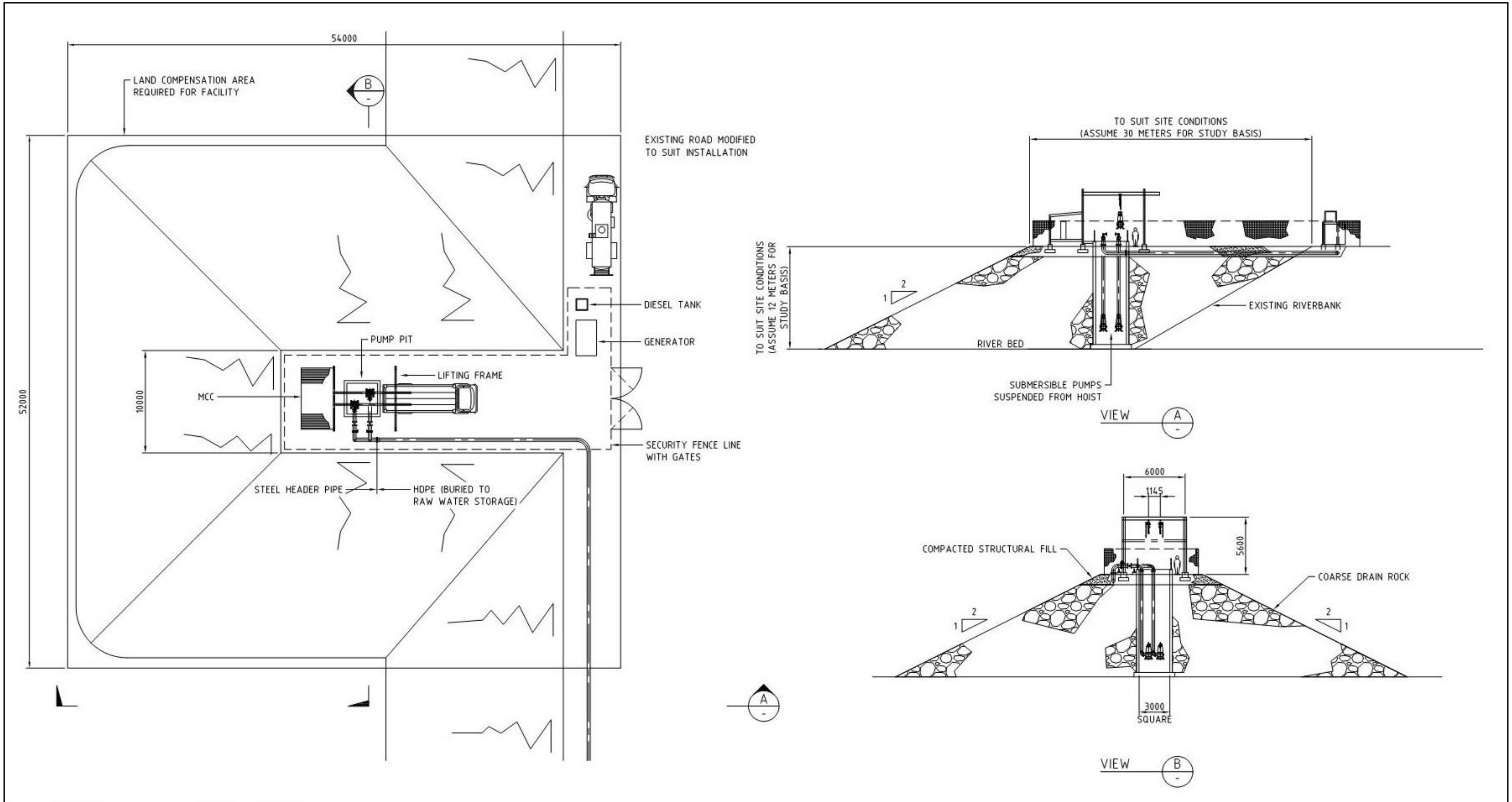
The entire area is enclosed by security fencing and will be monitored at all times by on-site security personnel when pumping water back to the raw water dam at the project site.

A 6.5 km buried pipeline will transfer raw water from the Kolur-Kalasur barrage to a raw water storage dam constructed on the plant site. The pipeline will be buried and will utilise easements next to existing roads. This design requires minimal government involvement and ensures that once the water is pumped it remains within Deccan Gold Mines Limited (DGML) control.

Figure 9.2 Kolur-Kalasur barrage

Figure 9.3 Kolur-Kalasur barrage

Figure 9.4 Water extraction facility at Kolur-Kalasur Barrage



Site water balance

CPC developed the overall site water balance for the project which included inputs from Prime Resources and Snowden to determine the raw water dam sizing.

Table 9.6 outlines the constant inputs used in the overall site water balance and Table 9.7 indicates the variable raw water inputs by month.

Table 9.6 Constant raw water inputs

Input	Value
Process plant raw water demand	13 m^3/h
Potable water	5 m^3/h
Mine area	10 m^3/d
Plant site dust suppression	10% of mine requirements
Process water	143 m^3/h
Internal process water recycle	124 m^3/h
Tailings discharge density	55% solids

Table 9.7 Variable raw water inputs by month

Month	Rainfall (mm)	Evaporation (mm)	Dust suppression (m^3/d)	TSF decant return (% of tailings water)
October	117	114	150	90
November	12	110	150	39
December	24	102	150	45
January	0	110	150	36
February	0	134	150	31
March	14	161	150	28
April	3	171	150	23
May	65	166	120	50
June	63	114	30	60
July	148	89	30	109
August	54	92	30	61
September	64	99	120	64
Yearly total/average	**564**	**1461**	**41,880 m^3/y**	**53**

Prime Resources provided the percentage of process water returned by month based on their TSF design. This was used to calculate decant water availability based on constant discharge of process tailings to the TSF. Based on this monthly amount of decant return water, requirements for process water use in the processing plant and internal process plant recycle streams, the amount of additional raw water make-up required was calculated.

In months with high decant return water there will be no raw water make-up requirement for process water in the plant. When the decant return water is in excess of the process plant needs, it will be stored and used in the following month.

There is an additional need for raw water in the processing plant where recycled process water is not suitable – this mainly includes gland water for pumps, carbon regeneration quench water, intensive leach reactor demand and flocculant makeup water.

Prime Resources also ran statistical analyse on the yearly rainfall data to determine the year that was closest to the 15th percentile in the 103-year dataset. The annual rainfall at the 15th percentile occurred in the year 1989 with a total of only 564 mm. This was the basis for rainfall in the overall site water balance.

Evaporation data was only available for a single year and this was used in both the TSF design and the overall site water balance. Decrease in the surface area of the raw water dam has been factored into the evaporation calculation.

Snowden provided an estimate of raw water required for dust suppression in the mine area. An additional 10% was added to this number to account for dust suppression requirements around the process plant site.

The potable water treatment plant requires 5 m^3/h of raw water feed. The potable water supplies both the plant and mine site personnel consumption as well as provides water for the elution circuit.

Process water and raw water flows within the processing facilities are from the CPC mass balance with flows normalized on an annual basis (100% availability).

Raw water dam design

The raw water dam will utilise the existing quarry in the southwest of the project site to minimise bulk earthworks and will be expanded to allow for sufficient storage capacity for one-year supply of raw water including a two-month contingency allowance.

The total raw water dam volume is approximately 300,000 m^3 which includes two months' contingency.

Design assumptions for the raw water dam include:

- The raw water dam is full at the start of operations.
- No water is required for the greenbelt vegetation. Vegetation will be planted during the monsoon season and plant types selected will be suitable to grow without further watering.
- The design side wall angle of the dam is 2:1 horizontal: vertical (26.5°) to minimise surface area for evaporation. The decreasing dam surface level has been utilised for determining monthly evaporation.
- The raw water dam will utilise the existing quarry and be extended to the east.
- The depth of the raw water dam is 20 m which includes the volume below normal ground level and above ground utilising built up embankments.
- A freeboard of 0.5 m above the maximum water level has been included. This surface area is used to calculate the catchment area for monthly rainfall.
- Pumping of water from the Varada River will occur over a four-month period each year to refill the raw water dam.
- No allowance has been made for camp water. The accommodation complex is located within town and the water supply will be sourced locally. Water for the office complex is allowed for in the mine infrastructure.
- Water contained within the ore is accounted for in the CPC water balance figures used (assumed at 3% on average).
- A contingency of two months of raw water storage is included in the overall volume. Rainfall and evaporation are included within the sizing calculations.

Based on the overall site water balance and the dam design, the monthly raw water requirements are shown in Table 9.8.

Table 9.8 Monthly raw water demand

Input	Value
October	17,414
November	19,637
December	20,216
January	27,111
February	24,884
March	28,080
April	28,209
May	20,818
June	14,700
July	11,274
August	13,743
September	17,303
Two months' contingency*	43,259
Total	**286,647**
Year 1 (including contingency)	**297,590**

Additional raw water is required in the first year to make up process water requirements during the first dry season when there is no excess tailings decant/process water supply from previous months of operation. The dam has been sized according to the required volume in the first year of operation.

9.1.5 Buildings and facilities

Table 9.9 lists the site buildings, size and use.

Table 9.9 List of site buildings

Building	Size
Administration building	480 m^2
Process plant office	110 m^2
Security office	110 m^2
Maintenance office	110 m^2
Workshop/Warehouse shed	540 m^2
Medical facility	135 m^2
Mining office	110 m^2
Site area gatehouse	12 m^2
Mining area gatehouse	12 m^2
Plant ablutions	36 m^2
Crushing area switch-room	55 m^2
Main (HV/LV) switch-room	108 m^2
Lunch room	36 m^2
Control room	12 m^2
Titration room	12 m^2
Laboratory/sample preparation building	325 m^2 / 66 m^2

Building construction

Layouts of the buildings proposed for the project can be found in Appendix 9B.

The buildings will be panelised with wall panels consisting of two cladding layers with a polyurethane substrate in between. The panels are tongue and groove providing weather tight seals between the panels.

The roof of the buildings incorporate pre-engineered steel trusses, purlins and sheeting.

All plumbing and electrical within the building are inclusive.

The buildings are delivered on site, flat packed in containers and are easy and quick to erect minimising construction time on site.

Laboratory/Sample preparation

The laboratory and sample preparation shed will be constructed by DGML and operated and maintained by Shiva Analyticals (India) Private Limited. The laboratory will include the following major equipment:

- Jaw and rolls crushers
- Rotary sample divider
- Sample pulverisers
- Drying oven
- Compressor and dust collector
- Fusion and cupellation furnaces
- Weighing balances, weighing and working benches
- Fume hoods, acid scrubber and neutralisation systems
- Pouring table with cast iron molds
- Atomic absorption spectrometer, ELTRA (LECO) carbon and sulphur analyser, etc.

9.1.6 Communications and computer network

Airtell was selected as the communications services provider for the project by DGML.

For the purposes of this project the communications services boundary has been assumed to be the communications services provider's fibre connections at the fibre optic breakout tray (FOBOT) in the administration building.

The project has allowed for firewall, hubs, switches and routers as required in the administration building to facilitate connection of ethernet and telephone services with the communications services provider and plant communications infrastructure.

Communications around the plant will be via optical fibre cable configured in a ring topology to enable a robust communications media with redundancy. Twelve core fibre cables have been specified enabling the fibre network to carry a variety of applications on dedicated fibres. They include:

- Business IT
- VoIP phones
- SCADA
- Security
- CCTV.

Refer to Appendix 9D for the overall site communications block diagram.

Communications internal to plant and building areas will be via suitable Cat 5 cabling. The plant will utilise ethernet as the communications protocol with ethernet switches allowed for in each building.

All communications equipment including FOBOTs will be mounted in 19 inch racks in each building.

Plant control will be via PLC mounted in a separate PLC panel in each substation. The PLC will be networked via a fibre optic ethernet communications ring configured as a self-healing ring. The PLC will be monitored via a SCADA network server in the central control room.

Office administration computers will be on a business IT network arranged in a ring configuration between buildings utilising the fibre optic network. Office computers internal to buildings will be connected to the communications rack via Cat 5 cabling in a radial configuration.

Telephone communications within the site will be via voice over internet protocol (VoIP).

Communications within plant areas will be via onsite personal radio communications.

9.1.7 Potable water treatment plant

A brackish water reverse osmosis water treatment plant will process raw water from the raw water dam for potable water use. The treated water will be stored in a tank with a 20-hour capacity to provide potable water to the office buildings, mine site and process area.

The potable water treatment plant has a design capacity of 5 m^3/h.

9.1.8 Fire protection

The plant site facilities will be protected with a pressurised fire protection system that comprises a fire water reserve, an electric driven jockey pump, an electric driven fire pump, and an emergency diesel driven fire pump. The fire water reserve will be contained in a dedicated portion of the raw water tank in the process area.

The firewater distribution system will consist of a dedicated buried firewater loop and hydrant system for the process, ancillary buildings and warehouse/workshop.

All areas will be provided with handheld fire extinguishers.

9.1.9 Sewage

Sewage will be handled in concrete septic tanks and leach drains. When the septic tanks are full, sewage will need be disposed legally by a contactor. The leach drains will be integrated within the greenbelt for fertilising.

Two septic tanks will be constructed. One for the site area infrastructure and the other for the mining area infrastructure.

9.1.10 Security

Security fencing around both the site area and mine area has been included to ensure only authorised personnel have access to the sites.

The site area fencing perimeter includes the ROM pad, water storage dam, TSF, process plant, administration office, medical facility, laboratory and sample preparation building, and the process plant.

Access will be via the eastern public/haul road through the main gatehouse monitored by security personnel. Entry into the site area will be monitored via CCTV camera and access card readers and will only be permitted to authorised personnel who possess an access card.

A second level of security fencing will be provided for the process plant. Entry into the process plant will be allowed to personnel with an access card with the required security clearance. The security building will have turnstiles controlled by the access card reader to allow entry/exit to the process plant and CCTV cameras for monitoring purposes.

The mining site area boundary includes the mining pit, waste dump and mining offices. Entry and exit will be through gatehouse and will be monitored by security personnel.

The site area and mining area are considered a low security risk and as such perimeter lighting has not been included.

The gold room will be a heavily secured building, with three CCTV cameras for monitoring. Limited access will be provided into the gold room to authorised personnel via the access card reader system.

The gold room will be a typically sheeted building with security meshing on the inside for added security. The gold bullion will be secured in a safe which will be located in a concrete vault room within the gold room.

9.1.11 Run of mine pad and ore stockpile

The ROM pad has been sized to stockpile 130,000 t of ore configured in six finger stockpiles, each 6 m high.

The six finger stockpiles will be used to blend the feed ore to the process plant.

The ROM pad will incorporate a skyway to allow trucks to directly tip ore to the dedicated stockpiles.

An area south of the ROM pad has been allocated to stockpile an additional 120,000 t of ore.

Both the ROM pad and ore stockpile will be 300 mm clay lined to prevent any seepage of acid mine drainage (AMD) solutions.

The ROM pad will be constructed in two stages. Stage 1 ROM pad will contain two ore stockpile fingers and will be built during the construction phase of the project and completed prior to ore processing.

The final stage ROM pad will be completed progressively over the first two years of ore processing using mine waste.

9.1.12 Stream diversion channel

The open pit is situated in a valley. The stream flows through the pit during the monsoon season. To minimise water ingress into the pit, a diversion channel will be constructed to direct the water around the western and northern side of the pit.

The channel will be sized for a 1:50-year rain event and be constructed with a 1:400 slope. The channel will be an unlined and predominately excavated structure.

The channel width at the two diversion roads crossing is approximately 25 m. Two concrete bridges have been allowed for the road crossing.

9.1.13 Accommodation camp

No allowance for an accommodation camp has been made. The construction workforce and the mining staff will be housed in nearby towns and villages such as Haveri, Ganajur and Karajgi.

9.2 Plant infrastructure geotechnical investigation

The following section was provided by Prime Resources environmental consultants.

9.2.1 Surface geotechnical investigation

A geotechnical investigation was undertaken to characterise the surface soils over the proposed Plant Infrastructure area. The investigation was jointly undertaken by Prime Resources and Sarathy Geotech & Engineering Services Pvt. Ltd (SGES).

Prime Resources conducted a preliminary site visit and geotechnical investigation with the SGES representatives, during which test pits were identified, excavated and profiled over the proposed Plant infrastructure area. Samples were taken for soil and geomechanical testing.

Once the preferred position of the plant infrastructure components had been selected and the position of key supporting infrastructure confirmed, Prime Resource selected a number of additional positions for geotechnical test pits and diamond core drillholes. The coordinates of the test pits and drill holes were then supplied to SGES for the Phase 2 site investigation.

9.2.2 Site geology

Proposed Ganajur gold processing plant area is part of Ranibennur Group of the late Archaean Dharwar-Shimoga (or the Shimoga) greenstone belt in the Western Dharwar Craton. The Shimoga belt contains innumerable parallel, bands of cherty iron formation that are folded at many places within a vast mass of greywacke. The iron formation towards west is magnetite bearing whereas the eastern part is of sulphide facies containing mainly pyrite and arsenopyrite. The sulphide facies iron formation is auriferous and the Dharwar-Shimoga schist belt has been known for ancient artisanal gold mining. The regional trend of bedding and foliation are parallel and vary from northwest to west-northwest with steep north-easterly dips.

The geology of the Ganajur-Karajgi cluster predominantly comprises of greywacke and inter bedded banded sulphidic chert (BIF). The general strike direction of the banded sulphidic chert varies between N40-60° west and dips at 35° to 50° towards the northeast. The litho-stratigraphy of the area is as follows:

- Dolerite dykes
- Quartz veins
- Basic intrusive (Gabbros and dolerite)
- Late Archaean Dharwar Super Group
- Chitradurga Group/Ranibennur formation Greywacke (Qz-chl-bi schist), sericite-chlorite schist/phyllite (“shale”)
- Banded sulphidic-magnetite-quartzite inter banded with narrow bands and lenses of felsic, intermediate and basic volcanic rocks and greywacke.

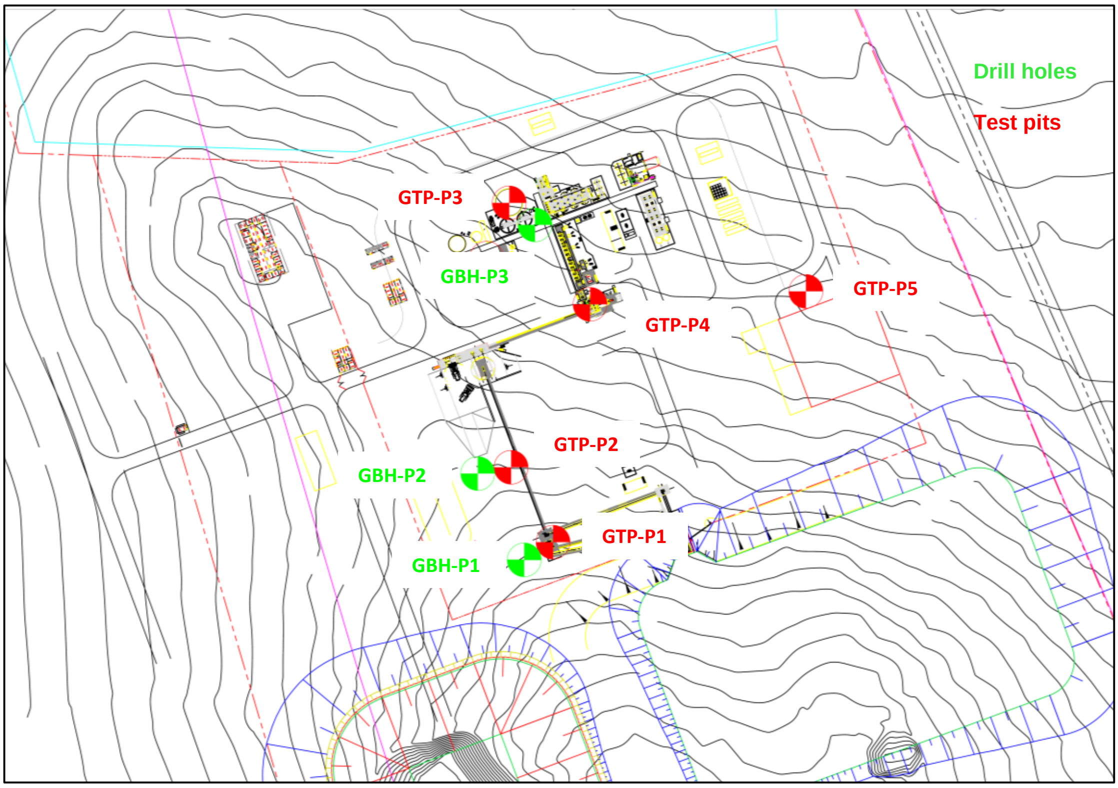
Gold mineralisation is confined to sulphide facies of banded iron formation, and the mineralisation is both syngenetic, strata bound and epigenetic. The gold mineralisation is associated with strong suphidation, chlorite-sericite and carbonate alteration along with minor biotite. The mineralised zones are characterised by the presence of significant sulphide minerals (10% to 12%) such as pyrite, chalcopyrite and arseno-pyrite. Positive correlation is noticed between gold values and amount of sulphides.

The Ganajur processing plant area is underlain predominantly by schistose greywacke traversed by narrow discontinuous quartz veins and chert bands that are parallel/oblique to the schistosity. The quartz veins and the host greywacke show minor to rare pyrite, pyrrohtite disseminations. The general strike direction of the country rock varies between N30° to 40°W and dips at 60° to 80° towards the northeast. The entire study area is a **peneplain** land with very gentle slopes, may be referring to matured topography. The plant area has limited exposure of rocks. However weathered greywacke is exposed towards the southern side which has been earmarked for the processing plant and water storage facility.

9.2.3 Site investigation and fieldwork

The near surface geotechnical site investigation included the drilling of three geotechnical diamond core drillholes and the excavation of a series of geotechnical test pits. Figure 9.5 below illustrates the positions of the drillholes and test pits relative to the proposed plant infrastructure.

Figure 9.5 Test pit locations over the process plant area



9.2.4 Diamond core drilling

The layout of the plant infrastructure components was provided to Prime Resources for the positioning of the drillholes relative to the heavier components. The three boreholes were positioned in close proximity to the crushing, screening and milling equipment, and beneath the thickener tanks.

The drilling was conducted by using diamond core drilling rig with HQ 63.5 mm core diameter. Each of the borehole were vertically drilled to a depth of 15 m below ground level. The drilled core was logged in the field key information recorded including description of layers, depth of ground water level, core recovery and rock quality designation (RQD).

The core logging of the three boreholes showed a uniform profile below the proposed plant infrastructure areas. No groundwater was encountered during the drilling. The general profile is shown in Table 9.10 below.

Table 9.10 General profile of the plant area boreholes

From (m)	To (m)	Description
0.00	1.2	Sandy clay with gravel, moderate top highly weathered greywacke
1.2	4.00	Moderately weathered, highly fractured greywacke
4.00	6.00	Moderately weathered to hard unweathered greywacke
6.00	15.00	Hard unweathered greywacke

9.2.5 Core laboratory testing and analysis

The core boxes were transported to the SGES laboratory in Bangalore to carry out a laboratory test program which included particle size distribution, Uniaxial Compression Strength (UCS) testing, direct shear- and point load testing and also core recovery and RQD. A summary of the laboratory testing on the drill cores are contained in Table 9.11/

Table 9.11 Summary of drill core test results

Test description	Result
Core recovery (%)	73% to 97% for weathered profiles 98% to 100% for fresh unweathered profile
RQD (%)	0% to 54% (weathered profile) 47% to 102% (fresh unweathered profile)
Direct shear	Internal friction angle Φ: 32° to 34° Cohesion c: 10 kPa to 12 kPa
UCS	12.21 MPa to 64.27 MPa
Point load index	2.19 MPa (single sample)

The purpose of the drill core testing was to determine the bearing capacity of the soils and rock for the foundations of the heavy plant and machinery. The minimum UCS value recorder during testing was 12.21 MPa at a depth of 1.3 m. The safe bearing pressure was determined as per the Bureau of Indian Standards: Code of Practice for Design and Construction of Shallow Foundations on Rocks (IS 12070:1987). The safe bearing pressure is determined from the product of the minimum UCS and a discontinuity dependent empirical coefficient. The minimum UCS was 12.21 MPa and the coefficient was selected as 0.1, resulting in a safe bearing capacity of over 1.22 MPa. It is recommended that footing be founded on the weathered greywacke rock as a minimum with a minimum foundation width of 0.75 m.

9.2.6 Geotechnical test pits

The five test pits over the plant area were manually excavated to an average depth of 1.5 m, profiled and logged. The respective surface soil profiles over the plant area have general profiles as detailed in Table 9.12.

Table 9.12 General surface soil profile over the plant area

From depth (m)	To depth (m)	Description
0.00	0.30	Brownish topsoil soil with plant roots and humus
0.30	0.60	Highly weathered greywacke and reddish brown soil with chert scree
0.60	1.50	Weathered greywacke

The test pit profiles showed minor variation in thickness for the profile layers with the pits toward the northern portion of the plant site containing reddish brown soil with chert nodules and chips of quartz and greywacke. No groundwater was observed in any of the pits.

The primary interest for the plant site surface geotechnical properties are the bearing capacity of the surface soils for the founding of light to medium heavy structures. The sampling program included (as a minimum) a 30 kg bulk sample being taken from each test pit for testing.

9.2.7 Surface soil laboratory testing and analysis

The laboratory testing of the surface soils over the plant area include:

- Foundation indicators: particle size distribution, Atterberg Limits and classification per the Unified Soil Classification System (USCS)
- Specific gravity
- Natural moisture content
- Free swell index
- Permeability
- Shear strength parameters.

These tests were selected in determining the suitability of the in-situ soils for its ended use as founding support for medium to heavy foundations. Other tests such as permeability were tested to further characterise the surface soils.

Selected laboratory tests were assigned to each of the samples. The testwork was undertaken by SGES with positioning provided by Prime Resources. A summary of the test results is shown in Table 9.13.

Table 9.13 Laboratory test results for the plant area upper surface soils

Test	Result summary
Classification	Sandy clay/clayey sand with gravel, silty gravel with sand
Clay size particles	2% to 21%
Specific gravity	2.67 to 2.70
Free swell index	0% to 20%, low to moderately expansive (clayey surface soils)
Maximum dry density	1.81 t/m^3 to 1.92 t/m^3
Plasticity	Slight to medium, with non-plastic soil samples
Permeability	1.07 x 10-4 cm/s to 4.33 x 10-4 cm/s
Shear strength parameters	Φ': 32°, C': 15 kPa

The complete laboratory results are attached in Chapter 8 Geotechnical and Tailings.

Classification

The surface soils over the plant area were found to consist of sandy clays with gravel (topsoil layer into greywacke), clayey sands with gravel (highly weathered greywacke) and two pits toward the south of the plant site comprised of silty gravel with sand (highly weathered greywacke).

Permeability

The permeability was tested to comprehensively characterise the soils and were found to be in excess of 1.00 x 10^{-4} cm/s which is higher than the maximum permeability required for low permeability liner. However, the plant site surface soils are not intended for this use.

Shear strength

The bearing capacity of the in-situ soil have been estimated with the shear strength parameters measured from a Direct Shear test. The test was conducted on a sample comprised of sandy clay and gravel. The results showed an effective friction angle of 32° and effective cohesion of 15 kPa. The ultimate bearing capacity was calculated as 1 130 kPa, and with a factor of safety of 3, an allowable bearing pressure of 377 kPa is specified. This bearing capacity applies to light and medium structures with pad or strip foundations with a minimum width of 0.75 m.

Free swell index

The classification and foundation indicator tests have indicated that the samples are comprised of a varying clay content in excess ranging from 2% to 21% (weathered greywacke (2%) to clay cotton soils (21%)).The free swell index ranges from 0% to 20%, with the majority of the soils showing a low expansive index at 10%. This swell index values were determined with no surcharge load. The pressure swell index for the soils will be lower when an additional pressure is applied. The low swell and heave potential will be further mitigated with the load of the plant infrastructure.

10 MARKETING INFORMATION

10.1 Market analysis for gold demand in India

India is a mineral rich country with wide availability of minerals in the form of abundant rich reserves and favourable eco-geological conditions. The Indian mining industry is characterised by a large number of small operational mines.

Given the traditional significance given to gold possession in India, it generates one of the highest quantum of demand in the world. In 2015, India accounted for nearly 26% of the global fabrications demand (jewellery, bar and coin and technology). The demand for gold in India is not only the highest in the world but also the fastest growing. With the Indian economy projected to grow at 8% during 12th plan, the demand for gold can only increase further.

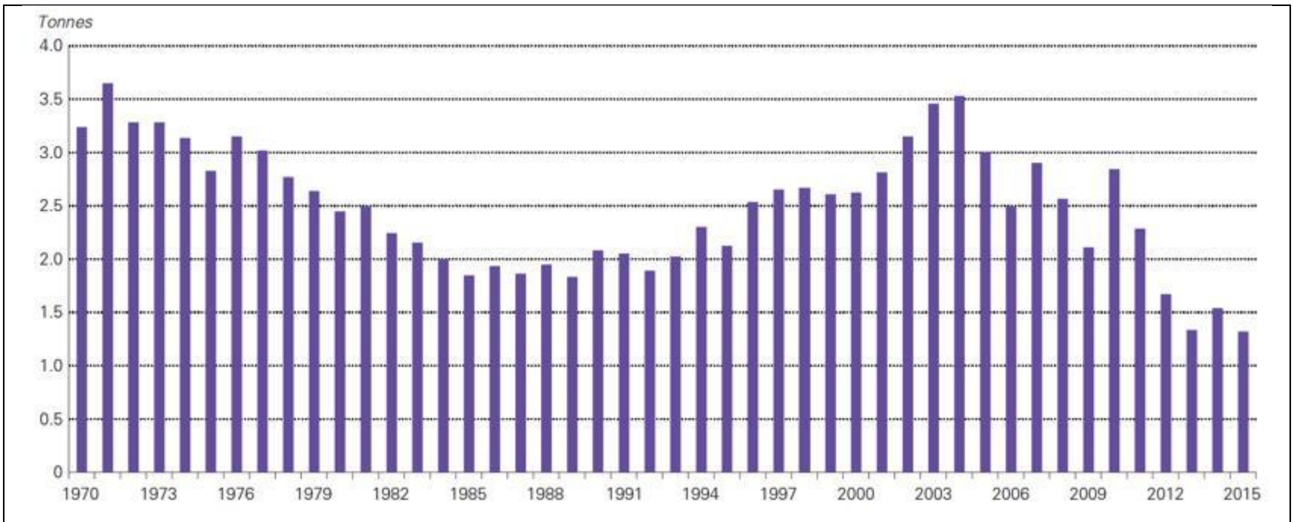
In India, the total resource in terms of metal (primary gold) is at 659.84 t. Out of this, 110.54 t are under Reserve category and 549.30 t under remaining Resource category, which also includes placer-type gold resources. Also, the “Obvious Geological Potential Area” in respect of gold is approximately 102,809 km^2.

By states, largest resources in terms of gold ore (primary) are located in Bihar (45%), followed by Rajasthan (23%), Karnataka (22%), West Bengal (3%), Andhra Pradesh and Madhya Pradesh (2% each). In terms of metal content, Karnataka remains on top followed by Rajasthan.

10.2 Gold production in India

Although India has a long history of gold mining, current production levels are very low; during 2014/2015, India’s gold production was a negligible 1.43 t (less than 2 t). Figure 10.1 shows Indian mine production of gold from primary sources for the period 1970 to 2015.

Figure 10.1 Indian mine production from primary source* (1970 to 2015)



** Mine production includes production from primary sources only and does not include gold output from secondary sources (e.g. Birla Copper Smelter which processes imported copper concentrate).*

Source: Indian Bureau of Mines; Metals Focus

The industry does, however, have potential. Mineral Reserves and Resources total 71.9 t and 574.3 t, respectively. Despite having significant resources, India reported only 1.43 t of gold production in 2015, whereas its imports went above 900 t, in quest of fulfilling its demand for gold. This is primarily attributable to:

- Low exploration activity

- Minimal private sector participation particularly in exploration of strategic and deep-seated mineral deposits
- Interest from players having state-of-the-art technologies in gold mining
- Low capital risk appetite.

Over the coming years, mine production is expected to grow modestly as new mines enter the production phase. But the industry faces significant challenges. For mining to develop in India, regulations need to be reviewed and the industry needs investment.

The mining industry in India is undergoing transformation due to globalisation. In order to bring in transparency and speedy disposal of mineral concessions, the new Government of India has brought in a number of changes in the mining sector by multiple enactments and procedures in 2015. The New Mines and Minerals (Development and Regulation) Amendment Act 2015 (MMDR) is likely to usher new hopes for the mineral industry. In order to encourage scientific exploration especially for minerals such as gold the new Act has introduced a Composite Licence system for exploration and followed by mining.

10.3 Future gold demand in India

India is one of the largest importers of gold in the world and the imports mainly take care of demand from its jewellery industry. During the period from April 2015 to February 2016, gold import stood at about 941.1 t while for the entire 2014/2015 fiscal, the same was 900 t[1]. The annual import bill on gold is between US$35 billion and US$40 billion.

India is a traditional and stable market for gold consumption imports of gold in significant quantities will continue. The present and future production of gold will not be sufficient to meet the ever-increasing demand. Therefore, efforts will be required to reduce the gap between production and demand.

This will require more reserves to be explored and established. Geologically, India has a shared history with Western Australia and parts of Africa as these were formerly part of the Gondwana super-continent that existed 300 million years ago. So, there remains a positive probability of more discoveries.

Gold demand is driven by a combination of factors that interact with each other. An econometric analysis done by World Gold Council to understand gold’s demand behaviour reveals the long and short-run determinants of gold demand.

The analysis, using annual data from 1990 to 2015, reveals two significant factors affecting gold consumer demand over the long term. All else being equal, gold demand is driven by:

- Income – gold demand rises with income levels; for a 1% increase in income per capita, gold demand rises by 1%
- Gold price level – higher prices deter gold purchases; for a 1% increase in prices, gold demand falls by 0.5%.

There are some very interesting factors that affect gold demand in the short term too. Holding everything else constant, these are:

- Inflation – for a 1% increase in inflation, gold demand increases by 2.6%
- Gold price changes – for a 1% fall in gold price, demand will increase by 0.9%.

Other factors that impact the gold demand in the short term are excess rainfall and tax regime.

[1] Source: profitndtv.com and inreuters.com

Going forward, the International Monetary Fund (IMG) has forecast per capital GDP to grow by 35% for 2015 to 2020 and the National Council of Applied Economic Research expects India's middle class to double, exceeding 500 million by 2025.

By 2020, it is expected that the Indian gold demand would average 850 t/a to 950 t/a. India's relationship with gold goes beyond income growth; gold is intertwined with India's way of life. And as we look ahead, India's gold market will evolve.

11 GEOCHEMISTRY

11.1 Introduction

Geostratum Groundwater and Geochemistry Consulting (Pty) Ltd (Geostratum) was appointed together with Prime Resources to perform a geochemical assessment on the proposed Ganajur Gold Mine.

The geochemical report prepared by Geostratum in support of the FS is provided as Appendix 11A.

The report includes the description and interpretation of geochemical analyses undertaken on ROM ore samples, tailings material and waste rock samples. Analyses of tailings material prior to arsenic stabilisation and post arsenic (As) stabilisation were also undertaken.

11.2 Methodology

The geochemical program sought to understand the acid-generating capacity and leaching behaviour of the ROM ore, tailings and waste rock material. This was undertaken through the following assessments, geochemical analyses and interpretations:

- Review of available information and preliminary assessment of potential issues and concerns that may be associated with ore, tailings, and waste rock material.
- Development of a sampling and an analytical plan.
- Geochemical analyses were conducted by ALS Environmental laboratories (Australia), Metron Laboratory (South Africa) and Shiva laboratories (India).
- Potential for long term acid generation from ore, tailings and waste rock material was evaluated (through the use of net acid generating (NAG) and acid-base accounting (ABA) experiments).
- Synthetic Precipitation Leaching Procedure (SPLP) and distilled water batch leaching experiments were undertaken on ore, tailings and waste rock material. Metals, metalloids and anions that were liberated during the static leach tests were identified.
- Toxicity Characteristic Leaching Procedure (TCLP) was undertaken on tailings and waste rock material in order to further investigate the toxicity of the material, as per current Indian regulations (Schedule II of Hazardous and Other Wastes (Management and Transboundary Movement) Rules, 2016). No mining specific regulations were available.
- An indication of the percentage sulphide (%S) relating to the potential for acid generation for rock material was given.
- Conceptual models for the TSF, mine pit and any stockpile were developed. These included the typical physico-chemical processes that will control acid mine drainage (AMD) generation.

11.3 Run-of-mine ore

11.3.1 Geochemistry

Five ore bearing composite samples were subjected to geochemical analysis. The samples had variable sulphide composition and were used by the mine to optimise their metallurgical extraction.

Quartz was the predominant mineral in all of the samples (48 wt% to 51 wt%) with major siderite (18 wt% to 26 wt%). The samples had varying pyrite composition (5 wt% to 12 wt%) and contained 1 wt% arsenopyrite. Dolomite-ankerite abundances were in the range of 7 wt% to 12 wt%.

Table 11.1 summarises the total elemental composition of the five ore composite samples. Gold abundances ranged from 2.5 g/t to 10.6 g/t. Arsenic (As) abundances ranged from 3,130 ppm to 8,600 ppm with an average of 6,302 ppm. The following elements were below reporting limits: Be (<20 ppm), Bi (<25 ppm), Cd (≤20 ppm), Co (20 ppm), Hg (<0.1 ppm), Li (<20 ppm), Mo (<20 ppm), Se (<5 ppm), Y (<100 ppm), and are not included in Table 11.1.

Table 11.1 Total elemental analysis of ore samples

Analyte	Unit	GM 2 sulphide variability comp.	GM 3 sulphide variability comp.	GM 4 sulphide variability comp.	GM 5 sulphide variability comp.	GM 6 sulphide variability comp.
Au (Average)	g/t	10.4	5.06	3.95	7.93	2.59
Ag	g/t	1.2	0.6	0.6	1.5	0.6
Al	%	2	2.6	4	2.88	4.76
As	ppm	8,600	6,790	4,400	8,590	3,130
Ba	ppm	100	160	240	140	280
C_{TOTAL}	%	3.87	3.93	3.21	3.75	3.39
$C_{ORGANIC}$	%	0.39	0.33	0.33	0.39	0.27
$C_{CARBONATE}$	%	17.4	18	14.4	16.8	15.6
Ca	%	1.9	1.74	2	1.91	2.03
Cr	ppm	75	50	50	50	75
Cu	ppm	40	36	20	72	34
Fe	%	19.7	16	14.7	16.7	12
K	%	0.475	1	1.1	0.875	1.53
Mg	%	1.16	1.2	1.28	1.32	1.56
Mn	ppm	800	580	800	760	780
Na	ppm	2,650	3,400	7,700	3,100	5,550
Ni	ppm	40	40	40	40	40
P	ppm	1,000	1,000	750	1,000	750
Pb	ppm	40	<20	20	20	<20
S_{TOTAL}	%	4.98	3.78	2.44	5.4	2.48
$S_{SULPHIDE}$	%	4.8	3.76	2.14	5.24	2.4
SiO_2	%	44.6	45	54.2	47	49
Sb	ppm	7.8	5	4.2	6.6	3
Sr	ppm	150	120	185	145	190
Ti	ppm	800	1,200	1,600	1,400	2,200
V	ppm	25	40	45	40	65
Zn	ppm	60	60	65	65	50

Table 11.2 summarises the ABA and NAG results. The oxidation of sulphide minerals (pyrite and arsenopyrite) produces acid. Three of the five samples analysed with sulphide content in the range of 3.6% to 4.8% were classed as likely to generate acidity in the long term. The two samples with sulphide content less than 3% have a low potential for acid generation and will not be acid generating unless significant preferential exposure of sulphides occur and/or the acid neutralising capacity (ANC) is insufficiently reactive.

The high carbonate content of the material indicates that some neutralizing of produced acidity will take place in the ore. However, most of the carbonate content is iron rich and oxidation of ferrous iron to insoluble ferric iron leads to the precipitation of iron hydroxides, a reaction which is acid producing. Siderite, the major carbonate present, has no net neutralizing capacity. The neutralising capacity gained through the dissolution of the carbonate mineral (reaction 1) is undermined by the associated oxidation (reaction 2) and acid releasing precipitation of iron (reaction 3).

$FeCO_3 + 2H^+ \rightarrow Fe^{+2} + CO_2 + H_2O$ … (1)

$Fe^{+2} + \frac{1}{4} O_2 + H^+ \rightarrow Fe^{+3} + \frac{1}{2} H_2O$ … (2)

$Fe^{+3} + 3H_2O \rightarrow Fe(OH)_3 + 3H^+$ … (3)

During the ABA experiments, it was likely that the above reactions did not proceed to completion. Therefore, there was an overestimation of the ANC and an underestimation of the net acid producing potential (NAPP) (Table 11.2). The NAG results however clearly indicate that three of the samples are acid producing.

Table 11.2 ABA and NAG results of ore material

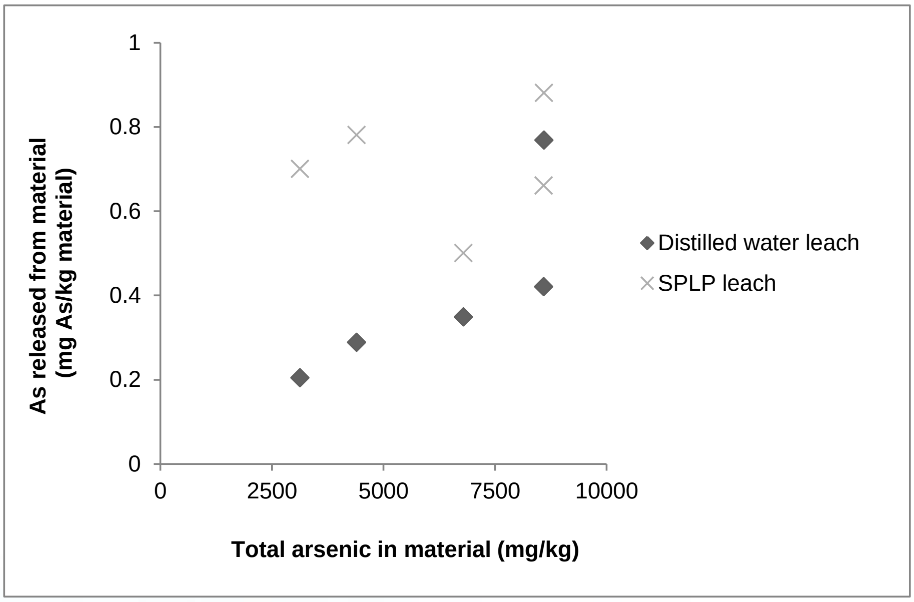
Sample ID	Paste pH	Sulphide %S	AP (kg H_2SO_4/t)	ANC (kg H_2SO_4/t)	NAPP (kg H_2SO_4/t)	ANC:AP	NAG pH*	NAG at pH 4.5* (kg H_2SO_4/t)	NAG at pH 7* (kg H_2SO_4/t)
GM 2 sulphide	7.6	4.28	131	211	-80	1.6:1	3.6	2.6	18.9
GM 3 sulphide	7.7	3.61	110	188	-78	1.7:1	3.6	2.3	16.8
GM 4 sulphide	7.8	2.45	75	160	-85.	2.1:1	6.5	<0.1	0.2
GM 5 sulphide	7.7	4.72	144	167	-23	1.2:1	3.6	2.2	18.5
GM 6 sulphide	7.9	2.28	70	137	-67	2.0:1	7.1	<0.1	<0.1

The readily soluble fraction of material was assessed in batch leaching experiments using distilled water and acidic SPLP solutions. All the leaching results were within General Effluent Standards for Discharge of Environmental Pollutants as prescribed by the Indian Pollution Control Board (Schedule IV of the Environment Protection Rules, 1986) (“General Effluent Standards”). However, these experiments do not take account of the leachability of metals from material that has been altered during oxidation and weathering.

The higher the sulphide contents of the ore material, the greater the amount of manganese (Mn) and As that was mobilised in the distilled water experiments. Even though only a small percentage of the As was dissolved, the net solution of the 1:20 SPLP leach yielded an As concentration greater than the Indian Drinking Water Specifications of 0.01 mg/l (preferred) and 0.05 mg/l (permissible).

The amount of As (mg) released from one kilogram of ore in the two leaching experiments is shown in Figure 11.1. The volume of the leaches has been taken into consideration. The initial solution of the SPLP leach is more acidic than the distilled water and has a greater ionic strength than distilled water. There is no clear linear correlation between the amount of As in the ore and the amount leached by the SPLP solution. It is possible that the concentration of As in solution is controlled by the solubility of an As bearing mineral. Examples of such minerals could be pharmocolite ($CaHAsO_4.2H_2O$) or symplesite ($Fe_3(AsO_4)_2.8H_2O$).

Figure 11.1 Arsenic (As) released from ore material during leaching experiments



11.3.2 Implications for on-site storage

Due to the stockpile being unsaturated and air moving freely through the pile, oxidation of exposed surfaces of material commences shortly after placement of the material. Rainwater percolates through the stockpile materials and the release of drainage from the ROM stockpile will be periodic. The chemistry of the released water will be affected by the length of time the material has been on surface and the degree of weathering, the time between rainfall events (when last the material was rinsed), and the intensity of the rainfall. The material is classed as potentially acid forming and the SPLP and distilled water leach tests indicated that a fraction of As was present in readily soluble phases. Therefore, the ROM stockpile would require appropriate lining to prevent seepage into groundwater resources and percolating rainwater should be diverted away from any water resources.

11.4 Tailings material

11.4.1 Geochemistry

Three tailings samples prior to As stabilisation and one sample that had undergone As stabilisation were analysed.

The As stabilised material was primarily comprised of quartz (39.6%) and siderite (32.7%) (analysed by Metron Laboratories). There was a significant pyrite content (7.7%). There could be minor amounts of arsenopyrite which were not detected using x-ray diffraction. Siderite and dolomite (11.5%) comprise the carbonate mineral fraction of the material. Dolomite offers net neutralising potential, however, as discussed previously, siderite does not.

The ABA and NAG (Table 11.3) tests indicated that one of the tailings samples (GM 5: Tails) had high potential for acid generation. Sample GM 6: Tails had lower acid generating potential, however the high sulphide content in the sample poses a long term acid generation potential. Sample GM 3: Tails was intermediate of the two and classed as uncertain. The As stabilised tailings sample was classed as potentially acid forming and has the capacity to generate acidity in the long term should oxidation occur.

The ANC and NAG results for the tailings before and after As stabilisation differ considerably (Table 11.3). Testing for the As stabilised waste took into account the kinetics of the siderite reactions (a longer reaction time was allowed for) and a siderite correction step (adding hydrogen peroxide to oxidise the iron content). Therefore, the results for As stabilised tailings material better represent the true reactivity of the material.

Table 11.3 ABA and NAG results of tailings material

Sample ID	Paste pH	Sulphide %S	AP (kg H_2SO_4/t)	ANC (kg H_2SO_4/t)	NAPP (kg H_2SO_4/t)	ANC:AP	NAG pH*	NAG at pH 4.5* (kg H_2SO_4/t)	NAG at pH 7* (kg H_2SO_4/t)
GM 3 tails	7.2	3.72	114	150	-36	1.32:1	3	2.6	9.3
GM 5 tails	7.8	5.83	178	141	37	0.79:1	3	2.3	11.3
GM 6 tails	6.9	2.44	75	143	-68	1.92:1	5.3	<0.1	<0.1
Tails (As stab)	7.2	3.43	105	88	17	0.83:1	2.5	59.7	

Table 11.4 summarises select results of the batch leaching experiments of the tailings samples. The pH of the leaching experiments was due to the neutralising capacity of the dolomite content. As was leached from the tailings material during the distilled water and SPLP tests (Table 11.4). Therefore, a portion of As exists in phases that are readily soluble. The As concentration in these leaches was above the limit recommended by the Indian Drinking Water Specifications (0.01 mg/l preferred and 0.05 mg/l permissible) and close to or exceeding General Effluent Standards (0.2 mg/l). Most of the omitted analytes in Table 11.4 (including Be, Cd, Cr, Hg, Se, Sn, V, W and Zr) were below reporting limits.

Leaching experiments indicated that the As stabilisation step was successful because on average, less As was released.

