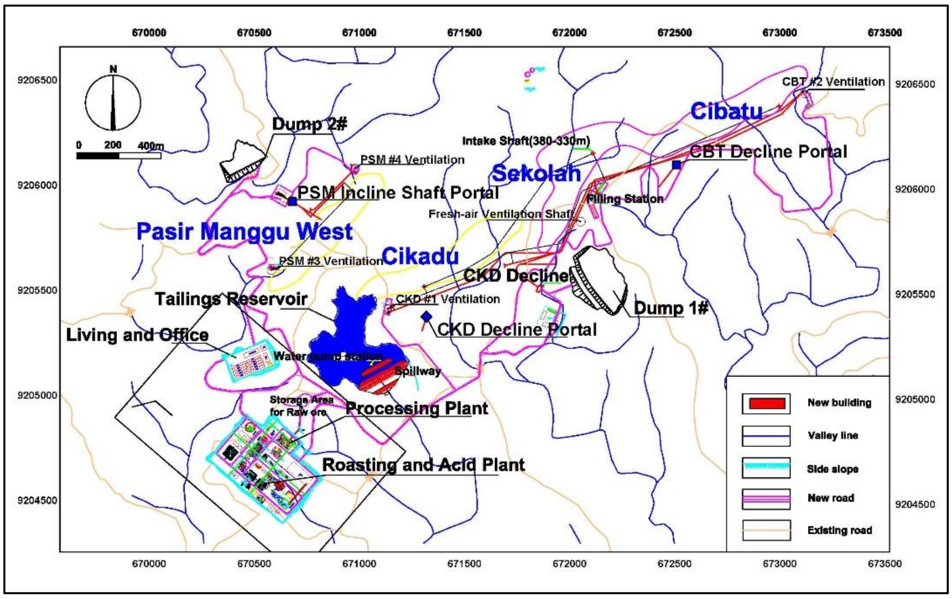
Update of the Independent Qualified Person's Report for the Ciemas Gold Project, Ciemas, Sukabumi Region, Republic of Indonesia

Report Prepared for

PT. Wilton Wahana Indonesia



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Executive Summary

In 2017, PT. Wilton Wahana Indonesia ("Wilton" or "the Company") commissioned SRK Consulting (China) Limited ("SRK") to conduct a feasibility study to develop the defined mineral resources of the Ciemas Gold Project ("Ciemas") located near the town of Pelabuhan Ratu in the Sukabumi Region of West Java, Indonesia. Furthermore, SRK was required to update the technical report on the project with new findings of the feasibility study and to issue an Independent Qualified Person's Report ("IQPR" or the "Report") for inclusion in documents to be submitted to the Stock Exchange of Singapore Limited ("SGX").

Summary of Principal Objectives

The purpose of this Report is to provide an independent technical assessment of the project based on all available technical data in compliance with the requirements for listing a mining company on the Singapore Stock Exchange; the Report is to be included in documents to be submitted to SGX.

Outline of Work Program

The work program involved three phases:

- Phase 1: Review of provided historical data and information, and a site visit to the Ciemas Gold Project near Ciemas, Sukabumi, Indonesia, in September 2017. Tasks include: inspection of the project sites, including general inspection of the prospects, drill-core storage, electricity and water supplies, current and proposed industrial sites, i.e. ore processing plant site, decline portals and mining industrial sites, tailings storage facility site, office and dormitory sites, etc.; discussion of issues with Wilton staff, collection of documents; and preliminary decision on the development scheme and method in different aspects;
- Phase 2: From September to October 2017, SRK proposed the development scheme and methods in different aspects with trade-off studies, and together with the Wilton personnel to determine the final development scheme and methods used in the feasibility study;
- Phase 3: From November 2017 to May 2018, SRK, together with Nerin, a subcontractor by SRK, conducted the detailed study and design for the feasibility study of developing the deposits of Pasir Manggu West, Cibatu, Cikadu, and Sekolah, and submit a draft report for the review of the client.
- Phase 4: From June to July 2018, by considering the comments from the client, SRK finalized the feasibility study;
- Phase 5: From July to August 2018, SRK compiled the update of the IQPR to include the new findings of the feasibility study.

Results

Overall

Wilton operates the Ciemas Gold Project in West Java, Indonesia with two mining licences covering a total area of approximately 30.8 square kilometres (km^2). The gold mineralization in Ciemas is hosted in quartz veins, structural altered rocks with tectonic breccia, or in quartz porphyry. A number of mineralized zones have been identified in the Project, among which four main prospects ("4 Main Prospects" or "4 Prospects"), namely Pasir Manggu West, Cikadu, Sekolah and Cibatu, and other two prospects, Cibak and Cipancar, were explored with adequate density of core drilling or shallow shafts.

SRK has reviewed the exploration work and the integrated database, and estimated that the Ciemas Project contains approximately 3.42 million tonnes ("Mt") of Measured + Indicated Resources averaging 8.6 grams per tonne ("g/t") of gold, and about 2.56 Mt of Inferred Resource averaging 6.5 g/t of gold at the areas of Pasir Manggu, Cikadu, Sekolah, and Cibatu, plus Cibak and Cipancar.

It is proposed that both underground mining and open-pit mining will be applied to mine the mineral resources. From 20 meters underneath the surface, underground mining method will be used, and the top 15 meters will be mined by using open-pitting method. Open-pit operation will be in the last two years of underground mining.

The overall production capacity designed for the Ciemas Gold Project is 1,500 tonnes per day ("t/d"). For underground mining, there will be two mining systems, one at Pasir Manggu West, and the other is for Cikadu-Sekolah-Cibatu. The primary access for the Pasir Manggu West consists of incline shaft for main access and vertical shafts for ventilation. The primary access of the mining system for other three deposits consists of declines for main access and vertical shafts for ventilation. The Pasir Manggu West system will have a production capacity of 300 t/d and the other system for Cikadu-Sekolah-Cibatu will have a capacity of 1,200 t/d.

Both mechanized cut-fill and traditional cut-fill stoping methods are proposed to be used for underground mining. Heights of levels are 20m and 40m for and mechanized cut-fill and traditional cut-fill stoping methods, respectively.

An ore metallurgical plant with 1,500t/d capacity will be constructed. Primary crusher, semi autogenous mill and ball mill ("SAB") are proposed for crushing and grinding, then flotation flowsheet will produce gold concentrates, and then the tailings of the flotation will be undergone Carbon-in-leach ("CIL") cyanidation process, and finally to produce gold doré. For the gold concentrates, they will be sold as final products in the beginning years, and then they will be further processed by roasting for oxidation, then the residue of the roasting process will be milled and undergone CIL to finally produce gold doré. The sulphur discharged from the roasting of gold concentrates will be recovered and further processed to produce sulphuric acid.

It is proposed that the mining and the ore processing plant will be built first, which will take one and a half years. The roasting and acid manufacture facility will be built at fourth year after the mining and ore processing production. It is estimated that the initial capital expenditure ("CAPEX") will be US dollar ("USD") 99.4 million ("M"); and sustaining CAPEX will be USD 68.4 M. The operating expense ("OPEX") for the mining and processing only is about USD 69.3/t ore, and the OPEX with the roasting will be about USD 85.9/t ore. Economic analysis by using the discount cashflow method projects that the Ciemas Gold Project has a net present value ("NPV") of USD 180.3 M, an internal rate of return ("IRR") of 43.0%, and the pay-back period is 3.4 years.

The analytical results indicate that the project possesses Probable Ore Reserve of 2,831 thousand tonnes with an average gold grade of 8.2 g/t gold for underground mining and 429 thousand tonnes with an average grade of 4.6 g/t gold for open-pit mining. The combined Ore Reserves of underground and open pit mines are 3,260 thousand tonnes with an average grade of 7.7g/t.

Operational Licences and Permits

SRK has sighted the original business licences for the Ciemas project, one for the Company and the other for the PT. Liek Tucha Ciemas ("Liek Tucha"). SRK has sighted an original supporting document indicating that the Company owns 95% of PT. Liek Tucha Ciemas. SRK has also sighted the two original Mining Business Licences ("IUPs") that have been issued for the Ciemas project. These were both issued by the Integrated Licensing Services Board Administration of Sukabumi District.

- SRK has sighted the relevant land documents indicating that the Company has secured land access rights to approximately 100 hectares ("ha") of land from the local residents in Pasir Manggu and Cileuweung gold bearing zone areas during past five years.