Table 11.4 Select results of distilled water and SPLP leaching experiments of tailings material

Parameter	No arsenic stabilisation			Arsenic stabilised tailings
	GM 3 tails	GM 5 tails	GM 6 tails	
1:4 Distilled water leach				
pH (value)	8.02	8.05	8.22	7.48
EC (mS/m)	490	403	938	207
Sulphate as SO_4	140	132	437	1 133
Total alkalinity as $CaCO_3$	102	73	211	45.6
Cl	24	26	42	65.5
Al	0.02	0.01	0.01	<0.06
As	**0.562**	**1.76**	0.101	**0.274**
Ca	58	40	170	292
Co	0.006	0.002	0.013	<0.01
Cu	0.004	0.002	0.005	<0.02
Mg	16	13	46	59.7
Mn	0.1	0.015	0.28	0.131
Ni	0.006	0.001	0.052	<0.01
Zn	0.146	0.061	0.354	<0.01
1:20 SPLP leach				
pH (value)	8.05	7.92	8.16	7.30
EC (mS/m)	181	157	338	63.3
Sulphate as SO_4	52	45	99	Not measured
Total alkalinity as $CaCO_3$	33	30	88	Not measured
Cl	12	12	21	28.9
Al	<0.1	<0.1	<0.1	<0.06
As	**0.39**	**1.00**	0.035	0.154
Ca	28	20	49	79.5
Fe	<0.05	<0.05	<0.05	0.130
Mg	8	6	14	18.1
Mn	<0.01	<0.01	0.05	<0.06
Ni	<0.01	<0.01	<0.01	<0.01
Zn	<0.1	<0.1	0.2	0.012

The TCLP leach tests have classified the tailings material as non-hazardous according to Schedule II of Hazardous and Other Wastes (Management and Transboundary Movement) Rules, 2016) (Table 11.5). However, the material may be altered during oxidation and this may affect its classification. The TCLP leach is not an accurate portrayal of field conditions and the concentrations reported by the SPLP, distilled water leach and the kinetic leach tests are likely to better represent the disposal situation.

Table 11.5 TCLP results of arsenic stabilised tailings material

Parameters (mg/l)	Reagent 1: pH 4.90		Class A TCLP concentrations*
	Tailings DLI 213	Tailings DLI 213 (duplicate)	
Al	0.073	0.070	
As	0.047	0.046	5
B	<0.01	<0.01	
Ba	0.051	0.063	100
Be	<0.01	<0.01	
Ca	303	332	
Cd	<0.003	<0.003	1
Co	0.026	0.026	80
Cr	<0.01	<0.01	5
Cu	0.507	0.736	25
Fe	2.48	2.83	
K	11.5	15.4	
Mg	78.3	81.2	
Mn	3.53	4.47	10
Mo	<0.01	<0.01	350
Ni	0.103	0.114	20
Pb	<0.01	<0.01	5
Sb	<0.02	<0.02	
Se	<0.02	<0.02	1
Sr	1.30	1.50	
V	<0.01	<0.01	24
Zn	0.410	0.466	250

** According to Schedule II of Hazardous and Other Wastes (Management and Transboundary Movement) Rules, 2016*

11.4.2 Implications for on-site storage

Given the high As abundance in the ore samples, it is likely that there is sufficient As in the tailings to pose a long term leaching risk at the site. Although the As stabilisation step was successful, As is still released from tailings material during batch leaches at concentrations close to or exceeding General Effluent Standards. The leaching profiles of acidity, As and sulphate (SO_4) from tailings material that has undergone As stabilisation are currently being assessed during long term column leaching experiments. These results are expected after June 2017.

Saturated tailings material is deposited onto a TSF. The fine-grained tailings material is pumped as slurry onto the TSF and oxygen ingress is limited to the outer edges of the TSF. The well packed nature of tailings material restricts oxygen ingress to some degree. The unsaturated zone will comprise of an outer oxic and deeper anoxic zone depending on the depth of oxygen diffusion into the TSF. Pyrite oxidation will only take place in the oxic zone and the interstitial water in the upper part of the unsaturated zone will have a much higher sulphate concentration than the saturated water deeper in the TSF. The oxygen concentration will be at its highest in material directly in contact with the atmosphere and due to its consumption, the oxygen concentration will gradually become depleted within only a few metres. The temperature in the material will eventually rise due to the oxidation of sulphides. Temperature differences will result in differences in gas pressure that initiate the process of oxygen advection. Advection is however minimal in the fine material (tailings) and more relevant in coarse material (e.g. waste rock).

Due to differences in oxygen content and pyrite oxidation rate in the TSF, as well as the slow water flow in the TSF, water quality in the TSF will vary in different parts of the TSF. The outer rim will include the unsaturated zone and the contact zone with the saturated zone. The water quality in the outer rim will have higher SO_4 content and will eventually become acidic. Seepage water at the toe of the TSF if uncovered/not rehabilitated will become progressively more representative of the water in the outer rim. Water in the inner saturated part, from the seepage drains, will not be acidic and will have a much lower SO_4 concentration as the SO_4 concentration will mostly be determined by gypsum saturation.

The TSF requires an appropriately lined facility (HDPE and clay) to prevent groundwater contamination. During construction or post closure the facility should be capped to minimise oxygen ingress and oxidation of tailings. Surface water decant should be contained and treated if necessary. Rainwater should be diverted where possible.

11.5 Waste rock material

11.5.1 Geochemistry

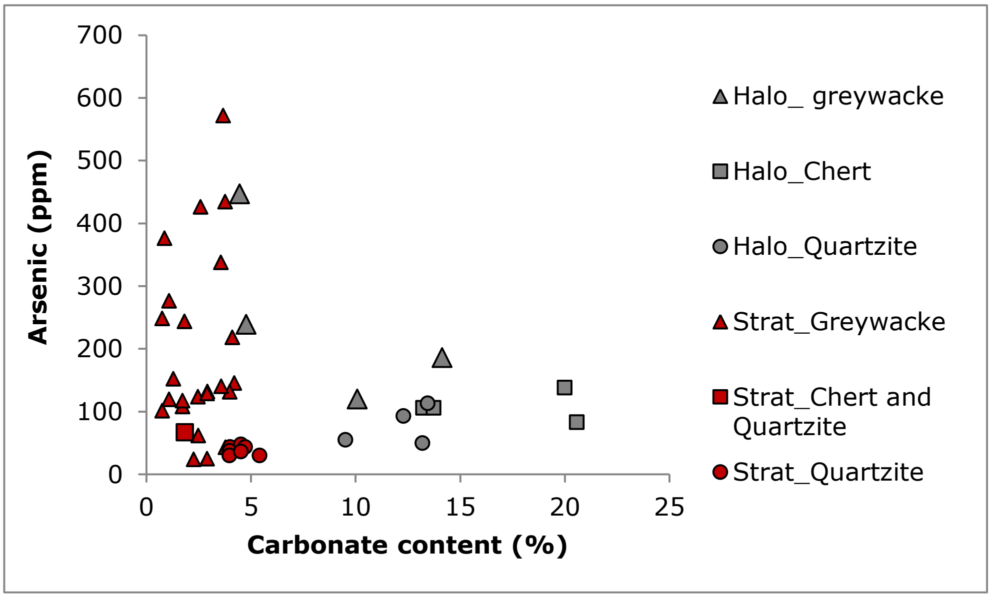
The waste rock is intended to be sold as aggregate. For a period of time prior to sale, waste rock might be stored on site as a waste rock pile. Geochemical tests were therefore undertaken to assess the release of metals and acidity from the waste rock.

Two groups of waste rock were analysed: (a) halo waste rock which is close to the orebody; and (b) stratigraphic waste rock which represents waste rock from outside the halo zone. ALS laboratories analysed 31 halo samples which selected from across 19 boreholes. Basic testing was recommended for all these samples. Further comprehensive testing on a selection of nine samples from the original batch of samples was undertaken. Composite samples of halo material were generated and sent to Metron Laboratory for long term column leaching experiments. Shiva laboratories analysed 32 waste rock samples selected to represent the hangingwall and footwall (termed “stratigraphic waste rock”) for Al, As, Mn, Fe, SO_4, carbonate, and sulphide content.

The halo and stratigraphy waste rock material was subdivided into greywacke, quartzite, brecciated chert and quartzite-chert. The samples were comprised of quartz and illite (or other minerals similar to mica). Siderite and ankerite/dolomite were the predominant carbonate minerals in the halo waste rock with cumulative abundances ranging from 6.7% to 29%. Two halo waste rock samples that had low carbonate content contained a large amount of goethite (6% and 9%). The iron within the original carbonates of these samples was likely to have been oxidised, resulting in the formation of iron hydroxide minerals.

Stratigraphic waste rock contained less carbonate minerals than halo waste rock (Figure 11.2). Quartzite samples contained on average less As than chert and greywacke samples.

Figure 11.2 Arsenic vs. carbonate content in stratigraphic and halo waste rock



Most of the halo samples had sulphide content less than 0.4%, with the exceptions being chert-containing samples (Table 11.6). These chert samples were close to the orebody and therefore were partially mineralised. The samples have a low capacity to produce acid which was confirmed in ABA and NAG experiments (Table 11.6). The high ratio between the ANC and AP indicates that the samples should be able to neutralise any liberated acid. All the stratigraphic waste rock samples had sulphide content less than 0.1%; therefore, these samples will also be non-acid generating in the long term.

Table 11.6 ABA and NAG results of halo waste rock material

Sample ID	Rock type*	Paste pH	Sulphide %S	AP H_2SO_4 kg/t	ANC H_2SO_4 kg/t	NAPP H_2SO_4 kg/t	ANC:AP	NAG pH	NAG at pH 4.5 (kg H_2SO_4/t)	NAG at pH 7 (kg H_2SO_4/t)
100956	G	8.8	0.21	6	110	-104	17.1:1	10.3	<0.1	<0.1
100958	G	8.5	0.35	11	123	-112	11.5:1	9.3	<0.1	<0.1
100974	G	8.6	0.02	1	103	-102	168.3:1	10.8	<0.1	<0.1
101027	Q	8.4	0.02	1	78.7	-78	128.6:1	10.6	<0.1	<0.1
101077	BC	8.4	0.53	16	128	-112	7.9:1	9.1	<0.1	<0.1
101083	QC	8.3	0.28	9	122	-113	14.2:1	9.9	<0.1	<0.1
101088	QC	8.4	0.67	21	156	-135	7.6:1	9	<0.1	<0.1
101321	BC	8.4	0.25	8	39.6	-32	5.2:1	10.5	<0.1	<0.1
101333	QC	8.5	0.09	3	111	-108	40.3:1	10.8	<0.1	<0.1
101345	G	8.1	<0.01	0	6.3	-6	20.6:1	8.4	<0.1	<0.1
101341	G	8.7	<0.01	0	50.4	-50	164.7:1	10.8	<0.1	<0.1
101388	Q	9	0.03	1	67.4	-66	73.4:1	10.5	<0.1	<0.1
102031	G	8.9	<0.01	0	91.1	-91	297.7:1	10.8	<0.1	<0.1
102035	BC	8.8	0.1	3	87.2	-84	28.5:1	10.7	<0.1	<0.1
102051	G	9	<0.01	0	69.4	-69	226.8:1	10.2	<0.1	<0.1
102091	G	8.8	0.02	1	115	-114	187.9:1	10.8	<0.1	<0.1
101943	Q	8.6	0.13	4	140	-136	35.2:1	10.8	<0.1	<0.1
102141	G	9	0.02	1	99.6	-99	162.7:1	10.2	<0.1	<0.1
102145	BC	8.7	0.38	12	83.5	-72	7.2:1	9.5	<0.1	<0.1
102149	BC	8.4	0.68	21	57.2	-36	2.7:1	8.9	<0.1	<0.1
102262	G	8.8	0.02	1	92.4	-92	151. :1	10.9	<0.1	<0.1
102272	G	9.1	0.05	2	120	-118	78.4:1	9.2	<0.1	<0.1
102335	Q	8.9	0.04	1	129	-128	105.4:1	10.9	<0.1	<0.1
102379	G	8.5	0.05	2	91	-89	59.5:1	10.8	<0.1	<0.1
102493	G	8.9	0.02	1	116	-115	189.5:1	10.8	<0.1	<0.1
102901	G	8.9	0.02	1	94.5	-94	154.4:1	10.5	<0.1	<0.1
102942	G	9	0.02	1	236	-235	385.6:1	10.5	<0.1	<0.1
102945	G	9.1	<0.01	0	106	-106	346.4:1	9.6	<0.1	<0.1
103028	G	8.9	0.02	1	154	-153	251.6:1	11	<0.1	<0.1
103031	G	8.9	<0.01	0	66.8	-66	218.3:1	10.5	<0.1	<0.1
103358	Q	9	<0.01	0	110	-110	359.5:1	10.9	<0.1	<0.1

Although not acid generating, the samples can generate saline or neutral drainage. This drainage can have high concentrations of As as observed in the leaching experiments (Table 11.7). The As in the waste rock is in a more readily soluble phase than in the ore material. This phase is likely to be associated with iron hydroxide or carbonate minerals. The leaching tests indicate the leachability of elements from waste material and do not necessarily predict the concentration of a leachate emanating in the field. The concentration of As in field leachates could be higher or lower depending on environmental conditions. Furthermore, the increased surface area due to crushing and pulverising the samples can lead to increased solubility of minerals and to elevated concentrations of metals.

Waste rock samples that contained significant abundances of goethite (samples 101321 and 101341) released aluminium (Al) and iron (Fe) during the leaching experiments (Table 11.7). Oxidation and subsequent leaching of the other samples could lead to the mobilisation of a similar suite of metals.

Table 11.7 Select results of distilled water and SPLP leach experiments of halo waste rock

Parameter	100956	101077	101083	101088	101321	101333	101341	102145	102149
Rock type*	G	BC	QC	QC	BC	QC	G	BC	BC
1:4 Distilled water leach									
pH (Value)	8.9	8.71	8.69	8.51	8.5	8.81	8.66	8.82	8.52
EC (mS/m)	371	281	301	328	205	205	170	268	416
Sulphate as SO_4	40	33	38	62	21	12	3	22	97
Total alkalinity as $CaCO_3$	135	91	86	76	60	80	64	101	87
Cl	15	14	16	15	10	10	11	13	14
Al	0.09	0.03	0.04	0.01	1.04	0.05	1.03	0.08	0.12
As	0.928	0.312	0.708	0.057	0.087	0.684	0.082	1.46	0.244
Ba	0.718	0.613	0.596	0.651	0.691	0.632	0.57	0.668	0.592
Ca	12	17	19	24	5	13	3	12	23
Cd	0.0003	0.0004	0.0004	0.0004	0.0002	0.0006	0.0002	0.0003	0.0004
Cr	<0.001	<0.001	<0.001	<0.001	0.002	0.001	0.002	<0.001	<0.001
Cu	<0.001	0.002	0.001	0.001	0.002	0.004	0.003	<0.001	0.003
Fe	<0.05	<0.05	<0.05	<0.05	1.19	<0.05	1.26	0.06	0.06
K	59	32	30	15	6	12	4	29	41
Mg	6	8	7	11	1	5	<1	5	9
Mn	0.002	0.003	0.002	0.007	0.007	0.003	0.009	0.002	0.007
Mo	0.012	0.021	0.027	0.015	0.008	0.007	0.005	0.043	0.016
Na	29	12	30	15	6	16	6	12	17
Ni	0.002	0.004	0.008	0.003	0.003	0.004	0.004	0.006	0.004
Zn	0.02	0.024	0.026	0.025	0.025	0.019	0.027	0.021	0.033
1:20 SPLP leach									
pH (Value)	8.93	9.06	8.98	8.99	8.87	9.05	9.06	9.06	8.81
EC (mS/m)	167	124	120	128	92	102	84	111	141
Sulphate as SO_4	8	6	7	11	3	1	<1	2	16
Total alkalinity as $CaCO_3$	70	53	45	48	43	46	37	48	48
Cl	6	6	6	6	5	5	5	5	6
Al	0.2	0.1	<0.1	<0.1	0.3	<0.1	0.4	0.2	0.1
As	0.689	0.167	0.298	0.037	0.080	0.214	0.041	0.383	0.158
Ba	0.4	0.4	0.4	0.4	0.5	0.4	0.4	0.4	0.4
Ca	10	12	12	14	7	10	6	8	11
Cd	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001	<0.001
Co	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
Cr	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
Cu	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
Fe	<0.05	<0.05	<0.05	<0.05	0.54	<0.05	0.34	<0.05	<0.05
K	20	10	10	4	6	4	5	9	11
Mg	3	4	4	5	1	3	1	2	4
Mn	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
Mo	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
Na	8	4	5	4	8	5	6	4	5
Ni	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01
Zn	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1	<0.1

** Rock type abbreviations: G = Greywacke; BC = Brecciated chert; QC = Quartzite and chert*

The As, Fe and Al released from waste rock material were a concern for the potential use of the waste rock as aggregate. Therefore, TCLP experiments were undertaken to determine whether the waste rock classifies as hazardous according to Schedule II of Hazardous and Other Wastes (Management and Transboundary Movement) Rules, 2016. Mining specific regulations are not yet available.

The halo waste rock composite samples were all classified as non-hazardous (Table 11.8).

Table 11.8 TCLP results of composite halo waste samples

Parameters	Reagent 2: pH 2.90			Class A TCLP concentrations*
	Comp 1 (Greywacke)	Comp 2 (Brecciated chert)	Comp 3 (Quartzite)	
Al	3.73	3.91	3.59	
As	<0.01	<0.01	<0.01	5
B	0.036	0.065	0.061	
Ba	0.080	0.087	0.117	100
Be	<0.01	<0.01	<0.01	
Ca	566	427	457	
Cd	<0.003	<0.003	<0.003	1
Co	0.053	0.077	0.039	80
Cr	0.140	0.181	0.106	5
Cu	0.081	0.141	0.126	25
Fe	154	243	272	
K	12.1	10.3	11.0	
Mg	106	172	179	
Mn	8.88	8.98	9.06	10
Mo	<0.01	<0.01	0.014	350
Na	9.11	6.34	9.12	
Ni	0.110	0.142	0.122	20
Pb	<0.01	0.012	0.074	5
Sb	<0.02	<0.02	<0.02	
Se	<0.02	<0.02	<0.02	1
Sr	1.94	2.53	2.69	
V	<0.01	<0.01	<0.01	24
Zn	0.198	0.228	0.288	250

** According to Schedule II of Hazardous and Other Wastes (Management and Transboundary Movement) Rules, 2016*

Geochemical screening showed that the abundances of metals and metalloids of concern (As, Mn and Fe) within the stratigraphic waste rock are within the range of abundances determined for the halo waste rock. However, halo waste rocks contained more sulphide and carbonate minerals than stratigraphic waste rock.

The stratigraphic samples have similar metal and As abundances to the composite samples and are highly unlikely to be classified as hazardous given the high TCLP concentrations in the Indian regulations. The only element close to the regulation limit was Mn. However, as shown in the SPLP and distilled leach tests, Mn was not highly soluble in rainwater leaching (Table 11.7).

11.5.2 Implications for sale of waste rock

The waste rock is classified as non-hazardous according to Schedule II of the Indian Hazardous and Other Wastes (Management and Transboundary Movement) Rules, 2016. If the waste rock will be sold as aggregate it is the responsibility of the seller (according to IS 383:1970) to determine the physical characteristics and provide further information regarding presence of reactive minerals as requested by the purchaser. Details stipulated in the regulations which may be required by the purchaser, subject to agreement, include specific gravity, bulk density, moisture content, adsorption value, aggregate crushing value, abrasion value, flakiness index, elongation index, presence of deleterious material, potential reactivity of aggregate and soundness of aggregate. The onus is on the purchaser to request this additional information.

11.5.3 Implications for on-site storage

In the event that waste rock is stored on site, the halo waste rock likely poses a short term environmental risk. The long-term risk will be determined by long term column leaching experiments that are underway. The present SPLP and distilled water leach tests of halo waste rock indicate that although not classed as hazardous, the effluent that is likely to emanate from the waste rock pile will contain As concentrations that exceed the Indian Drinking Water standards (0.01 mg/l preferred and 0.05 mg/l permissible) and in some cases exceeds the General Effluent Standards (0.2 mg/l). The SPLP tests run to date on halo waste rock were mostly undertaken on quartzite and brecciated chert. Therefore, it was prudent to conduct a small batch of additional TCLP, SPLP and distilled leach tests on stratigraphic greywacke waste rock if the waste rock will be stored on site in the long term. This is currently underway at Shiva laboratories. The interim results of one sample (with relatively high As of 572 ppm) indicate that the material is non-hazardous and has low leachability of arsenic.

11.6 Conclusions of Geochemical Study

Acidic to near neutral saline drainage is expected to be released following oxidation of the ROM stockpile and tailings material. The waste rock material is less sulphidic and does not have long term potential to generate acid, therefore neutral or saline drainage is expected to decant from any waste rock piles on site.

The abundance of As in the ore, tailings and waste rock material is of concern. Liberated As species are mobile across the pH scale. In the material, As is present as both readily soluble and insoluble phases. Potential soluble phases include adsorption onto or inclusion into iron hydroxide or association with carbonate minerals. Insoluble phases include arsenopyrite which would require oxidation for the As to become mobile.

The following mitigation strategies are recommended:

- The ROM stockpile would require appropriate lining to prevent seepage into groundwater resources and percolating rainwater should be diverted away from natural, fresh water resources.
- The TSF should be lined. In the long term, if the material is oxidised, acidic decant with significant As concentrations is likely.
- Capping of the TSF at closure will prevent water and oxygen ingress. Limiting oxygen ingress through continuous tailings sidewall covering and rehabilitation will minimise the oxidation of the tailings material and prevent the generation of an acidic leach.
- In the event that waste rock is not sold and is stored on site, the waste rock dump/s will require an appropriate base (e.g. compacted clay) to prevent point source contamination of groundwater. The long-term risk will be determined by the ongoing column leaching experiments.

12 HYDROGEOLOGY AND HYDROLOGY

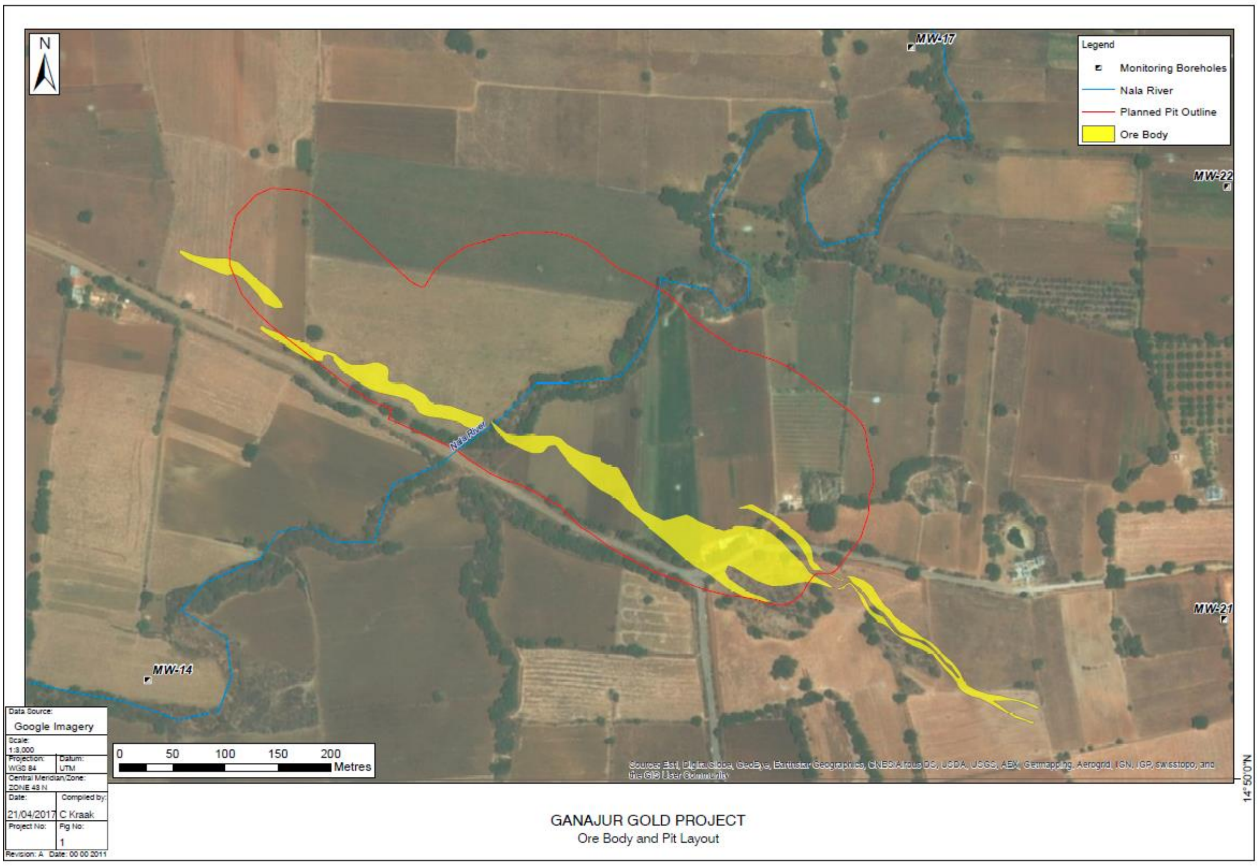
12.1 Overall water circuit

The following chapter was provided by Prime Resources based on a review of the **hydrogeological** work undertaken by Mr B. Jaya Kumar, and Mr M.C. Reddy (Appendix 12A), former Regional Directors, Central Ground Water Board, and the **hydrological** work undertaken by E I Technologies Pvt. Ltd (Appendix 12B).

Deccan Exploration Services Private Limited (DESPL) proposes to establish a 1,000 t/day gold mine with an interlinked 1,000 t/day gold ore processing and beneficiation plant near Ganajur, Karnataka. The Ganajur Gold Project is located 0.5 km to the southeast of the village of Ganajur and 3 km to the north of Haveri Town. The orebody will be mined in three phases with the ultimate pit having a maximum depth of 85 m, length of 700 m along strike (northwest-southeast) and width of 250 m (Figure 12.1).

Ground surface in the area varies from 540 mamsl to 544 mamsl, so the proposed pit bottom is expected to be at approximately 460 mamsl.

Figure 12.1 Geological map showing Ganajur Main Gold deposit orebody



The objective of the hydrogeological study in Appendix 12A was to develop a conceptual hydrogeological model that forms the basis for a numerical flow model to evaluate the potential impacts of the proposed mining project on the groundwater resources and vice versa.

Groundwater may be available from mine pit dewatering – as aquifer modelling has shown that mining pit flows may also reduce the water level in surrounding private borehole wells. An alternative surface water source may be required to replace this water if lost to private landowners and supplement water supplies in the area.

Total make-up water requirement for the Ganajur Gold Project is 3,000 kilolitres per day (kl/day) or 0.035 cubic metres per second (m^3/s). The main make-up water supply for the ore processing will be pumped from the nearby Varada River, over a four-month period during the monsoon season, for the months from July to October and will be stored on site in a HDPE lined water storage dam with an approximate volume of 300,000 m^3. A pump station at the Varada River and a buried HDPE pipeline will transfer the river water to the plant site raw water dam for further distribution.

The ephemeral stream flowing across the mine pit will also need to be diverted; details of which are included in the Hydrology Report by E I Technologies in Appendix 12B.

12.2 Hydrostratigraphy

The Ganajur Main Gold deposit is hosted dominantly by greywacke and interbedded banded ferruginous chert (banded iron formation) which are part the greenstone belt. The ferruginous chert is of sulphide facies, with pyrite, arsenopyrite and chalcopyrite, and the strata bound gold mineralisation occurring as disseminations and fine veinlets in this unit. The general strike direction is northwest-southeast dipping from 35° to 60° towards the northeast. The orebody is part of a north-westerly plunging antiformal structure and the sulphidic chert on surface shows strong brecciation and alteration (presence of gossans, limonite and goethite) with pyrite box-works.

Geological structures play a significant role in the development of the aquifer system. The major joints (but with low frequency) are N 60°W cross cutting the general strike direction of N 10°W and dipping sub-vertically to vertical. Minor joints (at higher frequency) occur in an east-westerly direction. Sections of the Varada River and minor stream beds (including the stream) are fault controlled.

The rocks are weathered from surface to varying depths, ranging from a couple of metres to about 35 m below ground level (mbgl) and the weathered zone is mostly unsaturated.

The main aquifer is associated with secondary porosity due to uplift from tectonic forces and weathering processes in the fractured and jointed greywacke, extending in depth from 40 mbgl to 120 mbgl. The thickness of the aquifer developed in the fractured rock is generally about 60 m (from 40 mbgl to 100 mbgl).

12.3 Aquifer parameters

Seven operational private boreholes near the proposed mine pit and the locations of which are shown in Figure 12.2 were test pumped to estimate the aquifer hydraulic parameters. Due to power limitations (only six hours of electricity available per day) and the fact that the pumps in the boreholes were used, the test pumping could not be carried out over a longer period of time, which would have been preferable (at least 24 to 48 hours followed by equivalent recovery).

The transmissivity T (in m^2/day) was determined using the Theis curve matching method during the pumping period and Cooper Jacob method for the late recovery data.

Table 12.1 shows the details of the boreholes test pumped and the approximate effective hydraulic conductivity (K in m/day), which is determined by dividing the transmissivity by the approximate thickness of the aquifer (b). The thickness was estimated as the final depth of the borehole less the depth to the first water strike (m). The boreholes are only cased through the weathered zone.

The transmissivity ranges from 1 to 50 m^2/day and resulting in the average hydraulic conductivity ranging from 0.01 to 1 m/day with an average of 0.4 m/day. This is a reasonable value for weathered and fractured metamorphic rocks with intermediate permeability. It is anticipated that with depth, the unfractured greywacke will have a lower permeability. No hydraulic testing of the ore zone has been undertaken to date but it is highly likely that this unit has a higher hydraulic conductivity, due to weathering and hydrothermal alteration (gossan on surface allows higher ingress).

No slug testing or packer testing data was undertaken to date.

Figure 12.2 Map showing location of test pumped boreholes

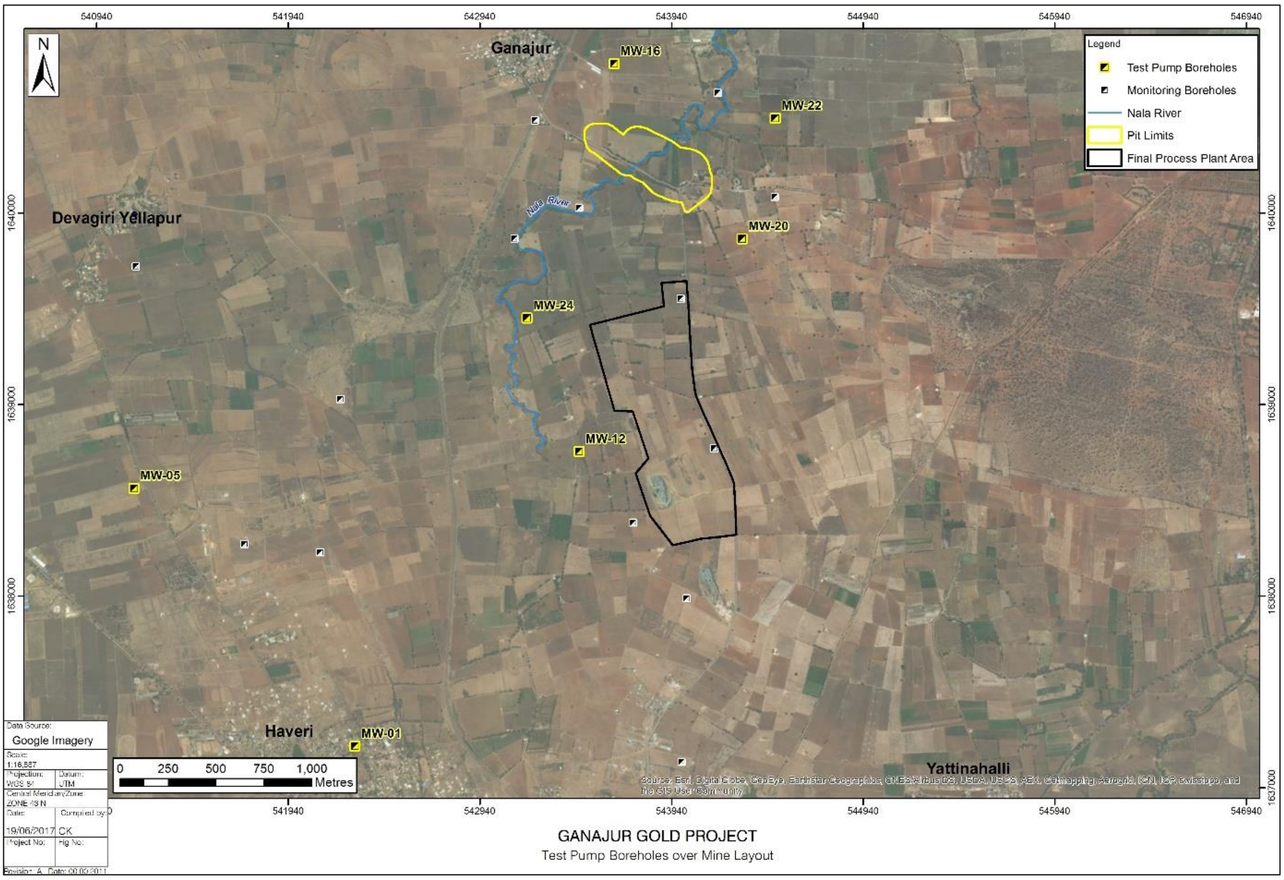


Table 12.1 Summary of test pumped boreholes and analysis

BW ID	Coordinates		Elevation (mamsl)	Depth (m)	Casing (m)	Water strike (mbgl)	Rest water level at start (mbgl)	Pumping rate (m^3/day)	Pumping time (minutes)	Maximum drawdown (m)	Transmissivity (m^2/day)		Thickness of aquifer (b)	Hydraulic conductivity (m/day) K=T/b
	Easting	Northing									Theis	Late recovery		
Mw-01	542288	1637218	580	76	6	46	49	100.2	100	8.29	4	12	30	0.4
Mw-05	541135	1638563	574	140	12	38	58	101.2	150	4.51	1	4	82	0.01
Mw-12	543458	1638755	558	55	6	6		112	100	19		30	50	0.6
Mw-16	543640	1640779	556	113	34	64	29	455	100	4.5	11	50	49	1.0
Mw-20	544308	1639865	556	122	18	91	25	295	80	46	2	6	30	0.2
Mw-22	544481	1640494	538	107	27	52	50	114	100	.07	?	20	57	0.4
Mw-24	543186	1639452	548	76	12	18	22	100	100	6	2	25	54	0.5

12.4 Groundwater recharge

The Ganajur Gold Project is located in a sub-tropical region with moderate temperatures and rainfall. The total annual average rainfall is about 790 mm/year and the total average evaporation is 1,460 mm/year for the Haveri District shown in Table 12.2. The rainfall occurs from June to September (southeast monsoon season) and from October to December (northeast monsoon season).

Figure 12.3 shows a plot of the mean 103-year rainfall and evaporation for the Haveri District. Recharge to groundwater will mainly occur when the rainfall exceeds evaporation in the months from June to October. During the remainder of the year, monthly evaporation exceeds monthly rainfall.

Figure 12.3 Graph of average monthly rainfall vs. evaporation (Haveri District)

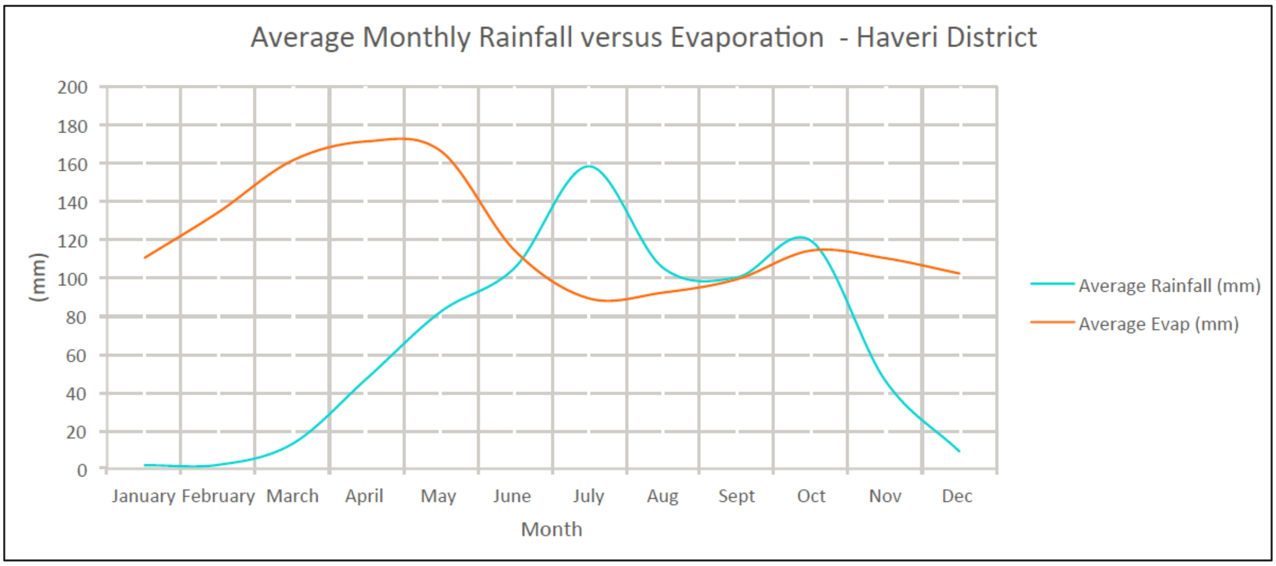


Table 12.2 Average monthly rainfall and evaporation for Haveri District

	Average rainfall (mm)	Average evaporation (mm)
January	2	110
February	2	134
March	13	161
April	47	171
May	82	166
June	105	114
July	158	89
August	105	92
September	100	99
October	119	114
November	46	110
December	9	102
Total	**790**	**1,461**

No site-specific recharge estimates to groundwater have been undertaken. Rainfall recharge to groundwater is considered to occur only when there is a rainfall of more than 10 mm in a month. From the month of April to November, the normal rainfall in a month ranges from 43.6 mm to 169.8 mm with 100 mm and above from June to October. The recommended recharge rate for areas underlain by metamorphic rock (schists) is 10% to 12% (GEC, 2009). Higher recharge is anticipated for the upper reaches of the catchment where the soils are more loamy and sandy and lower in the valleys where the soils are more clayey, restricting rainfall infiltration.

Recharge due to irrigation water is not considered significant due to the fact that no standing body of water is used for the dry crops that are grown in this catchment. The main crops of the region are chillies, cotton, jowar and vegetables. Irrigation is done from groundwater once in every two or three days and no surface ponding occurs.

12.5 Groundwater levels and flow direction

The Ganajur Main Gold deposit is located in a sub-catchment of the Varada River, with the ground surface elevation varying from about 610 mamsl in the south at the top of the catchment and sloping gently towards to the northeast and the Varada River at an elevation of 525 mamsl. The town of Haveri lies to the south of a hydraulic divide and in a different sub-catchment to the mine operations; however, the Ganajur village is in the same sub-catchment of the stream.

There is no record of historical ground water levels for the Ganajur Gold Project area. Groundwater levels monitored by Central Ground Water Board (CGWB) and Department of Mines and Geology (DMG) in the nearby locations have been collated. Since late September 2016, 24 private boreholes (MW01-MW24) in closer proximity to the Ganajur Gold Mine area (the locations of which are shown in Figure 12.4) have been monitored at intervals of roughly 10 days to present (April 2017). The water level data is summarised in Table 12.3.

In general, groundwater levels appear to follow topography, being at deeper depths under hills and shallower in low lands and valleys. Figure 12.34 also shows the groundwater piezometric contours in mamsl based on the most recent April 2017 monitoring data. Groundwater flow is towards the northeast although the localised impacts of pumping boreholes is apparent. The hydraulic gradient is lower in the upper reaches of the catchment and relatively moderate in the northern lower reaches of the catchment towards the confluence with the Varada River.

The groundwater table in the vicinity of the Ganajur Gold Mine pit area is about 30 mbgl to 40 mbgl. Figure 12.4 shows the hydrographs of the water level for the boreholes water levels closest to the mine pit area (MW13, MW14, MW20, MW21) and there is clearly a decreasing trend of at least 5 m to 10 m in all boreholes. As there is not yet a full year of monitoring data, it is not known if this is a seasonal response (following the end of the monsoon seasons) or a long-term trend due to over exploitation of the fractured rock aquifer.

On-going monitoring of the 24 boreholes for at least a year is required to establish the seasonal trends and establish a baseline prior to the mining operations. In addition, it is planned to install an additional three piezometers near the plant and mine pit areas. This data is required as mining of the open pit will impact on groundwater levels and so an understanding of the general trends (spatial and temporal) pre-mining is necessary.

Figure 12.4 Location of Ganajur Gold Project monitoring boreholes and piezometric contours

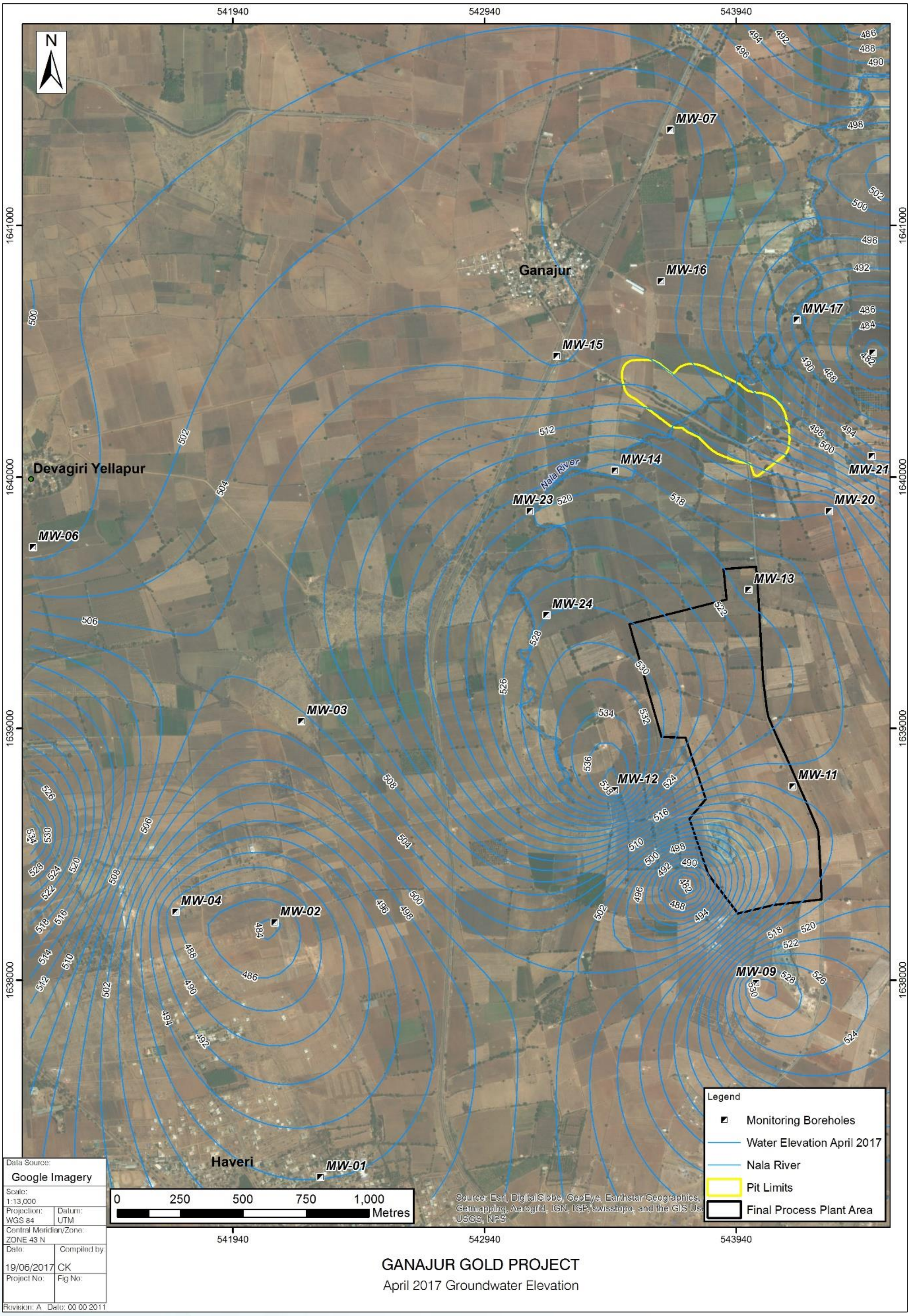


Figure 12.5 Hydrographs for monitoring boreholes closest to the proposed Ganajur mine pit

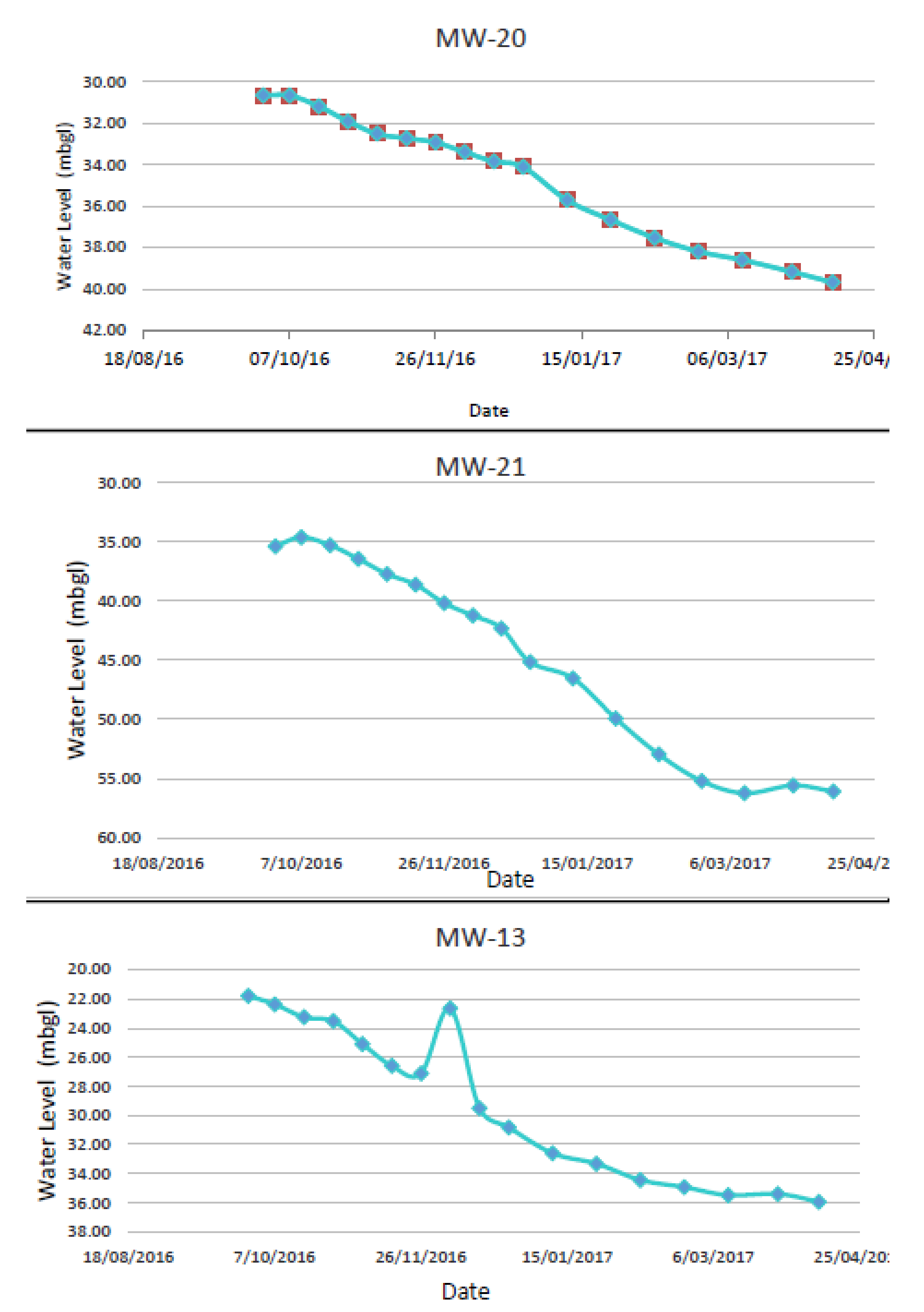


Table 12.3 Water level measurements (mbgl) for Ganajur Gold Project monitoring boreholes

Date	MW-01	MW-02	MW-03	MW-04	MW-05	MW-06	MW-07	MW-08	MW-09	MW-10	MW-11	MW-12	MW-13	MW-14	MW-15	MW-16	MW-17	MW-18	MW-19	MW-20	MW-21	MW-22	MW-23	MW-24
Collar elevation (mamsl)	580	583	561	583	574	571	554	554	568	561	560	558	555	549	557	556	548	552	549	556	551	538	556	548
28/09/2016	33.5	82.6	19.3	35.2	22.9	63.0	33.4	22.2	23.9	19.9	34.3	12.4	21.7	21.8	26.7	27.6	27.4	61.6	29.5	30.6	35.3	0.0	0.00	0.00
07/10/2016	23.7	75.8	19.3	91.50	23.2	62.0	33.6	23.6	25.8	20.1	35.9	13.4	22.3	21.8	27.5	29.2	27.7	61.6	29.1	30.6	34.6	31.4	22.9	13.7
17/10/2016	16.5	85.4	22.3	39.6	22.9	60.1	43.0	22.8	26.1	20.6	36.6	14.1	23.2	22.1	34.7	30.1	29.1	62.2	29.4	31.2	35.3	32.1	22.5	14.5
27/10/2016	49.9	80.3	24.6	37.8	23.4	64.7	40.5	23.1	26.5	20.8	39.3	14.5	23.5	22.4	44.4	32.5	30.9	62.9	30.0	31.9	36.4	33.6	24.6	15.4
06/11/2016	49.3	91.2	27.3	73.4	26.0	33.5	42.9	23.0	26.9	21.7	37.0	15.8	25.0	22.7	52.8	33.4	32.8	64.2	30.7	32.5	37.7	34.0	23.4	15.6
16/11/2016	19.6	99.6	30.3	66.2	27.3	37.1	39.9	23.3	27.4	22.1	41.1	16.1	26.5	23.2	54.0	34.5	34.6	65.7	31.1	32.7	38.6	36.2	24.5	16.0
06/12/2016	65.6	92.8	34.1	93.9	27.0	66.1	44.0	24.4	28.0	22.9	51.8	16.9	22.6	23.9	52.8	39.0	38.0	66.3	35.0	33.4	41.2	39.8	25.6	17.0
16/12/2016	24.8	99.6	38.5	78.2	27.4	67.1	46.4	24.6	28.5	23.2	51.0	16.7	29.5	24.4	48.7	40.2	41.0	66.5	40.1	33.8	42.3	41.6	26.3	16.7
26/12/2016	34.4	91.2	40.9	51.2	29.1	52.5	43.9	24.9	28.9	23.4	51.6	16.8	30.8	25.0	54.0	43.5	44.2	67.0	40.6	34.1	45.1	44.4	26.8	17.1
10/01/2017	30.1	94.6	44.9	77.0	33.9	70.2	45.5	25.6	29.3	23.7	52.9	17.0	32.6	25.6	52.2	48.3	47.5	68.4	44.9	35.7	46.5	48.0	27.2	17.4
25/01/2017	32.5	99.6	48.0	67.0	36.4	80.2	46.0	26.6	29.7	24.1	54.1	17.1	33.3	26.7	53.1	52.9	51.6	68.7	45.1	36.6	49.9	39.8	27.5	18.7
09/02/2017	38.3	99.6	50.3	87.0	36.9	53.5	44.4	74.1	33.9	24.1	53.3	17.2	34.4	27.3	53.6	55.4	54.8	69.4	49.2	37.5	52.9	43.7	36.7	19.6
24/02/2017	39.8	99.6	53.0	79.1	37.3	56.2	45.3	87.1	31.5	69.0	53.6	17.1	34.9	28.2	53.4	58.7	58.0	68.8	45.1	38.2	55.2	56.5	36.4	20.5
11/03/2017	55.8	91.6	55.6	78.6	37.7	70.2	45.6	86.8	36.0	65.5	53.8	17.2	35.4	29.0	54.2	59.7	59.3	69.8	54.3	38.6	56.2	56.6	36.4	20.9
28/03/2017	47.7	99.6	57.0	71.0	38.1	57.4	45.2	87.4	34.7	87.4	45.5	18.2	35.4	29.6	54.3	59.7	58.1	69.3	44.7	39.1	55.5	59.1	36.5	21.9
11/04/2017	84.0	99.6	55.6	95.4	38.7	72.5	47.8	46.9	35.8	82.14	43.0	18.0	35.9	29.9	54.2	50.2	59.0	68.1	43.9	39.6	56.0	57.5	36.1	19.9

12.6 Groundwater chemistry

The baseline hydrochemistry of the groundwater resources must be determined prior to the commencement of the mining, so that the potential impact of the mining operations on the groundwater quality can be assessed. For a FS, usually one year of monthly monitoring data is required to characterise the current status quo (pre-mining) with respect to spatial and seasonal variations in groundwater quality.

A total of 12 groundwater samples were collected from private boreholes during the hydrocensus (September 2016) in a 10 km radius around the study area (locations are shown in Figure 12.6). More recently 11 of the Ganajur Gold Project monitoring boreholes, closer to the project area, were sampled in March 2017.

The groundwater samples were analysed for physical parameters (pH, Total dissolved solids (TDS) and Total Hardness), major cations (calcium, magnesium, sodium, potassium) and anions (bicarbonate, chloride, nitrate, sulphate) and minor constituents (fluoride, iron). The analytical results are shown in Table 12.4 and Table 12.5, respectively.

Five of the Ganajur Gold Project monitoring boreholes (MW11, MW12, MW13, MW17 and MW21) were also sampled for heavy metals arsenic, zinc, lead, mercury, copper and cyanide (as total concentrations as samples were acidified immediately on collection).

The baseline water quality numbers in red indicate values in excess of the permissible limits and orange numbers indicate levels above the desirable limits.

Figure 12.6 Map showing location of boreholes sampled for hydrochemical analyses

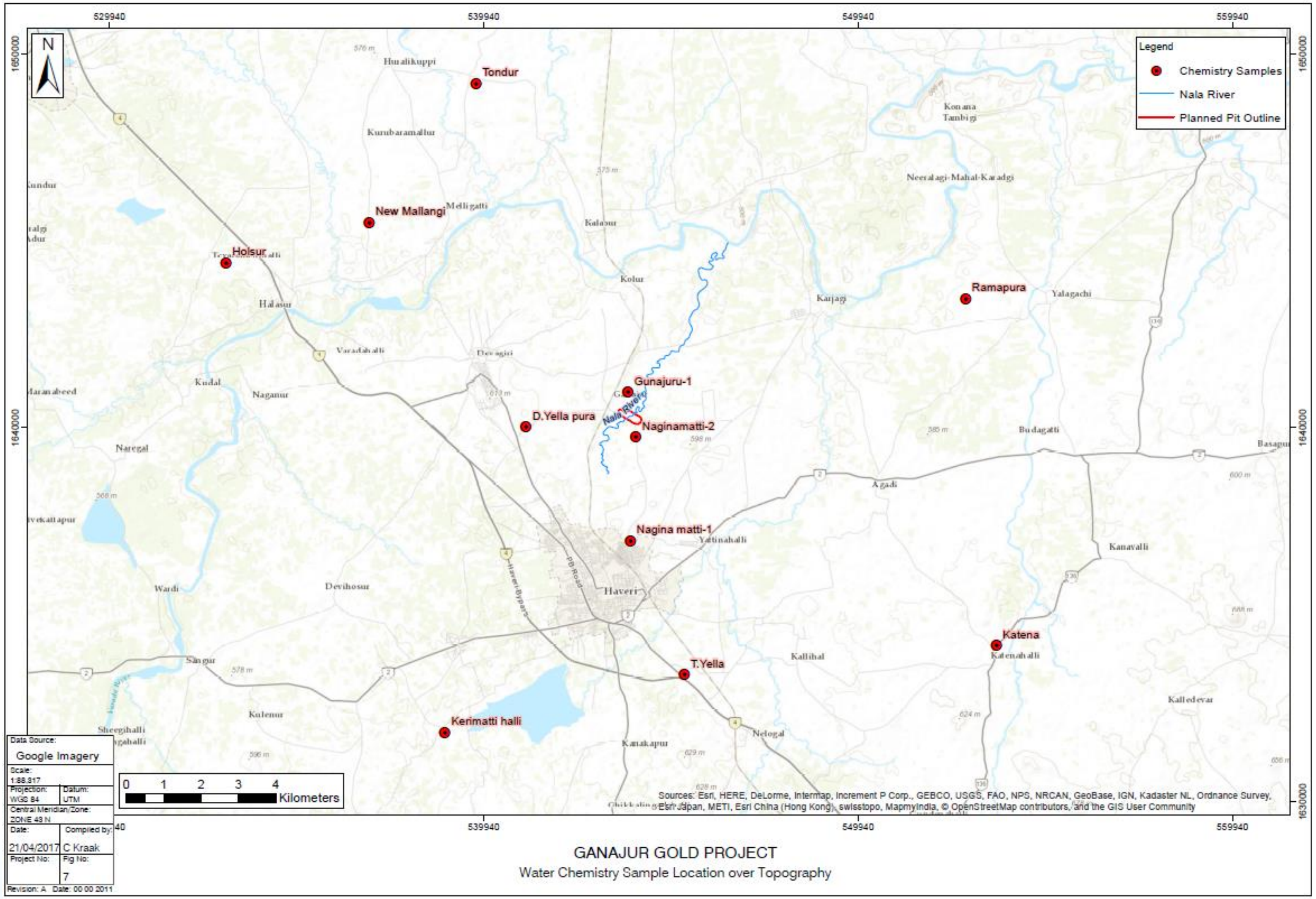


Table 12.4 Analytical results for groundwater samples collected regionally during hydrocensus

Chemical constituent	Unit	Desirable limit	Permissible limit	Sample location											
				Gunajuru-1 (MW17)	Holsur	New Mannangi	Tondur	D.Yella pura (MW05)	Kerimatti halli	T.Yella pura	Katena halli	Ramapura	Nagina matti-1	Ganajuru-2 (MW21)	Nagina matti-2
pH		*6.5 – 8.5*	*N.R.*	7.25	7.21	6.99	7.3	7.3	7.57	7.04	7.33	7.81	7.47	6.8	7.2
Total dissolved solids	mg/L	*500*	*2,000*	804	1,089	1,688	1,104	1,377	811	1,714	804	907	816	909	1,113
Electrical conductivity	µS/cm			1,249	1,655	2,150	1,435	1,886	990	25,40	1,068	1,343	1,198	1,310	1,591
Total hardness	mg/L	*200*	*600*	532	540	760	588	752	384	1000	448	356	508	512	648
Major cations															
Calcium	mg/L	*75*	*200*	142	144	192	142	141	98	304	131	86	168	120	166
Magnesium	mg/L	*30*	*100*	43	44	68	56	97	34	58	29	34	21	52	56
Sodium	mg/L			110	268	340	148	204	71	212	67	256	62	176	82
Potassium	mg/L			3	6	1	1	5	3	4	4	2	3	2	2
Major anions															
Carbonate	mg/L			0	0	0	40	0	40	0	0	40	32	0	0
Bicarbonate	mg/L			332	440	440	296	372	268	260	244	332	260	432	344
Chloride	mg/L	*250*	*1,000*	177	200	471	519	272	113	481	160	126	170	155	245
Nitrate	mg/L	*45*	*N.R.*	61	86	22	74	45	45	177	24	130	58	16	20
Sulphate	mg/L	*200*	*400*	33	103	108	82	171	20	82	41	256	42	76	110
Minor constituents															
Fluoride	mg/L	*1*	*1.5*	1.16	0.25	0.92	1	0.44	0.45	1.09	1.04	1.29	0.35	1.12	0.55
Iron	mg/L	*0.3*	*N.R.*	<0.01	<0.01	<0.01	<0.01	<0.01	<0.01	0.26	<0.01	<0.01	<0.01	<0.01	0.01

Note: Numbers coloured red exceed the permissible limit and numbers coloured orange exceed the desirable limit

Table 12.5 Analytical results for groundwater samples from Ganajur Gold Project monitoring boreholes

Chemical constituent	Unit	Desirable limit	Permissible limit	Sample location										
				MW-01	MW-05	MW-11	MW-12	MW-13	MW-16	MW17	MW-19	MW-21	MW-22	MW-24
pH	mg/L	*6.5 – 8.5*	*6.5 – 8.5*	6.8	6.6	6.6	6.6	6.7	6.5	6.9	6.8	6.7	7.3	7.1
Total dissolved solids	mg/L	*500*	*2,000*	882	905	1,220	858	1,027	1,442	728	756	1,032	685	890
Total hardness	mg/L	*200*	*600*	473	484	627	440	616	891	264	275	616	264	473
Major cations														
Calcium	mg/L	*75*	*200*	104	110	150	112	126	193	58	56	133	57	112
Magnesium	mg/L	*30*	*100*	52	51	60	38	74	99	30	33	68	30	46
Sodium	mg/L			85	86	121	75	58	89	123	137	76	112	69
Potassium	mg/L			BDL	1.6	1.4	1.1	BDL	2	BDL	BDL	2.8	BDL	1
Major anions														
Bicarbonate	mg/L			313	355	326	290	319	406	326	406	413	355	297
Chloride	mg/L	*250*	*1000*	118	120	216	88	158	235	61	0	134	42	117
Nitrate	mg/L	*45*	*N.R.*	24	16	46	78	27	59	BDL	BDL	7	BDL	BDL
Sulphate	mg/L	*200*	*400*	43	34	50	42	50	91	37	27	55	32	30
Minor constituents														
Iron	mg/L	*0.3*	*N.R.*	BDL	BDL	BDL	BDL	BDL	BDL	BDL	BDL	BDL	BDL	BDL
Arsenic	mg/L	*0.01*	*0.05*			BDL	BDL	BDL		0.027		0.030		
Zinc	mg/L	*5*	*15*			0.05	BDL	0.20		0.1		BDL		
Lead	mg/L	*0.01*	*N.R.*			BDL	BDL	BDL		BDL		BDL		
Mercury	mg/L	*0.001*	*N.R.*			BDL	BDL	BDL		BDL		BDL		
Copper	mg/L	*0.05*	*1.50*			BDL	BDL	BDL		BDL		BDL		
Cyanide	mg/L	*0.05*	*N.R.*			BDL	BDL	BDL		BDL		BDL		

Note: Numbers coloured red exceed the permissible limit and numbers coloured orange exceed the desirable limit

12.6.1 Physical parameters

Generally, the pH of the groundwater is neutral (6 to 7); however, the geochemical studies (Geostratum, 2017) have shown that the ore and tailings samples from the mining operations are likely to be acid generating if subjected to long term oxidation in atmospheric conditions when the sulphide content is above 2.5% to 3%. This means that acid rock drainage could impact on the groundwater quality resulting in a lower pH and propensity to dissolution of metals if present, both the tailings dam and the ore stockpiles are therefore going to be lined. The waste rock material (quartzite, greywacke and brecciated chert) is non-acid generating.

The groundwater in contact with the ore zone must also be sampled to determine if the water quality is different from the regional aquifer. The higher concentration of sulphides may result in localised poorer water quality once exposed to the atmosphere during mining and could result in the formation of an acidic pit lake post closure.

The Total Hardness is very high (classified as very hard water >180 mg/L), which can result in scaling of equipment and pipes due to the deposition of calcium carbonate.

The total dissolved salts (TDS) varies from 700 mg/L in the upper reaches of the catchment (MW01) increasing to >1,700 mg/L in boreholes closer to towns. In terms of the Bureau of Indian Standards, the desirable drinking water limit is 500 mg/L and the maximum permissible limit is 2,000 mg/L, indicating that all of the samples collected were more than the desirable limit but less than the maximum limit. Boreholes within the proposed pit area are within the range 800 mg/L to 1,100 mg/L.

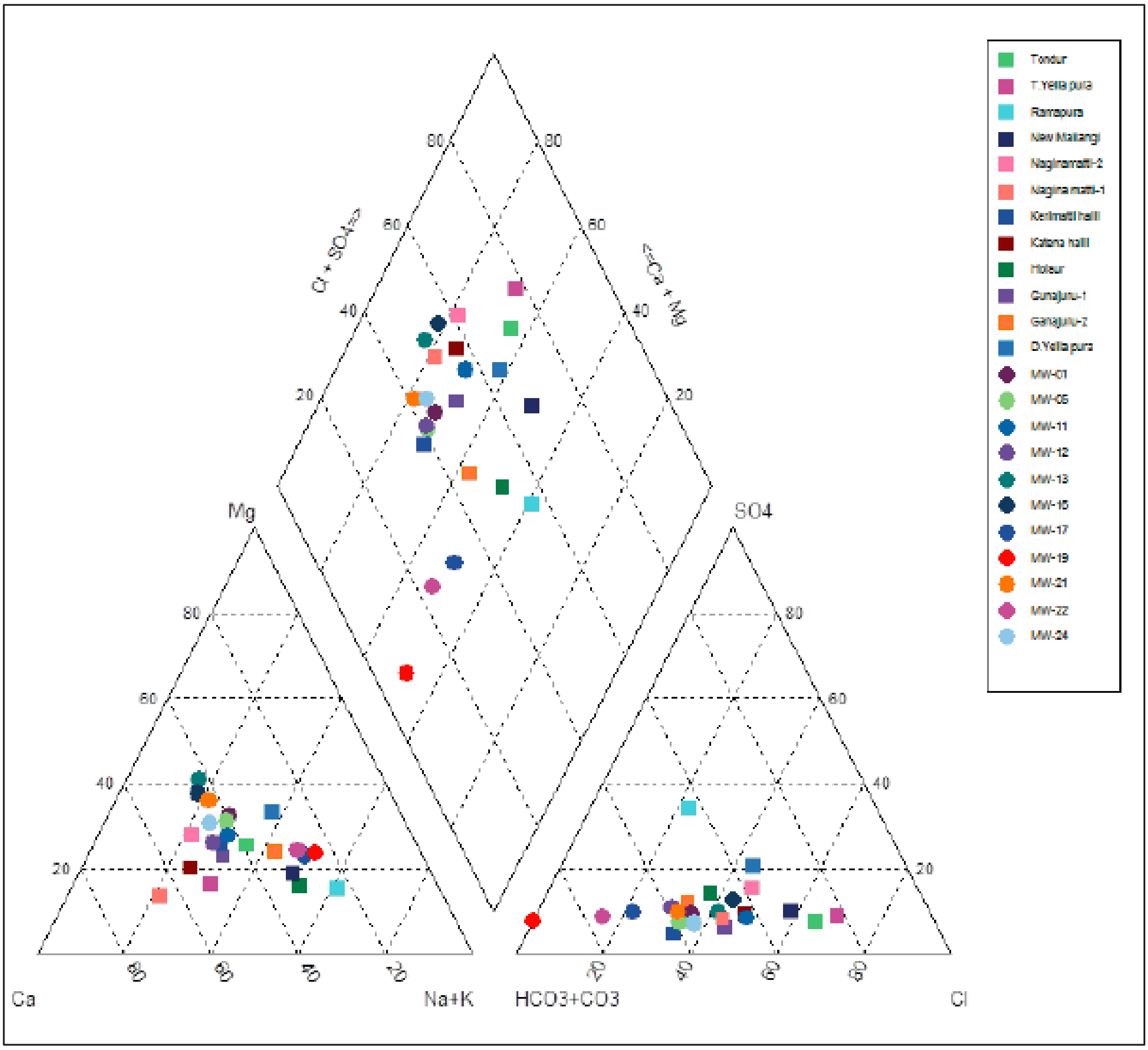
12.6.2 Major ions

The elevated TDS is due mainly to chloride and bicarbonate as the major anions and sodium and calcium as the major cations. These elevated levels are due to evapoconcentration of salts due to irrigation and a build-up of salinity in the soil profile that gets flushed through to the groundwater during the monsoon seasons.

A piper plot of the concentrations of major cations and anions (converted to milliequivalents/litre (mq/L) is shown in Figure 12.7. The trend due to accumulation of salts through evapoconcentration of the irrigation water in the soil profile and subsequent dissolution of salts, resulting in the samples plotting in the centre of the diamond-shaped field (parallelogram). Groundwater that is recently recharged from rainfall would normally plot tightly in the left-hand corner of the diamond-shaped field. The piper plot therefore confirms the impact of irrigation and salt accumulation on the groundwater quality.

Nitrate is generally above the permissible drinking water limit of 45 mg/L and this is due to the high usage of inorganic nitrogenous fertilisers and manures as well as animal and human waste. In the Project study area, the boreholes near the Ganajur village have elevated nitrate levels >60 mg/L; this is important to note, as increasing concentrations of nitrate are usually associated with mining activities due to the explosives that are used, so background values in the groundwater are important to establish pre-mining.

Figure 12.7 Piper diagram for groundwater samples from hydrocensus and Ganajur Gold Project monitoring boreholes



12.6.3 Heavy metals

Irrigation using groundwater is also concentrating the arsenic and other heavy metals in the soil and groundwater. Results of chemical analyses by State agencies have shown the presence of iron and arsenic in excess of the standard limits of 0.3 mg/L and 0.05 mg/L respectively in the villages, Haveri taluk and Savanur taluk.

The monitoring boreholes near the Ganajur Gold Mine proposed mine area MW17 and MW21 have arsenic levels exceeding the acceptable limit of 0.01 mg/L but lesser than the permissible limit of 0.05 mg/L. The geochemical studies (Geostratum, 2017) have shown that although arsenic is partitioned in a fairly insoluble phase as arsenopyrite requiring oxidation or strong acid for release from the material, leachates from the ore, tailings and halo waste rock material are higher than the Indian Drinking Water Standards but below the General Effluent Standards. In the short term, the halo waste rock material leaches more arsenic than the ore material although it has lower total abundances of arsenic and this phenomenon is being investigated further through kinetic leach testing.

The placement and construction of the mine infrastructure (TSF and ore stockpiles) must take into consideration the high potential for leaching arsenic as this could exacerbate the poor groundwater quality which is already contaminated with arsenic.

12.7 Groundwater users

A hydrocensus was conducted within a 10 km radius of the Project area. In most of the study area, hand dug wells exploiting the shallow phreatic or perched aquifer are not common and those observed during the hydrocensus were all dry. Groundwater is exploited from the fractured rock aquifer system from boreholes at depths of 50 to 150 mbgl. There are three villages within the core project area along with the northern outskirts of Haveri town. A total of 213 wells, which are actively pumped for about six to eight hours a day for eight to 12 months a year, were identified.

The groundwater is used for irrigating dry crops by wetting the surface and allowing it to infiltrate and so no ponding occurs. The major crops grown in the area are cotton, chillies, jowar and other millets, as the water requirements are low. Other than agriculture, groundwater is also abstracted for drinking water purposes.

The aquifer, in the schistose rock (greywacke), has moderate potential with an average borehole abstracting 40 m^3 in a day in monsoon season, 28 m^3 in a day in post-monsoon season and 20 m^3 in a day in the summer season. Higher extraction of 90 to 190 m^3/day was observed at some locations.

Depending on the power supply, ground water is extracted for about six to eight hours in a day and for eight to 10 months in a year. The abstraction rates, from boreholes in the stream catchment, varies from less than 1 m^3/hr to 10 m^3/hr with higher rates up to 30 m^3/hr at some locations. The reported withdrawal rates differ in each of the three main seasons (i.e. monsoon, post-monsoon and summer) as shown in Table 12.6.

The groundwater resource estimation conducted in March 2011 by the Department of Mines and Geology and Central Ground Water Board (CGWB) classified the Haveri Taluk of Haveri District as “Safe” as the groundwater development is only 68% of the available resource. In view of the rapid industrial development, increase in irrigation for food security, the social responsibility of industry is to protect and conserve the groundwater resources in parallel with development through rain water harvesting and recharge of groundwater. In terms of the available guidelines by these Departments, businesses shall adopt artificial measures to enhance natural recharge to groundwater if groundwater extraction exceeds 10 m^3/day in an area declared as “safe” category.

Table 12.6 Average groundwater abstraction (m^3/day) from boreholes in the stream catchment

Season	Average groundwater balance m^3/day
Monsoon season (June to October)	7.714
Non-monsoon winter season (September to February)	4.739
Non-monsoon summer season (March to May)	3.553

12.8 Conceptual hydrogeological model

The data collated to date has been used to develop a conceptual hydrogeological model for the Ganajur Gold Project and forms the basis of the numerical groundwater flow model. The conceptual model is shown in Figure 12.8.

The conceptual model consists of the top weathered zone, which is unsaturated and above the water table to depth of about 30 m. The main aquifer associated with the fractured rock occurs from 30 mbgl to 120 mbgl (460 mamsl to 555 mamsl). The effective hydraulic conductivity for this zone is assumed at 0.4 m/day. Below this, the fracturing and weathering decreases with depth into the fresh greywacke with lower hydraulic conductivity. Infiltration of rainfall is estimated at 10% to 12% but is seasonally controlled.

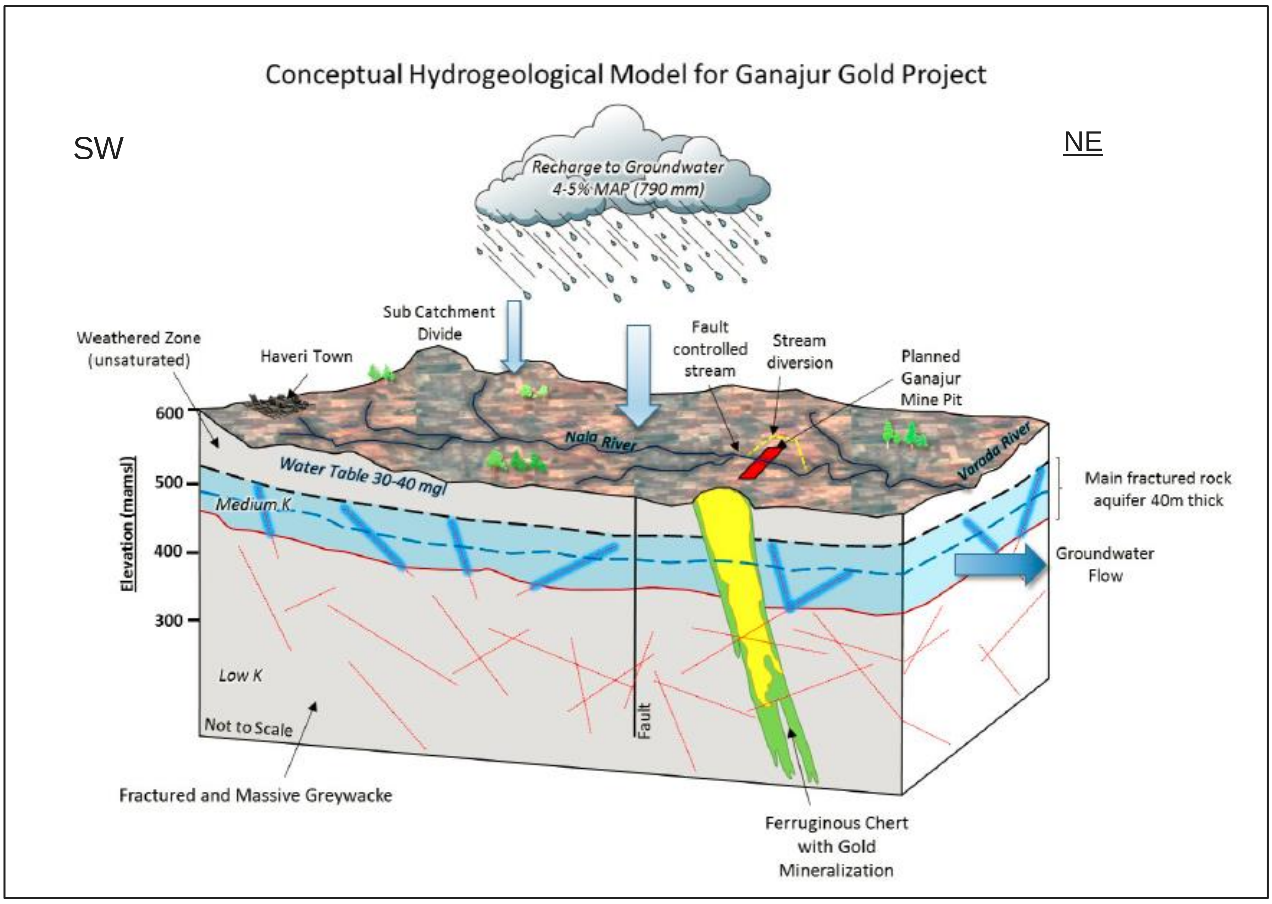
The mining operation will be excavated to a depth of 85 m, intersecting the water table from about 510 mamsl to the bottom of the pit at around 460 mamsl. Groundwater ingress into the pit will have to be managed appropriately by either active or passive dewatering (or both).

High pore pressures, particularly on the southern pit walls where the stream occurs, could be a risk in terms of slope stability. The greywacke footwall is dipping into the pit on this side and although the stream will be diverted, as it is fault controlled, groundwater flow along this structure will continue to occur.

The groundwater quality is within permissible limits but exceeding desirable limits and is characterised by elevated salinity and arsenic due to irrigation of the surrounding agricultural lands and human/animal wastes/fertilisers has resulted in elevated nitrate levels. The ore and TSF materials have the potential to generate acid rock drainage and to leach arsenic as well as increasing the nitrate loading due to explosives use, hence the TSF will be lined.

The mining operations (open pit mining, TSF, ore stockpiles) could therefore impact on the main aquifer in terms of water levels and water quality. Other users in the catchment are dependent on the groundwater resources for irrigation and water supply. To quantify the risks to the groundwater resources, a numerical flow model has been developed to simulate the mining activities and predict the impacts on the groundwater resources. A robust groundwater model can also be used to determine the dewatering/depressurisation requirements during the life of mine.

Figure 12.8 Conceptual hydrogeological model for Ganajur Gold Project



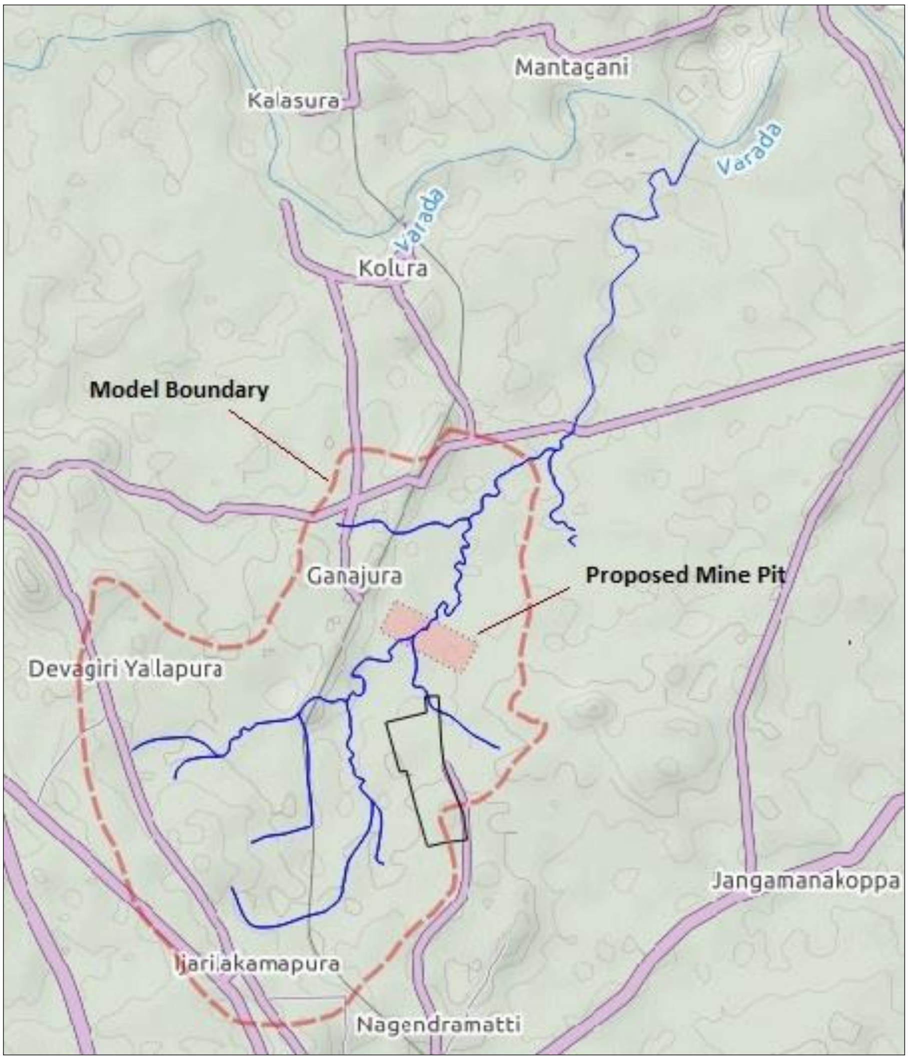
12.9 Hydrogeological flow simulation

A numerical flow model was developed on the basis of the conceptual hydrogeological model, using Visual Modflow software (Waterloo), which is based on the industry standard three-dimensional, finite-difference model code MODFLOW, (developed by United States Geological Survey), with the purpose of simulating groundwater flow for the Ganajur Gold Mine Project. The numerical flow model is then used to determine the impact of mining on the groundwater in terms of a dewatering cone developing around the open pit and to predict the volumes of ingress into the pit that would have to be managed during the mining operations.

12.9.1 Hydrologic study area

The catchment or hydrologic study area (HSA), selected for modelling, covers an area of 16 km^2 as shown in Figure 12.9. The stream starts from Haveri–Devagiri region, flows in a north-northeast direction, on the east of Ganajur village and finally joins the Varada River near Mantagani village. The catchment of the river is well defined, with the topographic highs forming the model boundaries.

Figure 12.9 HSA selected for the groundwater model boundaries

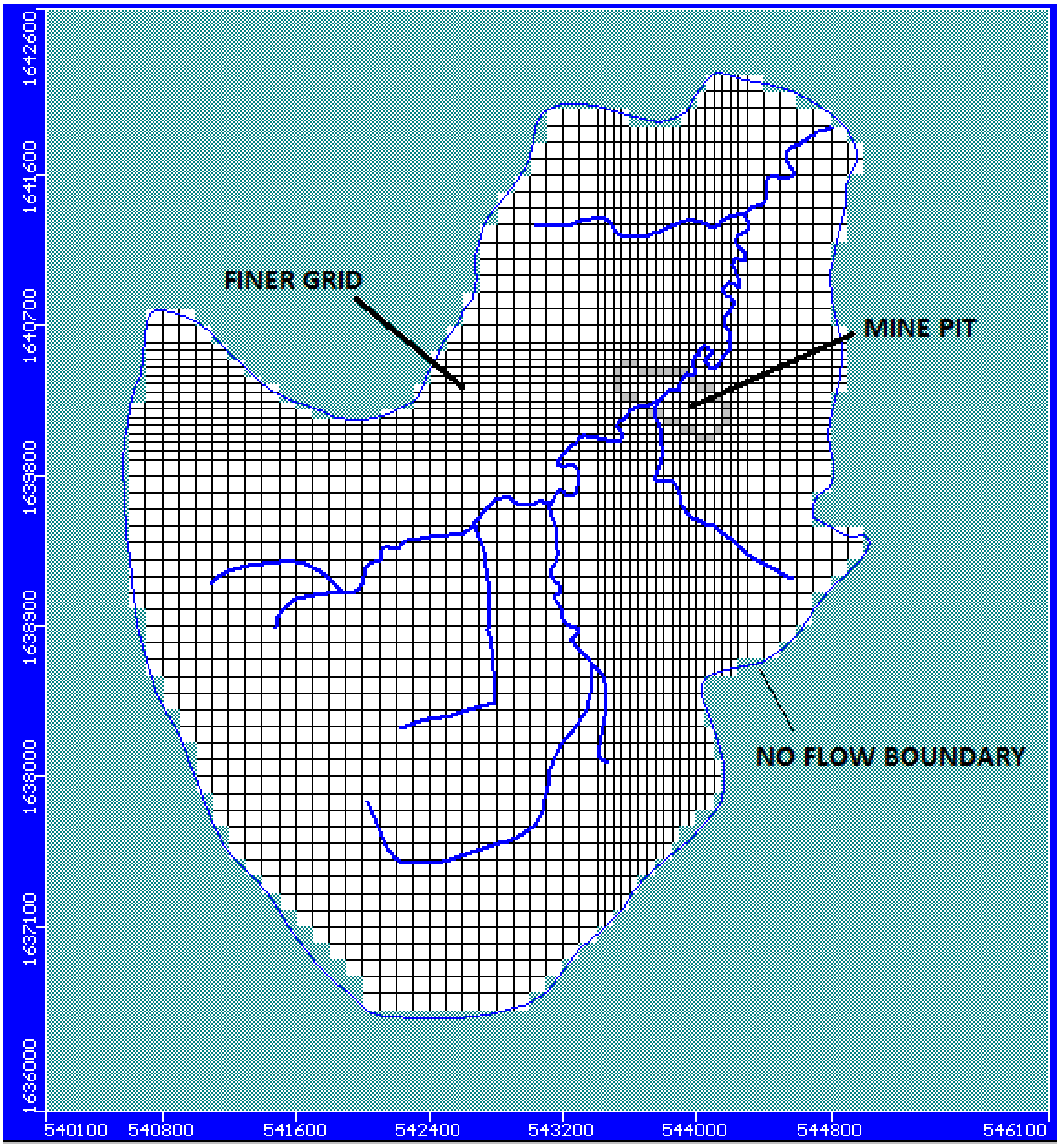


12.9.2 Model set-up

Model grid

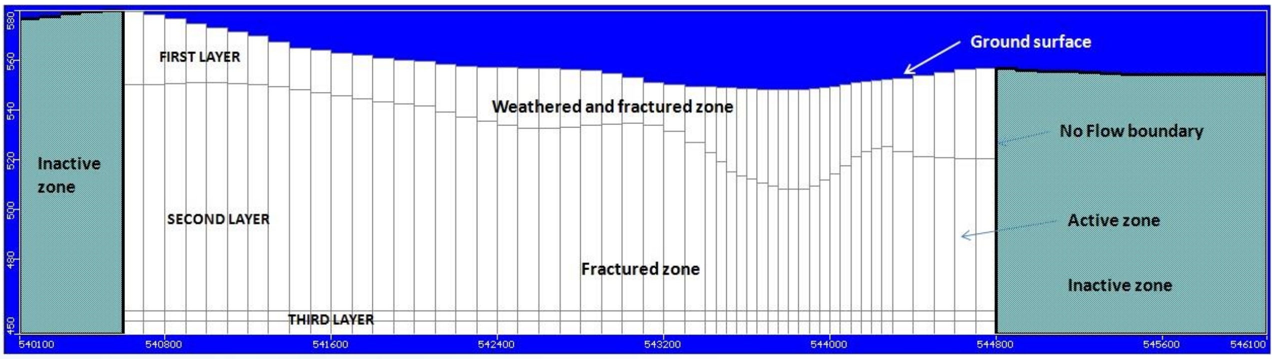
The model grid consists of 60 rows and 48 columns of approximately 98 m x 98 m blocks which in the vicinity of the mine pit, each block is more finely discretised as nine cells (3 m x 3 m), to obtain more precision as shown in Figure 12.10.

Figure 12.10 Model domain and grid showing proposed mine pit



The model thickness is approximately 100 m, consisting of three layers as shown in Figure 12.11. The top layer is bounded by the ground surface using SRTM data and the contact of the weathered zone with the fractured rock aquifer (mainly unsaturated). The second layer is the main fractured rock aquifer. The third layer is less fractured (lower permeability) and split into two thin layers of 5 m each, with the bottom of the model at 450 mamsl.

Figure 12.11 Cross section showing model layers



Model boundaries

Boundary conditions define the locations and manner in which water enters and exits the active model domain. The conceptual model for the aquifer system is that water (1) enters the system as recharge from precipitation (rainfall) and recharge from the delivery and application of surface-water irrigation, and (2) exits the system as streamflow, evapotranspiration, seepage and groundwater withdrawal.

No-flow boundary conditions were specified at the major topographic divide that coincides with the lateral model boundary, which is assumed to be groundwater divide. Subsurface groundwater outflow is simulated using a general-head boundary in the north-east corner of the model. The streams do not carry appreciable flows and as such the streams, which are of 1st to 3rd order, are not considered under boundary conditions.

Six types of model boundaries are used:

- No-flow boundary (groundwater divides) along the watershed divide
- General-head boundary near north-east corner of the area
- Recharge to the top active layer
- Evapotranspiration from top layer
- Groundwater extraction through boreholes
- Outflow as seepage draining into mine pit.

Recharge

Recharge is mainly through rainfall as there are no major surface water bodies in the model area and the variable rainfall distribution has been assigned according to soils in the catchment. The recharge due to irrigation is considered as negligible.

Three periods of stress on the groundwater system can be considered for simulation of groundwater flow. The stress periods are related to recharge and withdrawal in this area.

- Monsoon period from June to October – maximum recharge
- Post-monsoon period from November to February – withdrawal
- Summer season from March to May – less withdrawal.

Groundwater abstraction

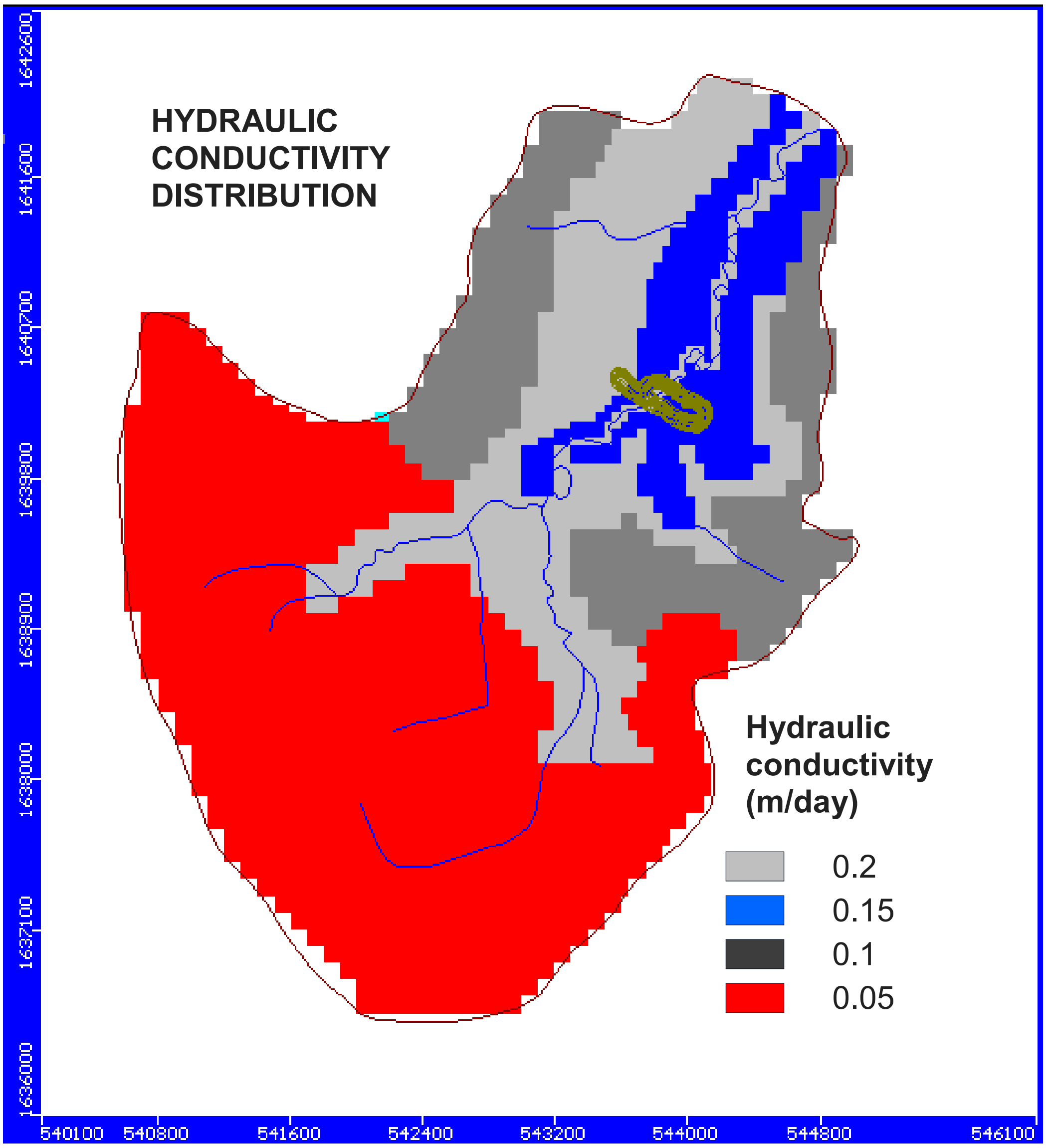
Pumping of groundwater from the 213 active private boreholes located mainly in the central and eastern parts of the catchment, is the main outflow in the area.

At present, there is no mining activity. The mine pit designs for a depth of 30 m, 60 m and 80 m were used for simulating groundwater flow conditions under transient conditions.

Hydraulic parameters

Hydraulic conductivity for the main aquifer (Layer 2) has been assigned with values ranging from 0.05 m/day to 0.2 m/day as shown in Figure 12.12. Specific yield of 0.25 has been used for first layer and storage coefficient of 0.0001 to 0.0006 for the second layer.

Figure 12.12 Hydraulic conductivity (m/day) assigned to second layer (main aquifer)

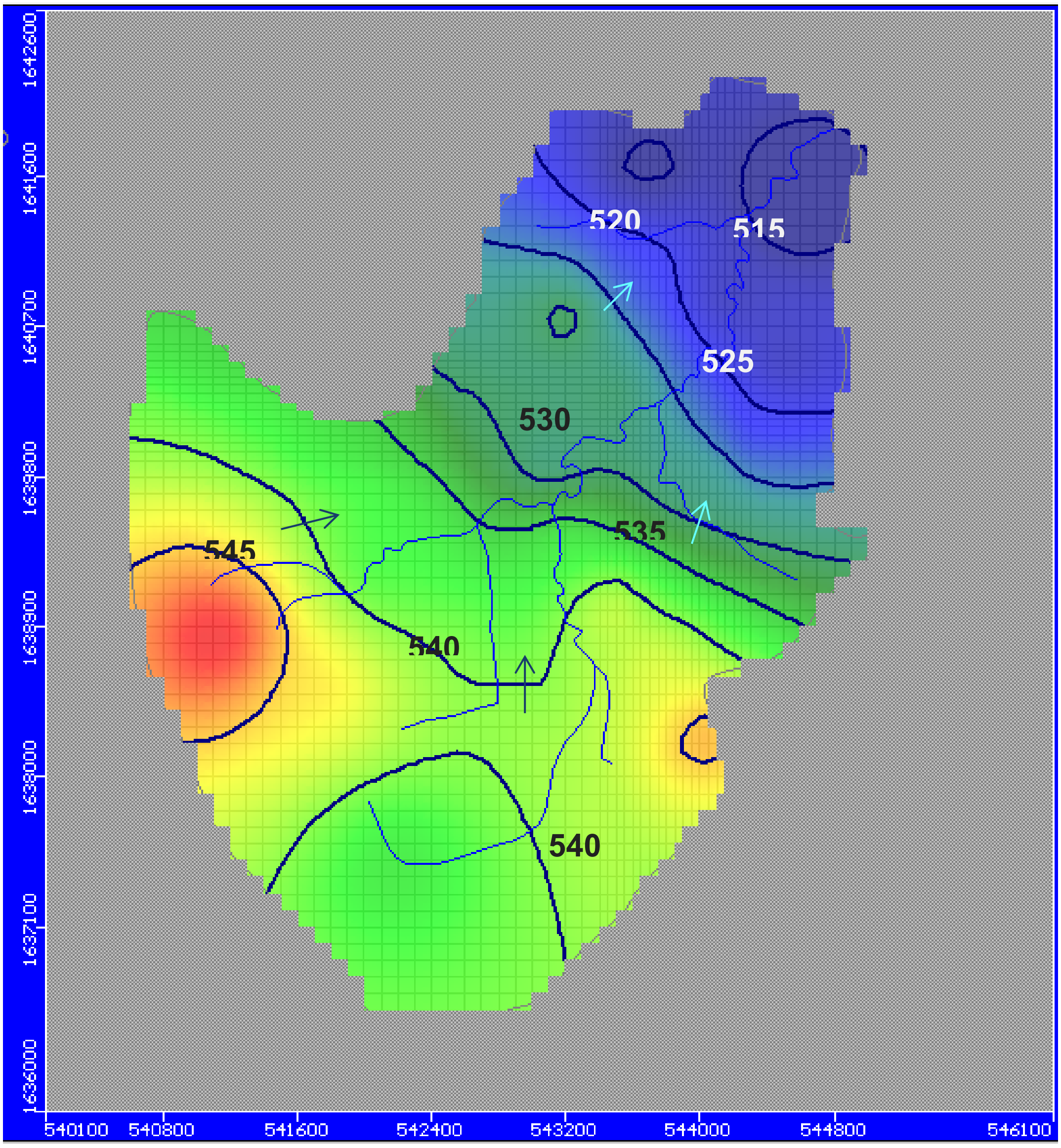


12.9.3 Steady state conditions

The model was set up using the water levels recorded in the 24 non-pumping boreholes (Table 12.3) at the end of September 2016 as the initial piezometric heads of the model (Figure 12.13). The map shows the general ground water flow from southwest towards northeast following the topography of the area.

For calibration purposes, recharge was considered for the month of October 2016. No recharge was considered for November and December 2016 as there was no significant rainfall. Initially recharge of 11% of the normal monthly rainfall was used and was eventually revised to a range of 4% to 5% spatially. This will be further refined as the modelling study progresses following more data collected.