Geology

The Ciemas Gold Project is situated within a volcanic polymetallic metallogenic belt in Ciletah Bay, Indonesia, containing gold ("Au"), silver ("Ag"), lead ("Pb"), zinc ("Zn"), and copper ("Cu"). The belt is formed mainly of volcanic breccias and mostly covered by Quaternary eluvium and alluvium as well as a post-mineralisation tuff blanket up to 20 m thick. Volcanic breccias, tuffs, and andesite are widely distributed in the Project area.

Two sets of fractures are developed, striking to the northeast and northwest with extensions varying from about 100 to 1,000 m; the fracture belts are generally 1 – 20 m wide. These fractures are the primary gold ore-controlling tectonics and ore-bearing zones in this area.

Most gold mineralized bodies present in the northeast zone contain brecciated chalcedony-quartz carrying pyrite, arsenopyrite, and small amounts of galena and sphalerite mineralization. The zone is covered by strongly silicified clay several metres thick and containing disseminated pyrite. The indistinct external propylitic alternation envelope features chlorite and scattered pyrite.

Pasir Manggu is made up of three (3) sets of quartz veins from southwest to northeast which occur in andesitic lava and andesitic pyroclastic rock. They generally strike northeast ("NE") at 45° and dip southeast ("SE") at 75° - 80°. Pasir Manggu West, located at the southwestern most end of Pasir Manggu, was explored by drill holes on 20 × 20 m and 40 × 40 m grids, which delineated a mineralized belt with four major veins extending about 600 m along the strike in accordance with the tectonic framework. According to the drilling findings, the gold mineralized veins are still open at depth and the defined down-dip extension exceeds 120 m at most but with an average controlled depth of 50 – 70 m. The true thickness of gold veins in Pasir Manggu West varies from 1 m or less up to 10 m, with average thickness about 4 m. The average grade of gold mineralized veins at Pasir Manggu West is about 7 g/t Au.

Cikadu is composed of two main mineralized bodies on a northwest ("NW") strike and dip of 60° to 75°, with a length of 700 m, a thickness of 1 to 10 m, and an average Au grade of about 9 g/t.

Cibatu-Sekolah comprises 11 mineralized bodies plunging NW and dipping 60° to 75°, including five main bodies striking for a total length of 1,500 m, 1 to 10 m thick, and with an average Au grade of about 9 g/t.

The structure and type of alternation in the northwest belt are similar to those found in the northeastern belt, but the NW belt contains small amounts of chalcopyrite, and more galena and sphalerite. This zone mainly occurs in the Ciaro region. There are several NW veins in the east which have been subject to extensive mining in the past.

There are several north-south ("NS") striking zones in various locations, but due to insufficient exploration works, their ore bearing potentials are unknown. Several veins around Pasir Manggu strike approximately east-west, and are regarded as related to the northwest zone.

There are few outcrops of intrusive rocks; quartz porphyry outcrops are observed in the Cileuweung block. Potential for further discoveries of numerous gold occurrences are scattered throughout the Ciemas exploration license. The primary mineral commodity is the gold ore.

Three types of gold ores were distinguished and can be described as quartz-vein, tectonically altered-rock, and quartz porphyry ores.

Exploration

In general, exploration work including geological mapping, drilling and surface outcrop exposure (i.e., trenching and pitting), soil and bedrock sampling, and geochemical and geophysical surveys over a significant portion of the Project's concession area were completed in a series of staged exploration programs.

Beginning in 1986, a former Australian company, Parry Corporation Limited ("Parry"), contracted with Liek Tucha (the concession holder at the time) and commenced exploration work in the project area. Detailed exploration work was concentrated in Pasir Manggu, consisting of geological mapping, geochemical and geophysical surveys, extensive outcrop sampling, trenching (called "costean" by Parry), pitting, reverse-circulation ("RC") drilling, and diamond drilling. Diamond and RC drilling, as well as pit sampling and trenching, were also conducted in the deposit areas of Cibatu, Cikadu, and Sekolah. Most of the diamond drill holes ("DDH") conducted in the Project were completed by Parry between 1986 and 1990.

Another Australian company, Terrex Resources NL ("Terrex"), joined the exploration from 1990 to 1994. Work carried out by Terrex included RC drilling, percussion drilling, and some trenching (costean). The exploration was focused on the targets of Pasir Manggu, Cibatu, Cikadu, and Sekolah;

and resources in these areas were preliminarily estimated based on extensive sample results. During this time, Terrex started prospecting on other deposits in the project area.