Figure 12.13 Initial piezometric heads in second layer used in model set up (September 2016 water levels)

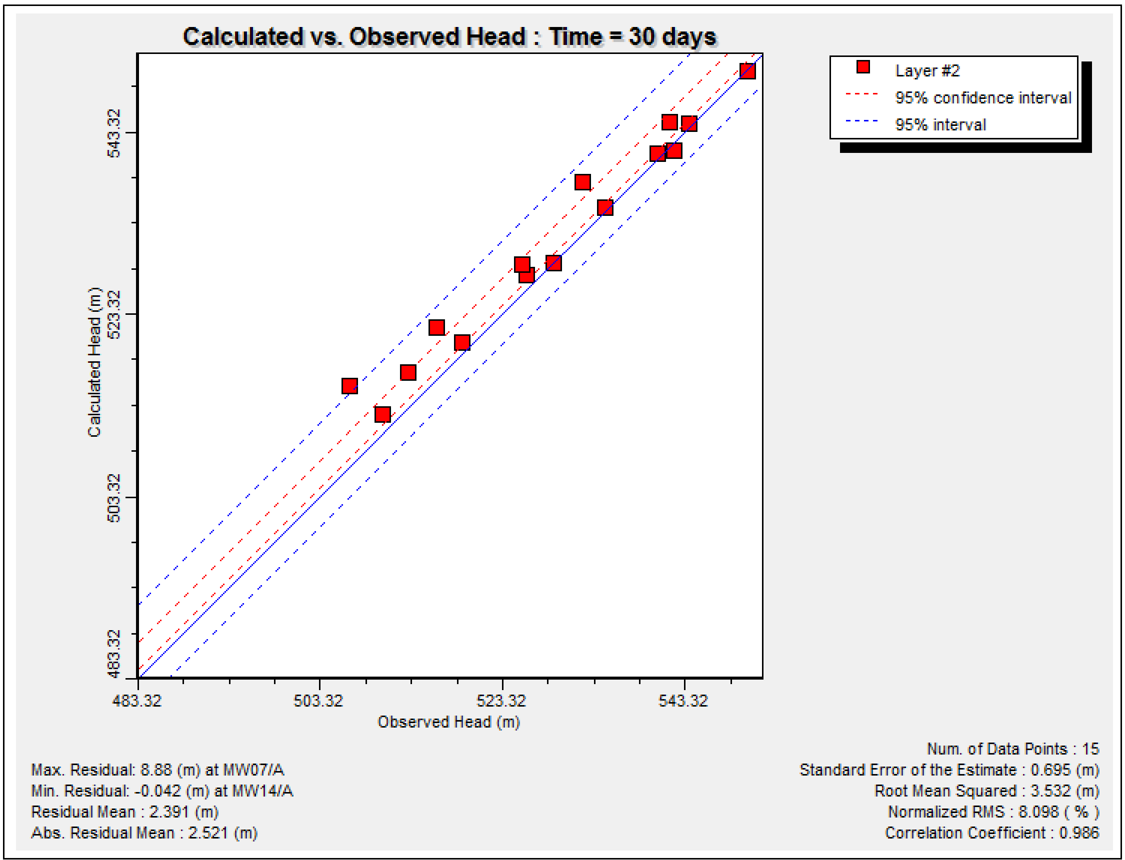


12.9.4 Model calibration

Hydraulic conductivity, storage coefficient, recharge values are the key input parameters that are adjusted manually with appropriate acceptable values until a fairly acceptable match is obtained between observed and the computed water levels. The calibrated model results for 30 days are shown as Figure 12.14. The water level data generated for a short period of three months, at 10-day frequency was used for calibration, comparing the measured water levels against the computed values until a reasonable fit was achieved.

The model shall be refined on a regular basis as more data is acquired.

Figure 12.14 Model calibration after 30 days – observed vs. computed heads (water level)



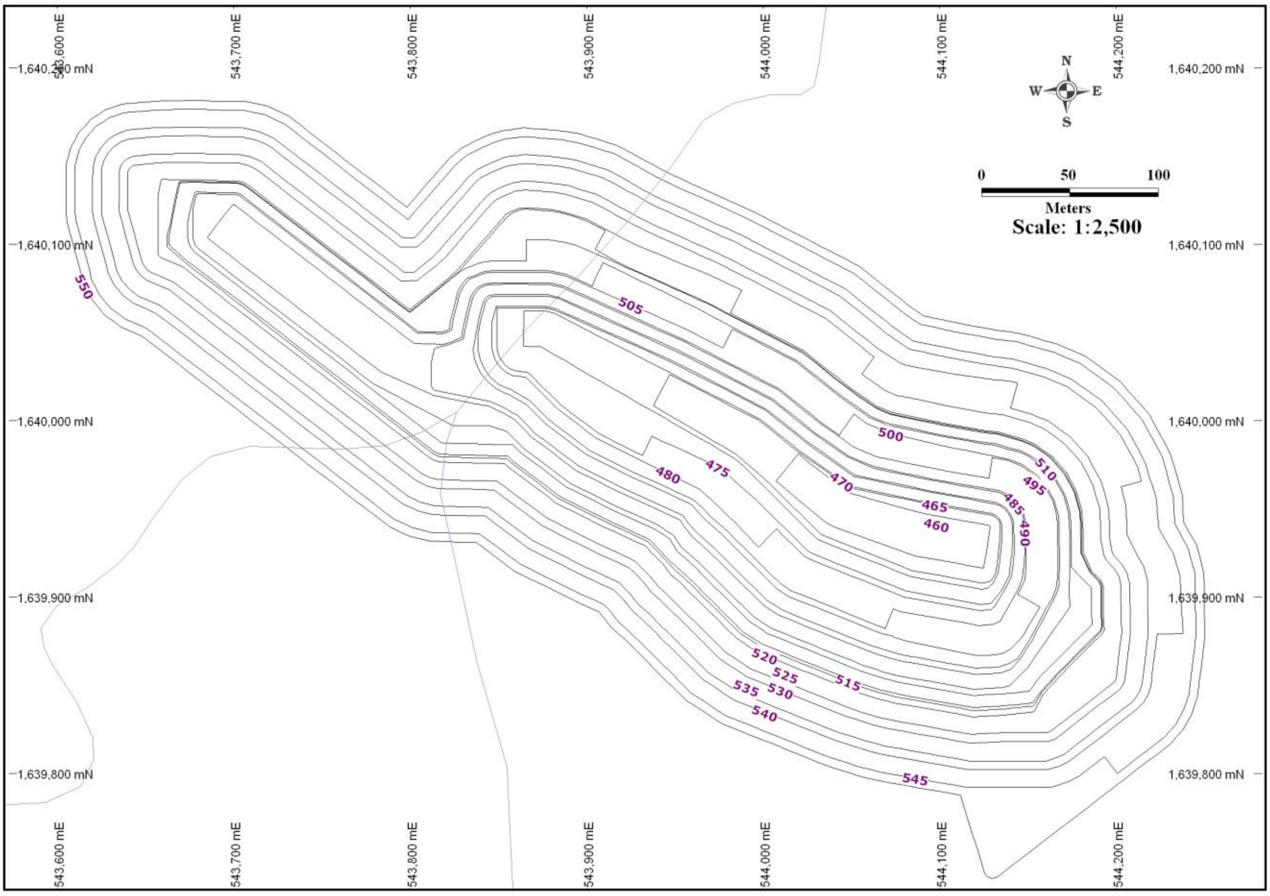
12.9.5 Transient conditions due to mining of Ganajur Gold Mine Pit

The model was run under transient conditions for 360 days to simulate the annual variations in rainfall and borehole abstraction and a reasonable fit was obtained.

The calibrated transient model was then run to simulate the commencement of the Ganajur Gold Mine mining operations using the mine plans provided and shown in Figure 12.15. Once the pit is excavated below 510 mamsl, the groundwater table will be intersected and ingress into the pit will begin to occur, increasing with depth due to the steeper hydraulic gradient and as new water bearing fractures/faults/joints are exposed. The transient model was used to predict the ingress rates in steps according to the mine plans (i.e. different depths and excavated areas).

The lower boundary of the model is 450 mamsl and the mine pit bottom level is 460 mamsl. Ideally, the lower boundary should be much deeper than the bottom of the pit, in order to minimise boundary effects.

Figure 12.15 Ganajur Gold Mine pit plan with elevations (mamsl)



12.9.6 Model outputs

Groundwater ingress rates

The volume of water, collected within the mine pit due to seepage from fractures and joints and from direct rainfall must be estimated in order to manage the water (i.e. size sumps, pumping systems and usage or disposal of excess water).

For estimation of the groundwater ingress, the mine boundary cells in the calibrated model are converted to drain boundary cells and the cells within the proposed mine pit area are made inactive. The deepening of the mine pit was simulated for incremental 5 m steps over a six-month period and the passive ingress into the pit was determined from 510 mamsl (from water table) for each pit shell.

Table 12.7 shows the predicted groundwater ingress rates which increases from 6 m^3/day to a maximum of 1,900 m^3/day when the lowest bench is mined at 465 mamsl. Analytical calculations using the Jacob Lohman Equation for T=20 m^2/day (average late recovery test-pumping results) are shown in Figure 12.16 and radius of pit 470 m (r_p) correlate well with the model results – groundwater ingress of 2,000 m^3/day when the lowest bench is mined (85 m depth).

Table 12.7 Predicted groundwater ingress into Ganajur Gold Mine pit (passive dewatering)

No. of days of operation	Mine pit level (mamsl)	Seepage estimate (m³/day)
180	525	0.0
360	520	5.9
540	515	140.8
720	510	403.1
900	505	651.1
1,080	500	859.2
1,260	495	1,038.0
1,440	490	1,193.8
1,620	485	1,373.9
1,800	480	1,526.7
1,980	475	1,663.9
2,160	470	1,789.4
2,340	465	1,908.6

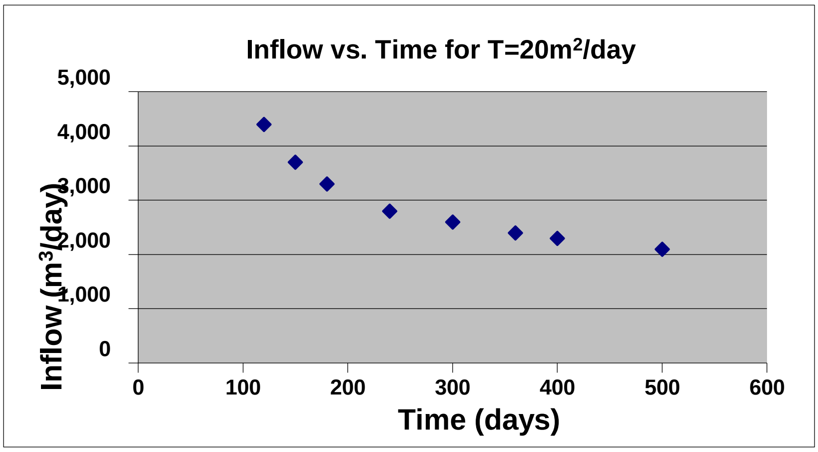
Figure 12.16 Jacob Lohman estimate of pit ingress

$$Q = f\left[\frac{4\pi T \Delta h}{\ln\left(\frac{2.25\,T\,t}{r_p^2 S}\right)}\right]$$

For 85 m deep pit

f	1	
T	20	m²/day
Δh	30	m
*r*p	470	m
S	5,00E-03	[ ]

10m seepage face



In addition to the ingress of groundwater into the pit, the volume of direct precipitation directly into the pit must be taken into consideration when determining the total volume of water to be pumped out of the pit during the monsoon seasons.

Table 12.8 shows the estimated volume of direct rainfall that collects in the pit, assuming the mine area is 119,900 m² and approximately 65% of the rainfall less evaporation. The total volume must be summed with the pit ingress when sizing pumps, volumes to be used/disposed of.

Table 12.8 Direct rainfall to be pumped out of pit

Season of the year	Average rainy days	Normal rainfall (mm)	Rainfall (m/d)	Rainwater to be dewatered (m³/day)	Total rainwater (m³)
Southwest monsoon	61.1	495.1	0.0081	631.3	38596
Northeast monsoon	10.3	169.9	0.0166	1293.7	13274
Non-monsoon	10.8	126.9	0.0118	919.6	9904

Cone of drawdown

As the depth of excavation of mine pit increases, the area of influence of drawdown of the groundwater table also increases and can be simulated using the groundwater model.

If the cone of drawdown (or zone of influence) extends as far as other boreholes in the catchment, it can lower the groundwater table and reduce the yield, negatively impacting other private users. The boreholes that are exploiting fractures that are deeper than the bottom of the pit, may in fact reduce groundwater ingress into the pit.

The drawdowns have been computed by the model for different levels of mine operation and are shown in Figure 12.17. The cone of drawdown expands until when mining the 465 mamsl bench, a drawdown of the water table by over 35 m along strike has reached the model boundaries and potentially impacts many users in the catchment including the Ganajur village. Future modelling updates should enlarge the model boundaries so that the drawdown cones do not expand to the edge of the model (and the bottom of the model should be deepened).

Figure 12.17 Drawdown cones (m) for different levels of Ganajur Gold Mine pit

12.10 Potential disposal of excess groundwater

The current estimated steady-state process water requirement for the Ganajur Gold Project is approximately 760 m^3/day and will initially be sourced from the Varada River by pipeline; as there is very limited groundwater ingress up to 20 mbgl.

However, once the mining operation progress below the water table, groundwater ingress will occur; preliminary modelling estimates have this inflow as increasing to a maximum of around 2,000 m^3/day based on the numerical flow model predictions toward the end of Phase 3 at 85 m pit depth.

Groundwater drilling is planned at the start of the mining and future modelling work is planned. The groundwater ingress predictions must be built into the seasonal water budget for the mine – to effectively optimise groundwater use, minimise water abstraction from the Varada River and to minimise disposal as part of the water management plans.

Three water supply boreholes are planned to be part of the mine water supply to deliver a total estimated volume of 162 m^3/day. It is recommended that these boreholes are drilled as closed to the mine pit operations as possible in order to start dewatering the mine pit ahead of the mine plans. This will also have the advantage that the water quality pumped from these boreholes as part of dewatering target is maintained and could be used to artificially recharge the aquifer once the mine becomes water positive.

Dewatering boreholes may be required upstream and downstream of the pit to reduce this water inflow into the mine workings. Any excess groundwater dewatered ahead of the mine can be tested and returned to the private farmers.

12.11 Sources of surface water

In Karnataka, water abstraction from groundwater resources is strictly regulated. With the potential absence, of sufficient and exploitable groundwater resources initially, it is necessary to source the additional required water from a surface water source nearby. An allocation of surface water has been obtained by DESPL from the Government of Karnataka, to utilise up to 3,000 m^3/day (3,000 kl/day) water from the Varada River for the mining operations at the Ganajur Gold Project.

The Varada River is a tributary of the Tungabhadra River. The Varada rises in the Varada Moola in Sagara Taluk, the hill range of the Western Ghats in Shimoga district of Karnataka. Its basin is located between the latitudes 14°06'N and 14°55'N and longitudes 75°02'E and 75°40'E. It has a total catchment area of 5,020 km^2 and traverses in Shimoga, Uttarakannada, Haveri and Gadag districts. It has a dendritic drainage pattern.

The Varada River basin lies roughly in a southwest to northeast direction. This southwest-northeast trend is due to the uplift of the Western Ghats and slight tilt of the Peninsular Indian mass to the east during the Miocene age (Krishnan, 1981). The Varada flows for about 300 km and breaks into innumerable small branches before flowing into the Tungabhadra River (K8 sub-basin) which in turn is a tributary of River Krishna. River Krishna being an interstate river traversing Maharashtra, Karnataka and Andhra Pradesh.

There is a specific water share allotted to Industries for usage from the River Krishna basin. Water allotments are considered and allocated by the Water Resources Department of Government of Karnataka. The DESPL proposal of 3,000 kl/day of water is available and approved – as far as utilisation of water is concerned.

The abstraction point is proposed to be sourced at the existing barrage near Kalasur. The required quantum of water is pumped and conveyed to the raw water storage dam located at the plant site. The conveyance is through raising main for a length of 6.5 km.

The rising main has to pass beneath the Bengaluru–Mumbai broad gauge railway line. A small culvert has been identified which can serve the purpose of allowing the rising main to pass under it. However, it is necessary to get permission from the railway authorities to take the pipeline below the railway line through the culvert structure.

12.12 Data collection

12.12.1 Regional climate and hydrometric data

The rainfall normal has been prepared for the Haveri District based on 50 years of data (1941 to 1990). Other climatic parameters have been prepared based on 30 years of data (1971 to 2000) (Table 12.9). Rainfall amounts refer to cumulative rainfall for the past 24 hours (measured at 08:30 hours IST). Relative humidity, wind speed and cloud amount values are calculated as an average of values measured at 08:30 hours and 17:30 hours IST. Mean temperatures are calculated as the average of maximum and minimum temperatures in the month.

Table 12.9 Climatological normals of Haveri District, Karnataka for the period (1971 to 2000)

Month	Rainfall (mm)	Temperature			Cloud (Oktas)	Relative humidity (%)	Wind speed (km/h)	Rainy days (days)
		Maximum	Mean	Minimum				
January	15.21	32.34	25.91	19.5	1.35	66.79	5.09	0.1
February	15.27	31.82	25.85	19.87	0.95	71.46	5.79	0
March	37.06	32.35	27.45	22.6	1.61	74	5.91	0.2
April	52.99	33.06	29.1	25.15	2.76	72.6	6.46	1.3
May	82.43	32.97	29.56	26.2	4.03	74.56	7.42	5
June	105.48	30.2	27.35	24.52	6.36	74.06	7.97	21.8
July	157.92	29.01	26.49	23.99	6.83	88.66	8.37	27.6
August	104.88	28.77	26.31	23.86	6.48	89.13	6.75	25.2
September	101.44	29.75	26.72	23.73	5.19	86	4.66	14.6
October	119.99	31.4	27.47	23.58	4.18	81.2	4.02	7.4
November	60.39	33.07	27.64	22.23	2.69	69.84	4.04	2.5
December	27.02	33.15	26.99	20.86	1.8	63.24	4.69	0.4

12.12.2 Regional hydrology

Rainfall occurs mostly in the Southwest Monsoon period from June to September. The Monsoon reaches a peak during July and August and tapers thereafter.

12.12.3 Project site rainfall data

The area enjoys sub-tropical climate with temperatures ranging in between 18°C and up to a maximum of 40°C. The annual rainfall varies from over 903 mm to less than 592 mm. July is the wettest month with normal monthly rainfall in excess of 150 mm.

12.12.4 Site evaporation

There is no local evaporation data available for the Project site. However, the site is located in proximity to the district HQ, Haveri. Haveri District falls under Agroclimatic Zone No. 8 in the State of Karnataka. The Minor Irrigation Department has prepared evapotranspiration and crop coefficients for different agroclimatic zones and the monthly evaporation values are indicated below Table 12.10.

Table 12.10 Daily evaporation value month-wise for the Project area

Month	Daily evaporation (mm)
January	7.04
February	7.77
March	7.50
April	8.90
May	8.35
June	7.27
July	4.24
August	4.12
September	4.58
October	5.09
November	6.44
December	6.99

12.13 Surface hydrology

12.13.1 Water availability

Additional make-up water for the Ganajur Gold Project is being sourced from the Varada River. The Central Water Commission (CWC) has a gauging station to measure the discharge at Marol on Varada River just before its confluence with River Tungabhadra. The catchment area at this station is 4,902 km^2. The river flow data is available from 1970.

There are many barrages built across Varada River for different uses such as irrigation and drinking water requirements. The nearest barrages to the proposed mine site are Kolur-Kalasuru and Karaigi barrages. Water is being stored in these barrages and is being pumped after September by farmers for irrigation and continuously for drinking water.

12.13.2 Water requirement for the Ganajur Gold Project

The initial total water requirement for the present project is 3,000 kl/day (0.035 m^3/sec). The average annual flow in the Varada River is 1,966 Mm^3/sec. The annual requirement of water for the proposed mining project is around 1.1 Mm^3 which is just 0.06% of the annual inflow in the Varada River. Hence, there is practically no impact on existing water uses from the river.

12.13.3 Water withdrawal from Varada River

Water is proposed to be sourced from Varada River from the upstream of the existing barrage near Kalasur. The required quantum of water will be pumped and conveyed to the storage dam located at the plant site. The conveyance is through a raising main of 6.5 km.

The pipeline has to pass beneath the Bengaluru–Mumbai Broad gauge railway line. A small culvert has been identified which can serve the purpose of allowing the rising main to pass under it. It is necessary to get permission from the railway authorities to take the pipeline below the railway line through the culvert structure.

Not to disturb the flows in dry season, it is proposed to draw the full mining requirement during four months in the monsoon season itself. Withdrawal of water during monsoon does not affect existing uses.

Based on the water balance statement, the quantity of water to be drawn from the Varada River is 74,982 m^3/month (i.e. about 2,500 ML/day) over a four-month period.

A raw water storage dam is proposed to be constructed at the gold processing plant site to cater to the daily needs of the project. The required quantum of water will be pumped and conveyed to the proposed raw water storage dam through a dedicated rising main. As per the CPC water balance statement in Chapter 9 (Project Infrastructure), the capacity of the raw water storage pond will be 297,590 m^3 including two months' spare capacity.

12.13.4 Flood frequencies in Varada River at Marol

The Varada River is gauged at Marol (Table 12.11) by the CWC and daily discharge data is available for 40 years from 1972 to 2011.

The annual flood peaks for the 40 years were picked out from the daily data, and are as follows:

Table 12.11 Annual flood peaks for the 40 years at Marol

No.	Year	Peak flow (m^3/sec)	No.	Year	Peak flow (m^3/sec)	No.	Year	Peak flow (m^3/sec)	No.	Year	Peak flow (m^3/sec)
1	1972	869.8	11	1982	1,430	21	1992	1,525	31	2002	380.9
2	1973	588	12	1983	1,010	22	1993	640.5	32	2003	315.1
3	1974	462	13	1984	752.5	23	1994	1,320	33	2004	605
4	1975	1,017	14	1985	576.9	24	1995	476	34	2005	919.6
5	1976	304.1	15	1986	751.7	25	1996	377.8	35	2006	780.4
6	1977	482.9	16	1987	305.5	26	1997	1,191	36	2007	1,540
7	1978	601.6	17	1988	1,180	27	1998	475	37	2008	1,317
8	1979	799.7	18	1989	644.9	28	1999	1,107	38	2009	1,076
9	1980	1,525	19	1990	485	29	2000	807.5	39	2010	677.5
10	1981	588.5	20	1991	1,194	30	2001	249.1	40	2011	700.4

The intake location would be somewhere near Kolur and no major tributary joins the river between the Intake site and Marol. Hence, the above flood magnitude can be taken as the basis to fix the intake levels (Table 12.12).

Table 12.12 Frequency analysis – Gumbel's method for Varada floods

Return period (years)	Peak flood (m^3/sec)
10	1,361
25	1,672
50	1,903
75	2,037
100	2,132
500	2,662
1,000	2,890

12.13.5 Flood inundation mapping of Varada River

Since the levels in the pump house are to be fixed with reference to the flood levels in Varada, a flood frequency analysis was carried out.

Floods generally are the excess flows which occur during the wet season (three to four months of the southwest monsoon season in India) when the flows exceed the transporting capacity of the river channel. This results in inundating the areas outside the water body/channel. It is a natural occurrence when the rainfall is heavy and discharge is high. An inundation map projects the spatial extent of probable flooding which may occur with a specific return period. It may be either quantitative or qualitative. The flood inundation map delineates the probable areas which could be affected due to flooding.

Seasonal monsoon rainfall is the prime factor which is responsible for flooding in the Varada River. The flooding usually affects the low-lying, flat topographic areas of the river basin. Intense rainfall in the upper regions of Western Ghats causes flooding on the downstream. It is usually seen that on account of the existing topography, the upstream areas (in the Western Ghats) will be flooded for short durations after intense and prolonged rainfall events. It is the downstream which will be affected for a longer duration of time in the case of flooding.

The objective of the present mapping is to analyse and delineate the areas which would be affected by considering various return periods of flood. In the present case, the return periods considered are 10, 25, 50, 75, 100, 500 and 1,000 years.

The study area chosen for the flood inundation study in the present case is between Halasur Barrage and Marol on the river course of the Varada. There are two barrages namely, Kolur–Kalasur and Karaigi on the Varada.

A digital elevation model (DEM) was generated for the study area and a triangular irregular network (TIN) was created. River geometry files and stream flow data were then used as input files for HEC-RAS to generate the water surface level along the river using HEC-GeoRAS interface. The water surface profile gives the probable areas which are likely to be submerged in case of flooding. The flood inundation maps are prepared for a various return periods.

The final level of floods for the above return periods are given in Table 12.13.

Table 12.13 Flood levels for the different return periods at various locations

No.	Return period (T) (years)	Flood (m^3/sec)	Water surface elevation (m)			
			At Halasuru Barrage	At Kolur-Kalasuru Barrage	At Karajgi Barrage	At Marol
1	1,000	2,890	535.81	535.50	535.15	532.60
2	500	2,662	535.57	535.27	534.93	532.43
3	100	2,132	534.94	534.68	534.37	532.01
4	75	2,037	534.81	534.57	534.26	531.92
5	50	1,903	534.63	534.40	534.11	531.81
6	25	1,672	534.30	534.08	533.81	531.55
7	10	1,361	533.79	533.61	533.36	531.16

12.13.6 Intensity-duration-frequency curve for Ganajur Gold Project

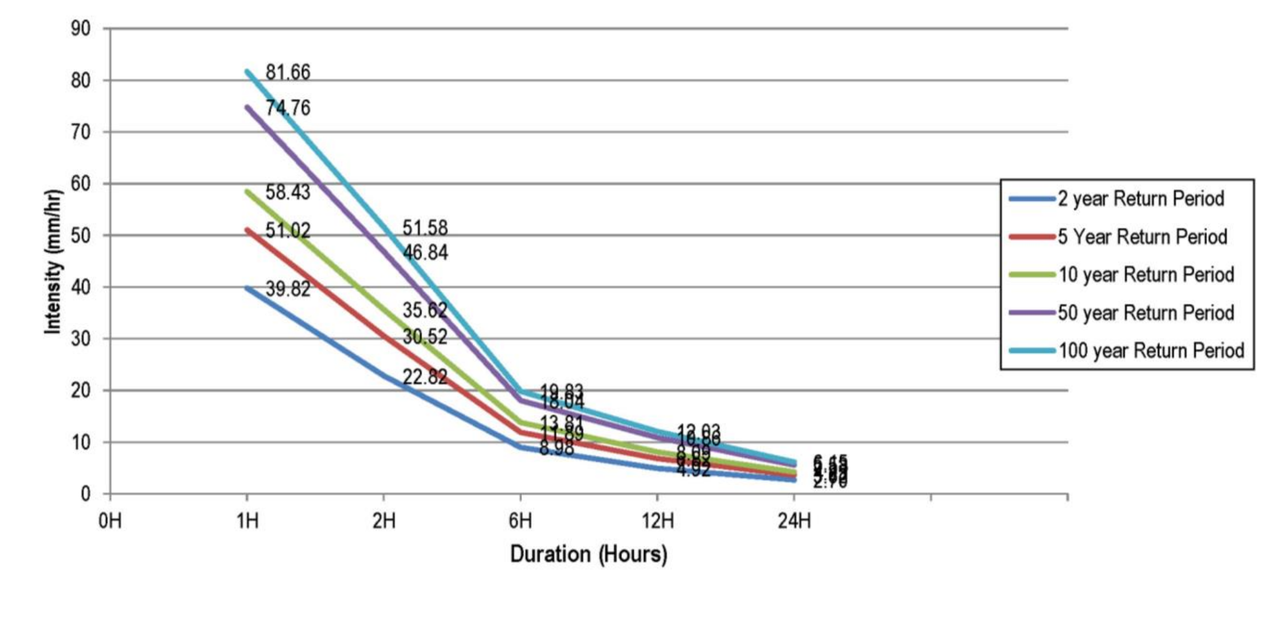
An intensity-duration-frequency curve (IDF curve) is a graphical representation of the probability that a given average rainfall intensity will occur.

It also gives the change in rainfall intensities vs. storm duration, usually, short storms will have higher rainfall intensities as compared to longer storms.

An IDF curve is created with long term rainfall records collected at a rainfall monitoring station. The data of 24 hours rainfall for 16 years of Haveri SRRG station has been collected.

In the present case, the return periods considered were 2, 5, 10, 50 and 100 years. Detailed analysis has been carried out by E I Technologies and the IDF curve for the above return period is shown in Figure 12.18.

Figure 12.18 IDF curves for the Ganajur Gold Project area



12.13.7 Diversion of stream

Two small ephemeral streams (or streams) flow northwards past the plant area and join just inside the mine pit area. They are to be diverted in a perimeter channel running around the western boundary of the mining area, which then reconnects to the stream at the downstream side of the mine area.

No flow measurement gauge data is available for the small streams, and hence the flood has been estimated by E I Technologies by using an empirical method. Fuller's formula is used in the US (Hydrology by K Subramanya, 1st Ed):

$$Q = C * A^{0.8} (1+0.8 \log T)$$

Where Q is the flood magnitude in m^3/sec with return period of T years, C is a coefficient between 0.18 and 1.88, and A is the catchment area. The flood magnitude depends on the value chosen for C, which can be determined from the flow data of the Varada River at Marol. From the 40 years of data available at Marol, floods of different return periods have been calculated by Gumbel's method (Table 12.14). Also given are the floods calculated from Fuller's formula with C=1. There is a reasonably good match between the two and hence C=1 can be used for the stream.

Table 12.14 Varada flood by Gumbel's method and Fuller's formula with C=1

Return period (years)	Varada flood (m^3/sec)	
	Gumbel's method	Fuller's formula
10	1,361	1,613
25	1,672	1,898
50	1,903	2,113
75	2,037	2,240
100	2,132	2,329
500	2,662	2,830
1,000	2,890	3,046

The streams have a combined catchment area of 11.9 km^2 at the upstream edge of the mine area. With C=1, the 50-year flood in the stream would equate to 17.11 m^3/sec.

12.13.8 Design of diversion channel

Since the existing stream cuts across the mine site, it is proposed to divert it to avoid inundation of the mine by the stormwater.

Accordingly, a wide trapezoidal section has been calculated for different return periods and corresponding flood discharge. E I Technologies is of the opinion that the diversion channel be designed for a 50-year flood magnitude of around 17.11 m^3/sec.

The stream design diversion by CPC is presented in Chapter 9 (Project Infrastructure).

12.14 Site-wide water balance

CPC reviewed the overall site water balance for the Ganajur Gold Project to determine preliminary raw water dam sizing – for both the process water (return water from the TSF) and raw water (from the Varada River).

This water balance has been based on the TSF water balance provided by Prime Resources. Accordingly, it is indicated that the total raw water pond volume of 300,000m^3 with a make-up water volume of 255,000 m^3 with an additional 45,000 m^3 for the two-month contingency.

As per the agreement entered into with the Government, a maximum of 3,000 kl/day is allocated and allowed to be lifted from the Varada River. Hence, in order to meet the requirement of 300,000 m^3 (raw water make-up), it is necessary to distribute the lifting of water to four months so that the limit of 3,000 kl/day is not exceeded.

Accordingly, the water balance statement has been compiled. Based on the CPC water balance statement the quantity of water to be drawn from Varada River is 74,982 m^3/month (i.e. about 2,500 ml/day) over the monsoon period for four months. The maximum make up water requirement as per the revised statement is 75,000 m^3 during the months of June, July, August and September.

12.15 Impact assessment

The following impacts due to the proposed Ganajur Gold Mine mining operations associated with the hydrogeological conditions should be taken into consideration:

- Groundwater quality:
  - Potential deterioration in groundwater quality due to acid rock drainage (from ore stockpiles, TSF)
  - Leaching of arsenic and other heavy metals from ore stock pile, waste rock dump and TSF
  - Increasing concentrations of nitrate and ammonium concentrations in the groundwater due to blasting residues (associated with waste rock dump, TSF)
  - Spillages and leaks from waste facilities, fuel storage/depots.
- Groundwater quantity:
  - Water table decreasing in private boreholes within the expanding cone of drawdown and therefore less water available for irrigation and water supply
  - Less baseflow to stream due to passive and active mine dewatering
  - Elevated pore pressures on southern footwall with stratigraphy dipping into pit, due to natural stream course which is fault controlled could result in stability concerns and may require flattening of slope.

12.16 Water circuit recommendations

The following recommendations are made to manage the potential impacts on the combined groundwater and surface water resources:

- Best practice guidelines to be implemented at potential sources of contamination to groundwater through bunding and lining of facilities
- Groundwater and surface water (rainfall) made in the pits to be used in ore processing plant as make-up water
- Active dewatering/water ahead of mining using dewatering/water supply boreholes close to pit so that water quality can be maintained whilst starting to drawdown water table ahead of mine plans
- Combined water balance to be developed for the mine for three seasons so that water use is optimised
- Excess mine water from dewatering can be directly supplied to the neighbouring private farmers, through reticulation providing water quality is acceptable.
- As the mine will eventually be abstracting >1,000 m^3/day artificial recharge of groundwater using roofs of selected mine infrastructure to harvest rainwater and recharging the aquifer through recharge pits as per the CGWA guidelines.
- Ultimately, groundwater may replace Varada River water at the supply to the mine if the results of the preliminary modelling can be confirmed
- TSF to be lined and footprint of ore stockpiles to be compacted and lined
- Additional hydraulic parameters to be determined for ore zone and halo zone in each geotechnical domain of the pit
- Accumulate one year of monitoring data (water levels and water quality) including heavy metals
- Expand model boundaries and re-calibrate model with new data
- Develop transport model for arsenic and nitrate plume development from waste rock dumps, TSF and determine risks to human health and aquatic environment
- Model mine closure scenarios to predict pit lake chemistry once the groundwater table rebounds.

13 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

13.1 Project environmental approvals and permitting

B S Envi-Tech Pvt Ltd (BSET) was contracted by Deccan Exploration Services Private Limited (DESPL) to address the environmental and social requirements (investigations, assessment and permitting) for the planned Ganajur Gold Mine, Haveri district, Karnataka. The environmental and social documentation has been compiled into separate chapters (provided as Appendix 13A) as follows:

- Chapter 1: Introduction
- Chapter 2: Project description
- Chapter 3: Description of baseline environment
- Chapter 4: Anticipated environmental impacts and mitigation measures
- Chapter 5: Disaster management and risk assessment
- Chapter 6: Post-project monitoring program.

13.2 Environmental and social analysis

13.2.1 Baseline conditions

The study area includes a 10 km radius around the proposed Ganajur Gold Project that includes the open pit mine, gold processing plant and related infrastructure near Ganajur village, Haveri Taluk and District of Karnataka State. The baseline environmental conditions determined represent the background environmental conditions in the study area and buffer zone of the project. Baseline environmental monitoring was carried out during the 2016 summer season (March, April and May 2016) per Indian legal requirements.

Climate

The tropical climate of the region comprises hot and humid summers, moderate monsoon seasons and mild winter seasons. May is the hottest month in the year. The months of December, January and February are considered to have pleasant climate. A weather station installed onsite for the monitoring period revealed the following minimum and maximum temperature and humidity conditions shown in Table 13.1.

Table 13.1 Temperature and humidity conditions recorded on site (summer 2016)

	Temperature (°C)	Humidity (%)
Minimum	18.0	11.0
Maximum	41.8	75.7

The predominant wind direction determined for the site as per data gathered from the on-site meteorological station during summer 2016 was from the WSW-W-WNW-NW-NNW sector (56.89% of the time). Wind speed during this period varied from 1 km/h to 15 km/h (calm winds less than 1.6 km/h occurred 13.77% of the time).

Ambient air quality

Ambient air quality was determined for PM_{10}, $PM_{2.5}$, SO_2, NOx and CO monitored at eight locations around the project area. The results (Table 13.2) all met the limits in terms of the National Ambient Air Quality (NAAQ) standards of Industrial, Residential Rural and other areas. No industrial emissions were identified in the vicinity of the Project site.

Table 13.2 Summary of Ambient Air Quality (µg/m^3)

Station code	Location name	Nature of area	98th percentile values			
			PM_{10}	$PM_{2.5}$	SO_2	NOx
A1	Project site	Current – rural, future – industrial	56.4	27.7	9.6	10.8
A2	Karajgi	Residential areas	48.4	22.5	8.7	10.1
A3	Agadi		54.3	25.9	8.5	9.7
A4	Virapura		55.6	25.5	8.5	9.7
A5	Haveri		52.5	23.8	10.1	11.2
A6	Lakmapura		52.2	23.9	8.9	10.1
A7	Yellapura		55.7	25.4	9.3	10.6
A8	Koluru		48.7	23.8	9.6	10.8
NAAQ Standards for Industrial, Residential, Rural and Other Areas			**100**	**60**	**80**	**80**

Noise

From noise levels monitored at eight locations around the Project area, it was determined that the daytime noise level equivalent was 68.1 dBA, while the night-time noise level equivalent was 59.4 dBA. These conditions meet the standard for an industrial area, however, it is noted that, except for rural traffic, there are no industrial or major noise-generating sources. The areas surround the Project location recorded noise levels compliant with residential areas (55 dBA daytime and 45 dBA night-time).

Hydrology and hydrogeology

The catchment of the Project site is drained by the Varada River in a west-east direction. The Dodda Halla stream in the south of the Project area flows in a southwest to northeast direction and drains to the Varada River. Drainage is ultimately to the major east-flowing Krishna River. Drainage generally follows the orientation of strike faults and joints in the regional geology.

The aquifer system in the Project area is typical, with water moving through the unsaturated, weathered zone of the host geology (5 mbgl to 20 mbgl, shallow aquifer system) along fractures and joint contacts in the deeper consolidated Greywacke material (20 mbgl to 90 mbgl, main aquifer system). The yield of this fractured aquifer depends on the openness of the fracture and interconnectivity of the fracture system at a particular location.

Groundwater levels in the study area are generally deeper than 15 mbgl, with shallower levels encountered near the Varada River, and deeper levels to the east and west of the proposed open-cast pit.

Groundwater is the major source for potable/domestic and irrigation use in the area. Abstraction is primarily through borehole-wells, hand-dug wells were not noted. There are no major irrigation systems, except small to medium reservoirs with canals (Heggeri Kere is the major reservoir observed in the buffer area).

A hydro census of groundwater use in the study area identified 41 borehole wells. The depths of the borehole wells varied from 45 mbgl to 150 mbgl (average depth was 60 mbgl to 90 mbgl), and yields were generally in the range of 2 to 6 litres per second.

The proposed mining project is situated upon a watershed, with groundwater generally flowing in a north-northeast direction. The general groundwater elevation and flow directions for the study area are shown in Figure 13.1.

Groundwater resource determinations are undertaken every second or third year by the Department of Mines and Geology, Government of Karnataka and Central Ground Water Board. Based on the most recent estimations undertaken, current groundwater use in Haveri taluk, where the Project is situated, is at 69% of the net available groundwater resources of 7,927 Ha-m per annum. The taluk has been categorised as “safe” in terms of groundwater use.

Groundwater and surface water quality was determined through sampling of eight surface water sites and eleven groundwater boreholes. All water samples were found to be compliant within the prescribed ranges of the Indian drinking water standard (IS 10500). More recently, 11 of the Ganajur Gold Project monitoring boreholes closer to the Project area were sampled in March 2017.

The monitoring results are tabulated in Chapter 12 of this FS; however, it is noted that, generally, the pH of the groundwater is neutral (6 to 7), the Total Hardness is very high (classified as very hard water >180 mg/L), and the total dissolved salts (TDS) varies from 700 mg/L in the upper reaches of the catchment increasing to >1,700 mg/L in boreholes closer to town.

The elevated TDS is mainly due to chloride and bicarbonate as the major anions and sodium and calcium as the major cations. These elevated levels are due to evapo-concentration of salts due to irrigation and a build-up of salinity in the soil profile that gets flushed through to the groundwater during the monsoon seasons. Nitrate is generally above the permissible drinking water limit of 45 mg/L and this is due to the high use of inorganic nitrogenous fertilisers and manures as well as animal and human waste. The monitoring boreholes near the Ganajur Gold Project proposed mine area MW17 and MW21 have arsenic levels exceeding the desirable limit of 0.05 mg/L.

For more information regarding the hydrogeology of the Project area, refer to Chapter 12 of this FS.

Land use, potential and soil quality

Ten soil samples from around the 10 km radius encompassing the Project site were analysed. It was found that:

- pH of all the soil samples was found to be neutral (6.12 to 8.36)
- Soils are clay, sandy loam and sandy clay loam in texture
- Organic carbon in the range 0.52% to 1.19%
- Phosphorus content in the range 35 mg/kg to 312 mg/kg
- Sodium and calcium content in the respective ranges 63 mg/kg to 442 mg/kg and 902 mg/kg to 4,349 mg/kg
- Nitrogen content in the range 288 kg/ha to 538 kg/ha.

In terms of land use patterns in the study area, the largest percentage comprised cropland (agricultural land accounts for 442 km^2 or 91.6% of the total study area, followed by wastelands (land with or without scrub), built-up (residential) land, forest (five forest reserves occupying a total footprint of 5.69 km^2 or 1.18% of the total land use area) and water bodies.

The majority of the farms in the district are small and marginal traditional farming operations as the majority of the area comes under rain fed farming.

The primary crops being cultivated in the area include millet, rice, maize, cotton, pulses, groundnut, green chilies, soya bean and turmeric.

Terrestrial ecology (flora)

According to the available records, there are no National Parks, Wildlife Sanctuaries or Biosphere Reserves in a 10 km radius of the Project area. Five forest reserve blocks were, however, identified as per Table 13.3 below.

Table 13.3 Forest blocks in the study area

Name of the forest block	Distance from plant boundary	Direction from plant boundary
Karajgi Reserve Forest (RF)	1.2 km	East-northeast
Katenahalli RF	7.4 km	Southeast
RF nearest to Virapura village	4.4 km	Southeast
RF nearest to Katenahalli village	8.4 km	Southeast
RF nearest to Kadamanahalli village	7.8 km	Southeast

The terrestrial ecosystem is characterised by scrub units and mixed dry and deciduous forests. Deciduous species, which thrive in drier climates, predominate in the region, which includes herbs, shrubs and trees. The core zone comprises herbaceous and shrubby vegetation which are scarcely distributed. Surveys undertaken across the study area identified the following floral life forms according to Raunkaier's system (Table 13.4).

Table 13.4 Floral life forms according to Raunkaier's system

Phanerophytes	Trees, shrubs and climbers where the growing buds are located on the upright shoot much above the ground surface and they are the least protected.
Therophytes	Plants which survive the adverse season in the forms of seeds. The plants produce flowers and seeds in the favourable season. They are annuals, predominantly found in extremes of dry, hot or cold conditions.
Hydrophytes	Water plants except plankton (free floating and submerged macrophytes).
Hemicryptophytes	Predominantly present in cold climatic regions. buds are present just under the surface soil and remain protected there. Mostly these are biennial or perennial herbs whose vegetative growth and aerial parts are conspicuous in warm seasons only. Buds may also be present at the soil surface but they are never exposed, they remain concealed under dead leaves and twigs.
Geophytes	Plants, with perennating parts buried in substratum such as bulb and rhizomes.
Epiphytes	Parasitic plants or plants without contact with ground.
Climbers	Lianas, stragglers and climbing plants.

A total of 187 plant species were recorded during the floristic survey. The Importance Value Index (IVI) is a statistical quantity which gives an overall picture of the importance of the species in the vegetative community. The highest IVI of plant species evaluated in the study area occurred in the Karajgi RF, while the lowest was recorded near the proposed plant location. Similarly, the highest plant species diversity was identified in the Karajgi RF, while the lowest was recorded near the proposed plant location.

In terms of rare, endangered and endemic plant species, the available literature (Red data books of Indian plant species and detailed lists of Rare and Endangered plant genera of Karnataka-Haveri district) revealed that no endangered, threatened or rare plant species were observed or recorded during site surveys; however, there is a possibility that surveys conducted during other seasons could reveal the presence of such species.

Terrestrial ecology (fauna)

The study area is not generally a suitable habitat for wildlife and further there are no National Park, Wildlife Sanctuaries, Biospheres Reserves, Tiger Reserves, Elephant Corridors, Conservation Reserves and Community Reserves in a 10 km radius from the plant boundary according to the Ministry of Environment Forests and Climate change and the Forest Department of Government of Karnataka. Areas situated near RFs do, however, provide suitable habitat for Schedule I species (which are protected species according to the Indian Wildlife Protection Act 1972) such as Indian peafowl. A few species within Schedule II (protected but less vulnerable than Schedule I species) are recorded in the literature for Karnataka.

During the site surveys undertaken, the following total number of species were observed/recorded:

- 11 mammals
- 13 reptiles and amphibians
- 59 birds
- 26 butterflies and invertebrates.

One of the above species is classified as protected in terms of Schedule I of the Indian Wildlife Protection Act, namely, the Indian Peacock.

Socio-economic environment

The demographic profile of the study area is as shown below in Table 13.5.

Table 13.5 Population demographics

Population, household size and sex ratio in the study area	Total (0 to 10 km)
No. of villages	20
Households	60,585
Population	292,696
Male population	150,671
Female population	142,025
Household size	4 to 5
Sex ratio	943 females : 1,000 males

Of the population of the study area,9.8% comprises Scheduled Castes (SC), while 10.2% comprises Scheduled Tribes (ST).

A comparative analysis of the regional workforce revealed a total number of workers as 139,950, 92,763 of whom were male and 47,187 of whom were female. The majority of these are “main workers” followed by agricultural labourers. The lowest number of workers occurs in the household industry category.

In terms of facilities, all villages in the study area have access to drinking water and electricity and the study area is well connected to the road and railway network. Major market facilities are available in Haveri. A total of 28 medical facilities service the 20 villages in the study area.

Educational facilities in the study area include 36 primary, 29 middle, four secondary and one senior secondary schools.

In terms of archaeological, heritage and cultural resources, Haveri has earmarked burial yards for all religions. However, no burial grounds were identified in any of the villages aside from Ganajur. *Purada Siddeshwara* Temple in Haveri, which is situated approximately 3.5 km from the processing plant site and 5 km from the centre of the mining lease area, is listed as a protected monument by the Archaeological Survey of India (ASI) according to the Ancient Monuments and Archaeological Sites and Remains Act 1958.

The Ganajur Gold Mine and Plant area does not involve any displacement of human settlements as the land is private agricultural land and will be purchased by DESPL. A farmhouse is present at the location of the proposed open-cast pit; however, it is assumed that this property will be dispensed with in terms of the land acquisition agreement arrived at.

A socio-economic survey and focus group discussions were undertaken for the study area. Table 13.6 indicates the summary of the outcomes thereof.

Table 13.6 Consolidated outcomes of focus group discussions

Name of the group	No.	Action points
Self-help groups (SHGs)	One group from each village	Requirements for revolving fund, marketing facilities to be extended, non-performing SHGs to be given handholding support by providing capacity building initiatives of NGOs and other stakeholders.
Non-government organisations (NGOs)	One NGO	Need for financial assistance.
SC/ST minority groups	Two groups from each village	Looking for better amenities, roads and sanitation facilities.
Central/State Government sponsored program groups etc.	MGNREGS-Scheme members participated	Require Anganwadi centres, minor irrigation works, Godown facilities, primary health centres, rural roads, rural markets. Rural service centres to be opened more for the accessibility for the local people.
Unemployed youth – technical/non-technical	Five members from each village	Employment opportunities for local people.
Yuvak Kendras and societies	One member from each village	Self-employment opportunities to be provided nearest to the village for the local people.

13.2.2 Impact assessment

Potential impacts to the physical, ecological and socio-economic environments which may arise as a result of the proposed mining and related activities were assessed for the following aspects:

- Air emissions
- Transportation and roads
- Noise generation
- Waste water generation
- Solid waste disposal
- Socio economic changes.

Air emissions

Two potential sources of air emissions were identified for the Ganajur Gold Project, namely gold ore production dust (open pit mining and material handling) and processing of mined ore.

Airborne particulate matter (dust) was identified as the main potential pollutant arising from open pit mining and material handling operations. Emission rates were determined for drilling, excavation/loading and transportation (to either the plant or the waste dump area) were determined using USEPA factors

Emissions arising from the ore processing plant were considered for crushing of the mined ore (minimal emissions due to dust suppression inherent in the design), tailings disposal (progressive vegetation may be utilised to limit dust emissions) and diesel generator sets (negligible impact as their use is only in emergency situations).

A worst-case scenario dispersion model considering the cumulative ground level concentrations of particulate matter with diameters of 10 and 2.5 microns (PM_{10} and $PM_{2.5}$) was prepared using meteorological data measures during the 2016 summer season on-site. This model revealed maximum incremental 24-hourly average cumulative ground level concentrations (µg/m^3) of 5.7 µg/m^3 for PM_{10} and <1 µg/m^3 for $PM_{2.5}$.

No village or habitation is located within the 0.9 km of the mine area and the potential impact in terms of air pollution is considered to be marginal. By adopting effective dust suppression measures such as water sprinkling methods, the significance of impacts in this regard can be reduced to negligible.

Transportation and roads

The potential impact in terms of heavy vehicle movement is limited considering that mined material and waste handling all take place on site. A short section of road between mine and the processing plant will be widened and upgraded. Ore will be transported between the mine and the ore stockpile on the open road. The only additional traffic to the road network will be the transportation of processed gold bullion off site. Considering the capacity of the existing road network, impacts in terms of congestions to road traffic or vehicular emissions are not anticipated.

Noise generation

The primary sources of noise generation anticipated for open-cast mining activities and the anticipated noise levels are indicated below in Table 13.7.

Table 13.7 Anticipated noise levels

Source	Noise level at source in dB(A)
Shovels	84 to 91
Front-end loaders	80 to 88
Backhoes	82 to 86
Trucks	87 to 96
Water trucks	82 to 92
Compactors	86 to 92

Anticipated noise levels above 80 dB(A) are not expected to occur for more than eight hours per day. Furthermore, the foreseen noise levels are expected to be confined to the immediate vicinity of the operations, thereby effecting primarily personnel. Noise impacts from open-cast mining are not expected to affect receptors in surrounding areas.

The main source of noise generation for ore processing activities will be the crushers, which generate cumulative noise levels of approximately 105 dB(A) (when all three will be in operation) at a continuous rate during plant operation. The resulting noise-level at the boundary of the plant as a result of crusher operation was determined to be less than 75 dB(A) without considering other attenuation factors such as encasement, buffering by the proposed green belt, interaction with physical objects and atmospheric pressure. It is anticipated that noise levels in the areas surrounding the mine may increase due to the current residential nature of these areas; however, management measures proposed are considered adequate to mitigate the impact in this regard.

Although the diesel generators may also produce significant noise levels, the use of these is limited to emergency situations.

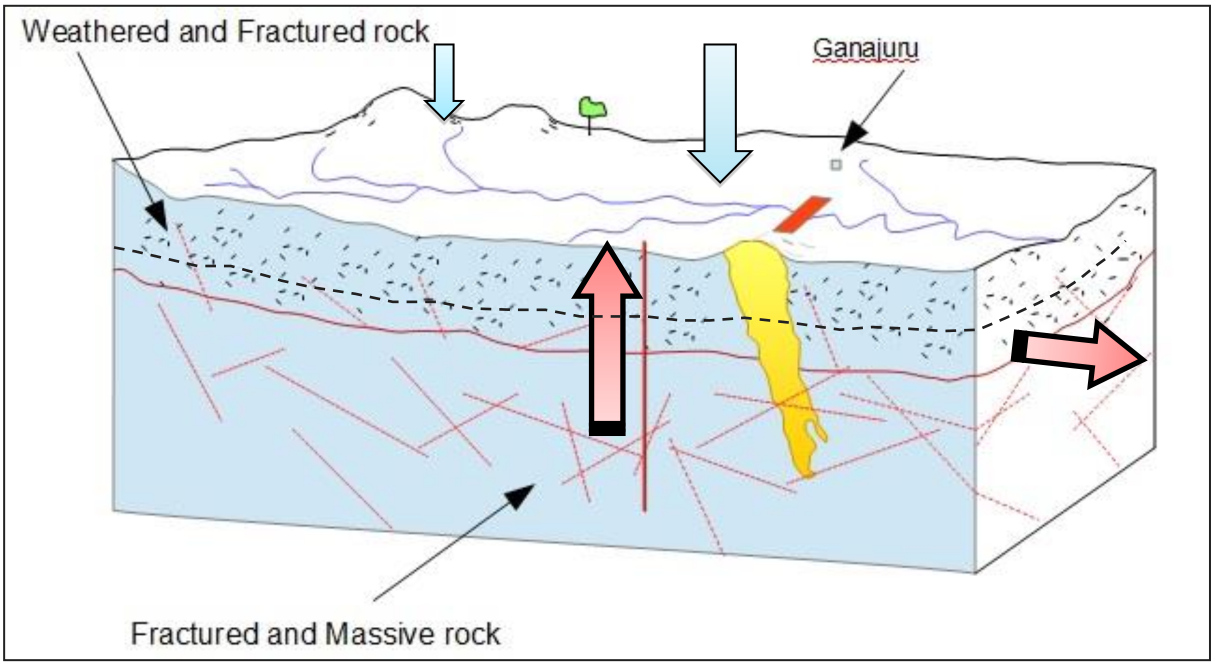
Groundwater environment

Groundwater levels

According to the groundwater modelling study undertaken (fully described in Chapter 12 of this FS), the open-cast pit will breach the aquifer system situated between 40 mbgl and 100 mbgl. The shallow weathered and fractured aquifer system is not saturated and no phreatic aquifers were identified, therefore groundwater inflow will be minimal aside from minor seepage until mining depth progresses beyond 510 mamsl. The deeper, fractured aquifer system will be encountered at this point and thus seepage into the open-cast pit will increase with depth. The requirement for dewatering will also increase as pit inflow increases, due to new fractures being encountered, the elevated hydraulic gradient towards the pit from adjacent areas and the overall increasing spatial extent of the mining area. The maximum estimated seepage volume is 1,909 m^3/day during Year 6 of mining.

Mine dewatering is anticipated to result in a drawdown in regional groundwater levels. A large number of users in the zone of influence depend on groundwater for potable and domestic use. This water is derived mainly through borehole wells excavated to below 20 mbgl and targeting the deeper aquifer system situated between 60 mbgl and 100 mbgl. The groundwater model illustrates that mine dewatering process will impact on the neighbouring borehole wells in terms of yield and water levels. This impact will not, however, be noted in deeper borehole wells targeting fractures at a depth of 120 mbgl, which is also deeper than the mine pit level. Pumping from such deeper fractures, below the mine pit level, may lower the hydraulic gradient and seepage into the mine pit. The current understanding of the situation is illustrated below in Figure 13.1.

Figure 13.1 Diagrammatic representation of groundwater flow in the Project area



In terms of water supply for operation of the plant, it is envisioned that water will be abstracted from the Varada River and provision has been made for the installation of a pump and pipeline for additional water in the monsoon season, however supplementary groundwater from the pit dewatering borehole wells can also be added to the processing circuit. The potential borehole well yields in this regard have been estimated in Chapter 12. There are, however, uncertainties in this regard as long-duration pump tests have not been undertaken with which to determine sustainably yield.

Groundwater quality

The impacts to water quality envisioned as a result of open-cast pit mining pertain to acidity from the sulphide ores and siltation as a result of erosion of exposed surface. The storage of excavated overburden, ore material, tailings from the processing plant and wash-off from these areas, as well as the process plant area, were identified as the main risk to groundwater quality through spills or leakage of hazardous chemicals. Inherent design considerations, including a low-permeability liner in the TSF, will serve to limit pollution arising, and the operation will be required to meet the Central Pollution Control Board requirements to limit cyanide and sulphur in gold processing waste water to 0.2 mg/l.

According to the hydrogeological and geochemical studies undertaken as part of this FS (refer to Chapters 10 and 11 of this report), ore and tailings material from the mining operations are classed as potentially acid forming and have the capacity to generate acidity in the longer term should oxidation occur, when the sulphide content is above 2.5% to 3%.This means that acid rock drainage could impact on the groundwater quality resulting in a lower pH and propensity to dissolution of heavy metals. The waste rock material (quartzite, greywacke and brecciated chert) is non-acid generating. These geochemical considerations regarding the interaction of groundwater with exposed host geology are not discussed in the Environmental Impact Assessment (EIA) report. The development of any contamination plumes arising from any of the above-mentioned sources has not been quantified as part of the hydrogeological modelling exercise and not elaborated on in the environmental impact study. It is thus not clear whether any external users bore wells in the zone of influence will be impacted in terms of declining water qualities in addition to the previously discussed groundwater levels.

Land environment

Land use

Open-cast pit mining, material handling and related mining activities will alter the profile of the land, however, the impact in this regard is not considered to be significant as the land is currently devoid of any vegetation, being private land, that will be acquired prior to the commencement of mining activities.

Terrestrial ecology

Due to the fact that forests in the study area were not noted to provide habitat for any Schedule I or other conservation-worthy species, except for Indian Peafowl *Pavo cristatus*, the potential impacts in this regard as a result of mining and related activities is considered to be low. The limitations of the baseline surveys should be borne in mind, however, as further conservation-worthy species may be identified by undertaking additional seasonal studies in different seasons.

Blasting and ground vibration

A technical note prepared by Deepak Vidyarthi indicates the following drill and blast design for the Ganajur Gold Mine (Figure 13.2 and Figure 13.3 below).

Figure 13.2 Drill and blast design for ore

Drill Hole Pattern in Ore

TECHNICAL DETAILS

Holes Diameter ...	150 mm	Firing Pattern ...	V - Cut
Hole Depth	11 m	No of Holes / Month ...	42
Spacing x Burden ...	5m x 4m	No of Blasts / Month ...	1

Figure 13.3 Drill and blast design for waste rock

Drill Hole Pattern in Waste Rock

TECHNICAL DETAILS

Holes Diameter ...	150 mm	Firing Pattern ...	Truncated V
Hole Depth	11 m	No of Holes / Month ...	112
Spacing x Burden ...	6m x 5m	No of Blasts / Month ...	2

It was determined that 42 holes will be blasted in the ore and 112 holes in the waste rock per month for the life of mine. The frequency of these blasts will be once per month in the ore and twice per month in the waste rock. The consumption of explosives per blast (ore) will be approximately 5.82 t, approximately 7.70 t in waste rock. The MCD (maximum charge per delay), calculated as a single hole firing per instance, was determined to be 133.33 kg in ore and 132.0 kg to 142.25 kg in waste rock.

From the above parameters, the potential impacts to the nearest sensitive structure was determined. The Hanuman Temple is situated approximately 130 m from the mining activities. Considering the calculated MCD, the Peak Particle Velocity (PPV) will likely meet the permissible limits of the Director General of Mines Safety of 0.5 mm/sec to 10 mm/sec for sensitive structures.

It should be noted, however, that detailed information regarding rock constants has not yet been determined and thus the actual PPV has only been estimated. It is further noted that muffle blasting and pre-splitting techniques (if required) will be employed to control fly-rock and thereby prevent any adverse impacts.

Socio-economic environment

Potential positive impacts

- Increased employment opportunities (directly and indirectly from economic activities in unrelated sectors) and overall skills training and development:
  - The Project is estimated to create approximately 200 direct employment opportunities and 200 indirect employment opportunities.
- Increase in trade, business and service sectors due to the availability of infrastructure in the post-Project period.
- Increase in wage rate to more market-related rates;
- Value appreciation of land and other immovable property.
- Development of organised markets due to improved living standards and an influx of mine workers into the area.
- Improved access to infrastructure and services.
- Impacts to woman and vulnerable groups, including:
  - better participation of women in economic activities
  - improved general outlook and awareness of women
  - improved security due to increased family income and improvement in social status
  - potential improvement to educational and healthcare facilities in the post-Project period
  - increased group interaction and improvement in social status.

Potential negative impacts

Only potential negative impacts related to the potential environmental impacts described elsewhere in this section are listed.

13.2.3 Management of impacts

Air emissions

The following risk areas were identified as the main sources for controlling fugitive dust emissions on site:

- Drilling
- Blasting
- Excavation

- Loading operation
- Transportation of ore and overburden.

The proposed environmental management measures for controlling air pollution are as follows:

- Utilisation of drilling equipment with built-in water injection systems.
- Regular wet suppression (spraying) on blasted heaps, dumps and haul roads. Water sprayers controlling conveyor-borne dust with an efficacy of 90% or greater. Additives should be added to the sprayer arrangements of stockpiles of the crushed material.
- Wet suppression should also be implemented on stockpiles of the ore at the processing plant. Crushed fine ore must be stored in closed bins. Dry dust collectors (using dust socks which are cleaned via high pressure reverse air jets) and water sprays should further be utilised at the crushers and transfer points of all the conveyors for ore handling.
- Implementation and maintenance of a green belt around the mining area (afforestation).
- Best-practice measures for drilling and blasting (sharp drill bits, using optimum blast charges and time delay detonators).
- Avoiding blasting during high windy periods, night times and temperature inversion periods.
- Regular grading of haul roads and service roads.
- Managing vehicle loads to avoid over-filling and spillages
- Ongoing maintenance and servicing of vehicles and machinery.
- Progressive revegetation of denuded areas to stabilise surfaces.

Noise generation

The following noise abatement measurements are proposed for implementation during the operational phase:

- Ongoing maintenance of vehicles, machinery and equipment.
- Limiting blasting activities to daylight hours and employing optimum explosive charges, proper delay detonators and proper stemming to prevent blowout of holes.
- Limiting time exposure of personnel to excessive noise and adequate provision of personal protective equipment.
- Limiting vehicle speeds appropriate to the type of vehicle and the working area.
- Noise generating sources at the plant should be adequately encased or sufficiently away from residential dwellings. Crushers, grinding mills and diesel generating sets should be housed in closed buildings to attenuate the noise level.
- In order to reduce noise generation/absorb noise from air compressors, pumps and diesel generators, the machinery will be placed on vibration isolators.
- The proposed 7.5 m wide green belt should encompass the plant area, office buildings, township and internal roads wherever possible to attenuate noise.

Terrestrial ecology (fauna and flora)

To manage potential impacts to flora and fauna of the study area, the following conservation measures should be undertaken in the study area:

- Plantation of suitable native species preferably indigenous species on degraded or waste-land and open degraded forest:
  - It is noted that a comprehensive list of suitable species is not provided in this regard, nor is an actual implementation plan.

- Planting of palatable grasses will be undertaken to support the herbivore population.
- Artificial waterholes will be created and natural water sources will be maintained on a spatial – temporal distribution basis.
- Development of the greenbelt around mining area (mentioned elsewhere).
- Planting of native fruit and fodder species within the buffer zone.

Water management

Refer to Section 13.4.

Blasting and ground vibration

The drilling and blasting technical parameters specified (see "Blasting and ground vibration" discussion in Section 13.2.2) must be strictly adhered to in order to limit the potential for fly-rock and ground vibrations and to ensure that the PPV remains within the allowable limits at all times. The following measures are planned for controlling ground vibration and fly rock:

Ground vibrations should be limited to less than 6 mm/sec by implementing the following measures:

- Free face must be provided for each hole and the charge per delay must be kept within permissible limits.
- Sand-covered, delayed detonating fuses must be used during blasting.
- The burden of holes in the first row, as well as the effective burden of other blast holes, must be optimised. Blast holes must also be charged with the optimum quantity of explosives.
- A staggered pattern of blasting must be adopted.
- Benches must only be blasted one at a time.

Afforestation

The proposed green belt, which is recommended to mitigate various potential impacts arising due to mining and related activities at the Ganajur Gold Project, includes the following areas (Figure 13.5):

- 26% (36 ha) of the plant area
- Waste rock stock pile – 10.87 acres
- Temple Buffer Zone and Safety Zone (5.8 acres).

Socio-economic environment

Although the EIA report describes the following measures by which to improve the socio-economic conditions of the study area, many of these measures are not directly related to the Project or are beyond the duty of care for DESPL to implement:

- Establishment of satellite soil testing laboratories by the Agricultural Department, who should also facilitate the issuing of soil health cards.
- Financial institutions should promote extending credit to tenant farmers/share croppers and marginal farmers. Modern agricultural tools and implements should be financed. Agricultural graduates should be targeted for establishing agri-clinics through proper financial structures.
- Solar water pumping systems should be encouraged for uninterrupted power supply to farmers.
- A dairy scheme should be implemented to generate additional employment income to uneducated villagers.
- The availability of power supply (including alternative sources) and road connectivity should be addressed

- The education sector has to be improved by providing good buildings, teachers, and transport facilities.
- Availability/usage of manpower for the construction of infrastructure is essential.

Disaster management

A disaster management risk assessment and emergency preparedness has been prepared which identifies risks may arise as a result of unforeseen circumstances or unusual operating conditions and which pose a risk to employee, public and environmental safety. The following aspects are addressed therein:

- Filling up the mine pit due to an excessive storm event:
  - Interceptor trenches constructed along the outer boundary of the pit area will control inflow of runoff into the mine pits. Water that collects in the mine pit from rainfall will be pumped out from the pits, using centrifugal pumps and used for dust suppression or return to the mine raw water dam and not be discharged without water quality testing.
- Failure of pit or dump slopes:
  - Slope stability will be determined from existing quarries after determining various physical properties of the ground mass. Chapter 5 of this report deals with the selection of a suitably designed stable mine pit slope.
  - The safety factor will be determined against overall slope failure as well as against individual bench slopes by circular failure, planer failure and wedge failure.
  - Slopes will be monitored at regular intervals to check for any possible failures.
  - Slopes of external dumps will initially be set at the angle of repose of dump material.
  - As the edge approaches the final position, the slopes will be terraced and proper vegetation will be laid which will decrease slope angles and bind the soil, thereby preventing any slope failure.
- Fly-rock from blasting operations:
  - The blasting and drilling design parameters calculated to ensure the safety of blasting operations (including limiting the PPV to allowable standards, preventing fly-rock and limiting ground vibrations) must be strictly adhered to.
- Heavy machinery/equipment accidents:
  - Excavators, tippers, drills to incorporate a high level of automation
  - Preventive maintenance to reduce noise and prevent breakdowns
  - Maintenance of roads, with proper water drainage to prevent slippery surfaces
  - Utilisation of personal breathing and other appropriate protection equipment, such as safety shoes, belts, ear muffs, masks etc.
  - Warning notice boards indicating time of blasting and prohibiting access to the area
  - Spotters with suitably signalling systems to be used while loading
  - Ongoing training regarding safety measures to be practiced
  - Display safety standard sat important areas of operation
  - Periodic and regular medical check-ups.
- Surface fire (electrical and oil):
  - Equipment must be provided with on-board firefighting devices
  - Procedures for evacuation and notification of the regional fire brigade and emergency services in the event of any fire are described.

The emergency preparedness plan, which is generic in nature, incorporates the above risk factors and further addresses:

- Firefighting equipment
- Personnel safety, storage and handling of chemicals, including cyanide
- Tailings material
- Safety management
- Personnel of suitable qualifications
- Safety education and training
- Safety committees and auditing.

13.3 Environmental monitoring program

Recommendations are made regarding the implementation of a monitoring program on site. Regular monitoring of the following aspects should be undertaken:

- Pollution arising at the plant and in its vicinity
- Gather and analyse data for predicting trends regarding pollution arising
- Evaluate the efficiency of pollution control systems installed at the plant
- Assessing environmental impacts.

Monitoring should be undertaken by an independent team of experts and the outcomes of such monitoring should be evaluated by an Environmental Management Committee incorporating representatives from Gram Panchayat, professors, retired government officials and senior citizens.

The monitoring program will incorporate:

- Air quality measurement.
- Regular and frequent inspections on the TSF for any signs of slope failure and excess erosion.
- Inspection and regular cleaning of settling tanks, drainage systems, retaining walls and sewage treatment plants.
- Monitoring of water quality:
  - Downstream of all the seasonal streams/natural drainage lines within the Project area, for all the seasons
  - At the inflow and outflow of all the settling tanks to evaluate the efficiency thereof
  - Groundwater quality.

13.3.1 Tailing storage facility leachate monitoring

Leachate will be analysed at four sumps/borehole wells in the corners of the TSF according to legal requirements and specified procedures specified.

Measures such as cyanoprobes will be utilised for periodic cyanide monitoring at streams and water bodies around the TSF.

13.3.2 Groundwater monitoring

Groundwater from monitoring wells within a 5 km radius will be monitored for all heavy metals on a regular (monthly) basis. A total of three piezometric wells (nest) are planned around the Project area for monitoring the impact on the ground water regime and also for collection of water samples.

13.3.3 Air quality monitoring

Ambient air quality will be monitored at three locations in nearby villages for suspended particulate matter, PM_{10}, $PM_{2.5}$, CO, SO_2 and NOx, during summer, winter and post-monsoon periods.

13.3.4 Noise-level monitoring

Noise level monitoring will be undertaken for two seasons (dry season and wet season) at a minimum of 10 locations in the vicinity of the mine.

13.3.5 Soil quality monitoring

Soil samples will be monitored to improve soil characteristics for utilisation by the correct species, to meet requirements for vegetation, and to determine fertilisation requirement. Locations for soil quality monitoring will be based on the type of land use. Samples will be collected in the dry and wet seasons. Samples from the TSF will be analysed for hazardous constituents.

13.3.6 Drinking water quality

Potable water sources will be monitored on a monthly basis for the parameters specified as per IS 10500.

13.3.7 Blasting

Blast-induced ground vibrations will be determined utilising a vibrometer recording the following information for each blast:

- Number of holes blasted
- Total explosive charge (kg)
- MCD (maximum charge per delay)
- Air blast (dB)
- Frequency T (Hz)
- Resultant vibrations – (mm/s)
- Maximum L - (mm/s)
- Frequency L (Hz)
- Maximum T - (mm/s)
- Frequency T (Hz)
- Maximum V – (mm/s)
- Frequency V (Hz)
- Instrument distance from blast site (m)
- PPV (mm/s) – as measured.

The above data will be used for regression analysis and computation of the scaled distance and PPV.

13.4 Project water management

13.4.1 Silt loads

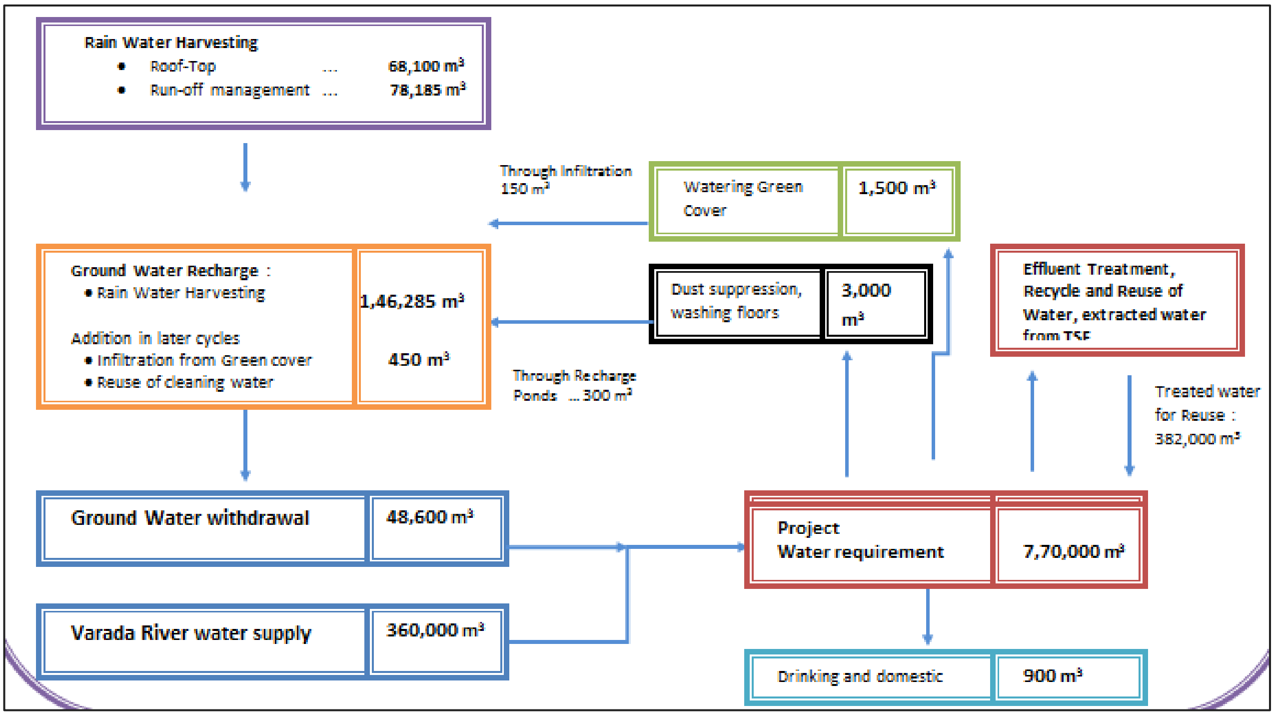
To manage soil erosion and wash off from dumps garland drains with silt traps will be constructed along the working mining pit and temporary waste rock dumps. Although the EIA indicates that a check dam will be constructed at the discharge end and that silt-free water will be discharged into the seasonal stream, subsequent hydrogeological and geochemical studies undertaken as part of the FS (see relevant Chapters 12 and 11 respectively) indicated that the mine pit water quality may be slightly acidic and may contain arsenic and silt; and therefore, any discharge to the stream will be avoided. However, further sampling and analysis of the pit water during mining operation will be carried out to confirm this.

Any final waste dumps and the TSF will be stabilised (vegetation cover) to control the runoff from the surfaces.

13.4.2 Resource management

In terms of bulk water supply, demand on available resources must be managed and the resources enhanced where possible by artificially increasing recharge. It is proposed that rainwater harvesting structures are constructed in order to recharge the groundwater reservoir (Figure 13.5), enhance its resource and make it sustainable. These measures will also serve to limit the impact of dewatering and improve the groundwater quality. It is calculated that 68,100 m^3 per annum can be harvested from rainfall, while 78,185 m^3 can be conserved through runoff management, allowing for a total of 146,285 m^3 per annum to be recharged into the groundwater environment. General management of water resources in the Project area are represented in Figure 13.4.

Figure 13.4 Annual water management flow diagram

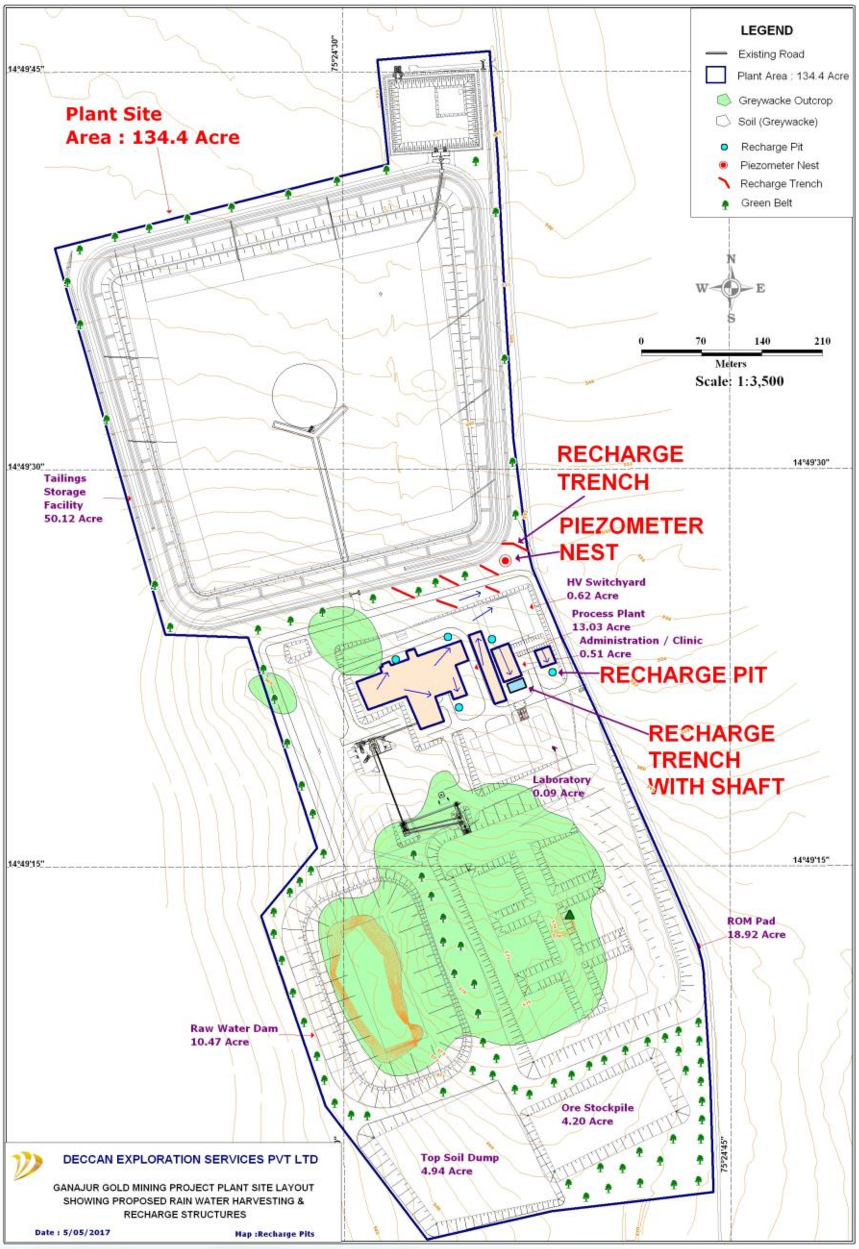


The infrastructure required to fulfil the water management plan and the overall configuration are indicated in Table 13.8 and the location of proposed rainwater harvesting and recharge structures are shown in Figure 13.5..

Table 13.8 Rainwater harvesting and recharge structures recommended

Rainwater harvesting from	Harvesting and recharge structures to be constructed	Dimensions	No. of structures
Rooftop of site office and stores	Connecting pipes from rooftop; recharge trench with shaft	Trench: 10 x 3 m Borehole shaft: 156 mm diameter Depth: 150 m (top 30 m to be cased)	One
Rooftop of other buildings/covered areas	Recharge pits	Pits: 2 m x 2 m x 5 m	Four pits
Open area	Recharge trenches – discontinuous type	10 m x 0.75 with 1 m depth	Area shown on map; 10 trenches or more in each section
Open area	Storage/Recharge pond	12 m x 12 m x 3 m	One pond
Open area (nearer to stormwater drains)	Recharge pits with de-silting chambers connected to stormwater drains	2 m x 2 m x 5 m; de-silting chamber/silt traps with minimum of three chambers	Minimum of two; location not shown on map

Figure 13.5 Location of proposed rainwater harvesting and recharge structures



13.4.3 Wastewater generation, treatment and disposal

An estimated 2.4 m^3/day of sewage will be generated and will be treated in septic tank/soak-pit system.

The final residue/waste stream from the process plant that will have the residual cyanide and soluble arsenic concentrations below the recommended international environmental discharge standards (i.e. the International Cyanide Code and the US EPA respectively), prior to reporting into the TSF.

The sulphur dioxide/oxygen (Inco method) process will be used for the removal of WAD (weak acid dissociable) cyanide and arsenic will be removed by precipitation after the cyanide destruct stage via the addition of ferric sulphate to form a stable arsenical ferrihydrite compound.

Due to the potential acid forming nature of the tailings material, the TSF and its water management infrastructure such as the dams and channels will be lined with a compacted clay layer overlain by an HDPE geomembrane or concrete in the case of canals. This barrier system will avoid any long-term residual leaching of metals, arsenic and sulphides into the surface and groundwater resources.

13.5 Mine closure, remediation and reclamation, and decommissioning

13.5.1 Reclamation of the open-cast mining area

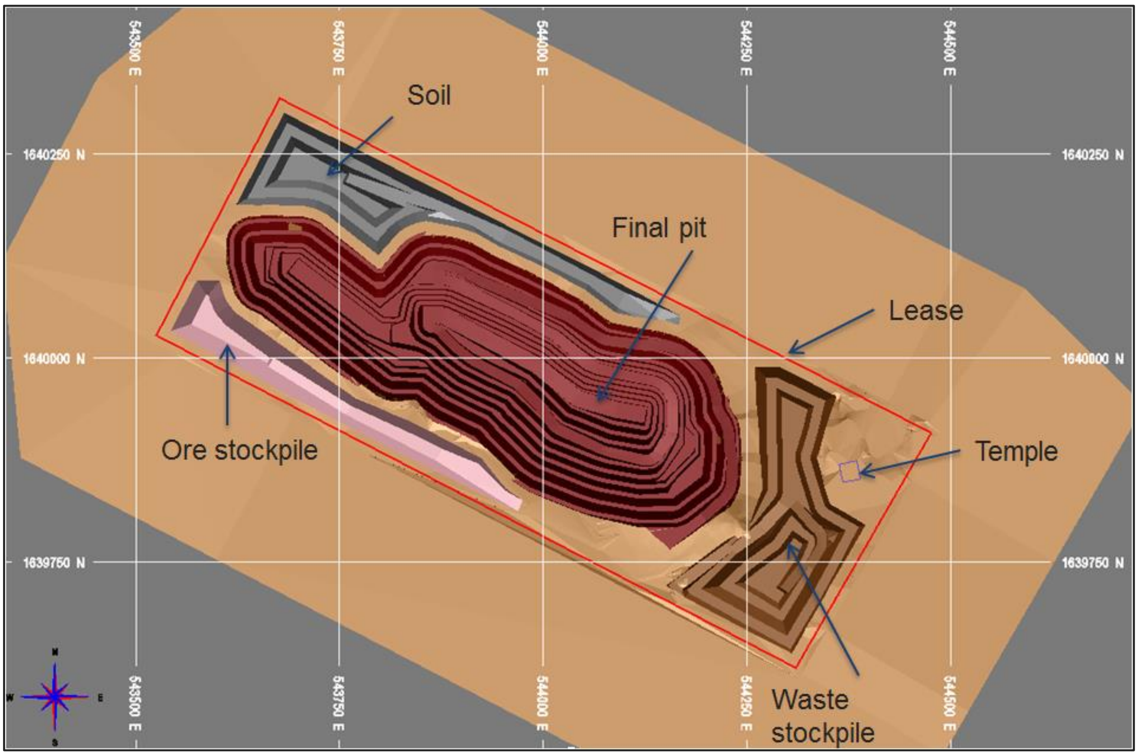
Table 13.9 shows the breakup of the area in post-mining phase.

Table 13.9 Post-mining land use pattern of the mining area only (ha/acres)

Description		Area in ha	Area in acres
Mined out areas	Stage 1	5.40	13.34
	Stage 2	8.50	21.00
	Stage 3	0.30	0.74
Stabilised soil stockpile		3.80	9.39
Stabilised oxide stockpile		2.80	6.92
Roads		1.60	3.95
Stabilised waste stockpile		4.40	10.87
Total		26.80	66.20
Temple buffer zone and safety zone developed under greenbelt		2.34	5.80
Total		**29.14**	**72.00**

The conceptual layout plan of the mine site is shown in Figure 13.6.

Figure 13.6 Conceptual mine site layout plan



13.5.2 Reclamation of the tailings storage facility

Once the maximum height of the TSF has been reached, reclamation processes will be initiated for closing the active zones from the edge of the stacking area. Topsoil will be used for rehabilitation of the area and revegetation/plantation will be implemented. Full details of the envisioned end-use and rehabilitation of the tailings facility is discussed under the TSF section of this FS report.

13.6 Recommendations

The data presented in this chapter summarises the data contained in the EIA report for the Ganajur Gold Mine, prepared by BSET. The aim of this section is to provide recommendations for additional studies and investigations by which to address gaps in the knowledge, confirm assumptions made and improve confidence in the available data.

The baseline ecological specialist studies took place over a single season, namely summer (March, April and May 2016). Terrestrial ecosystems are dynamic and complex and thus a more reliable assessment would entail additional sampling, as sampling, by nature, implies that not all species in the area will be recorded, due to seasonality, microhabitat requirements and management practices. In order to rule out the possibility that further protected or conservation-worthy species utilise the habitat in the study area, additional ecological studies should be undertaken during the winter and post-monsoon seasons and the findings of the EIA report updated accordingly.

The meteorological data used for assessing potential impacts in terms of air quality were based on data recorded on site between the months of March and May 2016. It is recommended that this data is verified through evaluation of meteorological conditions for the balance of the year and further compared to available historic records for the region so as to ensure that the predicted model outcomes are accurate.

The geohydrological impact assessment is based on a groundwater model derived from sampling of water levels in regional borehole wells as well as short-duration pumping tests on six groundwater boreholes. In order to better evaluate the potential impacts of the mine in the both the short and the long-term, it is recommended that the following tasks are undertaken:

- The installation and testing of site-specific groundwater monitoring borewells (three such boreholes are currently planned). These boreholes should be subjected to longer duration pump tests in order to confirm the characteristics of the aquifer. This data, together with the information gathered from sampling of existing borehole wells, can be used to update the groundwater flow model after two years.
- The updated groundwater model should seek to simulate regional impacts by increasing the modelling area to include adjacent watersheds.
- The model should further aim to identify the rate of groundwater recovery in the post-mining environment. The altered landscape may result in decant points arising which, depending on the quality of recharging groundwater, warrant additional management by the mine.
- The groundwater model should be updated to incorporate the geochemical assessment data of the mine materials under neutral and acidic conditions so that a pollution plume model can be prepared which illustrates any contaminants of concern which may be transported into the environment and whether any sensitive receptors (water courses and external users) are situated in the zone of influence. This aspect will also be important for management in the post-closure environment.
- Long-duration/sustainability pumping should be undertaken for boreholes to be used to supply groundwater to the mine in order to determine sustainable yields.

An implementation plan should be prepared which elaborates on the afforestation and green belt plans. This plan should be prepared by a suitably qualified specialist and should cover aspects such as suitable species, timing of the implementation, mechanisms to monitor the interventions implemented and to ensure that no unforeseen ecological impacts arise as a result. The plan should also cover the long-term aspects of this area in order to ensure that the mine will not inherit a long-term liability.

The estimated PPV used to evaluate potential impacts on the sensitive structures near the proposed mine should be confirmed once the rock constants have been determined.

The recommendations made in the monitoring program should be further elaborated in more detail. Monitoring locations should be selected and the monitoring requirements should be separated into the various phases of mining (construction, operation, closure and long term).

A detailed mine decommissioning and rehabilitation plan should be prepared which identifies all areas of potential liability, provides mechanisms for the removal of infrastructure not to remain permanently in place and rehabilitation of areas with the aim of producing a stable landscape which meets the intended end land use. The cost of such measures should be calculated and incorporated into the mines financial planning. The revised groundwater model to be prepared will assist in determining whether the mine needs any future plans for the installation of any water treatment infrastructure to manage quality groundwater and the duration for which such treatment may be required.

The EIA should be further elaborated by separately evaluating impacts for the various phases of the mine. The impacts should further be evaluated utilising a risk matrix which derives significance from aspects such as magnitude, scale, duration and probability for each potential impact during each phase.

14 COST ANALYSIS

The following chapter was provided by CPC Project Design Pty Ltd (CPC).

14.1 Process operating cost estimate

14.1.1 Introduction

The operating cost estimate for the Ganajur Gold Project is calculated in US dollars and uses prices obtained in, or escalated to, first quarter 2017 (1Q2017). Process operating costs by area are summarised in Table 14.1 as total dollars, dollars per tonne and dollars per ounce of gold for both sulphide and oxide ore. The operating costs have been calculated solely for the process plant; mining and site administration costs are not covered in this section.

The projected life of mine (LOM) average process operating cost for the 300,000 t/a Ganajur Main Project is $23.53/t of sulphide ore processed and $18.36 /t of oxide ore processed. This cost excludes all mining operating costs, taxes, permitting costs, non-process administrative costs and other government imposed costs unless otherwise noted.

Table 14.1 Summary of process operating costs

Category	Cost (US$)		
	$M/year	$/t	$/oz
300,000 t/a sulphide ore	7.06	23.53	249.31
300,000 t/a oxide ore	5.51	18.36	243.00

All operating cost data can be found in Appendix 14A.

14.1.2 Estimate currency and base date

The following exchange rates have been used in the compilation of the estimate:

- 1.00 US$ = 1.134 Australian dollar (A$)
- 1.00 US$ = 66 Indian Rupee (INR).

14.1.3 Basis of estimate

The estimate is based on the processing 300,000 t/a of ore through the process plant as either sulphides or oxides. Ore is to be campaigned throughout the LOM based on lithology and therefore processing costs have been developed specifically for each ore type.

Operating costs have been developed solely for the processing facilities, mining and administration costs are covered by others and not included in this report.

The following key inputs form the basis of both the process operating cost estimates:

- Electrical power draw quantities derived from the mechanical equipment lists, based on utilisation and expected demand
- Power costs were provided by the client at US$0.11/kWh
- Reagent and consumables costs are based on consumption outlined in the process design criteria (PDC) and flowrates from the mass balance
- Reagent and consumables unit costs are based on 1Q2017 pricing from local suppliers provided by the client
- Costs for comminution wear parts and for other indirect maintenance items are provided by the client

- Labour workforce requirements have been estimated for direct maintenance and operations personnel by the client
- Labour costs are client supplied and contain all burdened costs
- Mobile equipment and vehicle costs are based on CPC in-house data
- Metallurgical and process control sample analysis costs are provided by the client.

Escalation

Operating costs have a base date of 1Q2017 with no allowance for escalation.

Freight costs

Freight costs have been estimated for all reagents and consumables based on point of supply and similar quotes for the area.

Contingency

No contingency or other allowance has been included in the operating costs.

Exclusions

This operating cost estimate excludes the following:

- Costs external to the battery limits, including but not limited to mining, exploration, and TSF
- Owner's costs
- General and administrative costs not relating directly to the processing facilities including accommodation and transportation costs for all process plant personnel
- Capitalised costs including commissioning and first fills
- Sustaining capital costs associated with the replacement of depreciated equipment
- Mobile plant purchase
- Taxes and duties
- Accuracy provisions and contingency
- Corporate overhead charges including insurances or compliance costs
- Licences and land use fees or other such charges
- Financing costs
- Rehabilitation and closure costs.

14.1.4 Process operating costs

The average process operating costs for the LOM are $23.53/t of sulphide ore and $18.36/t of oxide ore. Table 14.2 shows a breakdown of the process operating costs for sulphide ore which include labour, power, reagents and consumables, maintenance and process general and administration (G&A) costs. Process costs for oxide ore processing are shown in Table 14.3. Details of the process operating costs can be found in Appendix 14A.

Table 14.2 Summary of process operating costs for sulphide ore

Category	Sulphide processing cost		
	$M/year	$/t	$/oz
Labour	0.61	2.04	21.59
Power	2.26	7.52	79.73
Reagents and consumables	2.14	7.14	75.67
Maintenance	1.60	5.34	56.59
Process G&A	0.45	1.48	15.72
Total	**7.06**	**23.53**	**249.31**

Table 14.3 Summary of process operating costs for oxide ore

Category	Oxide processing cost		
	$M/year	$/t	$/oz
Labour	0.61	2.04	26.96
Power	1.58	5.28	69.86
Reagents and consumables	1.27	4.22	55.87
Maintenance	1.60	5.34	70.67
Process G&A	0.45	1.48	19.63
Total	**5.51**	**18.36**	**243.00**

Process labour

The total process labour costs are calculated to be approximately $2.04/t of ore and are the same for both sulphide and oxide ore processing.

The process manning has been provided be the client and is summarised in the process operations organisational chart in Figure 7.9.

Expected monthly salaries have been provided by the client and account for all employee benefits (leave, public holidays, bonuses, etc.) and other contributions.

Crusher and mill relines along with all other major shutdown maintenance will be contracted out to local specialists and have been included under process maintenance costs.

Table 14.4 shows the process labour operating costs and the number of workers covering each position. The manning level is the same for processing both sulphide and oxide ore.

Figure 14.1 Process operations organisational chart

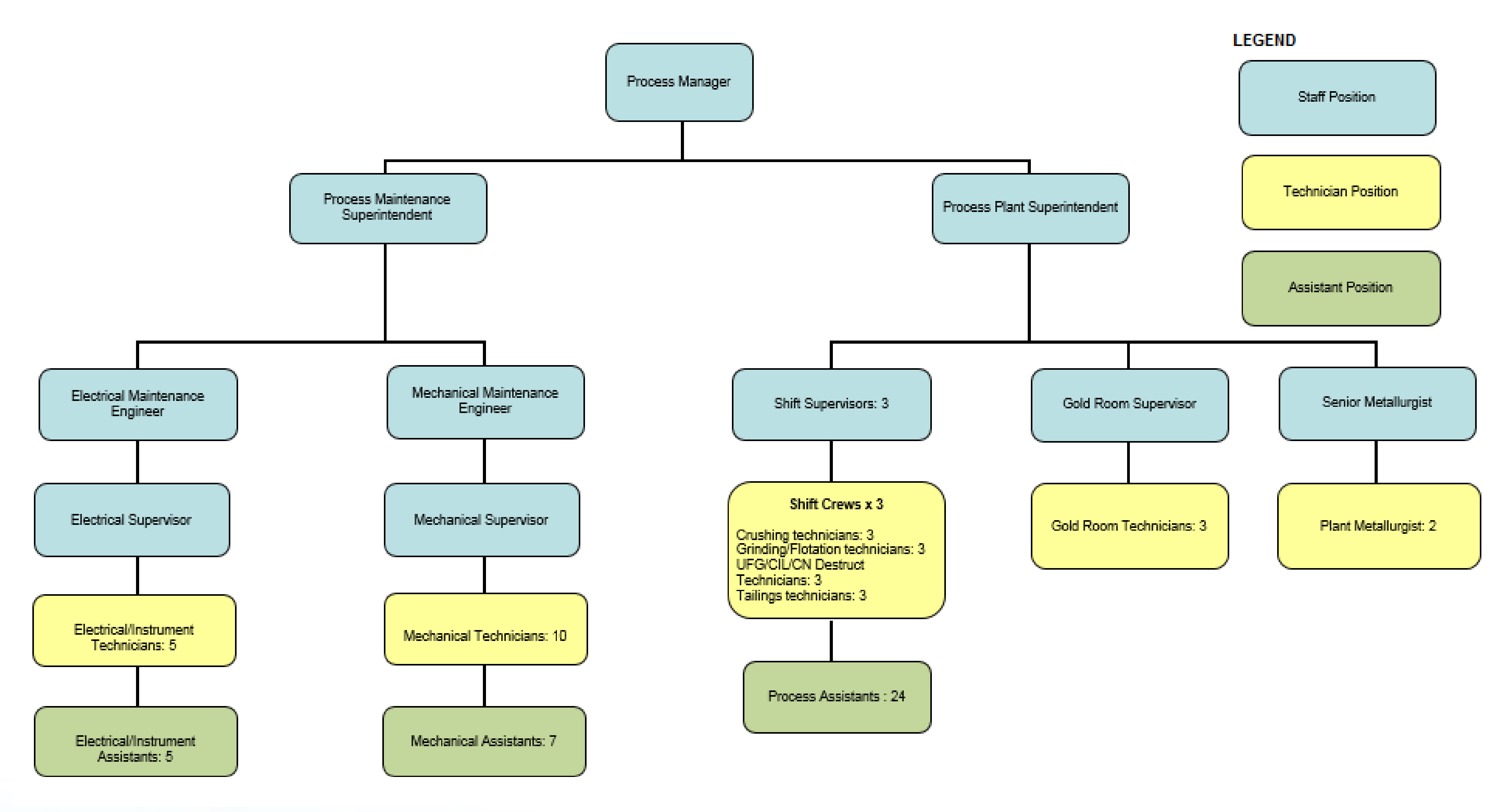


Table 14.4 Process labour operating costs and manning

Position	No. required	Cost ($/year)	Total annual cost ($)
Management and Technical			
Process Manager	1	21,818	21,818
Process Superintendent	1	18,182	18,182
Maintenance Superintendent	1	18,182	18,182
Maintenance Engineers	2	14,545	29,091
Gold Room Supervisor	1	14,545	14,545
Metallurgists	3	14,545	43,636
Operators			
Loader Operator (ROM feed)			
Shift Supervisor	3	12,727	38,182
Crushing	3	10,909	32,727
Crushing Assistant	6	2,182	13,091
Grinding/Flotation	3	10,909	32,727
Grinding/Flotation Assistant	6	2,182	13,091
UFG/CIL/CN Destruct	3	10,909	32,727
UFG/CIL/CN Destruct – Assistant	9	2,182	19,636
Gold Room	3	10,909	32,727
Tailings	3	10,909	32,727
Tailings – Assistant	3	2,182	6,545
Process Maintenance			
Maintenance Supervisor	2	10,909	21,818
Mechanical	10	10,909	109,091
Mechanical Assistant	7	2,182	15,273
Electrical/Instrumentation	5	10,909	54,545
Electrical/Instrumentation – Assistant	5	2,182	10,909
Total	**80**		**611,273**

Power

Power will be supplied by the local utilities with transmission lines installed specifically for the Project. The installed power demand for the process facility and site infrastructure is approximately 4 MW.

Power consumption is based on the load list developed from the mechanical equipment lists for each process area, accounting for load and motor efficiency factors, and equipment utilisation. Separate power calculations for sulphide and oxide ore are used in the power costs based on the differences in equipment used in processing the two ores.

The total power operating costs are calculated to be approximately $7.52/t of sulphide ore and $5.28/t of oxide ore. Power costs are decreased for oxide ore processing as the flotation circuit is not used and therefore the power consumption is decreased.

A 5% system loss factor is included to account for the losses incurred in transferring power from generation equipment to process inputs. These losses include transformers, power transmission and cable runs.

The unit power cost is $0.11/kWh as supplied by the client.

Additionally, there will be a requirement for a generator to supply power to the pumps located at the Varada River which will pump raw water to the raw water dam during the monsoon season.

Power costs have been calculated for these pumps based on:

- An estimated generator fuel consumption of 83.6 L/h at an efficiency of 75% (based on vendor datasheet for a 500 kVA Cat generator)
- A fuel cost of $0.75 /L (local supply cost estimate)
- A yearly operating time of 1,440 hours (assume two months at full capacity).

The power consumption by area is summarised in Table 14.5 for sulphide ore processing and in Table 14.6 for oxide ore processing.