An Australian-Indonesian joint venture, PT. Meekatharra Minerals ("Meekatharra"), conducted a detailed follow-up exploration in the project area from 1995 to 2000. Meekatharra reviewed and evaluated previous geological data, and additional exploration completed during this period included detailed geological mapping and additional sampling from trenches and pits, as well as evaluation diamond drilling. In the Ciaro porphyry copper-gold deposit area, a total of eight additional holes were drilled to further the geochemical and geophysical prospecting.

Geophysical prospecting including Induced Polarization ("IP") and a ground magnetic survey was conducted across the Pasir Manggu quartz veins in 2008. Wilton also completed some trenching and pitting as well as surface sampling in the Project area.

Of all the deposits, Pasir Manggu was considered the most advanced in terms of exploration, followed by Cibatu, Cikadu, and Sekolah. Feasibility study reports were prepared for the Pasir Manggu deposit in 1997 and 2010.

In 2012, Wilton completed a total of 17 DDHs to verify the historical data and explore the gold mineralization at Pasir Manggu West, Cikadu, Sekolah, and Cibatu. Core samples were prepared by the Intertek Laboratory in Jakarta and were analysed with fire assays.

To date, the major exploration work completed in the Ciemas Gold Project area consists of detailed geological and topographical mapping, geophysical and geochemical surveys, 360 costean/trenches/pits, 217 DDHs, 114 RC drillholes (reverse circulation hole or RCH), 7,500 hand auger drillholes, and 120 percussion drillholes.

Samples and Data Compilation

Samples from the Project were collected mainly from DDHs, RCHs, trenches, and pits. The compiled exploration database for Pasir Manggu, Cikadu, Sekolah, and Cibatu has been reviewed by SRK. For other properties of this Project, exploration is represented by trenching and pitting; however these data are insufficient for a JORC Code compliant resource review/estimation. The delineation of mineralized bodies for the Ciemas Project is based primarily on the drilling results. As the historical pitting and trenching data records are incomplete, the resource estimation in this Report only involves the DDH and RCH drilling.

Core and channel sampling comprised the primary sampling methods. The sampling grids were generally 20 m × 20 m (only in Pasir Manggu West), 40 m × 40 m, and 80 m × 80 m. Most of the DDHs were drilled with a dip angle of 60°. Drill cores were split into two halves and the basic sample length was around 1 m. Channel samples were collected from trenches and pits. Channel samples were about 1 m long.

Most of the drill cores were HQ-sized, which was considered adequate for splitting and sampling. In Pasir Manggu, a total of 691 core samples with an average length of 0.94 m were taken from 80 DDHs. In Cikadu, Sekolah, and Cibatu, a total of 1,290 core samples with an average length of 0.97 m were taken from 118 DDHs.

In Pasir Manggu West, a total of 769 samples with an average length of 1 m were taken from 64 RCHs. In Cikadu, Sekolah, and Cibatu, a total of 443 chip samples with an average length of 0.98 m were taken from 42 RCHs.

In Pasir Manggu West, a total of approximately 450 samples with an average length of 0.90 m were taken from 16 trenches and pits. Trenches and pits excavated in Cikadu, Sekolah, and Cibatu have not been compiled in a complete database for review.

The Ciemas Gold Project has been explored and evaluated with staged and separate works and by various companies or consultants, and historical data were not appropriately inherited during the changes of owners and stages. Data compilation and integration was performed by Wilton with its technical consultants prior to SRK's review. The samples were assayed by laboratories Kep Seksi Kimia Mineral, Inchcape Testing Service, and PT. Inchcape Utama Service. SRK sighted part of the original laboratory sample results for the historical exploration (all works conducted before 2008);

however, there were no detailed indications regarding the assaying methodology or QA/QC measures. To evaluate the reliability and accuracy of the historical sampling and assays, Wilton conducted verification drilling following SRK's recommendations made in March 2012.