Table 14.5 Sulphide ore processing – power summary

Area	WBS	Annual consumed power (kWh/y)	Consumed power (kWh/t)	Annual cost ($M/year)	Cost ($/t)
Crushing and screening	310	2,017,120	6.72	225,797	0.75
Grinding and classification	320	7,142,199	23.81	799,500	2.66
Gravity circuit	321	145,357	0.48	16,271	0.05
Flotation and regrinding	330	5,721,872	19.07	640,508	2.14
Leaching and adsorption	340	438,129	1.46	49,044	0.16
Gold recovery	350	290,653	0.97	32,536	0.11
Tailings	370	350,785	1.17	39,267	0.13
Reagents	380	159,241	0.53	17,825	0.06
Services	390	804,127	2.68	90,014	0.30
Process plant infrastructure	550 and 560	1,462,095	4.87	163,667	0.55
Water supply at plant site	680	101,026	0.34	11,309	0.04
Water supply at river (diesel generator)				67,379	0.22
System losses	5%	931,630	3.11	104,287	0.35
Total		**19,564,235**	**65.21**	**2,257,405**	**7.52**

Table 14.6 Oxide ore processing – power summary

Area	WBS	Annual consumed power (kWh/y)	Consumed power (kWh/t)	Annual cost ($M/year)	Cost ($/t)
Crushing and screening	310	2,017,120	6.72	225,797	0.75
Grinding and classification	320	7,142,199	23.81	799,500	2.66
Gravity circuit	321	145,357	0.48	16,271	0.05
Leaching and adsorption	340	668,506	2.23	74,833	0.25
Gold recovery	350	290,653	0.97	32,536	0.11
Tailings	370	384,459	1.28	43,036	0.14
Reagents	380	136,149	0.45	15,241	0.05
Services	390	554,930	1.85	62,119	0.21
Process plant infrastructure	550 and 560	1,462,095	4.87	163,667	0.55
Water supply	680	101,026	0.34	11,309	0.04
Water supply at river (diesel generator)				67,379	0.22
System losses	1.5%	645,125	2.15	72,215	0.24
Total		**13,547,618**	**45.16**	**1,583,904**	**5.28**

Reagents and consumables

The total reagents and consumable operating costs are calculated to be approximately $7.14/t of sulphide ore which is comprised of reagent costs totalling $5.28/t and consumables totalling $1.86/t. For oxide ore, the total reagents and consumable operating costs are $4.22/t made up of $2.78/t for reagents and $1.44/t for consumables.

The reagent and consumables costs for processing sulphide ore are higher due to the additional reagents and the regrind mill consumables in the flotation circuit. The reagent, consumables and other supply costs are based on 1Q2017 pricing from local suppliers provided by the client.

Reagent consumptions and costs are summarised in Table 14.7 for sulphide ore processing and Table 14.8 for oxide ore processing.

Table 14.7 Reagents consumption and costs – sulphide ore processing

Area	Reagent	Unit	Annual consumption (unit)	Unit cost* ($/unit)	Total annual cost ($/year)	Total annual cost ($/t)
Flotation/Regrind	PAX	t	45	1,983	89,228	0.30
Flotation/Regrind	Interfroth 50	t	3	2,919	8,756	0.03
Flotation/Regrind	Copper sulphate	t	30	1,349	40,455	0.13
Flotation/Regrind	Lead nitrate	t	40	6,393	257,010	0.86
Thickening	Flocculant	t	5	3,281	16,735	0.06
CIL	Hydrated lime	t	122	292	35,453	0.12
CIL	Cyanide	t	246	1,819	446,739	1.49
CIL	Oxygen	m^3	192,000	0.21	44,584	0.15
Detoxification	Hydrated lime	t	76	292	22,158	0.07
Detoxification	Copper sulphate	t	141	1,349	189,735	0.63
Detoxification	SMBS	t	201	326	65,550	0.22
Arsenic precipitation	Ferric sulphate	t	241	471	113,544	0.38
Arsenic precipitation	Sulphuric acid	t	352	393	138,548	0.46
Elution	Cyanide	t	14	1,819	24,639	0.08
Elution	Caustic	t	17	587	10,048	0.03
Elution	Hydrochloric acid	t	49	460	22,640	0.08
Gold room	LPG	m^3	180	0.31	56,418	0.19
Gold room	Silica	kg	156	1,125	175	0.00
Gold room	Borax	kg	624	1,752	1,093	0.00
Gold room	Soda ash	kg	48	976	47	0.00
Gold room	Nitre	kg	9	4,744	43	0.00
Total					**1,583,598**	**5.28**

**Unit cost includes estimated freight cost delivered to site*

Table 14.8 Reagents consumption and costs – oxide ore processing

Area	Reagent	Unit	Annual consumption (unit)	Unit cost* ($/unit)	Total annual cost ($/year)	Total annual cost ($/t)
Thickening	Flocculant	t	30	3,281	98,440	0.33
CIL	Hydrated lime	t	295	292	85,988	0.29
CIL	Cyanide	t	153	1,819	278,277	0.93
CIL	Oxygen	m^3	192,000	0.21	44,584	0.15
Detoxification	Hydrated lime	t	166	292	48,506	0.16
Detoxification	Copper sulphate	t	60	1,349	80,910	0.27
Detoxification	SMBS	t	252	326	82,182	0.27
Elution	Cyanide	t	14	1,819	24,639	0.08
Elution	Caustic	t	17	587	10,048	0.03
Elution	Hydrochloric acid	t	49	460	22,640	0.08
Gold room	LPG	m^3	180	0.31	56,418	0.19
Gold room	Silica	kg	123	1,125	138	0
Gold room	Borax	kg	495	1,752	867	0
Gold room	Soda ash	kg	36	976	35	0
Gold room	Nitre	kg	6	4,744	28	0
Total					**833,701**	**2.78**

Unit cost includes estimated freight cost delivered to site

Consumables usage and costs for both sulphide and oxide ore processing are summarised in Table 14.9.

Table 14.9 Consumables usage and costs

Area	Consumable	Sulphide ore total annual cost ($/year)	Sulphide ore total annual cost ($/t)	Oxide ore total annual cost ($/year)	Oxide ore total annual cost ($/t)
Crushing	Crusher liners (allowance)	223,881	0.75	223,881	0.75
Crushing	Screen liners	included in allowance		included in allowance	
Grinding	Ball mill liners				
Grinding	65 mm grinding balls	179,104	0.60	179,104	0.60
Flotation/Regrind	3.5 mm ceramic media	143,194	0.48	-	-
CIL	Carbon	2,686	0.01	20,042	0.07
Gold room	Furnace liners	2,539	0.01	2,539	0.01
Gold room	Furnace crucibles	5,749	0.02	5,749	0.02
Total		**557,153**	**1.86**	**431,315**	**1.44**

Maintenance

With the exception of major service consumables (liners etc.), maintenance costs have been provided by the client.

Table 14.10 outlines the operating cost allowance for maintenance which are the same for both sulphide and oxide ore processing.

Table 14.10 Maintenance costs

Area	Annual cost ($/year)	Cost ($/t)
Fixed maintenance costs	1,341,045	4.47
Contract labour for major shutdowns	261,194	0.87
Total	**1,602,239**	**5.34**

Process G&A

Process G&A costs cover all ancillary costs associated with operating the process facility. These include ROM rehandling, process vehicle fuel and maintenance, yearly assay costs and contingency. Yearly process G&A costs have been calculated to be $1.48/t of ore for both sulphide and oxide ore processing.

A summary of the processing G&A costs can be found in Table 14.11 including the basis for the costs.

Table 14.11 Process G&A costs

Item	Annual cost ($/year)	Cost ($/t)	Basis
ROM rehandle	179,104	0.60	Client provided
Fuel and vehicle maintenance	30,308	0.10	CPC estimated
Assay costs	195,224	0.65	Client provided
Contingency (10%)	40,464	0.13	CPC estimated
Total	**445,100**	**1.48**	

ROM rehandle:

- A cost per tonne for the rehandling of ROM ore has been provided by the client. This cost includes labour and cost of operating the front-end loader.

Process vehicles:

- Five 4WD dual cab utility vehicles, one integrated tool carrier, a 35 t mobile crane, a forklift and a flatbed truck have been accounted for under the process vehicle costs.
- The cost includes an annual maintenance allowance and fuel consumption.

Processing laboratory costs:

- The yearly estimated laboratory analysis cost for the process samples have been provided by the client.

Contingency:

- A contingency of 10% has been added to the process G&A costs to account for additional items not yet defined.

14.1.5 Risks and opportunities

Risks

Freight costs are preliminary and therefore there is a small risk associated in this area.

Costs of diesel, LPG, power and reagents/consumables are taken at a fixed period of time and are subject to fluctuations in the market.

Most prices were supplied in either Indian Rupee or Australian dollars and converted to a base of US dollars. Major changes to exchange rates would impact the project costs.

Major shutdown maintenance is relying on local contractors to perform the work and is assuming their personnel is experienced in these tasks. There may be increased costs associated with training local contractors during initial shutdown periods that have not been included.

Opportunities

There are opportunities to reduce reagent and consumable costs by entering into contracts direct with suppliers and therefore negotiate supply pricing.

An opportunity exists to optimise freight costs for reagents and consumables.

14.2 Mining cost estimate

Mining costs were estimated and supplied to DESPL in a quotation process, and the unit costs estimated on an average cost per tonne in Indian Rupee. These were converted to US$ for financial modelling.

14.2.1 Scope

Battery limits

The mining costs include:

- Mining area surface preparation (clearing and stripping)
- Load and haul
- Drill and blast
- Rehandle of stockpiles to the ROM
- Dewatering
- Mining infrastructure
- Haul road construction on the mining lease
- Maintenance of all haul roads.

The costs are inclusive of:

- Fuel costs
- Maintenance costs (inclusive of GET, consumables, tyres, accidental damage and contingency)
- Labour costs (operators, maintenance personnel and mining management, technical and administration)
- Sampling costs
- Equipment ownership costs (including transportation).

The mining costs exclude:

- ROM rehandle
- Closure including rehabilitation of the mining lease area
- Construction of pit roads external to the mining lease
- Ex-pit dewatering bores
- Any costs incurred from the primary crusher on (i.e. crushing excluded).

The costs do not include an allowance for contingencies.

Accuracy

The cost model is a P50% estimate, implying a 50% probability of being too high or too low.

The mining cost estimate is supported by:

- Budget estimate from five contractors
- Estimation of haulage profiles for each source and destination over the life of the Project.

The supporting evidence is sufficient such that Snowden is confident that this cost is estimate is accurate to within ±10% for the given inputs.

Methodology

The cost modelling was completed using Microsoft Excel software. The estimate is based on the average of four of the five contractor estimates, plus an allowance for owner's costs.

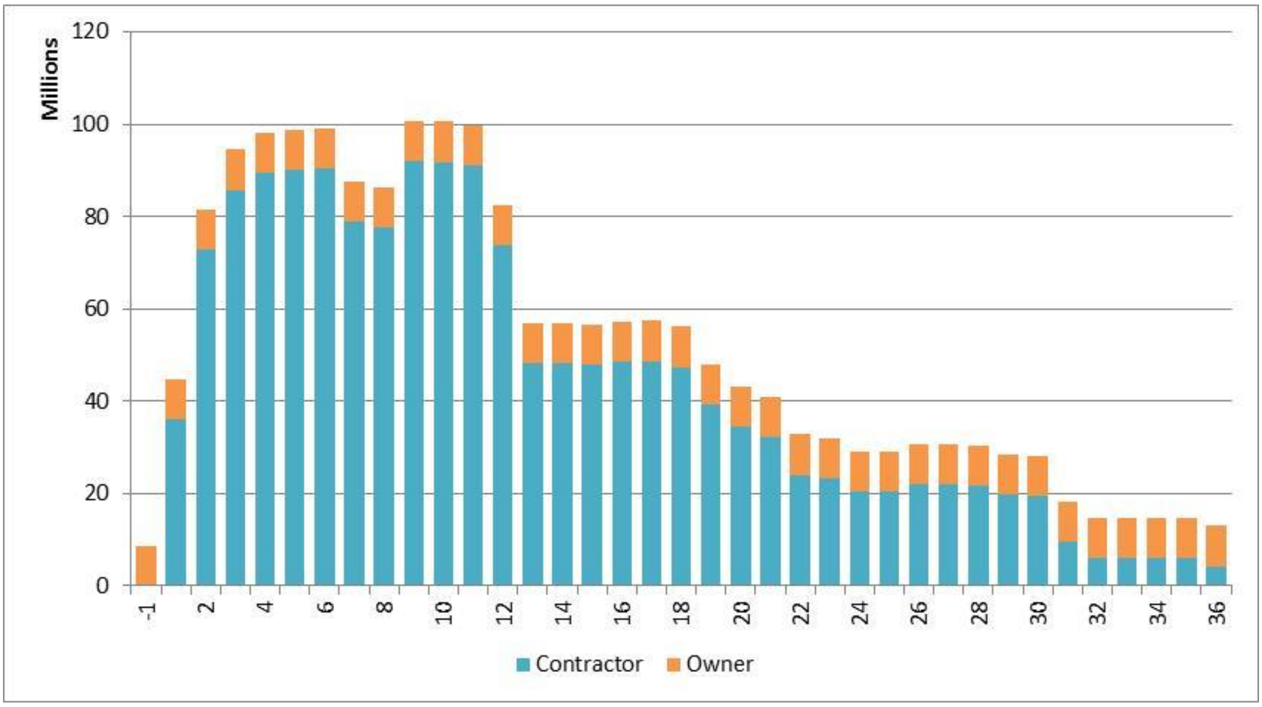
Currency

All costs are shown in Indian Rupee (₹) and are based to 2017.

14.2.2 Summary

Figure 14.2 summarises the total mining costs by quarter. The total LOM mining costs are estimated at ₹1,917 million which is equivalent to ₹174/t moved.

Figure 14.2 Cost summary (₹)



14.2.3 Capital costs

Summary

As the preferred option is to use a mining contractor, capital costs were expensed as an operating cost. The mining contractors are to provide their own infrastructure.

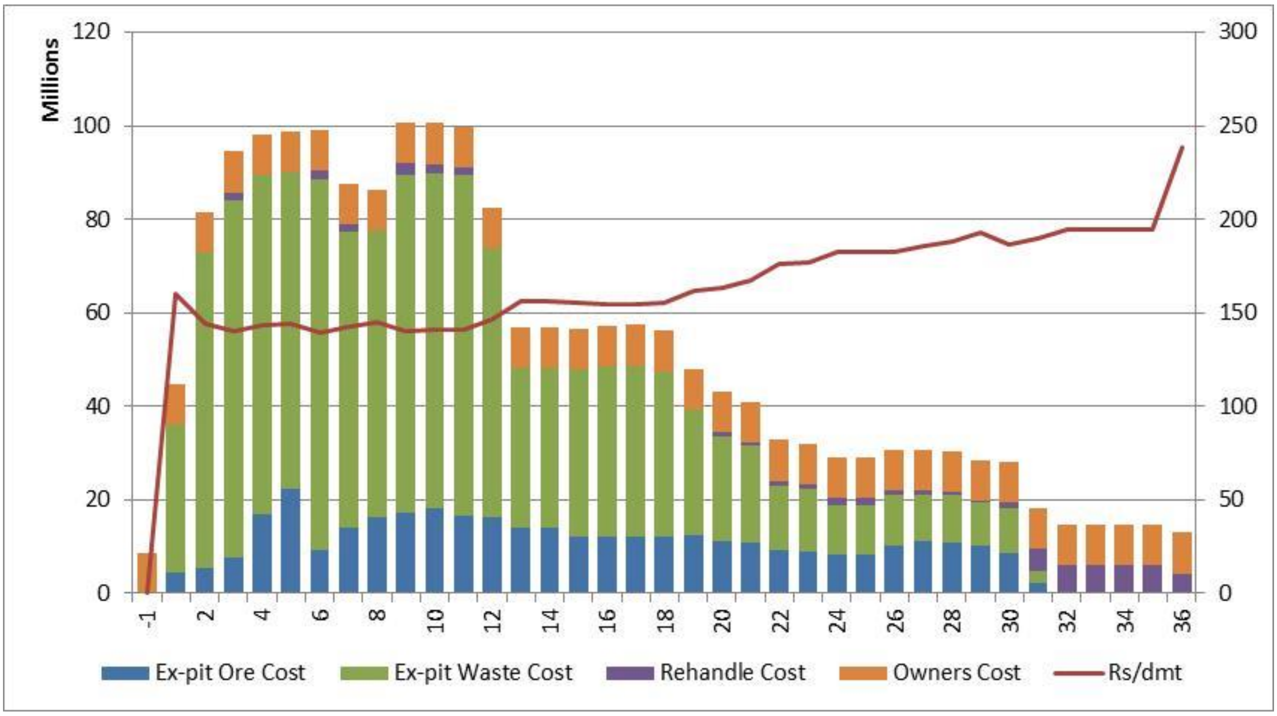
DESPL will operate out of their existing offices.

14.2.4 Operating costs

Summary

Snowden estimated a LOM operating cost of ₹1,917 million. The operating cost schedule is shown in Figure 14.3.

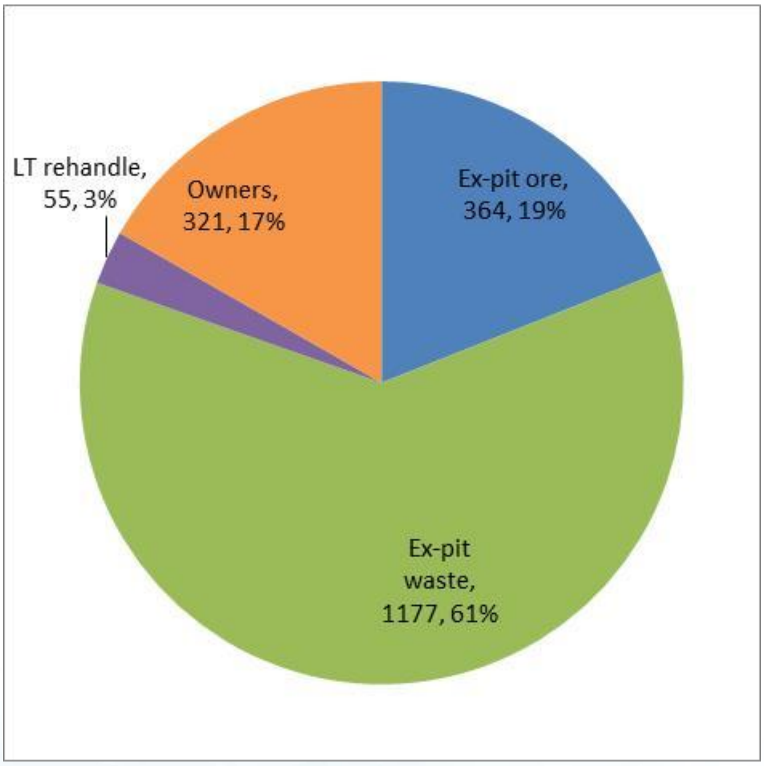
Figure 14.3 Operating cost summary (₹ million)



At the peak in quarters 9 to 11, operating costs are estimated to be around ₹100 million per quarter. The unit operating cost is relatively consistent at approximately ₹150/t moved for the first half of the mine life before rising over the second half to around ₹200/t moved.

Figure 14.4 summarises the total mining operating cost over the mine life. Nearly two-thirds of the total cost is attributable to mining waste.

Figure 14.4 Total operating cost spilt (₹ million)



Contractor cost

The mining contractors were supplied a proforma by DESPL to complete. Table 14.12 summarises the average costs from four of the five[2] contractors. The rates provided do not vary over time or mining location.

Table 14.12 Average contractor rates (₹/t)

Type	Ore	Waste
Drill	32.75	27.00
Blast	33.00	23.00
Load	26.75	26.25
Haul	31.25	30.00
Maintenance	8.75	8.75
Miscellaneous	12.50	12.50
Total	**145.00**	**127.50**

Figure 14.5 summarises the contractor cost by quarter based on the rates in Table 14.12 and the scheduled movements.

Figure 14.5 Contractor cost summary (₹ million)

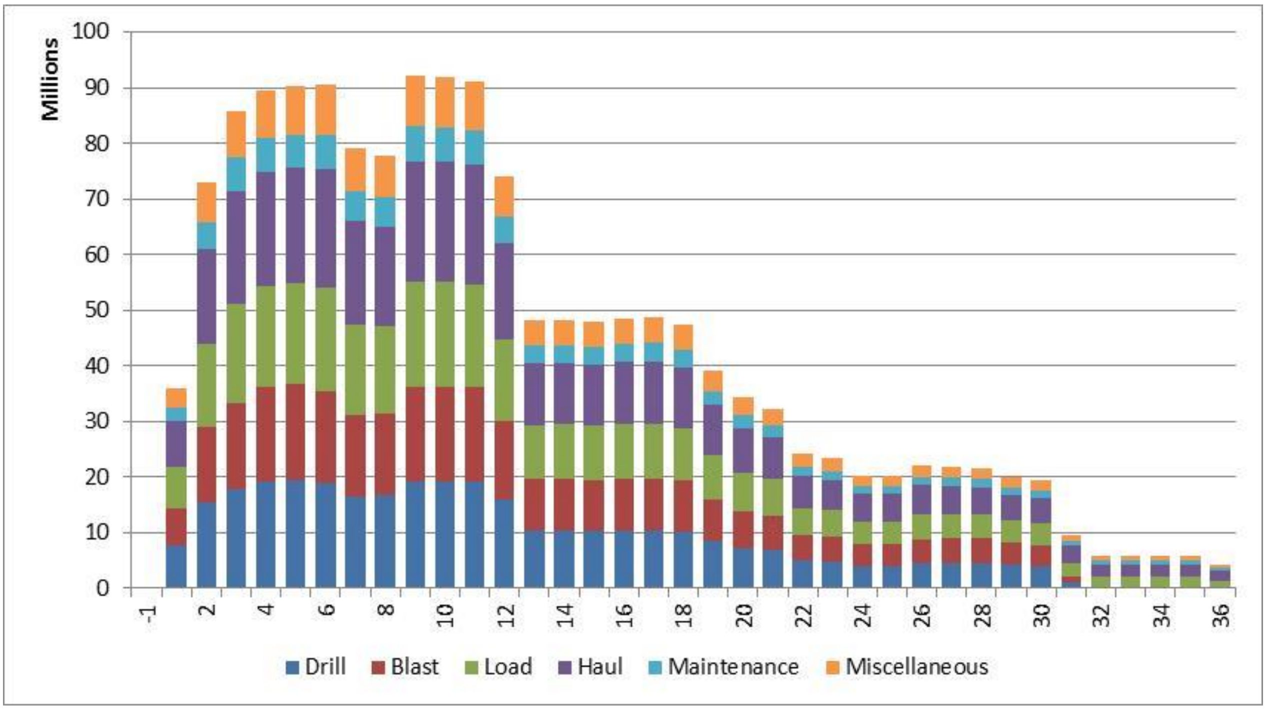
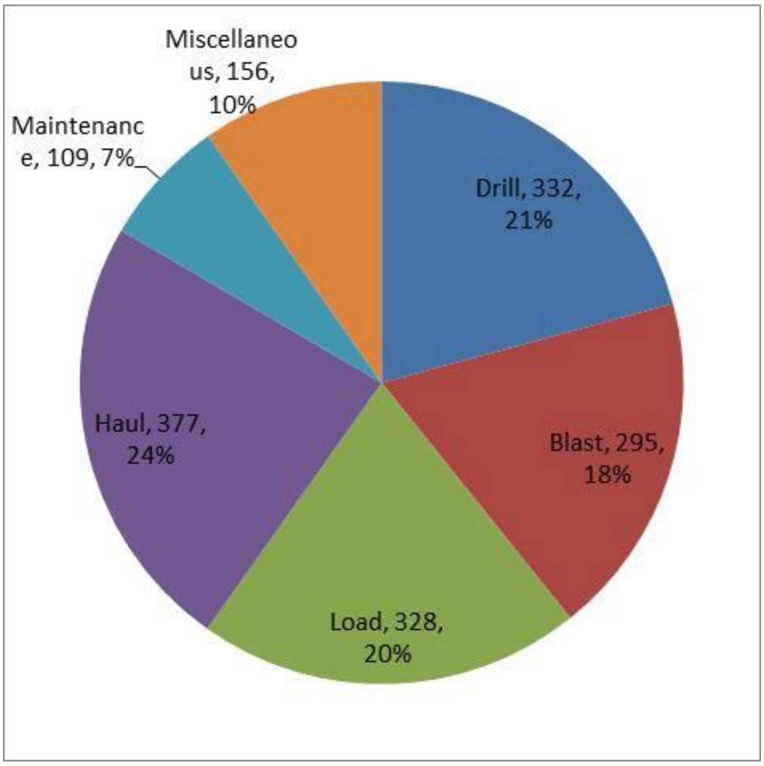


Figure 14.6 summarises the total contractor cost over the mine life. Drill, blast, load and haul each represent just under a quarter of the costs.

[2] One contractor was removed as they did not submit in the requested format.

Figure 14.6 Total contractor cost spilt (₹ million)



Owner’s cost

Table 14.13 shows a summary of the owner’s labour costs inclusive of on-costs (allowances for meals, transport, PPE and benefits). The costs were provided by DESPL.

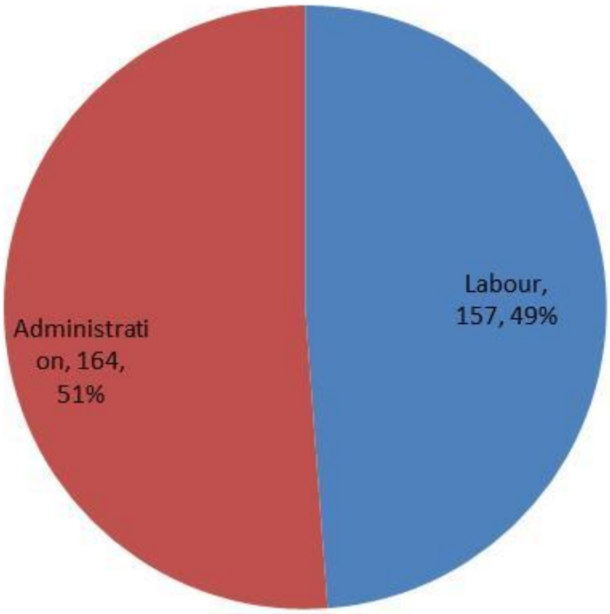
Table 14.13 Owner labour costs (₹)

Labour type	Full-time employees	Quarterly salary	Quarterly cost
Senior mine management	2	360,000	720,000
Mine planning	2	240,000	480,000
Mine geology	2	240,000	480,000
Pit technician	4	36,000	144,000
Environmental	2	240,000	480,000
Survey	2	180,000	360,000
Geotechnical	1	270,000	270,000
Production superintendent	2	300,000	600,000
Site clerk	6	36,000	216,000
OH&S officer	2	240,000	480,000
Total	**25**	**169,200**	**4,230,000**

An allowance totalling ₹4,435 k/qtr for administration items was included in the owner’s cost. This allowance covers items such as office supplies, software licences, survey and geology supplies, bank charges, recruitment and insurance.

Figure 14.7 summarises the owner’s costs, which are split almost evenly between labour and administration.

Figure 14.7 Owner's cost summary



14.2.5 Sensitivity

The cheapest and most expensive contractor costs were evaluated using the same methodology. This resulted in mining costs of ₹161/t moved and ₹184/t moved which are within 10% of the average estimate.

14.3 Capital cost estimate

14.3.1 Cost summary

CPC has prepared the total capital cost estimate for the Ganajur Main Project, which is summarised in Table 14.14.

Table 14.14 Capital estimate summary – Ganajur Main Project (1Q17, ±15%)

WBS	Description	US$
1	Mining	874,470
3	Process plant	19,028,226
5	Process plant infrastructure	5,042,640
6	Infrastructure plant and equipment	4,283,556
8	Construction indirects	2,152,503
9	Indirect costs	14,961,410
Total		**46,342,805**

The capital cost estimate is summarised by construction discipline in Table 7.1. A detailed report of the overall capital cost estimate can be found in Appendix 14B.

Table 14.15 Cost summary by discipline

Discipline	US$
Earthworks	5,760,165
Roads	446,091
Concrete	1,321,241
Steelwork	995,202
Platework and tanks	1,217,011
Mechanical	9,306,362
Pipework	716,643
Electrical cable	540,941
Electrical equipment	5,100,840
Instrumentation and control	168,669
Architectural	1,038,539
Construction (SMP, E&I)	2,164,689
Freight	1,256,517
Indirects	12,914,022
Contingency	3,395,873
Total	**46,342,805**

While CPC has compiled the majority of the estimate scope and pricing, Prime Resources provided the engineering quantities for the TSF.

DGML assisted with obtaining indicative in country rates for earthworks and concrete. DGML provided the owners' costs, power supply costs and the bitumen road construction rates in India.

A Level 2 summary, which provides detail for each cost area relevant for a gold process plant is presented in Table 14.16 for direct costs and Table 14.17 for indirect costs and total.

Table 14.16 Level 2 cost summary (direct costs)

WBS	Area	US$
	DIRECT COST	
1	**Mining**	**874,470**
3	**Process plant**	**19,028,226**
300	General	1,971,557
310	Crushing and screening	3,776,929
320	Grinding and classification	3,120,604
321	Gravity	717,861
330	Flotation	1,457,904
335	Regrind	1,203,504
340	Leaching and adsorption	3,046,906
350	Gold recovery	940,324
370	Tailings and detoxification	859,525
380	Reagents	911,613
390	Services	1,021,499
5	**Process plant infrastructure**	**5,042,640**
510	Earthworks and drainage	998,998
530	Tailings storage facility	2,491,131
550	Plant buildings	393,847
560	Laboratory and sample prep	489,062
580	Fencing	10,511
590	Non-plant building	659,091
6	**Infrastructure plant and equipment**	**4,283,556**
611	Power supply	1,348,485
612	Emergency power generation	122,339
663	Sewage	8,445
671	LPG storage	10,662
680	Water supply	2,793,625
8	**Construction indirects**	**2,152,503**
	DIRECT COSTS SUBTOTAL	**31,381,395**

Table 14.17 Level 2 cost summary (indirect costs)

WBS	Area	US$
	INDIRECT COSTS	
920	EPCM	3,148,943
941	Vendor representatives	480,600
961	Owners costs (personnel, management team)	854,424
962	First fills and spares	1,789,134
963	Permitting and statutory approvals	184,964
964	Light vehicles and cranage	668,864
966	Office costs and expenses	147,921
967	Client corporate (taxes and duties)	652,127
968	Site land acquisition and development costs	3,593,561
969	Camp and messing facility	45,000
992	Contingency	3,395,873
	Indirect costs subtotal	**14,961,410**
	TOTAL	**46,342,805**

The work breakdown structure (WBS) is based on the standard CPC WBS for capital projects and is presented in Appendix 14D.

14.3.2 Estimate currency and base date

The estimate is expressed in US dollars (US$) based on prices and market conditions current at first quarter 2017 (1Q17).

The following exchange rates have been used in the compilation of the estimate:

- 1.00 US$ = 1.34 Australian dollar (A$)
- 1.00 US$ = 66 Indian Rupee (INR)

Foreign currency exposure is shown in Table 14.18.

Table 14.18 Foreign currency exposure

Currency	Percentage of capital cost estimate
US$	46.5%
INR	27.5%
A$	25.9%
Other	<0.1%

14.3.3 Basis of capital cost estimate

General arrangement drawings (2D) and a 3D earthworks model were developed with sufficient detail to estimate the engineering quantities for earthworks, concrete, steelwork, mechanical and electrical for the processing plant and infrastructure.

Construction unit rates that reflect current market conditions have been established for bulk materials and capital equipment via an extensive budget quotation request (BQR) process. Budget pricing for equipment were obtained from suitable suppliers.

The direct labour costs were derived on the following basis:

- Earthworks – schedule of rates provided by DGML that reflect indicative earthworks rates in India
- Concrete – schedule of rates provided by DGML that reflect indicative rates in India
- Structural, mechanical and piping (SMP) – schedule of rates quote obtained from an Indian contractor
- Electrical and instrumentation (E&I) – man-hours estimated from first principals multiplied by a productivity factor for Indian labour and multiplied by an average gang rate.

Indirect labour costs were determined using a factor of direct labour.

EPCM costs were based on a percentage of total project direct costs.

The capital cost estimate has been derived to within an estimated accuracy of ±15%.

Estimate structure

The structure of the estimate consists of the following major categories:

- Direct costs
- Indirect costs
- Owners costs
- Contingency

A description of these cost categories is outlined below.

Direct costs

Direct costs are expenditures that include the supply of the equipment and materials, freight to site, project site labour to construct the plant and assembled equipment, supporting facilities and services, and growth allowances.

Indirect costs

Indirect costs are expenditures that cover temporary construction facilities plus engineering, procurement and construction management services (EPCM), and contractor and EPCM indirect costs.

Owners costs

Owner's costs include expenses associated with owner's team, insurances, owner's contingency, foreign currency rate of exchange variation, government duties, taxes, permit fees, licence fees, land cost, right of way, royalties and business readiness.

Contingency

Contingency is an allowance additional to the base cost estimate to cover unforeseeable elements of cost, risk and uncertainty within the defined scope of work. This is an allowance to cover unforeseen costs due to lack of complete, accurate and detailed information.

14.3.4 Direct cost development

Direct cost summary

The direct costs include:

- Supply of bulk materials and fixed equipment
- Labour to install and manage the construction activities including wages and salaries with loadings for site labour, supervision and management
- Contractors and suppliers' mark-up and profit
- Freight expenses for bulk material and equipment to site.

Direct cost quantity development

The direct costs have been derived based on the estimated quantities as listed in Table 14.19.

Table 14.19 Material quantities

Classification	Quantity	Unit
Earthworks – bulk earthworks, water storage dam, ROM pad	869,437	m^3
Earthworks – TSF	283,227	m^3
HDPE lining – water storage dam	35,420	m^2
HDPE/Geotextile lining – TSF	338,663	m^2
Concrete – Process Plant	3,140	m^3
Concrete – TSF	981	m^3
Structural steel	367	t
Platework/tankage	285	t
Piping	17,360	m
Electrical/Instrumentation cable	67,115	m

Earthworks

Quantities for the process plant bulk earthworks and water storage dam were derived from the 3D model. The model produced cut to fill quantities.

Prime Resources provided the earthworks quantities for the TSF construction.

Concrete

Concrete quantities were derived from preliminary engineering using CPC standard details and design basis or estimated from sketches/redline mark-ups from similar project designs.

Structural steel

Structural steel quantities were derived from preliminary engineering utilising the design software, Space Gass. Preliminary equipment load data and the CPC standard design basis were used to size structural steel members to develop the quantities for the majority of the structures. Minor structures were estimated from relevant previous project designs or factored from similar facilities.

Platework/Tankage

Design calculations were performed on all tanks as outlined in the process design criteria (PDC). Using the API 650 code and CPC's standard design requirements, tank shell, floor plate, baffles and stiffeners were sized to determine an accurate weight of each tank.

Platework items were sized using experience and/or existing similar design drawings to determine the mass of plate and lining surface area.

Piping

A detailed piping line list was developed from the process flowsheet diagrams and piping line lengths were estimated using the general arrangement drawings. Fittings were factored on a per meter pipe length basis and valves were factored as a percentage on the overall supply cost of pipe materials.

Mechanical equipment

The mechanical equipment list was developed from the process flowsheet diagrams and process design criteria. Scope of work documents complete with duty datasheets and standard specifications where prepared for the majority of mechanical equipment.

Electrical and instrumentation

The electrical equipment list was developed from the mechanical equipment list. Scope of work documents complete with duty datasheets and standard specifications were prepared to outline the requirements for the major electrical equipment such as transformers, high voltage switchgear, low voltage motor control centres, variable speed drives and the control system.

A cable schedule for high voltage power distribution, low voltage motor controls centres, control and instrumentation was developed based on the mechanical equipment list and the lengths measured from the general arrangement drawings.

An instrument list was prepared from the process flow diagrams and the level of instrumentation and process control was defined by DGML.

Electrical and instrumentation bulk materials such as cable ladder, conduits, small power and lighting, termination kits, earthing, local control stations, distribution boards, etc. were estimated from first principles and presented in the electrical material take-off list.

Direct cost pricing basis

Estimate pricing was derived from a combination of the following sources:

- Budget quotation – budget pricing solicited specifically for the project
- 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.

Structural steel fabrication

A schedule of rates request for quotation (RFQ) enquiry package was issued to five structural steel fabricators. The RFQ enquiry package contained a scope of work, CPC's standard terms and conditions, pricing schedule, CPC's standard specifications, typical steel detail drawings and a structural steel bill of quantities (BoQ).

The following fabricators where issued with the RFQ:

- Summa NK Contracting Company (Thailand)
- McConnell Dowell (Thailand)
- China Grand (China)
- Epiterma (Indonesia)
- AG&P (Philippines).

Summa, McConnell Dowell and China Grand submitted quotations. Summa's rates, as presented in Table 14.20, were used for the capital cost estimate as they were the closest to the average rates of all three contractors. Summa is well known to CPC for their quality of work on previous projects that CPC has been involved with.

During the project implementation phase, the intent is to potentially use suitable Indian fabricators to fabricate the structural steel, thus saving on import duties and taxes and freight.

Table 14.20 Schedule of rates – steel fabrication

Commodity item	Unit	Rate (US$/unit)	
		Range	Summa
Light structural steel <25 kg/m	t	2,093 – 2,625	2,093
Medium structural steel 25-75 kg/m	t	1,992 – 2,359	2,057
Heavy structural steel 75-125 kg/m	t	1,656 – 2,217	1,983
Floor grating	m^2	62.0 – 92.3	62.0
Handrail	m	48.6 – 92.3	48.6
Stair treads	each	47.2 – 51.7	47.2
Cladding	m^2	13.3 – 24.2	13.6

Platework fabrication

A schedule of rates RFQ enquiry package was issued to five platework fabricators. The RFQ enquiry package contained a scope of work, CPC's standard terms and conditions, pricing schedule, CPC's standard specifications, typical platework detail drawings and a platework BoQ.

The following fabricators where issued with the RFQ:

- Summa NK Contracting Company (Thailand)
- McConnell Dowell (Thailand)

- China Grand (China)
- Epiterma (Indonesia)
- AG&P (Philippines).

Summa, McConnell Dowell and China Grand submitted quotations. Summa's rates, as presented in Table 14.21, were used for the capital cost estimate as they were the closest to the average rates of all three contractors.

As per the structural steel fabrication, the intent is to potentially use suitable Indian fabricators.

Table 14.21 Schedule of rates – platework fabrication

Commodity item	Unit	Rate (US$/unit)	
		Range	Summa
Fine ore bin – mild steel, Paint Spec P1	t	2,330 – 2,555	2,543
Chutes – mild steel, Paint Spec P1	t	2,308 – 2,555	2,308
Chutes – mild steel, Paint Spec P2	t	2,386 – 2,708	2,386
Boxes – mild steel, Paint Spec P2	t	2,373 – 3,029	3,029
Hoppers – mild steel, Paint Spec P2	t	2,515 – 2,729	2,729
Tanks – mild steel, Paint Spec P2	t	2,458 – 2,903	2,903
Tanks – 304 stainless steel	t	7,105 – 10,680	7,701
Tanks – 316 stainless steel	t	8,810 – 12,680	9,635
Bisalloy lining 6 mm thick bolted	m^2	132.0 – 207.2	207.2
Bisalloy lining 12 mm thick bolted	m^2	232.0 – 376.8	376.8
Bisalloy lining 12 mm thick – loosed	m^2	186.0 – 330.2	330.2
Rubber lining 6 mm thick	m^2	205.0 – 351.0	225.9
Rubber lining 12 mm thick	m^2	323.0 – 591.2	350.0

Mechanical equipment supply

RFQ enquiries were issued to multiple Indian and international reputable suppliers for the majority of the mechanical equipment. Each RFQ enquiry package contained a scope of supply, CPC's standard terms and conditions, equipment datasheet(s) and standard project specifications.

The value of equipment priced from current enquiries represents 94% of the total equipment supply. Historical database pricing older than six months and escalated to the current estimate base date comprised 4% of the equipment value with the remaining 2% of equipment value factored from costs with a basis.

Piping supply

Cost for supply of piping and fittings was developed from CPC's historical pricing.

An allowance was made for the supply costs for manual and actuated valves.

Electrical equipment and bulk materials supply

RFQ enquiries were issued to multiple Indian and international reputable suppliers for the following major electrical equipment:

- Transformers
- Low voltage motor control centres (MCCs)
- High voltage switchgear
- Variable speed drives

- PLC equipment and programming
- High voltage cables.

Each RFQ enquiry package contained a scope of supply, CPC's standard terms and conditions, equipment datasheet(s) and standard project specifications.

Bulk materials supply costs such as low voltage and instrumentation cable, cable ladder, lighting, termination kits, etc. were obtained from CPC's historical pricing database.

Earthworks

Earthworks pricing was derived from schedule of rates provided by Aparna Engineers, a project and engineering consultancy in Bengaluru, India. DGML engaged Aparna Engineers to provide representative earthworks rates in India.

Drill and blast, excavation and haulage rates from the mining contractor were used for the excavation of the water storage dam. The mining contractor will be mobilised to site earlier than the mining schedule in order to complete the required work on the water storage dam.

The rates in Table 14.22 were used to estimate the costs for the bulk earthworks and water storage dam construction.

Table 14.22 Schedule of rates – bulk earthworks and water storage dam construction

Commodity item	Unit	Rate (INR/unit)
Clearing of vegetation	m^2	10
Strip and stockpile topsoil	m^2	85
Bulk earthworks (cut, moisture condition, move, place, compact material)	m^3	80
Borrow, moisture condition, transport and place compacted structural fill	m^3	265
Borrow, moisture condition, transport and place compacted select fill	m^3	260
Borrow, moisture condition, transport and place compacted clay liner	m^3	180
Drill and blast, excavate, transport (0.5 km) quarry rock, by mining contractor	m^3	US$6.0
Drill and blast, excavate, transport (6.0 km) quarry rock, by mining contractor	m^3	US$8.4
Detailed earthworks, drains, pads	m^3	170
Excavation		

The rates in Table 14.23 were used to estimate the costs for the construction of the TSF.

Table 14.23 Schedule of rates – TSF construction

Commodity item	Unit	Rate (INR/unit)
Clearing of vegetation	m^2	10
Strip and stockpile topsoil	m^2	85
Bulk excavation in class a material	m^3	85 – 100
Base preparation of in-situ material and imported material	m^3	170 – 180
Backfill with selected and approved material	m^3	150
Placement and compaction of selected material for road layer works	m^3	180
Supply and install A2 BIDIM geotextile	m^2	100 – 110
Supply and install A4 BIDIM geotextile	m^2	120 – 125
Supply and install A6 BIDIM geotextile	m^2	120 – 125
Supply and install 1.5 mm HDPE geomembrane	m^2	150 – 160

Concrete

Concrete pricing is based on a schedule of rates provided by SGES, an engineering consultancy located in Bengaluru, India. DGML engaged SGES to provide representative concrete rates in India.

The rates in Table 14.24 were used to estimate the costs for the concrete works.

Table 14.24 Schedule of rates – concrete

Commodity item	Unit	Supply cost (INR/unit)	Install cost (INR/unit)	Total cost (INR/unit)
Ground slabs, <300 mm thick	m^3	17,899.70	2,990.00	20,889.70
Concrete walls	m^3	16,407.85	3,470.22	19,878.07
Suspended slabs, >300 mm thick	m^3	17,899.70	2,990.00	20,889.70
Minor equipment pedestals, <50 m^3	m^3	34,316.46	3,432.00	37,748.46
Structural footing	m^3	40,323.37	3,432.00	43,755.37
Concrete columns	m^3	39,381.89	3,432.00	42,813.89
Tank ring beams	m^3	18,100.99	4,992.00	23,092.99
Precast concrete sumps	m^3	18,496.95	5,101.20	23,598.15
Blinding concrete	m^3	6,985.85	1,926.60	8,912.45
Tank ring beam structural fill	CCM[#1]	1,781.82	491.40	2,273.22
Tank ring beam infill oil impregnated sand	CCM[#1]	2,137.52	590.20	2,727.72
Steel cast-in items/holding down bolts	t	124,444.32	34,320.00	158,764.32
Fencing 2.4 m high	linear m	1,874.82	2,392.22	2,392.22

[#1] CCM – compacted cubic metre

Structural, mechanical and piping installation

The SMP installation costs were derived by issuing a schedule of rates RFQ enquiry package, with preliminary quantities to the following Indian contractors.

- RVPR Construction Pvt. Ltd (RVPR)
- Petron Engineering Construction Ltd
- Simplicity Projects Pvt. Ltd
- Design Tribe (India) Pvt. Ltd.

The RFQ enquiry package contained a scope of work, CPC's standard terms and conditions, pricing schedule, standard specifications, mechanical equipment list, layout drawings, and preliminary steelwork, platework, piping quantities and mechanical equipment weights.

RVPR was the only contractor to submit a budget price for the preliminary quantities. RVPR also submitted an estimate of man-hours to erect steelwork, platework, mechanical equipment and piping.

The rates in Table 14.25 were used to estimate the costs for the SMP installation for the final estimated quantities.

Table 14.25 Schedule of rates – SMP installation

Commodity item	Unit	Install cost (US$/unit)
Structural steel, light, medium, heavy duty	t	319
Grating	m^2	4.00
Stair treads – Webforge	each	1.50
Handrail	m	1.75
Checker plate, 6 mm thick	m^2	15.00
Purlins – falvanised	m	4.00
Cladding – Colorbond	m^2	4.00
Platework – mild steel	t	319
Platework – 304 stainless steel	t	372
Tanks – mild steel	t	319
Tanks – 304 stainless steel	t	372
Tanks – 316 stainless steel	t	372
Safety guards	m^2	115
Conveyor skirts	m	120
Liner – Bisalloy 6 mm bolted	m^2	12
Liner – Bisalloy 12 mm bolted	m^2	14
Liner – Bisalloy 6 mm site welded	m^2	35
Rubber lining – 6 mm or 12 mm thick	m^2	85
Mechanical equipment installation	t	283

The rates in Table 14.26 were used to estimate the costs for the piping installation for the process plant.

Table 14.26 Schedule of rates – piping installation

Commodity item	Pipe size (mm)	Install cost (US$/metre)
High density polyethylene(HDPE) piping		
Specification PE1/PE8	16	1.20
	20	2.00
	63	4.70
	90	6.70
	110	8.20
	160	11.90
	225	16.70
Carbon steel piping		
Specification SC1/SC2/SC6/SC9	15	1.80
	25	3.00
	40	4.80
	50	6.00
	80	9.60
	100	12.00
	150	18.00
	200	24.00
Galvanised steel piping		
Specification SG1	15	1.80
	25	3.00
	40	4.80
	50	6.00
Carbon steel rubber lined piping		
Specification SR3	100	14.90
	150	22.50
	200	30.00
Stainless steel piping		
Specification SS1	15	2.10
	25	3.50
	50	6.90
	80	11.00

The rates in Table 14.27 were used to estimate the costs for the overland and buried piping installation for the process plant.

Table 14.27 Schedule of rates – overland and buried piping installation

Commodity item	Pipe size (mm)	Install cost (US$/metre)
High density polyethylene (HDPE) piping		
Buried PE1	32	4.15
Buried PE1	63	6.45
Buried PE5	110	9.95
Buried PE5	160	13.65
Buried PE1/PE2/PE3	355	28.10
Above ground (overland) PE1/PE2	90	6.70
Above ground (overland) PE1/PE2	110	8.15

Electrical and instrumentation Installation

The capital cost for the electrical and instrumentation installation was developed from first principles utilising CPC's experience on similar construction projects.

Direct installation man-hours for each activity were estimated on Australian labour. A productivity factor was applied to the estimated Australian man hours to account for the working practices in India.

The productivity factor estimate was derived from the RVPR's man-hour estimate submitted for the SMP installation and compared to the man-hours required to complete a similarly activity in Australia. A productivity factor of 4.214 was derived on this basis.

The high productivity factor is due to the different construction methods adopted in India compared to Australia and is not a reflection of the quality or availability of skilled labour in India.

In India, labour rates are relatively low and construction activities are more manual labour focused and mechanical equipment which is expensive, are less utilised. In Australia, labour rates are high, and manual labour is typically minimised by utilising more mechanical equipment such as cranes, automatic welding machines, forklifts, hydraulic elevated work platforms, etc.

An average gang rate of US$9/hour was derived to develop the direct installation costs. The rate was developed from RVPR's average cost per man-hour for the SMP installation and adjusted to include the following:

- Personal protection equipment (PPE)
- Tooling and consumables
- Food and transportation.

Power supply

DGML obtained costs for the grid power supply from Gemini Structures. Gemini Structures is an Indian class 1 electrical contractor located in Bangalore, India and specialise in high voltage transmission lines, switchyards and substations.

Gemini Structures submitted a budget price for the power supply to the project and includes the following:

- Liaising with the relevant regulator government bodies including the Karnataka Power Transmission Corporation Limited (KPTCL)
- Survey, detail drawings and engineering
- Procurement
- Civil works
- 8/10 MVA, 110 kV to 11 kV substation
- Take-off Bay at KPTCL substation
- Erection of towers and stringing of conductors and earthing for 7.5 km transmission line.

The quotation for the power supply can be found in Appendix 14E.

Roads

The costs for the project roads including the pit diversion roads and haul road upgrade were provided by DGML and were based on information from the Office of the Superintending Engineer, PRE Circle, Dharwad.

A cost of 6,000,000 INR per kilometre of bitumen road was applied for the capital cost estimate.

Buildings

The supply and installation cost for the buildings was derived by sending RFQ enquiries to international reputable suppliers. Each RFQ enquiry package contained a scope of supply, CPC's standard terms and conditions, building layout drawings and standard project specifications.

MS Projects from South Africa submitted the most competitive bid and their pricing was used for the capital cost.

The laboratory building and equipment was derived from a quotation received from Shiva Analyticals (India) Private Limited. A quotation for the laboratory can be found in Appendix 14G.

Construction indirects

The indirect costs for each major construction contractor was derived as a percentage for direct costs to cover the following:

- Preliminaries and generals
- Mobilisation and demobilisation
- Construction office set-up
- Construction equipment such as cranes
- Construction management.

Table 14.28 shows the percentages used on direct costs to determine the indirect construction costs for the major installation contractors.

Table 14.28 Indirect costs percentage to direct costs

Contractor	Percentage
Earthworks	10%
Concrete	10%
Structural, mechanical and piping	10%
Electrical and instrumentation	5%
Road construction	10%

Freight

The freight cost estimate was derived from first principles. Unit container freight rate costs were received from Antrak Logistics for five different international port origins to the port of Nhava Sheva in India. The container freight rate cost was inclusive of delivery fees, terminal handling charges, shipping charges, bills of lading, clearance fees, ocean freight, customs clearances, and destination handling charges.

A cost to transport each works package or supply contract was determined taking into account the quoted port of origin, and the number of container units required to transport the package.

An allowance for bulk break freight was added for any out of gauge items.

An allowance was made to unload and handle the package items on site at a final laydown yard on site ready for handover to the installation contractor. In country items were allocated a road freight allowance from within India to the project site.

14.3.5 Indirect cost development

EPCM services

An engineering, procurement and construction management (EPCM) implementation approach, whereby the EPCM engineer will provide design, procurement and construction management services on behalf of the owner according to the project schedule.

As directed by DGML, the estimate for EPCM services costs has been based on 10% of the total project direct costs.

The EPCM costs are low when benchmarking against previous CPC projects where the engineer is Australian or internationally based. However, the costs allowed appear to be adequate for engaging an Indian EPCM engineer.

Owners costs

As part of the construction stage, it is expected that DGML will provide a project management team.

DGML's construction team will interact closely with operations management personnel recruited during the construction phase of the project.

In addition to the above, the following allowances have been made in the estimate:

- Owners project expenses
- Pre-production costs
- First fills (grinding media, fuel and reagents)
- Office costs and expenses
- Mobile equipment and light vehicles
- Capital and maintenance spares
- Maintenance tools
- Vendor representative and training costs for the process plant
- Communication costs
- Greenbelt vegetation costs
- 10-person camp and messing facility in Haveri.

DGML provided the owners' cost estimate.

Land acquisition

The costs for land acquisition was provide by DGML based on advice and recommendations from the Karnataka Industrial Area Development Board (KIADB).

KIADB recommended a cost of 900,000 INR per acre, which includes costs for compensation, stamp duty and registration. Refer to Appendix 14F for details.

An acquisition of 255 acres has been estimated for the project.

Duties, taxes and insurances

<u>Government duties and taxes</u>

The capital cost estimate includes a 10% import duty on all imported equipment.

Indian sales and services taxes and withholding tax for the EPCM engineer's scope of work and vendor representatives are excluded from the estimate.

Project insurances and permits

DGML provided an allowance of 2.0% of total owner's costs for project insurances and permits.

Contingency

Contingency is an allowance additional to the base cost estimate to cover unforeseeable elements of cost, risk and uncertainty within the defined scope of work. This is an allowance to cover possible costs that cannot be explicitly foreseen or described at the time the estimate is prepared due to lack of complete, accurate and detailed information.

Contingency amounts included in the cost estimate are intended to cover:

- Variance in equipment and material prices
- Variance in labour rates and productivity
- Miscellaneous minor design changes including addition of small equipment and instruments, size changes in equipment, piping, wiring, and changes in pipe and cable tray routing.

Contingency amounts are not intended to cover:

- Schedule delays
- Changes in economic and environmental conditions
- Changes in execution approach
- Project capacity changes or the process flowsheet
- Changes to facility life expectancy
- Major changes in environmental regulations
- Force majeure events.

Contingency has been applied to the estimate on a line-by-line basis by assessing the level of confidence in each of the defining inputs to the item cost, these being engineering, estimate basis and vendor or contractor information, and then applying an appropriate weighting to each of the three inputs. 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. The resultant overall contingency for the project is 7.3%. A detailed breakdown of contingency costs can be found in Appendix 14C.

A summary of contingency percentage is provided in Table 14.29.

Table 14.29 Contingency percentage summary

Discipline	Percentage
Buildings	0%
Earthworks	15.0%
Concrete	10.0%
Mechanical equipment supply	10.0%
Piping supply	0%
Structural steelwork supply	0%
Platework supply	0%
Electrical and instrumentation supply	5.0%
SMP installation	15.0%
Electrical and instrumentation installation	15.0%
Power supply	15.0%
Freight	10.0%
Owners costs	5.0%
Management costs	10%
Project total	**7.3%**

Note: The project total contingency consists of additional minor components not included above.

There was no contingency allowance for buildings, piping, structural steelwork and platework supply as the rates used in the estimate were from international suppliers. It has been assumed that lower in country rates will be obtained.

As highlighted previously, the intention is to utilise Indian contractors for the above items and the higher rates used in the estimate is assumed to provide contingency for these construction commodities.

14.3.6 Deferred capital

No allowance for deferred capital has been allowed in the capital cost estimate.

14.3.7 Escalation

There is no allowance for project escalation in the capital cost estimate.

14.3.8 Qualifications and assumptions

The capital estimate is qualified by the following assumptions:

- The base date for the bulk of pricing for the capital cost estimate is the first quarter 2017 (1Q17).
- 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 utilising native currencies wherever possible.
- The bulk earthworks rates that include imported material are based on the assumption that suitable construction/fill materials will be available from borrow pits within 2 km of the work fronts.
- Stage 1 ROM pad earthworks construction costs (two ore stockpile fingers) for the startup phase have been included in the estimate.
- Engineering quantities for the TSF have been provided by Prime Resources.
- Rates for concrete and earthworks were provided by DGML. Contractor rates include indirect costs. An additional allowance for indirect costs was included in the capital cost estimate.
- Electrical and instrumentation installation was derived from first principles by applying contractor rates from the SMP contractor.

- With the exception of waters storage dam, there is no allowance for unforeseen blasting in the bulk earthworks cost estimates.
- Owner's mobile equipment that will be used for during construction and later for operations is included.
- The freight estimate allows for a majority of equipment and materials to be procured in India, however some items were sourced outside of India.
- 10% import duty on all imported equipment and material was allowed for in the capital cost estimate.
- Road construction costs in India was based on rates provided by DGML.
- EPCM cost estimate was based on 10% of total project direct costs as directed by DGML.

14.3.9 Exclusions

The following items are specifically excluded from the capital cost estimate:

- Project sunk costs
- Exchange rate variations
- Escalation
- Deferred capital
- Lubricant first fills
- Indian sales and services taxes
- Withholding tax for EPCM engineer and vendor representatives.

15 ECONOMIC ANALYSIS

All $ values in this section are US$.

15.1 Project economic headline results

Table 15.1 and Table 15.2 provide the Project headline results before and after taxation for a gold price of $1,250/oz of gold (base case).

Table 15.1 Economic model headline results before taxation

Item	Unit	Value at $1,250/oz Au
Net cash flow	$ M	133.0
NPV_5	$ M	91.6
IRR	%	39.1

Table 15.2 Economic model headline results after taxation

Item	Unit	Value at $1,250/oz Au
Net cash flow	$ M	93.1
NPV_5	$ M	61.4
IRR	%	29.6

15.2 General criteria

Snowden prepared an economic cash flow and financial analysis model based on inputs derived from mining and processing schedules, as well as capital and operating cost estimates, including royalties for the project. The model was prepared from construction and mining schedules estimated on a quarterly basis for project life. All inputs are consolidated annually in this report. The cash flow model was based on the following:

- 100% equity ownership
- Costing from January 2017
- 1.75-year production period for plant construction
- No cost escalation
- All costs reported in US$ and where costs were estimated in Indian Rupee (INR), the exchange rate used was INR66 to the US$.

The objectives of preparing the cash flow model was to:

- Collate all the inputs for the following disciplines into a single model:
  - Mining
  - Processing
  - Metallurgical
  - Metal pricing
  - Pre-production capital costs
  - Production sustaining capital
  - Operating costs
  - Rehabilitation and closure costs
  - Royalties
  - Taxation.

- Provide sufficient information to management so that they are supported in any decision-making process.
- Provide the basis for future studies.

The economic cash flow model was then interrogated to determine the following values after taxation:

- Headline values:
  - Net cash flow
  - NPV at 5% discount rate (NPV_5)
  - IRR
  - Breakeven gold grade
  - Breakeven (NPV_5) gold price
  - C1 cost per ounce of gold (Brooke Hunt Methodology)
  - Production year payback.
- Key performance indicators (KPIs):
  - Operating costs
  - Total costs
  - Production payback years.

15.3 Economic model inputs

Table 15.3 shows the inputs were used in the economic cash flow model.

Table 15.3 Economic model inputs

Item	Unit	Value
Pre-production	years	1.75
Life of process production	years	8.35
Project life	years	10.1
LOM ore mined	kt	2,506
LOM waste mined	kt	9,237
LOM total material mined	kt	11,743
Strip ratio w:o		3.68
LOM ore processed	kt	2,506
LOM average Au grade	%	3.38
LOM average Au recovery sulphide	%	79.0
LOM average Au recovery oxide	%	90.0
LOM average gold recovery	%	81.7
LOM contained ounces	koz	273
LOM recovered ounces	koz	221
Average annual gold produced	koz	27
Plant throughput (average)	Mt/a	0.30
LOM Au price	$/oz	1,250

15.4 Costs

A summary of total LOM costs is shown in Table 15.4 below. Note that no depreciation of capital was included in the taxation estimation.

Table 15.4 Total LOM costs

Item	Unit	Value
Pre-production capital	$ M	46.6
Production sustaining capital	$ M	3.1
Total capital costs	**$ M**	**49.7**
Total mining	$ M	21.6
Total processing	$ M	55.8
On-site labour	$ M	1.2
Total operating costs	**$ M**	**78.5**
Royalties	**$ M**	**14.9**
Taxation	**$ M**	**39.8**
TOTAL ALL COSTS	**$ M**	**183.0**

15.5 Production summary

The physical production of the project is based on the mining, processing and recovery of metal reported in Chapter 5.

15.6 Key performance indicators

The Project LOM KPIs after taxation are presented in Table 15.5 below.

Table 15.5 KPIs after taxation

Item	Unit	Value at $1,250/oz Au
Total value of product sold	$ M	276.1
Cash cost	$/oz	423
Total cost	$/oz	829
Production year payback	year	2.7
Brooke Hunt methodology C1 cost	$/oz	356
Brooke Hunt methodology C2 cost	$/oz	356
Brooke Hunt methodology C3 cost	$/oz	423

The cash costs include all direct operating costs plus royalties, the total costs include the cash costs plus capital costs and taxation. The Brooke Hunt methodology C1 costs include all direct operating expenses but do not include royalties, C2 is C1 plus depreciation and C3 is C2 plus royalties.

15.7 Sensitivity analysis

The economic cash flow model was used to prepare a sensitivity analysis for the NPV_5 for the Project after taxation. The sensitivity analysis was completed on the following variables:

- Grade of Au
- Recovery of Au
- Price of Au
- Pre-production capital
- Production capital

- Mining cost
- Processing, including on-site labour cost.

The sensitivity analysis determines how the NPV_5 is affected with changes to one variable at a time while holding the other variables constant. The results of the sensitivity analysis are presented in Table 15.6. In this table, "B/E" represents the breakeven and it indicates the change in the variable that will bring the project NPV_5 to $0.0. Elasticity is a measure of sensitivity that indicates for a 1% change in the variable what change in the NPV_5 will occur. A value greater than 1 indicates that the change in the variable will induce a higher value change in the NPV_5 than the change in the variable and indicates a higher sensitivity to change.

Table 15.6 Sensitivity table for the base case NPV_5 – after taxation

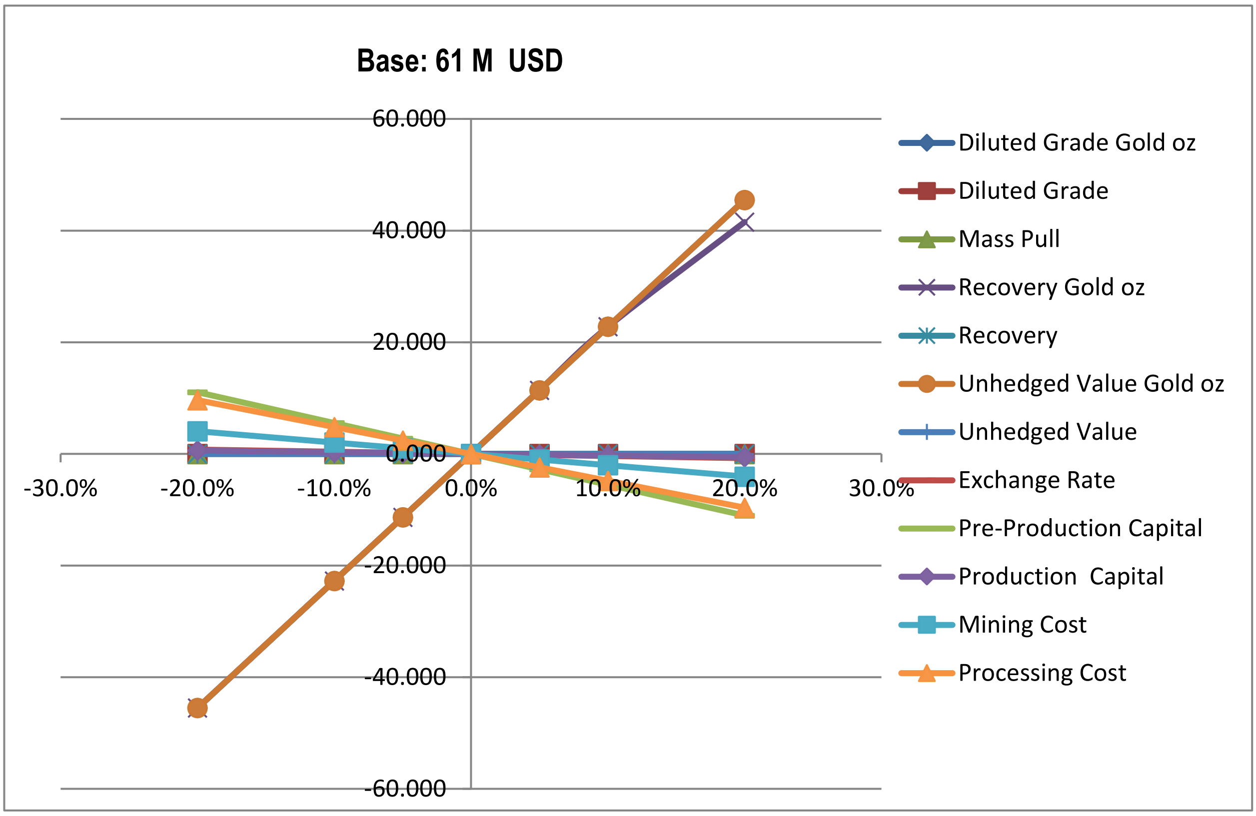
	-20%	-10%	-5%	0%	5%	10%	20%	B/E	Elasticity
Grade Au	33	47	54	**61**	68	75	89	**-44%**	2.3
Recovery Au	33	47	54	**61**	68	75	87	**-44%**	2.3
Price Au	33	47	54	**61**	68	75	89	**-44%**	2.3
Pre-production capital	68	65	63	**61**	60	58	55	-	0.6
Production capital	62	62	62	**61**	61	61	61	-	-
Mining cost	64	63	62	**61**	61	60	59	-	0.2
Processing cost	67	64	63	**61**	60	58	56	-	0.5

The B/E indicates the breakeven change for the variable if all other variables remain unchanged. For example, if the grade of Au grade reduces by 44% the Project will break even on NPV_5.

The Elasticity indicates the change in the Project NPV_5 that is induced by a 1% change of the variable if all other variables remain unchanged. For example, if the grade of Au changes positively or negatively by 1% it will induce a 2.3% change of the NPV_5. A change of greater than 1.0 indicates that the project is sensitive to a change of the variable as the NPV_5 will change by a greater amount than the change of the variable.

The sensitivity chart (Figure 15.1) covers a range of variable changes from -20% to +20%. As the variable line increases towards the vertical it indicates the project is more sensitive to changes of the variable.

Figure 15.1 Sensitivity graph – Ganajur Gold Mine (sensitivity range -20%/+20%)



15.8 Breakeven analysis

A breakeven analysis after taxation was undertaken on the gold price and gold grade for NPV_5. This analysis is conducted on the sensitivity analysis data and provides the gold price which will bring either the NPV_5 to $0.0. The results of this analysis are presented in Table 15.7.

Table 15.7 Breakeven analysis after taxation

Item	Unit	Breakeven
Gold price	$/oz Au	701
Gold grade	g/t Au	1.90

15.9 Conditional simulation

A conventional cash flow model provides a single point analysis based on the values of variables that have been used. A simple cash flow model will use the same variables for the life of the mine. A more advanced cash flow model will schedule the variables on a period basis. A sensitivity analysis is undertaken to determine the influence that a change in any variable will have on the project while holding all the other variables constant. The scenario analysis is a more advanced sensitivity analysis where each of the variables may be changed at the same time by different sensitivity change conditions and the results are updated in real time. This is a comprehensive form of a "What if" analysis and is used by the client or banks to stress test a project to see what changes to various variables are required to "break" the project. Whether the cashflow model is a simple form or scheduled, the values applied are most unlikely to occur in a real-life operational project.

The conditional simulation is applied to simulate all the possible ranges, within reasonable limits, that the variables may be within for each of the annual periods for the life of the project. To set the model up, risk limits (positive and negative) to the anticipated input variables are set. These may be skewed and they can have different ranges for each variable. A random number generator then selects a value within the range for each variable and for each period separately and applies that value to the period in the cash flow model. For each iteration, the conditional simulation will generate a large number of new variable values across the whole LOM (all variables to which a range has been applied) and will provide a single output result, e.g. the NPV, Cashflow or IRR for that set of data. This process is repeated a large number of times, generally between 100 and 10,000 times, and all of the output values are collected. The results for the conditional simulation for NPV_5 are provided in Table 15.8.

Table 15.8 Conditional simulation variable ranges

Variable	Unit	Risk limits		Value ranges		
		+ve%	-ve%	High	Base	Low
Grade Au	g/t	2	3	3.45	3.38	3.28
Recovery Au	%	1	3	82.5	81.7	79.3
Price Au	$/oz	5	10	1,313	1,250	1,125
Costs						
Pre-production	$M	25	5	58.3	46.6	44.3
Production	$M	25	5	3.9	3.1	2.9
Mining	$/t	10	5	9.46	8.60	8.17
Processing	$/t	10	5	24.49	22.27	21.15
On-site labour	$/t	10	5	0.51	0.47	0.44

15.9.1 Statistical analysis

A statistical analysis is done on the collected conditional simulation values to determine a real-life Cashflow, NPV or IRR depending on the metric applied. The conditional simulation is generally done on NPV. In addition, because there are a large number of values collected, a probability analysis can also be applied which will give the range of possible outcomes for the Project at different confidence levels.

A statistical analysis for a sample of a population is valid for a sample greater than 80 values. The conditional simulation was run for 1,000 iterations and the results for the statistical analysis are provided in Table 15.9.

Table 15.9 Statistical and probability analysis for the Ganajur Gold Project

Item	Unit	Value	Value
Single point NPV_5	$M		61.4
Conditional simulation (10,000) mean NPV_5	$M		51.3
Standard error	$M		0.106
Standard deviation	$M		3.3
Minimum value	$M		39.5
Maximum value	$M		62.2
Risk index (coefficient of variation)	%		7%
Range at 99.7% confidence	$M	41.4	61.1
Range at 95.0% confidence	$M	44.7	57.8
Probability of > value	$M	45.0	97%
Probability of > value	$M	49.0	75%
Probability of > value	$M	51.3	50%
Probability of > value	$M	53.0	30%
Probability of > value	$M	58.0	2%

An analysis of the statistical data indicates that the real life NPV_5 will be in the order of $51 million; this is realistic as the history of projects is that the variables have been optimistically assessed and there have been a number of costs that are higher than originally anticipated and they have not been covered adequately by contingencies.

15.10 Conclusion

In general, the Ganajur Gold Project, if it is managed to the study inputs and outcomes, is a robust project. This is demonstrated by the moderately low Risk Index, the ranges of NPV values at the two confidence levels are close, and the probability values are also tightly grouped.

16 PROJECT IMPLEMENTATION

16.1 Introduction

This section outlines the implementation philosophy and project execution strategy for the delivery of the Ganajur Main Project including engineering, procurement construction and handover to Deccan Gold Mines Limited (DGML).

The Project scope includes the delivery of the process plant, TSF, water storage dam, river extraction facility and pipeline for freshwater, infrastructure and support services, for a gold plant capable of treating 300,000 tonnes per year (t/y) of gold bearing ore over the current 10-year mine life.

The Project Implementation Plan will provide a cost-effective solution with the shortest practical completion period, without comprising on safety, quality and schedule.

The design and implementation of the Project will conform to Australian and/or Indian statutory laws and regulations.

16.2 Project location and access

The Project is located in the northern-western part of the Karnataka state, approximately 350 km from Bengaluru.

The Project site is well connected by road and rail and has reasonably established infrastructure in place. Haveri town, Ganajur and Karajgi villages are located in close proximity of the Project. Karajgi and Haveri are the nearest railway stations, all within 5 km of the Project site, on the Bengaluru-Mumbai broad gauge railway.

The nearest seaports are Mormugao in Goa and Jawaharlal Nehru Port (Nhava Sheva) located east of Mumbai. Mormugao is approximately 260 km by road from the Project site, while Nhava Sheva is 640 km. The study recommends that both ports be considered during the project execution phase as the point of entry of overseas cargo.

16.3 Project execution model

The recommended development methodology for the design and construction of the Project is engineering, procurement and construction management (EPCM). This approach allows DGML to monitor and control the budget, schedule and quality through all stages of project development and execution.

It is intended that the procurement of all equipment and bulk materials will be done by the EPCM engineer and will be free-issued to the construction contractors for installation. This will ensure control over the critical procurement activities to achieve the desired completion schedule and ensure control of quality that meet DGML’s requirements.

Other development approaches such as EPC, lump sum and turnkey, carry additional project risks that will require further assessment of the selected design engineer’s ability to deliver comprehensive, cost-effective projects on schedule with a high quality of work.

The Project capital cost estimate has been developed on the basis that a single organisation will provide the EPCM services necessary for the process plant and associated infrastructure and services, with the assistance of specialist sub-consultants as required.

16.4 Project objectives

The strategic objectives for the Project are to:

- Deliver the Project with zero lost time and medical treatment injuries
- Zero major environmental incidents
- 100% compliance with all approvals
- Positive community relations
- Low impact on surrounding communities
- Implementation and delivery of an operational process plant which achieves the availability, reliability and metallurgical performance given in the process design criteria
- Low cost, fast track, high quality implementation of the process plant and associated infrastructure
- Utilise Indian manufactured equipment and materials where practically possible and cost effective.

16.5 EPCM scope of services

The engineer will provide EPCM services associated with the development of the process plant and associated infrastructure and services including the following:

- Process engineering
- Design engineering and drafting for earthworks, concrete, structural, mechanical, piping, electrical and instrumentation
- Project services including cost control, scheduling, reporting, claims processing
- Procurement including purchasing, inspection of materials and equipment and expediting
- Contract administration including tendering, awarding and management of major contracts
- Logistics (transportation) coordination
- Construction management including site management, control and inspection of all construction activities, quality and safety management
- Commissioning.

Drawings showing the process plant and surrounding site infrastructure described below are provided in Appendix 16C.

Specialist consultants will be engaged by DGML to provide the following services:

- EPCM services associated with the TSF
- Surveying services during construction
- Vendor services for construction and commissioning
- Mining consulting
- Training and business readiness support.

16.5.1 List of services and deliverables

The engineer will prepare and issue the deliverables detailed in Table 16.1. The deliverables list will be updated throughout the Project as necessary.

Table 16.1 List of service deliverables

Description	Included/ Excluded	Comments
Engineering		
Design		
Process	✓	
Process design criteria	✓	
Mass and energy balance	✓	
Treatment plant water balance	✓	
Overall site water balance	✓	By DGML consultant
Civil including main site access road and internal plant area roads	✓	
Structural	✓	
Geotechnical assessments	✓	
Mechanical	✓	
Electrical and distribution	✓	
Piping	✓	
Architectural	✓	
Control system design	✓	
Process control philosophy	✓	
Control system programming	✓	By Vendor
Equipment specifications	✓	And data sheets as required
Independent design audits, if required	×	
Design reviews	✓	
HAZOP	✓	Facilitated by others outside the core design team
Drafting		
Process flow diagrams	✓	
General arrangement	✓	Including site layout, 3D plant elevations
Civil	✓	
Structural	✓	
Structural shop detailing	×	By Fabricator
Platework	✓	
Piping P&IDs	✓	
Piping layouts	✓	
Piping isometrics	×	
Electrical equipment layout	✓	
Electrical single line diagrams	✓	
Electrical schematics	✓	
Electrical loop diagrams	✓	
Electrical termination diagrams	✓	
As built drawings		
Civil	×	
Structural	×	
Mechanical	×	
Electrical and instrumentation	✓	
Piping	✓	P&IDs and buried services only
Architectural	×	
Control system programming	✓	

Description	Included/ Excluded	Comments
Lists and schedules		
Mechanical equipment list	✓	
Electrical equipment list	✓	
Electrical load list	✓	
Instrument list	✓	
Valve list	✓	
Lubrication schedule	✓	
Piping line lists	✓	
Concrete reinforcement scheduling	×	By Civil Contractor
Cable schedule	✓	
Spares schedule	✓	Procurement of spares by DGML
Construction specification		
Civil	✓	
Structural	✓	
Mechanical	✓	
Electrical	✓	
Instrumentation	✓	
Piping	✓	
Architectural	✓	
Control system	✓	
Vendor data reviews		
Mechanical	✓	
Electrical	✓	
Instrumentation	✓	
Control system	✓	
Equipment, operating and maintenance manuals	✓	Preparation of manuals based on documentation supplied by Vendors
Project management		
Scheduling/Planning	✓	
Cost control		
Systems management	✓	
Data input	✓	
Invoice approval	✓	
Invoice payment	×	By DGML
Estimating	×	
Procurement		
Tender document preparation	✓	
Tender issue	✓	
Tender adjudication recommendation	✓	
Purchase order award	✓	
Purchase order administration	✓	
Change order award and documentation	✓	
Spare parts orders	✓	Assist DGML with development of Spares List
Inspection		
ITP follow-up	✓	
Shop visits	✓	

Description	Included/ Excluded	Comments
Expediting		
Phone follow-up	✓	
Vendor data follow-up	✓	
Shop visits	✓	
Logistics		
Transport logistics and admin management	✓	
Transport to site or consolidation yard	✓	By Transport Contractors
Shipping documentation	✓	By Freight Forwarder
Customs clearance	✓	By Freight Forwarder
Project management		
Monthly reporting	✓	
Weekly coordination meetings	✓	
Project execution plan	✓	
Contract administration	✓	
QA plan and management	✓	
HSEC plan and management	✓	
Commissioning plan	✓	
Construction management		
Site management	✓	
Site supervision and inspection	✓	
Site contract administration	✓	
HSEC documentation	✓	
Inductions (construction)	✓	
QAQC documentation	✓	
Materials receipt	✓	
Commissioning		
Pre-operations testing	✓	
Wet commissioning	✓	
Process commissioning	✓	Support role to DGML

16.6 Project implementation stages and schedule

This project implementation strategy provides the overall methods of managing the Project from the detail design, procurement and construction through to commissioning. To meet the schedule proposed, the project implementation is structured into four stages:

- FEED engineering
- Detail design
- Construction
- Commissioning and handover.

The milestone dates for the development of the project are:

- May 2017 – Commence FEED engineering
- July 2017 – DGML approval for the Project
- September 2017 – Award of EPCM contract
- December 2017 – Mobilisation of mining contractor

- January 2018 – Site works earthworks
- May 2018 – Water storage dam and river extraction facility and pipeline completed
- October 2018 – Project completion and ore commissioning.

16.6.1 FEED engineering

At the completion of the study phase, and prior to Project approval, the study engineer will undertake FEED engineering and procurement activities to tender the long lead and critical equipment including:

- Ball mill
- Regrind mill
- Crushing plant
- Flotation cells
- Thickeners
- Steel-framed buildings
- Transformers
- HV switchgear.

Award of the ball mill, regrind mill and steel framed buildings will need to be made prior to project approval to meet the schedule.

The steel-framed buildings are not a long lead item; however, they are critical to the implementation schedule as the intent is to use the workshop/warehouse, administration building, lunch room and ablution block during construction.

The remaining long lead and critical equipment will not be awarded prior to project approval; however, orders will be placed for vendor data, at a minimal cost, to allow the finalisation of the process plant layout and associated design.

Full detail design of the water storage dam will be completed during this stage.

Indian construction and fabrication contractors will be identified during this period and shortlisted to streamline the award of contracts.

16.6.2 Detail design

Following Project approval, an EPCM contractor will be engaged and based on the agreed flowsheets and overall layouts developed in the FEED engineering stage and detailed design of the facilities will proceed including the award of the tendered long lead and critical equipment.

During this stage, the engineer will:

- Tender, evaluate and recommend the procurement of equipment and bulk materials
- Progress detail engineering of the process plant and support infrastructure
- Tender, evaluate and award the contract for the structural steel and platework fabrication packages for the process plant
- Tender, evaluate and award the contract for the process plant bulk earthworks and water storage dam construction
- Prepare tenders for site contract packages including concrete works, structural, mechanical and piping (SMP) installation, and electrical and instrumentation (E&I) installation.

16.6.3 Construction

If site bulk earthworks are commenced early January 2018, it is anticipated construction completion will be late third quarter 2018 (3Q2018).

During this stage, the engineer will:

- Complete detailed engineering of the process plant
- Complete procurement of equipment and bulk materials
- Tender, award and manage lump sum or fixed price schedule of rates contracts for construction of the process plant and remaining support infrastructure
- Complete construction of the treatment plant.

16.6.4 Commissioning and handover

During this stage, the engineer will manage and coordinate commissioning of the process plant and support infrastructure. Ore commissioning is expected to commence in September 2018.

16.6.5 Project schedule

The project schedule shown in Appendix 16A is based on the following:

- Off-site: 40-hour week, no work on public holidays, between Christmas and New Year and the first week of January.
- On-site: The engineer and construction contractors will work 13 days per fortnight, 10 hours per day, with no site activities between Christmas and New Year.

Critical activities

Following the necessary approvals, critical or near critical activities include the following:

- Delivery of ball mill
- Earthworks design
- Construction of water storage dam
- Concrete design
- Structural steel supply
- SMP and E&I installation.

The project master schedule will be expanded during the project implementation phase to the next level of detail with the inputs from contractors and suppliers.

All activities that are measurable will be incorporated in the schedule and weighted to define the project "S" curve. This schedule and "S" curve will be the project baseline. Procedures for measuring and weighting actual progress for the various project activities will be determined. These procedures will then be implemented and used throughout the term of the project to provide an accurate measure of the actual progress.

Input will be provided both from the engineering office, vendors and construction team. It is expected that the schedule will be updated at least weekly during the engineering phase and on a fortnightly basis during construction.

There will be no change to the baseline without DGML's approval. If the project is demonstrated to be failing to achieve the baseline forecast by the progress measurement, then appropriate corrective action will be taken to recover the schedule.

Key performance indicators

The engineer will develop a set of KPIs expressed in a range, against which the project will be measured. These include: safety, capital cost, schedule and quality. The project will report against these, and immediately highlight any deviation that will put any KPI at risk.

Detailed work or task schedules will be developed as required to determine progress and hence input into the project schedule including the following:

- Design deliverables (drawings, specifications, etc.)
- Procurement deliverables (orders placed, contracts)
- Fabrication (tonnes of steel/platework)
- Construction (cubic meters of concrete poured tonne of steel installed, number of pieces of equipment installed, meters of pipework installed, meters of cable installed, etc.).

16.7 Work breakdown structure

The project work breakdown structure (WBS) is shown in Appendix 16B and outlines the project in terms of activity levels that can be clearly defined, managed and controlled. The WBS will represent the total scope of the project work, with no deficiencies or duplication, and the way in which the work is to be performed, in a uniform, consistent and logical manner.

The WBS will define the method by which the project will be subdivided into smaller, manageable components for estimating, budgeting, cost control, planning and scheduling, progress measurement and earned value management functions. The WBS will provide a framework of common reference for all project elements and will be the fundamental linkage between cost and schedule information on the project.

The project WBS is a hierarchical coded control structure represented by numerical digits and derived from the project scope of work.

In order to achieve uniformity, to enable meaningful project monitoring and reporting and to facilitate communication among project team members, the same basic structure will be used for all facets of the project. This includes definition of work, cost estimates, budget allocation, cost monitoring and control, change management, time estimates, planning and scheduling, resource allocation, productivity, progress measurement and earned value management. The WBS will also facilitate efficient electronic migration of data between cost estimates, cost control budgets, project cost forecasts, project schedules and progress measurement, and the corresponding management reports developed from this data.

16.8 Project organisation and responsibilities

The project management organisation chart in Figure 16.1 shows the overall reporting structure of the project during the implementation phase.

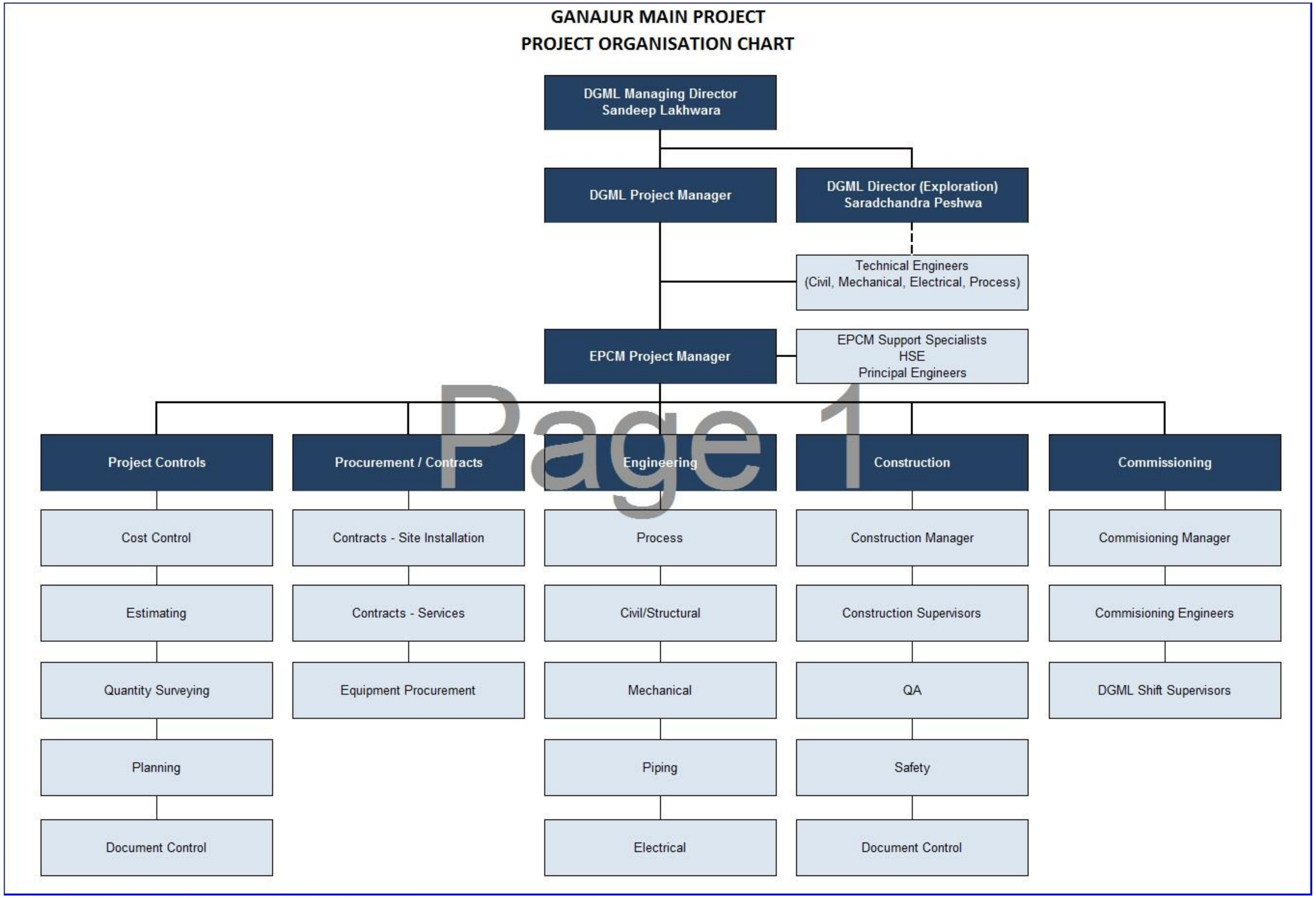
Working with DGML's project team, the engineer will undertake the design, procure equipment and services, manage the project and supervise the construction and commissioning of the process plant and associated infrastructure.

The DGML project manager, reporting the DGML's managing director will oversee the management of the project with the support of DGML director (exploration) and his technical engineers, to ensure DGML's requirements are met for schedule, costs and quality.

The engineer's project management team will be led by the EPCM project manager who will be responsible for the overall management of the project for and on behalf of the engineer to ensure that EPCM services meet the project objectives.

The EPCM project manager is responsible for regularly informing DGML's project team of the detailed status of the project and managing the interface between the engineer's various groups ensuring that all parties are communicating and aware of the short, medium and long term targets and objectives of the project, and that all are working to the pre-established project plans and procedures to successfully deliver the project.

Figure 16.1 Project organisation chart



16.9 Engineering and design

The Project FS serves as the starting reference for design by providing project background and initial design documentation.

Key documents contained within the feasibility study include:

- Process flowsheets
- Process design criteria
- Process plant layouts
- Facilities general arrangements
- Equipment lists.

The project will comply with the following design criteria (as referenced in Appendix 16D):

- 7056-DC-R-001 Process Design Criteria
- 7056-DC-B-001 Earthworks and Drainage Design Criteria
- 7056-DC-C-001 Civil/ Structural Design Criteria
- 7056-DC-M-001 Mechanical Design Criteria
- 7056-DC-P-001 Piping Design Criteria
- 7056-DC-E-001 Electrical Design Criteria
- 7056-DC-J-001 Control System and Instrumentation Design Criteria.

Key drivers for the engineering design include:

- A robust process plant that optimises gold recovery with minimum operating costs
- A compact plant layout that minimises capital costs whilst still providing ease of access to all equipment for operation and maintenance and allowance for future upgrading
- Equipment selection based on suitability for duty, reliability and ease of maintenance.

16.9.1 Design sub-consultants

It is envisaged that the following sub-consultants will be required for the project:

- Tailings dam design (engaged by DGML, managed by the engineer)
- Mining (engaged and managed by DGML).

16.9.2 Design deliverable approvals and design reviews

Key design and drafting documents will require formal approval by DGML.

Design reviews will be conducted in conjunction with DGML's project management team, as and when agreed.

A hazards and operability assessment (HAZOP) will be performed on the design when the control philosophy and process and instrumentation diagrams (P&IDs) are sufficiently developed.

16.10 Procurement and contracts

This section addresses the major procurement and contracting activities and details the strategies, methodology, procedures and controls that will be adopted during the delivery of the project.

16.10.1 Philosophy and approach

Equipment supply will be through a combination of competitively tendered and sole sourced packages taking into account suppliers' past performance.

Contractors for site works will be selected based on their safety record, their industrial relations record, previous experience with similar projects, cost, schedule, availability and capability to perform the work.

Local contractors and suppliers will be encouraged to tender for specific project works and contracts may be awarded based on their ability to meet the required conditions. To encourage local sourcing of project requirements for specific contract packages, it is planned that direct negotiations will be undertaken with smaller local business groups and agreed with DGML.

Construction contracts, will be tendered as horizontal packages which offer the following advantages:

- Provide greater flexibility to engage the expertise of contractors as required during the project.
- A horizontal contract allows the engineer to have direct control of the contractor undertaking the specific scope of work.
- Contractors normally maintain strength in one or two major disciplines. In a vertically packaged scenario, the work in those disciplines for which the principal contractor does not possess in-house expertise must be to sublet to other groups over which the principal contractor may not have effective control and which may incur additional cost.
- The horizontally packaged contractor may have a shorter duration on site which may provide time and cost benefits as well as health, safety and environment (HS&E) benefits.
- A horizontal contract may assist the engineer with greater control of specific areas of cost and schedule.
- A horizontal contract is not dependent upon identifying available contractors, large and experienced enough, to undertake vertical packages.

16.10.2 Types of contracts

Contracts will be based on standard terms and conditions prepared by the engineer for DGML, with approved specific templates for each of the following contract types.

Lump sum

In a lump sum contract the contractor is compensated based on a fixed, all-inclusive lump sum amount for performance of a defined scope of work.

Payment terms under lump sum contracts will be against a series of milestones (based on completion of specific deliverables). Milestone deliverables will be clearly defined and agreed prior to award of a contract. Milestone payments provide a good incentive to contractors to achieve schedule targets in a specific month.

Schedule of rates

In a schedule of rates with estimated quantities contract, the contractor submits a tender based on the estimated quantities, but is compensated against actual quantities being measured and validated by the engineer on a monthly progressive basis.

Source selection

In source selection, tenderers will be agreed through a consultation process between the engineer and DGML. For packages of high value, risk or complexity, the engineer will pre-qualify and recommend tenderers to DGML for approval. All other tenderers will be approached to determine their willingness and suitability to tender and be approved by DGML.

The purpose of pre-qualification is to determine a tenderer's willingness and suitability to tender. Tenders not being sole sourced will only be sent to companies with a realistic chance of success. Proposed pre-qualification lists will be approved by DGML.

Sole source

For those packages where it is determined only one company is to be requested to tender, whether for technical, commercial or any other reason, the sole source is to be approved by DGML.

A listing of the proposed contract packages and contract types is provided in Table 16.2.

Table 16.2 Contract packages

Contract description	Quantity	Contract type
EPCM Services	1	Schedule of rates
Tailings Dam Design Consultant	1	Schedule of rates
Purchase Orders	>50	Lump sum
Structural Fabrication	1	Schedule of rates
Platework Fabrication	1	Schedule of rates
Transport and Logistics	1	Schedule of rates
Fencing	1	Lump sum
Buildings Installation	1	Lump sum
Process Plant Earthworks and Water Dam Construction	1	Lump sum
Tailings Dam Construction		Lump sum
Concrete Works	1	Schedule of rates
Structural, Mechanical and Piping Installation	1	Schedule of rates
Electrical Installation	1	Lump sum

16.11 Project control

An accurate project controls system is critical to manage the project budget and schedule. The principal function of the project controls system will be the monitoring and reporting of actual progress and cost against the baselines established at the commencement of project implementation phase. Secondary functions of control will provide progress reports, establish quality standards and maintain a comprehensive document control system. The project management plan will include the basis for the project control and reporting systems.

Following the completion of the FS and confirmation of the project capital cost estimate, the estimate will become the project control budget. The budget for various scopes of works and contractors will be entered in the database cost control system utilising a coding structure that identifies all equipment, materials and discipline activities.

Throughout the course of the project all commitments and expenditure will be entered in the respective cost control systems. Forecasts to complete and variances will be updated monthly and contingency drawdown will be managed only with the approval of DGML.

Efficient and accurate cost control will be critical to ensure the project is completed within budget.

16.12 Reporting

During the implementation of the project there will be the need for ongoing communication of project status to DGML. Key matters to be reported will include:

- Occupational, health and safety
- Environmental and community
- Scheduling and progress in all areas across the project (measurement and narrative)
- Procurement and contract status
- Costs
- Quality
- Project issues and concerns.

The principal reporting forums will be:

- Fortnightly progress meetings
- Fortnightly written report comprising progress narrative
- Formal monthly report including narrative and updated project controls reports and forecasts.

16.13 Quality

It is a fundamental requirement that all aspects of the project are executed in accordance with specified technical requirements, be they performance, compliance with technical specification or manufacturing and fabrication standards. A quality management plan will be developed to define quality objectives, identify all standards applicable to the project and practices, such as design reviews, vendor inspections and audits etc., to be performed in order to ensure the quality objectives are met.

16.14 Construction

16.14.1 Construction management

Project construction will be carried out by suitably experienced contractors which will be selected by the EPCM contractor on the basis of proven experience, capability and a record on harmonious industrial relations.

The team will have the following key responsibilities:

- Occupational, health and safety
- Contract management
- Verification of contractors' quality compliance
- Interface management
- Progress monitoring and reporting
- Contractor progress claim certification
- Materials management.

16.14.2 Construction sequence

The early construction activities will be mainly focused on processing and associated infrastructure.

In addition, temporary facilities are required to be established prior to any significant construction work commencing, among others these are:

- Site construction offices and facilities
- Communications.

16.14.3 Construction equipment

Construction contractors will provide their own equipment, tools, construction power, fuel, supplies, consumables and facilities required to complete their work in accordance with their contract scope of work. Contractors will also provide plant equipment and materials, other than that free issued by DGML which will be defined under their scope of work.

16.14.4 Quality control/quality assurance

Contractors will furnish their own QC program in accordance with contractual obligations. This will include compaction, concrete strength, weld and electrical installation testing.

16.14.5 Construction personnel

Contractors will be responsible for provision of all supervisory and trades personnel. It is anticipated personnel will generally originate from the Karnataka region.

16.14.6 Potable and construction water

Water for earthworks compaction, road watering, concrete and other non-potable uses will be supplied by the contractors. Water for potable use will also be sourced by the contractors.

Bore water is available nearby the site. During the project execution phase, an investigation into whether bore water can be used for construction will be completed.

16.14.7 Concrete

Concrete will be batched on site by the concrete contractor for all project requirements.

16.14.8 Construction accommodation

No construction accommodation will be provided. The contractors will responsible to provide their own accommodation in the local area.

16.14.9 Communications

Two-way, multi-channel UHF handheld radios with one base station will be used at site.

16.14.10 Temporary power

Contractors will be responsible to provide their own construction power.

16.15 Commissioning

Commissioning will occur progressively and carried out in a number of phases, following the traditional approach adopted for projects of this nature and size. These phases are:

- Construction verification
- Dry commissioning
- Wet/ore commissioning.

Practical completion is defined as the point in time at which construction works have been completed, except for minor defects, testing and the contractor has provided appropriate certifications and documentation.

Once all the plant facilities have achieved practical completion then wet/ore commissioning will commence where ore, slurry and reagents are introduced and at that point, control of the facility operation will be transferred to DGML's operations team.

Commencement of testing and commissioning activities will be subject to the availability of power and raw water.

It is envisaged that the project facilities will be commissioned progressively. A commissioning plan will be developed describing the plan for starting up the project process facilities and defines the various stages of commissioning and will identify responsibilities for activities conducted during those stages. The plan will include a preliminary schedule, personnel structure and will outline the methodology to be followed and forms a reference document to assist with project scheduling, as well as preparation for commissioning.

16.15.1 First fill

There are several first fill products required for project start-up. The significant items in terms of quantity and logistics are as follows:

- Grinding media
- Oils and grease
- Reagents.

The first fill requirements will be determined during the engineering phase and procured, stored at the facility ready for commissioning.

16.15.2 Spares

During the project development phase, sufficient quantities of spares will be procured to cover commissioning, one year's operation and insurance requirements. The spares purchased will be based upon vendor recommendations, past operating experience and lead times for delivery of critical items.

16.15.3 Construction verification

Each process system or ancillary facility will be checked for compliance with the relevant drawings, piping and instrumentation diagrams, specifications, vendor data and lubrication charts.

Mechanical equipment will be checked for proper installation, alignment and rotation. Tanks and piping will be water/air tested.

Electrical equipment and circuits will be checked for proper installation. Instrumentation circuits will be checked and instruments will be zero-calibrated.

Equipment vendor representatives will be presented to verify and sign-off respective installation of the supplied equipment.

16.15.4 Dry commissioning

Dry commissioning will comprise energising of all electrical equipment and circuits, no-load running of drives (coupled and uncoupled), conveyor alignment, checking of control systems, safety devices, interlocks, instrumentation, control loops and valve actuation of all equipment. It will include, where possible, the circulation of water through systems and testing of control sequences, and operating the equipment grouped together into systems or modules, but without ore or reagents or other process material.

It includes:

- Extended (typically 4 hours) unloaded “dry” runs of equipment, such as conveyors
- Filling tanks and vessels with water
- Pumping water along liquid and slurry pipelines
- Simulation of operating conditions, using water and air, where practical
- Testing of all instruments, systems and equipment
- Ensuring plant operates in accordance with the control philosophy
- Taking and recording measurements required by the appropriate check sheets
- Surveillance for abnormal conditions.

The EPCM commissioning team will be responsible for conducting dry commissioning with the assistance of DGML’s operations team where applicable. This phase forms an important part of plant familiarisation and training for the plant operators.

The EPCM contractor will appoint a commissioning manager which will be responsible for the coordination of dry commissioning activities with the various contractors.

Equipment vendor representatives will be presented to verify and sign-off respective tests of the supplied equipment.

16.15.5 Wet/Ore commissioning

Wet commissioning will start when ore, reagents, grinding media and other process requirements are introduced to the process facilities and it is operated as a whole. The operation of the facilities will be performed by DGML’s operations personnel with the assistance of the contractors and equipment vendors specialised personnel which will sign-off on performance of the equipment. The contractors will have technical personnel and construction trades’ personnel available for assistance as required and also monitor the progress of the commissioning. Following the introduction of ore, the plant will be progressively ramped up to full production. During this time, the operations personnel will receive extensive training in all aspects of the plant’s operation.