Collar, survey, and sample data for 80 DDHs with a cumulative depth of 6,797 m and 64 RCHs with a cumulative depth of 3,295z m at Pasir Manggu were incorporated into the exploration database. The compiled database also contains 118 DDHs with a cumulative depth of 11,436.2 m and 42 RCHs with a cumulative depth of 2,011 m conducted at Cikadu, Sekolah, and Cibatu. SRK notes that additional exploration work has been completed in the Project area, but due to incomplete reviews, low data quality, or unverifiable sources, they have been excluded from the final database.

SRK inspected a number of drilling collars and surface trenches on site and reviewed drill logs. Drilling, logging, bulk density testing, sampling procedures, and data quality aspects were discussed and reviewed with Wilton staff.

Mineral Resources

Only the 4 Main Prospects, namely Pasir Manggu W (PSM), Cikadu (CKD), Sekolah (SEK) and Cibatu (CBT) will be the extent of the study and design for mining. Cibak and Cipancar have been estimated with Inferred Resources. The Mineral Resource Statement of Project as of 30 June 2018 are as below.

Table ES-1: Mineral Resource Statement for the 4 Prospects – as of 30 June 2018

Property	Type	Category	Resource (kt)	Au (g/t)	Au (kg)
Pasir Manggu West	Oxide	Indicated	109	7.2	783
		Inferred	36	5.6	200
	Fresh	Measured	100	7.3	731
		Indicated	380	7.3	2,776
		Inferred	206	4.7	975
Cikadu	Oxide	Indicated	81	6.2	496
		Inferred	20	6.9	134
	Fresh	Indicated	1,008	9.1	9,126
		Inferred	280	9.7	2,718
Sekolah	Oxide	Indicated	89	5.8	510
		Inferred	128	4.9	621
	Fresh	Indicated	612	9.6	5,869
		Inferred	326	8.3	2,689
Cibatu	Oxide	Indicated	129	6.2	794
		Inferred	78	3.0	233
	Fresh	Indicated	907	9.1	8,216
		Inferred	377	7.8	2,951
4 Prospects Total	Oxide	Indicated	407	6.3	2,583
		Inferred	261	4.5	1,188
	Fresh	Measured + Indicated	3,007	8.9	26,718
		Inferred	1,188	7.9	9,332
	Oxide + Fresh	**Measured + Indicated**	**3,415**	**8.6**	**29,301**
		Inferred	**1,449**	**7.3**	**10,520**
Cibak	Oxide + Fresh	Inferred	660	5.6	3,717
Cipancar	Oxide + Fresh	Inferred	450	5.6	2,520

Property	Type	Category	Resource (kt)	Au (g/t)	Au (kg)
Cibak & Cipancar Total	**Oxide + Fresh**	**Inferred**	**1,110**	**5.6**	**6,237**
4 Prospects + Cibak & Cipancar Total	**Oxide + Fresh**	**Measured + Indicated**	**3,415**	**8.6**	**29,301**
	Oxide + Fresh	**Inferred**	**2,559**	**6.5**	**16,757**

Note: Cut-off grades applied for Mineral Resource statement are 1.0 g/t Au for the 4 Prospects; and 2.5 g/t for Cibak and Cipancar.

kt – 1,000 tonnes.

Mineral Resources are not Ore Reserves and do not have demonstrated economic viability.

All figures are rounded to reflect the relative accuracy of the estimate. All composites have been capped where appropriate. Figures for Au metal in this table are estimated based on the resource tonnages and grades, and do not represent the exact amount of extractable metal for this Project. They should be treated differently from the expected production of gold bullion. The information in this Report which relates to Mineral Resource estimates is based on information compiled by Dr Anson Xu, and Mr Pengfei Xiao, employees of SRK Consulting China Ltd. Dr Xu, FAusIMM, and Mr Xiao, MAusIMM, have sufficient experience relevant to the style of mineralisation and type of deposit under consideration and to the activity which they are undertaking to qualify as Competent Persons as defined in the JORC Code (2012 Edition). Dr Xu and Mr Xiao consent to the reporting of this information in the form and context in which it appears.

Hydrology and Hydrogeology Assessment

The typical monsoon tropical climate is characteristic on this project site, and the rainy season is between November and April, and the rest of the year is dry season. The annual cumulative precipitation is ranged between 3,000mm and 4,000mm. No major watercourse traverses the project area, and only some seasonal creeks exist, such as creek Cikanteh, creek Cipipisan, creek Cireundeu, and creek Sanggarawa, ect. No permanent lakes or ponds exist in the project area. It seems that no historical flooding events occur on this site due to undulating terrain, and rainfalls do not cumulate there. Currently there is no site storm water management system, and all the precipitation either infiltrates into the ground or runs off to the surrounding areas, eventually flowing into the Pelabuhan Ratu bay around 10km in the west.

Based on the currently available information, the project area belongs to poor groundwater productivity areas with very low groundwater yield or without exploitable groundwater. It is likely that groundwater is recharged by rainfall infiltration within preferential surface area and moves downwards under steep hydraulic gradients and is then deflected laterally, most likely in the weathered breccia and saprolite zone. It seems that only a small portion of rainfall recharge is expected to continue moving downwards through the volcanic breccia and to recharge the regional groundwater flow system in the underground mining area. Based on the limited hydrogeological data, SRK estimated groundwater inflow into the underground mining area at up to 150m below ground surface with Dupuit equation. The inflow is estimated at 450 m^3/d at Pasir Manggu and 1,600 m^3/d at Cikadu, Sekolah, and Cibatu, and the maximum inflow is estimated at 675 m^3/d at Pasir Manggu and 2,400 m^3/d at Cikadu, Sekolah, and Cibatu respectively with a safety factor of 1.5. However, SRK is of opinion that a further comprehensive hydrogeological investigation is recommended within underground mining area to build a numerical groundwater model, based on the fact that most ore bodies are located below the water table.

Mining Method

The Feasibility Study proposed both open pit and underground mining methods will be used for the resource extraction, which contained 4 Prospects as mentioned above. Only Measured and Indicated Resource categories will be considered for mine planning. The top 20 meters of oxide ore zone will be mined by open-pit method, and the remained part of the ore bodies will be mined by using underground scheme.

Underground Mining

The underground design scope is 20m beneath the topography. To ensure the safety of mining operation, orebodies close to the surface within 20m (mainly oxide ore) are designed as crown pillars, which will not be recovered underground. Open cast is considered for recovering part of these resources, which is about 15 m from surface. The rest part of 5 m pillar will be remained as safety pillar left there.

The designed capacity of underground mine is totally 1,500 tpd.

The main access method is proposed trackless decline in CKD, SEK, and CBT complex, and railway incline shaft in PSM section. A trade-off study has been conducted by SRK. It is determined that the PSM section will be developed independently by using a main incline shaft for primary mine access, and other three prospects, i.e. CKD, SEK, and CBT, will be developed together as one system, with a main decline ramp at CKD and a supportive decline at CBT.

The horizontal access method is proposed truck drive in CKD, SEK, and CBT complex, and railway drift in PSM section related to main access method. The transportation of ore in the PSM section will consist of tracked locomotive cars from the stopes to the main incline shaft, and the cars hoisted through the incline to the surface, and then trucking to the ore processing plant. The transportation of ore from the 3 prospects will be from the stopes to the main decline by trucks, and then from the decline to the surface of CKD portal and then to plant by trucks.

Two types of cut and fill mining methods are proposed, which are mechanized cut and fill for CKD, SEK and CBT complex, and traditional cut and fill for PSM section. A back-filling facility and system is designed. It will use about 50% of tailings from the processing plant and cements for filling materials.

The supportive systems for underground mine will be designed according to the mining method and transportation method determined.

Based on the deposit occurrence length and the access system, the middle intake and two-wing exhaust strategy is proposed for both CKD, SEK and CBT complex, and PSM section. The fresh air flows through fresh air shaft and declines, flows over the work headings, then exhausted by the fan located in #1 and/or #2 ventilation shaft. The fresh air shaft is serve as ventilation shaft during construction period. In PSM section, the similar air flow from incline shaft, work headings, then exhausted via #3 and/or #4 ventilation shaft.

The principal method of backfill at the project is cemented full tailing fill. The cemented tailing fill could lower the capacity of tailing storage facility (TSF) than rock fill, particularly, it is not enough of this project if all tailing store in the TSF. The tailing is pumped from processing plant to backfill plant vertical sand silos. The cemented tailing prepared as 68% - 70% slurry in the backfill plant at the top of air shaft in SEK section. As the elevation and the length of ore bodies, the backfill material could not reach the stopes if no pumping. The slurry pumped underground via pipes in air shaft or surface pipes to CBT or PSM section.