17 RISKS AND OPPORTUNITIES

17.1 Project risk and opportunity assessment

A risk and opportunity assessment for the Ganajur Gold Project was performed via a one-day workshop held on March 2017 at CPC's office in Perth, Western Australia. The risk assessment was attended by representatives from Snowden, CPC and DESPL. Prime Resources input was via a conference call link (Table 17.1).

Table 17.1 Participants in the Ganajur Gold Project FS risk assessment workshop held on 17 March 2017

Participant	Company
Tom Back (Facilitator)	CPC
Dimitrios Felekis	CPC
Drew Noble	CPC
Jacqueline London	CPC
John Fodor	DESPL
Sandeep Lakhwara	DESPL
SCR Peshwa	DESPL
Frank Blanchfield	Snowden
Peter Theron (dial in)	Prime Resources

The current risks and opportunities identified were ranked on the following basis (Figure 17.1).

Figure 17.1 Risk ranking methodology

	Likelihood	Consequence 1 Insignificant	 2 Minor	 3 Moderate	 4 Major	 5 Catastrophic
5	5 Certain	11	16	20	23	25
4	4 Likely	7	12	17	21	24
3	3 Possible	4	8	13	18	22
2	2 Unlikely	2	5	9	14	19
1	1 Rare	1	3	6	10	15

Risk Level	Priority	Actions to Minimize Risk	Actions to Maximize Opportunity
Extreme	1	Detailed research and planning required; determine whether activity or task should be stopped pending further investigation	Detailed research and planning required; high payoff potential; pursue opportunity aggressively
High	2	Senior management attention; immediate corrective and preventative action required	Near term opportunity with above average rate of return; pursue diligently
Medium	3	Conditionaly acceptable risk - management responsibility assigned; corrective and preventative action plan developed	Opportunity to realise average rate of return with certainty pusue with existing plans
Low	4	Manage by routine procedures; accept risk	Manage by routine procedures

17.2 Current risk summary

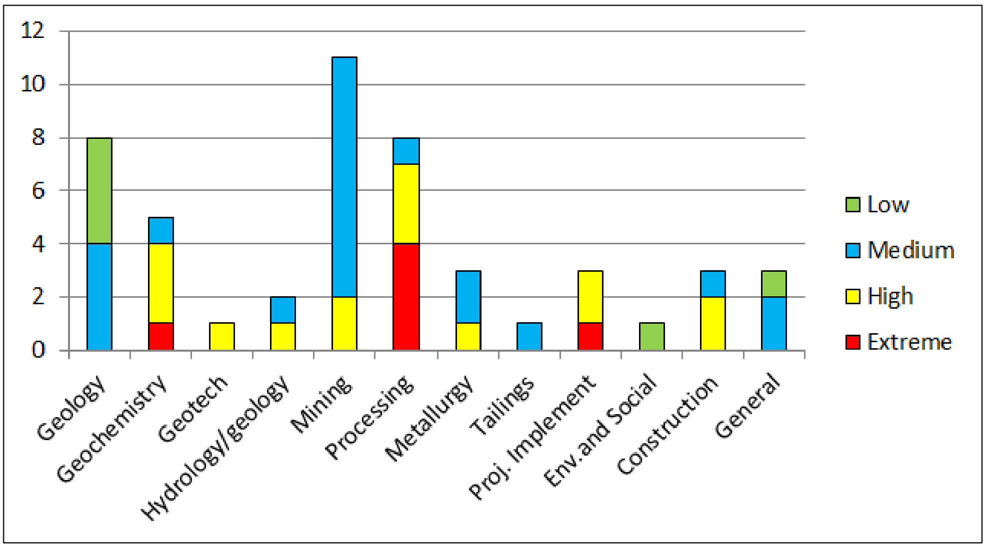
Forty-nine (49) Current Risks were identified during the risk workshop. The identified risks were divided into relevant project categories with the rankings split between Extreme, High, Medium and Low (Table 17.2). Note that these risks have been ranked on the basis of the current state of the project prior to any control or mitigation measures being identified.

Table 17.2 Identified current risks split between project categories and risk levels

Category	Extreme	High	Medium	Low	Total
Geology			4	4	**8**
Geochemistry	1	3	1		**5**
Geotechnical		1			**1**
Hydrology and hydrogeology		1	1		**2**
Mining		2	9		**11**
Processing	4	3	1		**8**
Metallurgy		1	2		**3**
Tailings			1		**1**
Project implementation	1	2			**3**
Environmental and social				1	**1**
Construction		2	1		**3**
General			2	1	**3**
TOTAL	**7**	**14**	**22**	**6**	**49**

The risks per project category are compared graphically (Figure 17.2).

Figure 17.2 Comparison of current risks across all project categories



17.2.1 Risks rated Extreme

A total of six risks were identified were rated as “Extreme” with two of the six risks related to health and safety.

Due to the high toxicity of cyanide, which is a key chemical reagent used in the processing of the gold ore, the consequence of significant injury or death can occur if this reagent is not properly managed.

The process plant design has incorporated static HCN detectors, automatic pH and cyanide addition control, which will significantly reduce the risk of HCN generation within the plant. It is also proposed to develop a training programme during the project implementation phase for all process personnel so as to ensure that the workforce understands the risks associated with cyanide and the other process reagents. With these future controls in place, this risk has been re-rated to high.

The other health and safety "Extreme" risk is related to the potential for significant injury due to work tasks associated with working at heights, confined space and the isolation of energy sources that are used in the plant (high pressure air, electricity, moving/rotating equipment etc.). Again it is proposed to develop and implement a comprehensive training program for the process workforce, which should help to mitigate the risk, and minimise the chance of significant injuries to the workforce.

The four remaining "Extreme" risks are associated with potential financial impact to the Ganajur Gold Project caused by:

- Water supply issues, either by the late construction of the raw water storage dam (RWSD) or abnormal weather. The RWSD has to be built prior to the start of the monsoon season to enable storage of water for processing for the remainder of the year. If this critical path item is delayed, then the commencement of the process operations could be impacted for a further year. Also, if a significant drought occurs, then insufficient water will be available for harvesting from the Varada river. The only potential control to mitigate against both these potential risks is to develop access to the aquifer resources in the region.
- A low skilled workforce has also the high potential for causing equipment damage to high value processing equipment and also prevents the attainment of the production targets. India is fortunate that it has some of its workforce skilled in the resources sector; however, the Ganajur Project will be required to recruit a high portion of unskilled labour from the local villages. Again, comprehensive training programs coupled with the recruitment of some skilled process personnel are the recommended controls to mitigate the risk and minimise the impacts.
- The current TSF design is based on upstream construction (using tailings for ongoing embankment construction) and that the PAF tailings can be managed by progressive rehabilitation of the TSF walls. If the column geochemistry leach testwork identifies that the sulphides oxidise rapidly then water treatment of the TSF solution on closure will require treatment prior to discharge back into the environment. Also, it may be more cost effective to change the design to downstream construction so as to inhibit the sulphide oxidation and the associated metal release into the tailings solution.

17.2.2 Risks rated High

Of the 15 "High" risks recorded in the register, seven relate to negative financial impacts to the project, and the remaining risks relate to health and safety, environmental and community relations. The supporting details on these high-rated risks are:

- Financial:
  - Three "High" risks are associated with project management, which may be impacted by the not meeting the implementation schedule, project cost over-runs and contractor quality issues.
  - Three "High" risks are due to the potential issues related to open pit waste and TSF tailings. If the ongoing geochemistry testing identifies further issues regarding arsenic release, then additional costs may be incurred to ensure environmental guidelines are met.
  - Pit wall stability, its potential for failure which may thereby cause significant financial impact, is always a high risk for an open pit operation. To minimise this risk, a high level of wall monitoring and mapping together with good control blasting must be incorporated/ implemented from the geotechnical management plan.
- Health and safety:
  - A key requirement for the engagement of the construction teams and the mining contractor will be their proactive safety culture. Due to the inherent "High" risk associated with construction and mining activities, DESPL will ensure that only contractors with a good historical safety performance be considered for potential engagement with the Ganajur Gold Project.

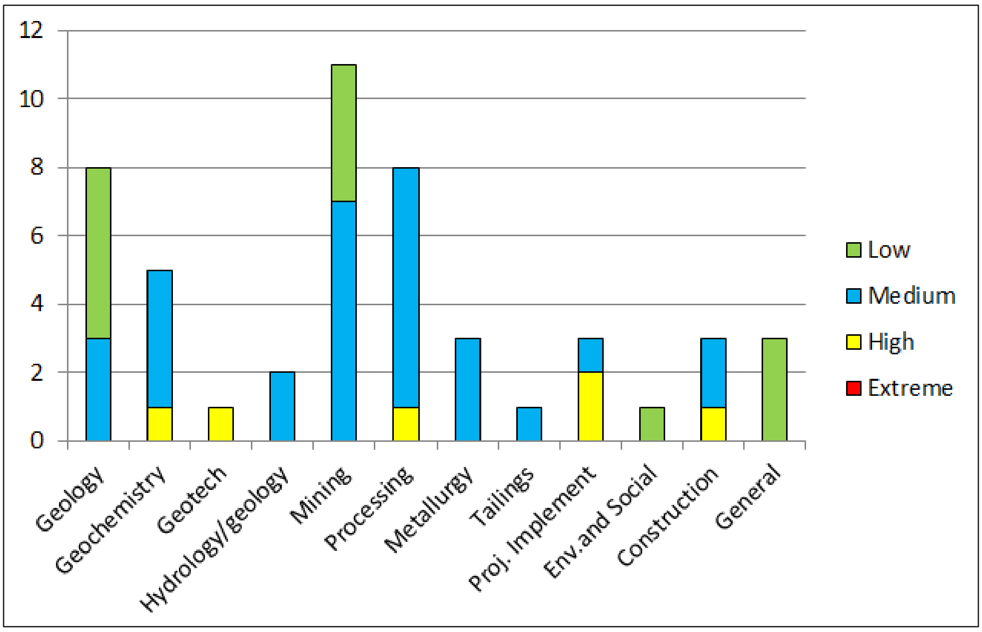
- Traffic management related to the transport of chemicals will be an important task due to the high traffic in the region and via the local villages and farming district. A traffic and transport management plan should be incorporated into DESPL's ongoing management strategies.

- Environmental:
  - Three "High" risks are associated with the uncontrolled release of either tailings (pipe break or dam wall), chemicals from a transport incident and the potential issue of arsenic release from the open pit waste material. Future controls include continual geochemical monitoring at the site and amending procedures as required to ensure there are no uncontrolled chemical releases.
- Community relations:
  - Potential noise and dust issues impacting the local Ganajur village and farming areas will necessitate good management practices and community engagement as part of DESPL's ongoing implementation and operational plans.

17.3 Risks after mitigating controls identified

Following review of the Current Risks, controls were identified and applied to each risk to reduce the potential risk level. The risk level of the 49 risks after mitigating controls were identified and applied is shown in Figure 17.3.

Figure 17.3 Risks after identified controls were applied



The mitigating controls are recommended to be put in place during the implementation phase of the project and be part of the ongoing management practices. The risks and the controls in place to manage them should be reviewed at least every six months to ensure the control is adequate to mitigate or manage the risk as is appropriate.

17.4 Opportunities

Opportunities were identified using the capital project criteria listed in Table 17.3.

Table 17.3 Opportunity table – capital projects criteria

Level	Project Design	Geology/Resource Development
1 >$500	Saving equivalent to one hour of production interruption; Schedule improves one day	Improved management of platform construction; Improved management of Drill or Assay Lab contracts
2 >$5,000	Saving equivalent to twelve hours of production interruption; Schedule improves one week	Improved drill access and planning; Utilizing local labor for drill program
3 >$50,000	Saving equivalent to one week of production interruption; Schedule improves one month	1% gold grade bias (add 3,000 ounces to base); Minor increase in Density used; Minor improvement in gold shape (continuity, size)
4 >$500,000	Saving equivalent to one month of production interruption; Schedule improves 3 months	Up to 5% improvement in gold grade Previously unaccounted metal resource Moderate improvement in gold shape (continuity, size); Moderate pit slope improvement due to changes in alteration model; Discovery of further mineralization
5 >$5,000,000	Saving equivalent to one year of production interruption; Schedule slips one year	>5% improvement in gold grade; Major improvement in gold shape (continuity, size); Major pit slope improvement due to changes in alteration model

Six identified opportunities were divided into relevant project categories with the rankings split between Extreme, High, Medium and Low (Table 17.4).

Table 17.4 Identified opportunities split between project categories and risk levels

Category	Extreme	High	Medium	Low	TOTAL
Hydrogeology		1			**1**
Metallurgy		1			**1**
Processing		2			**2**
General		2			**2**
Total	**0**	**6**	**0**	**0**	**6**

The key opportunities identified during the workshop are outlined as follows:

- The potential for improved gold recovery for the resources located east and west of the major ore tonnage area termed the “belly” portion appears to be high. This potential is due to the lower arsenic levels away from the “belly”, which generally indicates less refractory gold content
- Presently the access of power from the state power grid is located 7.5 km from the plant site. DESPL is negotiating with the power authority to confirm if a closer location can be accessed, which should result in significant capital cost savings
- Due to the potential risks associated with water supply, an exploration program to identify a deep aquifer (that will not be impacted by the local farmers) for water supply would be beneficial.
- In-pit water could be used to support the raw water make-up requirements for the process plant. Additional raw water-make up over 1,000 m^3 per day would potentially be available for local farmers if the quality was within the required standards.

18 RECOMMENDATIONS, INTERPRETATION AND CONCLUSIONS

18.1 Recommendations

Recommendations have been drawn from each of the technical chapters as provided by the various authors.

18.1.1 Geology

Snowden makes the following recommendations for future drilling and resource estimation:

- Create a geological model based on lithology (i.e. chert) as well as the mineralised interpretation. This will aid with the definition of bulk density.
- In future drilling campaigns, assay all variables so that there is more data for analysis. This is particularly relevant for any variables identified during the Feasibility Study as being important for processing.
- Look at using an industry standard database going forward and assess the use of a digital logging system (e.g. onto tablets) with digital data transfer.
- Review the QAQC procedures for future drilling programs including:
  - use of fully certified standards (CRMs) and trialling CRMs from another source
  - inclusion of fields duplicates
  - inclusion of QAQC samples for all sample types.
- Review the use of a scoop for subsampling the pulps and implement the use of a small rotary splitter instead. Given the nature of the mineralisation this is not likely to cause any material issues with the quality of the data.
- Older pulps samples were returned from the laboratory in well-structured and labelled boxes while more recent pulps have been returned in poorly-structured, recycled boxed which are harder to store and not as robust. Snowden recommended that DESPL request a return to the original boxes for pulp sample returns, which has subsequently been initiated.
- Ensure sufficient bulk density measurements are taken in the non-mineralised domains to provide more confidence in these values.
- Carry out a test bulk density program using wax coating of the samples to test the impact of porosity, particularly on the weathered samples.
- Samples within the unmineralised zones surrounding the mineralisation should ideally be assayed, perhaps using a larger sample interval (e.g. 2 m) to allow estimation of the waste grades for mining dilution and waste rock characterisation.
- The Ganajur Main Gold deposit is surrounded by a number of satellite prospects designated as Ganajur South, South East, Central, Karajgi Main and Hut etc. Exploration carried out by DESPL in these prospects to date using surface geological mapping, trenching, drilling and IP geophysical survey indicates significant gold mineralisation and the possibility of adding additional resources across the entire Ganajur-Karajgi Block. It is recommended that DESPL continue its exploration in the satellite prospects in pursuit of augmenting additional gold resources.

18.1.2 Mineral processing and metallurgical testing and recovery

The accuracy of the sulphide sulphur model and hence the estimated gold recovery for the sulphide resource could be improved with the inclusion of more representative drillhole samples.

To enable the opportunity for developing reliable gold production and sulphide concentrate tonnage forecasts, assaying of the blast hole samples for sulphide sulphur is recommended.

Laboratory flotation tests on grade control drill samples will aid in developing reliable production forecasts.

Further UFG tests at a finer liberation size may result in improved gold recovery.

Further arsenic precipitation testwork to identify improved stability would help reduce the environmental risks around the TSF.

Bacterial/oxidative testwork on the sulphide leach residues may lead to an opportunity to increase the gold recovery from the sulphide resource. If further resource/reserves are identified, then this may warrant the installation of a BIOX or Albion circuit.

The CIL gold recovery for the sulphide resource located to the west of the “belly” portion has the potential to have a higher gold recovery. This is due to the interpreted lower sub-microscopic gold content for the ore located in the western area of the Ganajur resource. Further metallurgical testwork on representative samples are necessary to realise this potential.

18.1.3 Geotechnical design

Based on the geotechnical studies, pit slope design recommendations were developed with an acceptable level of risk of small-scale failures developing on batter faces. The recommended slope design angles are summarised in Table 18.1 and design sectors provided in Figure 18.1.

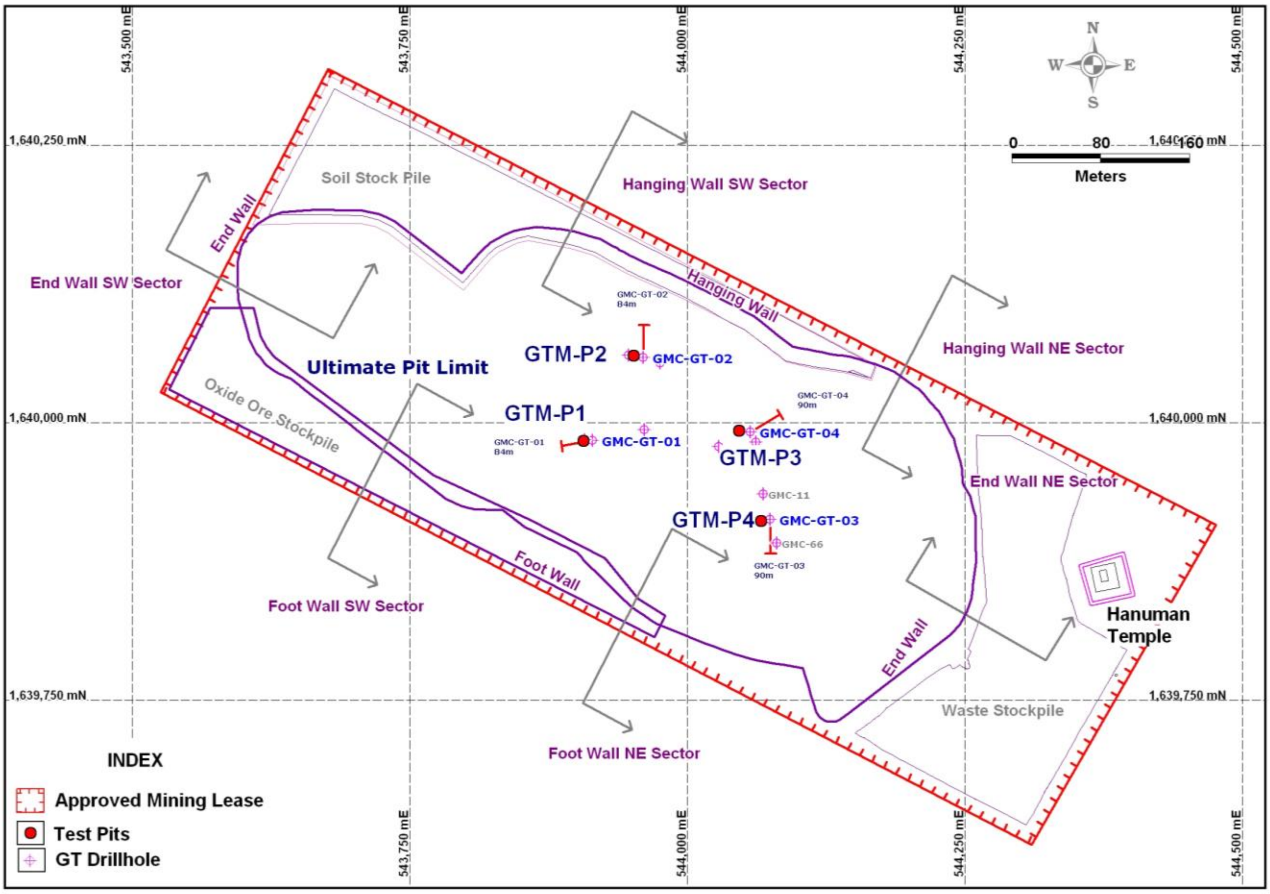
Table 18.1 Pit slope design recommendations

Sector	Wall	Batter angles (°)		Maximum inter-ramp angle (°)	
		Weathered	Fresh	0 to 50 m	50 m to 100 m
Northwest	Footwall	45	60	46	51
	Hangingwall	45	80	48	56
	End wall	45	75	46	56
Southeast	Footwall	45	55	46	51
	Hangingwall	45	80	48	56
	End wall	45	75	46	56

Notes to design recommendations:

- “Weathered” applies to top two 10 m benches
- “Fresh” applies to all benches more than 20 m below surface
- All batters to be maximum 10 m height
- All berms to be minimum 5.0 m width.

Figure 18.1 Pit design sectors



The overall slope angle (crest to toe) of the footwall slope should not exceed 40° for a maximum bench stack height of 50 m. If the ramp is not present on the slope, a double-width berm (10 m) should be incorporated at 50 m depth.

Operational controls to mitigate batter-scale hazards should include the following:

- Weekly inspections of pit walls to identify and map geotechnical hazards
- Final batter faces should be mined with controlled blasting techniques
- Loose rock should be scaled off batter faces with excavator buckets after blasting
- Loose rock on berms and against batter toes should be cleaned up before the subsequent bench is taken.

18.1.4 Mining

The recommendations are made for ongoing work, with items that may need further definition and resolution during early mine production.

Drilling and blasting

For final pit walls, the geotechnical study recommended that perimeter blasting on walls and batters will be required. This is to preserve the integrity of the final pit walls and promote wall stability. There may be additions to the drilling and blasting designs that were recommended on 10 March 2017 by Deepak Vidyarthi. Designs to augment the standard ore and waste drill and blast design patterns may consider perimeter blasting in the form of peripheral drill and blast designs that are:

- Modified production blasts, that reduce the local powder factor close to walls, thereby reducing damage to the fabric of the wall

- Pre-split blasts, where a tensile failure in the rock mass causes a crack on the mining limit and this crack is used to vent explosive energy away from the wall.

Both these peripheral drill and blast methods will need to be trialled when mining reaches the competent rock mass, 30 m below the surface. Based on the effectiveness of standard ore and waste drill and blast outcomes compared to the peripheral drill and blast outcomes, a decision may be made to implement peripheral drill and blast designs.

It is not likely that the peripheral drill and blast modified designs will significantly increase the mining costs.

Trafficability studies

Trafficability was not assessed. Snowden recommends that this be included in future studies so that truck bearing capacity and sheeting requirements can be determined.

Pit dewatering studies

All future dewatering designs need to be validated by a hydrogeologist. The outcome of water inflows into the pit should be provided to the mining contractor for accurate costing. The current design has the water that collects in the mine pit from rainfall being pumped out from the pits, using centrifugal pumps. This water would be used for dust suppression or return to the mine raw water dam and not be discharged without water quality testing. If execution studies result in pit water being utilised in the plant, then this will change the contractor pricing estimates.

Binding mining contract and waste pricing

Currently the mining contractors have estimated non-binding contract mining costs. The scope of mining works including dewatering should be finalised and a single mining contractor should be engaged in a legally binding mining contract between DESPL and the chosen mining contractor so that mining costs can be finalised. The mining contractor will also need to commit to a final purchase price of the waste rock in their pricing.

Equipment utilisation

The production schedule should be discussed with the mining contractors to see if the production schedule is achieve including a spike in explosive requirements in quarter 3.

18.1.5 Process plant

There are several opportunities to simplify the process design to reduce upfront capital costs through either changing selection of equipment or by staging construction of unit operations. The opportunities are detailed below:

- Tabling of the gravity concentrate rather than including an intensive leach reactor and direct smelting of the concentrate is dependent on security and other operating constraints.
- Inclusion of the flotation scavenger cells is based solely on other operations and not testwork for the Ganajur Gold Project. It may therefore be possible to remove this circuit without significantly impacting the overall gold recovery. Further testwork on the flotation rougher tailings should be completed to investigate the viability of this circuit.
- Currently an IsaMill™ has been installed for the UFG of flotation concentrate. It may be possible to install similar equipment for achieving a P_{80} grind of 10 µm that is not proprietary technology and therefore less costly.
- The reagent make-up systems could be simplified by sourcing some of the reagents with lower consumption (PAX, copper sulphate, lead nitrate, caustic) in liquid form or in smaller 25 kg bulk bags. This may increase slightly the cost of the supply, however, would remove bulk unloading systems, mixing tanks and agitators.

- Water supply is based on harvesting water from Varada River over four months and storing sufficient water for the year. Lower rainfall or higher evaporation may impact the supply of water. It would be prudent to explore the potential for accessing deep aquifer supply water, investigate bore water access at the river, investigate pit dewatering flows and reuse possibilities.
- Ongoing testwork should be maintained to understand the potential of arsenic precipitants re-dissolving in the TSF.

18.1.6 Surface geotechnical and tailings disposal

Long term kinetic testing is currently being undertaken on the tailings material to determine the long term intrinsic oxidisation rate of the tailings material. The final results (expected in June 2017) should be analysed together with other design criteria to confirm the suitability of an upstream facility.

The geotechnical investigation has shown that the surface soils comprise of medium to high clay content. The laboratory results have not conclusively shown that the soils are suitable for use as a primary low permeability liner. Additional soil testing is required during the detailed engineering stage where the focus should be placed on sampling and testing of the cotton soils found over the project site.

It is recommended that during the detailed engineering phase additional slope stability be undertaken and including seepage and the interface between the geomembrane and the in-situ soils.

The compilation of a construction and operations manual during the detailed engineering phase is required, specifically on drainage and seepage water management.

18.1.7 Project infrastructure

If the state grid power (KPTCL) power stability proves to be unreliable, install a diesel generating power station on site.

If the quantity of groundwater inflow into the Ganajur open pit proves to be high, then this source of water maybe more suitable as process water make-up. The impacted farmers could be provided with water supply from the Varada river in exchange.

18.1.8 Geochemistry

Acidic to near neutral saline drainage is expected to be released following oxidation of the ROM stockpile and tailings material. The waste rock material is less sulphidic and does not have long term potential to generate acid, therefore neutral or saline drainage is expected to decant from any waste rock piles on site.

The abundance of arsenic (As) in the ore, tailings and waste rock material is of concern. Liberated As species are mobile across the pH scale. In the material, As is present as both readily soluble and insoluble phases. Potential soluble phases include adsorption onto or inclusion into iron hydroxide or association with carbonate minerals. Insoluble phases include arsenopyrite which would require oxidation for the As to become mobile.

The following mitigation strategies are recommended:

- The ROM stockpile would require appropriate compacted clay lining to prevent seepage into groundwater resources and percolating rainwater should be diverted away from natural, fresh water resources.
- The TSF should be double lined with HDPE and clay. In the long term, if the material is oxidised, acidic decant with significant As concentrations is likely.
- Capping of the TSF at closure will prevent water and oxygen ingress. Limiting oxygen ingress through continuous tailings sidewall covering and rehabilitation will minimise the oxidation of the tailings material and prevent the generation of an acidic leach.

In the event that waste rock is not sold and is stored on site, the waste rock dump/s will require an appropriate base (e.g. compacted clay) to prevent point source contamination of groundwater. The long-term risk will be determined by the ongoing column leaching experiments.

18.1.9 Hydrology and hydrogeology

The following recommendations are made to manage the potential impacts on the combined groundwater and surface water resources:

- Best practice guidelines to be implemented at potential sources of contamination to groundwater through bunding and lining of facilities
- Groundwater and surface water (rainfall) made in the pits to be used in ore processing plant as make-up water
- Active dewatering/water ahead of mining using dewatering/water supply boreholes close to pit so that water quality can be maintained whilst starting to drawdown water table ahead of mine plans
- Combined water balance to be developed for the mine for three seasons so that water use is optimised
- Excess mine water from dewatering can be directly supplied to the neighbouring private farmers, through reticulation providing water quality is acceptable
- As the mine will eventually be abstracting >1,000 m^3/day artificial recharge of groundwater using roofs of selected mine infrastructure to harvest rainwater and recharging the aquifer through recharge pits as per the CGWA guidelines
- Ultimately, groundwater may replace Varada River water at the supply to the mine if the results of the preliminary modelling can be confirmed
- Additional hydraulic parameters to be determined for ore zone and halo zone in each geotechnical domain of the pit
- Accumulate one year of monitoring data (water levels and water quality) including heavy metals
- Expand model boundaries and re-calibrate model with new data
- Develop transport model for arsenic and nitrate plume development from waste rock dumps, TSF and determine risks to human health and aquatic environment
- Model mine closure scenarios to predict pit lake chemistry once the groundwater table rebounds.

18.1.10 Environmental and social

The baseline ecological specialist studies took place over a single season, namely summer (March, April and May 2016). Terrestrial ecosystems are dynamic and complex and thus a more reliable assessment would entail additional sampling, as sampling, by nature, implies that not all species in the area will be recorded, due to seasonality, microhabitat requirements and management practices. In order to rule out the possibility that further protected or conservation-worthy species utilise the habitat in the study area, additional ecological studies should be undertaken during the winter and post-monsoon seasons and the findings of the EIA report updated accordingly.

The meteorological data used for assessing potential impacts in terms of air quality were based on data recorded on site between the months of March and May 2016. It is recommended that this data is verified through evaluation of meteorological conditions for the balance of the year and further compared to available historic records for the region so as to ensure that the predicted model outcomes are accurate.

The geohydrological impact assessment is based on a groundwater model derived from sampling of water levels in regional borehole wells as well as short-duration pumping tests on six groundwater boreholes. In order to better evaluate the potential impacts of the mine in the both the short and the long-term, it is recommended that the following tasks are undertaken:

- The installation and testing of site-specific groundwater monitoring borewells (three such boreholes are currently planned). These boreholes should be subjected to longer duration pump tests in order to confirm the characteristics of the aquifer. This data, together with the information gathered from sampling of existing borehole wells, can be used to update the groundwater flow model after two years.
- The model should further aim to identify the rate of groundwater recovery in the post-mining environment. The altered landscape may result in decant points arising which, depending on the quality of recharging groundwater, warrant additional management by the mine.
- The groundwater model should be updated to incorporate the geochemical assessment data of the mine materials under neutral and acidic conditions so that a pollution plume model can be prepared which illustrates any contaminants of concern which may be transported into the environment and whether any sensitive receptors (water courses and external users) are situated in the zone of influence. This aspect will also be important for management in the post-closure environment.

An implementation plan should be prepared which elaborates on the afforestation and green belt plans. This plan should be prepared by a suitably qualified specialist and should cover aspects such as suitable species, timing of the implementation, mechanisms to monitor the interventions implemented and to ensure that no unforeseen ecological impacts arise as a result. The plan should also cover the long-term aspects of this area in order to ensure that the mine will not inherit a long-term liability.

The estimated PPV used to evaluate potential impacts on the sensitive structures near the proposed mine should be confirmed once the rock constants have been determined.

The recommendations made in the monitoring program should be further elaborated in more detail. Monitoring locations should be selected and the monitoring requirements should be separated into the various phases of mining (construction, operation, closure and long term).

A detailed mine decommissioning and rehabilitation plan should be prepared which identifies all areas of potential liability, provides mechanisms for the removal of infrastructure not to remain permanently in place and rehabilitation of areas with the aim of producing a stable landscape which meets the intended end land use. The cost of such measures should be calculated and incorporated into the mines financial planning. The revised groundwater model to be prepared will assist in determining whether the mine needs any future plans for the installation of any water treatment infrastructure to manage quality groundwater and the duration for which such treatment may be required.

The EIA should be further elaborated by separately evaluating impacts for the various phases of the mine. The impacts should further be evaluated utilising a risk matrix which derives significance from aspects such as magnitude, scale, duration and probability for each potential impact during each phase.

18.1.11 Project implementation

On the proviso that all statutory approvals can be finalised by 2017, further equipment orders in addition to long lead items could be placed in parallel with the statutory approval timeline

Due to the small equipment size for the process plant, this enables the modularisation for some of the process plant circuits. This should reduce the time for on-site installation and the associated construction labour requirements.

An operational readiness plan would help improve the commissioning process as well as potentially decrease the time taken to reach full production.

18.2 Interpretation and conclusions

18.2.1 Geology and Mineral Resources

The previous resource estimate was completed by SRK in August 2011. The estimate was reported at a 1 g/t cut-off as detailed below in Table 18.2. To provide a valid comparison Snowden reported its August 2016 Mineral Resource estimate at a 1 g/t Au cut-off in Table 18.3.

Comparison of the estimates shows that there is a 125 kt increase in tonnes and a slight decrease in grade for the total Measured and Indicated Resources at a 1 g/t Au cut-off. The updated Mineral Resource has converted the majority of the previously reported Indicated Resources to Measured Resources. This is due to the extra drilling which has increased confidence in geological knowledge and grade continuity of the deposit.

There is also an increase in the Inferred tonnes and grade due to extra drilling completed to increase the size of the deposit and better define the mineralised boundaries

Table 18.2 Previous Mineral Resource estimate – SRK 2011 reported at a 1.0 g/t Au cut-off

Classification	Deposit	Tonnes (kt)	Au (g/t)
Indicated	Oxide	631	3.19
	Sulphide	1,921	3.83
	Total Indicated	2,552	3.67
Inferred	Oxide	17	3.26
	Sulphide	93	1.82
	Total Inferred	109	2.06

Note: Small discrepancies may occur due to rounding

Table 18.3 Ganajur Mineral Resource estimate as at August 2016, reported above 1.0 g/t Au cut-off

Classification	Deposit	Tonnes (kt)	Au (g/t)
Measured	Oxide	569	2.85
	Sulphide	1,671	3.99
	Total Measured	2,240	3.70
Indicated	Oxide	117	1.96
	Sulphide	321	2.14
	Total Indicated	438	2.09
Measured + Indicated	**Total Measured and Indicated**	**2,678**	**3.44**
Inferred	Oxide	103	2.32
	Sulphide	96	2.46
	Total Inferred	199	2.39

Note: Small discrepancies may occur due to rounding

Ganajur Main Gold Deposit is surrounded by a number of satellite gold prospects such as Ganajur SE, Ganajur South, Karajgi Main, Karajgi Hut and Ganajur Central prospects. DESPL’s limited exploration carried out in these prospects so far, have indicated the possibility of finding additional resources. A sustained exploration effort in the satellite prospects will be required to explore the possibility of finding additional resources.

18.2.2 Metallurgical testwork

The metallurgical testwork performed for the FS has identified the following key outcomes and conclusions:

- The oxide and sulphide ore are competent in terms of comminution characteristics and are expected to consume 25 kWh/t to 28 kWh/t of ore processed at a grind liberation grind size of P80 75 microns.

- An average gold recovery of 90% can be achieved via CIL for the oxide ore.
- An average gold recovery of 79% can be achieved from the sulphide ore via flotation for the recovery of the sulphides (predominately pyrite), followed by UFG/CIL. The sulphide concentrates require further grinding to a liberation size of P80 10 microns.
- The reason for the lower gold recovery for the sulphide ore is due to a portion of the gold losses in the final residue is not recoverable by cyanidation leaching (partially refractory). These gold losses are due to sub-microscopic gold hosted in both arsenopyrite and pyrite.
- The sulphide ore samples tested were sourced from the “belly” portion of the Ganajur Main resource which contributes the major ore tonnes for the sulphide resource. The sulphide ore located to the northwest and southeast of the belly area has lower arsenic levels which indicate that higher gold recovery may be achievable.
- The sulphide ore gold recovery is strongly linked to the sulphide sulphur content of the ore, which has been estimated via a regression model using gold and arsenic content as the predictors. It would be prudent to carry out further sulphide sulphur analyses to further refine the regression model and hence improve the accuracy of the gold recovery estimate.
- The final residues require cyanidation destruct and arsenic removal before discharge into the TSF in order to comply with environmental conditions.

If additional sulphide resources are identified by on-going exploration in close proximity to the Ganajur Main Project, then the justification of adding an additional process circuit using oxidative techniques to further improve gold recovery may be warranted.

18.2.3 Mining and Ore Reserves

Ore Reserve reporting

Proved and Probable Ore Reserves were reported using the 2012 edition of the “Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves” (JORC Code 2102).

The Ore Reserve estimate for the Ganajur deposits, as at the end of April 2017, is provided in Table 18.4. Note that tonnes and ounces have been rounded and this may have resulted in minor discrepancies.

Table 18.4 Ganajur in-situ Ore Reserve estimate as at April 2017

Classification	Weathering	Tonnes (kt)	Au (g/t)
Proved	Oxide	568	2.76
	Sulphide	1,567	3.94
Total Proved		**2,135**	**3.63**
Probable	Oxide	122	1.78
	Sulphide	250	2.08
Total Probable		**372**	**1.98**
Total		**2,506**	**3.38**

The Ore Reserve is robust based the higher average gold grade of the deposit and will be able to be mined subject to:

- Finalisation of outstanding permits and approvals
- Finalisation and award of the mining contract, including the sale of waste rock to the contractor, as a legally binding arrangement.

18.2.4 Process plant

A conventional three-stage crushing and primary ball mill has been selected for the front end of the process plant as it offers the project the lowest risk in terms of achieving the target production rate of 300,000 t/a.

Due to the significant difference in metallurgical properties for the oxide and sulphide ore, two separate processing methods have been incorporated into the treatment plant. The oxide ore only requires direct cyanidation leach to achieve optimum gold recovery.

The sulphide ore will be processed via flotation for the recovery of a sulphide concentrate, followed by UFG and CIL. Due to the partial refractory nature of the sulphides, gold recovery is limited to the low 80% range.

The AARL elution method has been selected for the recovery of gold from activated carbon.

Prior to residue disposal to the TSF, cyanidation destruct via sulphur dioxide/oxygen and arsenic removal as an arsenical ferri-hydride precipitate are performed in order to meet environmental standards/conditions.

18.2.5 Geotechnical and tailings studies

The Ganajur Gold Project infrastructure will include a TSF, a return water dam and stormwater or event dam, and other surface water management measures.

A geochemical assessment of the tailings material was undertaken and the tailings have been classified as potentially acid forming and with residual traces of arsenic despite the stabilisation step in the process plant. Due of the geochemical classification, the selection of a suitable deposition and construction methodology required an analysis of the design criteria of both an upstream and downstream facility. A risk-based approach was used in selecting a preferred construction method, taking into account aspects such as oxidation rates and other mitigation measures, drainage and long term environmental impacts. The key criteria for the selection of the TSF construction and operation methodologies included a cost-effective solution, environmentally acceptable practice, maximum water conservation and minimum land use. A final recommendation and design option was made in favour of an upstream facility based on a smaller footprint, suitable pollution mitigation solutions and maximising water recovery.

Geo-mechanical laboratory testing of the tailings material was undertaken to determine its behaviour under conditions expected during the life of the TSF. These include density, permeability, shear strength and consolidation and were used in the design of the TSF.

A geotechnical investigation was undertaken to determine the suitability of the in-situ soils for its intended use as low a permeability liner and the construction of containment embankments. The results show that the in-situ soils have suited mechanical properties for embankment construction, but require additional compaction and permeability testing.

The TSF has been positioned adjacent to and downstream of the processing plant. The site selection was influenced by the required storage capacity and footprint, construction and development methods, local structural geology and topography, land ownership, rehabilitation requirements and existing significant surface infrastructures and features.

The TSF is designed to store a total dry tailings tonnage of 2.6 Mt over the 8.4-year life of mine requiring a volume capacity of 1.63 Mm^3 and footprint of 15.7 ha. The facility reaches maximum height of 19 m at a final rate of rise of 1.95 m per year and overall slope of 1V:3H.

The basin of the TSF will be lined with a low permeability clay layer, overlain with a geotextile and an HDPE geomembrane. Tailings will be deposited via a ring main distribution system. Drainage of supernatant water and rainfall will be via vertical penstock intake structures. Seepage water will be drained thought toe and blanket drains positioned along the basin of the TSF.

Decant and seepage water will be collected in a return water dam for reuse in the process water. Stormwater runoff from the side slopes and the top surface of the TSF will collect in the perimeter catchment paddocks from where it will drain into the stormwater or event dam.

The project site is located in a low risk seismic zone. A slope stability analysis, which incorporated tailings and soils mechanical properties and the TSF geometric profile, indicated a suitable FoS against major slope failure.

A monthly deterministic water balance model has been compiled for the TSF, return water and stormwater dams. The input parameters to the water balance include the tailings characteristics, TSF layout and climate data. The rainfall data has been statistically analysed to incorporate the seasonal variability and to compile a conservative water balance. The annual return water volume was estimated at 130,500 m^3, approximately 53% of the slurry water deposited on the TSF per annum will return to the process plant.

The side slopes of the TSF will be progressively rehabilitated with a clay and topsoil layer and vegetation. Once the TSF has been decommissioned, the top surface of the TSF will also be capped with a rock, clay and topsoil layer, and vegetated.

18.2.6 Project infrastructure

The Ganajur Gold Project in terms of location has good access via road and rail to support the delivery of materials to the project site.

Services for power supply will be accessed via the Kanataka State HV power grid and water supply will be provided by the harvesting of water over a four-month period from the nearby Varada River during the monsoon season. This water supply will be stored in a 300,000 m^3 dam to enable the process plant to operate over the full year.

Some contingency for water storage has been provided, however adverse weather may impact the ability to process ore over a full year. Groundwater sources may provide an additional source of water, however this resource is also used by the local farmers in the district.

18.2.7 Marketing

India is one of the largest importers of gold in the world and the imports mainly take care of demand from its jewellery industry. During the period from April 2015 to February 2016, gold import stood at about 941.1 t while for the entire 2014/2015 fiscal, the same was 900 t.

Going forward, the International Monetary Fund (IMG) has forecast per capital GDP to grow by 35% for 2015 to 2020 and the National Council of Applied Economic Research expects India’s middle class to double, exceeding 500 million by 2025.

By 2020, it is expected that the Indian gold demand would average 850 t/a to 950 t/a.

The mining industry in India is undergoing transformation due to globalisation. In order to bring in transparency and speedy disposal of mineral concessions, the new Government of India has brought in a number of changes in the mining sector by multiple enactments and procedures in 2015. The New Mines and Minerals (Development and Regulation) Amendment Act 2015 (MMDR) is likely to usher new hopes for the mineral industry. In order to encourage scientific exploration especially for minerals such as gold the new Act has introduced a Composite Licence system for exploration and followed by mining.

India’s relationship with gold goes beyond income growth; gold is intertwined with India’s way of life. And as we look ahead, India’s gold market will evolve. DESPL’s Ganajur Gold Mine will be well placed to supply the expanding gold demand in India and has the added advantage of providing income and prosperity for her citizens as well.

18.2.8 Geochemical study

Acidic to near neutral saline drainage is expected to be released following oxidation of the ROM stockpile and tailings material. The waste rock material is non-hazardous and does not have long term potential to generate acid, therefore neutral or saline drainage is expected to decant from any waste rock piles on site.

The abundance of arsenic (As) in the ore, tailings and waste rock material is of concern. Liberated As species are mobile across the pH scale. In the material, As is present as both readily soluble and insoluble phases. Potential soluble phases include adsorption onto or inclusion into iron hydroxide or association with carbonate minerals. Insoluble phases include arsenopyrite which would require oxidation for the As to become mobile.

The following mitigation strategies are recommended:

- The ROM stockpile would require appropriate lining to prevent seepage into groundwater resources and percolating rainwater should be diverted away from natural, fresh water resources.
- The TSF should be lined. In the long term, if the material is oxidised, acidic decant with significant As concentrations is likely.
- Capping of the TSF at closure will prevent ongoing water and oxygen ingress. Limiting oxygen ingress through continuous tailings sidewall covering and rehabilitation will minimise the oxidation of the tailings material and prevent the ongoing generation of an acidic leach.

In the event that waste rock is not sold and is stored on site, the waste rock dump/s will require an appropriate base (e.g. compacted clay available on site) and bunding to prevent point source contamination of groundwater. The long-term risk will be determined by the ongoing column leaching experiments.

18.2.9 Hydrogeology and hydrology

The following impacts due to the proposed Ganajur mining operations associated with the hydrogeological conditions should be taken into consideration during the final planning, execution and management stages of the project:

Groundwater quality:

- Deterioration in groundwater quality due to acid rock drainage (from ore stockpiles, TSF)
- Leaching of arsenic and other metals from ore stock pile, waste rock dump and TSF
- Increasing concentrations of nitrate and ammonium concentrations in the groundwater due to blasting residues (associated with waste rock dump, TSF)
- Spillages and leaks from waste facilities, fuel storage/depots.

Groundwater quantity:

- Water table decreasing in private boreholes within the expanding cone of drawdown and therefore less water available for irrigation and water supply
- Less baseflow to Nala Stream and therefore Varada River due to passive and active mine dewatering
- Elevated pore pressures on southern footwall with stratigraphy dipping into pit, due to natural Nala Stream course which is fault controlled could result in stability concerns and may require flattening of slope.

Recommendations to mitigate the impacts include:

- Active dewatering/water ahead of mining using dewatering/water supply boreholes close to pit so that water quality can be maintained whilst starting to draw down water table ahead of mine plans

- Implementing best practice guidelines to prevent potential groundwater contamination and monitor across the site
- Managing onsite water to maximise water harvesting, optimise usage and minimise wastage
- Update groundwater models and the water balance on a regular basis with monitoring data across project area.

18.2.10 Environmental and social studies

Baseline specialist studies took place over a single season, namely summer (March, April and May 2016). Following the completion of the baseline and the analysis in the Environmental Impact Assessment, recommendations have been made for additional specialist studies and investigations by which to address gaps in the knowledge, confirm assumptions made and improve confidence in the available data.

An implementation plan should be prepared which elaborates on the afforestation and green belt plans. This plan should be prepared by a suitably qualified specialist and should cover aspects such as suitable species, timing of the implementation, mechanisms to monitor the interventions implemented and to ensure that no unforeseen ecological impacts arise as a result. The plan should also cover the long-term aspects of this area in order to ensure that the mine will not inherit a long-term liability.

The estimated peak particle velocity (PPV) used to evaluate potential blasting impacts on the sensitive structures near the proposed mine should be confirmed once the rock constants have been determined. The recommendations made in the monitoring program should be further elaborated in more detail. Monitoring locations should be selected and the monitoring requirements should be separated into the various phases of mining (construction, operation, closure and long term).

A detailed mine decommissioning and rehabilitation plan should be prepared which identifies all areas of potential liability, provides mechanisms for the removal of infrastructure not to remain permanently in place and rehabilitation of areas with the aim of producing a stable landscape which meets the intended end land use. The cost of such measures should be calculated and incorporated into the mines financial planning.

The revised groundwater model to be prepared will assist in determining whether the mine needs any future plans for the installation of any water treatment infrastructure to manage quality groundwater and the duration for which such treatment may be required.

These items should be built into the implementation plan for the project as it progresses towards construction and should be monitored thereafter as part of the ongoing operation of the mine and process plant.

18.2.11 Costs and economic analysis

Operating costs have been sourced from local providers to ensure the costing is appropriate the country and region.

Capital costs have been sourced from reputable EPC providers with local country knowledge to ensure that the costing is appropriate the country and region.

In general, the Ganajur Gold Project, if it is managed to the study inputs and outcomes, is a robust project. This is demonstrated by the moderately low Risk Index, the ranges of NPV values at the two confidence levels are close, and the probability values are also tightly grouped Sufficient equity will need to be raised by DESPL to capitalise the project and Snowden has not modelled any debt financing in its financial analysis.

18.2.12 Project implementation

To achieve the target of full gold production by the fourth quarter of 2018, an aggressive work program is required. The key tasks and milestones are:

- Preliminary orders for long lead equipment items (ball mill) require placement by Q3, 2017
- Environmental approvals are achieved by the end of 2017
- A competent EPCM and mining contractor will require engagement by Q3, 2017 and Q4, 2017 respectively
- The water storage dam and river extraction facility and pipeline require completion by Q2, 2018
- Completion of process plant and related infrastructure by Q3, 2018 to enable the commencement of ore commissioning.

18.2.13 Risks and opportunities

A risk and opportunity assessment for the Ganajur Gold Project was performed via a one day workshop held in March 2017 at the CPC office in Perth, Western Australia. The risk assessment was attended by representatives from Snowden, CPC and DESPL. Prime Resources input was via a conference call link.

Forty-nine (49) Current Risks were identified during the risk workshop and these were assessed and ranked with a common methodology (Table 18.5). Following the initial ranking of these current risks, controls were identified to manage or reduce the risk. The risks were then re-ranked. Subsequently the risks were found to have a reduced ranking. Six opportunities for the project were also identified.

Table 18.5 Current risks with initial ranking and subsequent ranking with controls identified

	Extreme	High	Medium	Low	TOTAL
Initial risk ratings	7	14	22	6	49
After controls identified	0	6	30	13	49

18.2.14 Recommendations

Following completion of the technical studies, recommendations have been provided by the authors that cover the technical, environmental and social aspects of the project. These recommendations are items for consideration, or work to be completed, during the next phase of the Ganajur Gold Project and for the future operating state of the Ganajur Gold Mine.

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