The surface air compressor station is located near the air shaft in SEK, for the supply of demands in CKD, SEK and CBT complex, as well as PSM section. According to the requirements of mining technology, the main underground air consumption equipment is jack leg driller and shotcrete machine, with total air consumption of 157 m^3/min.

The project electricity is from Pelabuhan Ratu, which is about 40 km away from the site. The power supply system study battery limit starts from the high-voltage outlet cabinet (cable head) of the general step-down substation at the site, excluding the external power supply and the general step-down substation. The standard voltage of Indonesia national low-medium voltage: 20kV, 380/220V, AC voltage frequency of 50Hz. The demarcation point is the general step-down substations 20kV high-voltage outlet cabinet, which is in processing plant.

The summary of mining method criteria is presented in Table ES-2 below.

Table ES-2: Summary of Underground Mine Design Criteria

Item	Unit	CSC	PSM
Ore Reserve	t	2,291,000	437,000
Reserve grade	g/t	8.34	7.19
Gold contained	kg	19,110	3,140
COG	g/t	3g/t	3g/t
Coefficient of volumetric expansion of ore and rock	factor	1.6	1.6
Specific gravity of waste rock and ore	t/m^3	2.7	2.7
Mining method		MCAF	TCAF
Stope geometry (length by height)	m	100 by 20	100 by 40
Level height	m	20	40
Vein average width	m	4	3
Stope quantity	QTY	123	33
Recovery rate	%	88.8	92.5
Dilution rate	%	8.6	7.6
Stope development / stope mass	Standard m/kt	26	8
Mine access method		Trackless decline	Incline shaft
Horizontal level haulage method		Truck	Railway locomotive and tramcar
Rock mucking method		LHD	Rock loader
LOM length of vertical/decline/incline development	m	3,101	505
LOM length of horizontal development	m	9,187	1,677
Mining productivity	t/d per stope	100	60
Production capacity daily	t/d	1200	300
Production capacity annually	kt/a	396	99
LOM	a	8.5	7
Including: construction	a	1.5	1.0
ramp-up	a	1	1
full capacity	a	4	3
reducing	a	2	2
Working scheme	hour/shift/day	8/3/330	8/3/330
Engineering quantity for construction	X 1000 m^3	71	14

Open-pit Mining

A study on the open-cast mining for the mineral resources on the surface. In each of the mining section, at least one pit was designed to mine the resources.

Mining operation in open pits is to be outsourced to the contractor. The contractor will bring small size equipment to supplement the owner's equipment to move approximately 2,832 kt materials throughout the producing years. The hydraulic shovel proposed by SRK is 1.0 m^3. The trucks are same as that of underground operation, which has a nominated loading capacity of 15t.

The mine operation will involve just two procedures of loading and hauling. No drilling and blasting is necessary due to the materials to be moved are not consolidated hard rocks.

Mining starts at the top bench, then is driven down to the pit base bench by bench. No pushback is proposed due to short life of open pit mining. The active bench is 2m high with a bench face angle of 60 degrees. Each four benches will be vertically stacked to an 8m high bench at the final pit location with a bench face angle of 55 degrees and a berm width of 3m. Both the width of pit ramp and the minimum pit base are 8m.

Ore Reserves

The underground mine is based on a cut-off grade (COG) of 3.0 g/t Au. A selective mine unit (SMU) size of 4m×4m×2m (X×Y×Z) along each direction is suitable for this project. The designed mining methods is cut and fill ("CAF") mining method with two sub-types of mechanized CAF and traditional CAF with different mining equipment. One cut is designed 4m high and around 3-4m wide.

The proposed complex dilution and mining recovery for the underground mine design is presented in Table ES-3 below.

Table ES-3: Summary of Recovery and Dilution – Underground

Section	Mining Method	**Recovery Rate**	**Dilution Rate**
CSC	MCAF	**88.8%**	**8.6%**
PSM	TCAF	**92.5%**	**7.6%**

The open-pit mining is planned to start at the end of underground mining to maintain a throughput of 1,500tpa ore for processing plant. The safety pillar leaved at surface is about twenty meters thick, and it has been assumed that the cut and fill method (or MCAF) will be applied to underground mining operation. Usually, there will be void space occurrence in a stope below the surface safety pillar after several years shrinking of fill body. So, filling the stope again is required to support safety operation of open pit mining.

The marginal cut-off grade for open-pit Ore Reserves was calculated and rounding to 1.5g/t based on the parameters which are addressed in details of open pit parameters. SRK has finished the resource estimate for the Project using a user block size of 8m×8m×4m (X×Y×Z). Usually, the block size that applied to the resource estimate is not appropriate to be used directly to estimate the Ore Reserves.

The economically mineable part of the Measured Resources was converted into Proved Ore Reserves. The economically mineable part of the Indicated Resources was converted into Probable Ore Reserves, in compliance with the JORC Code 2012 Edition.

Table ES-4 shows a summary of the Ore Reserves estimated for the Project, based on the mineral resource model and studied modify factors, as at June 30, 2018. Reported Ore Reserve is within the limits of the Company's proposed mining license area. Please note that the reported Ore Reserves are included in the Mineral Resource.

Table ES-4: Summary of Ore Reserves, as of June 30, 2018

Section	Category	Reserve	Grade	Gold
		kt	g/t	kg
CKD	Probable	986	8.0	7,849
SEK	Probable	679	8.1	5,511
CBT	Probable	1,008	7.9	7,945
PSM	Probable	587	6.6	3,898
Total		3,260	7.7	25,203

Note: The information in this report which relates to Ore Reserve conversion is based on information compiled by Mr Falong Hu, MAusIMM, and Mr Qiuji Huang, FAusIMM, employees of SRK Consulting (China) Ltd. Both Mr. Huang and Mr. Hu have sufficient experience relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking to qualify as Competent Persons as defined in the JORC Code (2012 Edition). Mr. Huang supervised the work of Mr Hu. Mr. Huang and Mr. Hu consent to the reporting of this information in the form and context in which it appears.

kt – 1,000 tonnes

For the underground mines, at a BCOG of 3.0g/t au and including dilution material, the Project has 2.8 million tonne of Probable Ore Reserves averaging 8.2g/t Au or contains 23,233 kg of gold (see Table ES-5).

Table ES-5: Summary of Underground Ore Reserves, as of June 30, 2018

Section	Category	Reserve	Grade	Gold
		kt	g/t	kg
CKD	Probable	913	8.3	7,560
SEK	Probable	622	8.5	5,303
CBT	Probable	859	8.4	7,229
PSM	Probable	437	7.2	3,141
Total		2,831	8.2	23,233

Note: The information in this report which relates to Ore Reserve conversion is based on information compiled by Mr Falong Hu, MAusIMM, and Mr Qiuji Huang, FAusIMM, employees of SRK Consulting (China) Ltd. Both Mr. Huang and Mr. Wu have sufficient experience relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking to qualify as Competent Persons as defined in the 2012 Edition of the "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves". Mr. Huang supervised the work of Mr. Hu. Mr. Huang and Mr. Hu consent to the reporting of this information in the form and context in which it appears.

kt – 1,000 tonnes

For the open pitable ore reserves, at a BCOG of 1.5g/t Au and including dilution material, the open cast mine has 429 thousand tonnes of Probable Ore Reserves averaging 4.6g/t Au or contains 1,970 kg gold metal, as presented in Table ES-6 below.

Table ES-6: Summary of Open Pit Ore Reserves, as of June 30, 2018

Section	Category	Reserve	Grade	Gold
		kt	g/t	kg
CKD	Probable	73	4.0	289
SEK	Probable	57	3.6	208
CBT	Probable	149	4.8	716
PSM	Probable	150	5.0	757
Total		429	4.6	1,970

Note: The information in this report which relates to Ore Reserve conversion is based on information compiled by Mr Yonggang Wu, MAusIMM, and Mr Qiuji Huang, FAusIMM, employees of SRK Consulting China Ltd. Both Mr. Huang and Mr. Wu have sufficient experience relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking to qualify as Competent Persons as defined in the JORC Code (2012 Edition). Mr. Huang supervised the work of Mr. Wu. Mr. Huang and Mr. Wu consent to the reporting of this information in the form and context in which it appears.

kt – 1,000 tonnes

Mineral Resource and Ore Reserve summary tables prepared "in the form of Appendix 7D" in accordance with SGX Catalist Notice 4C are presented in Appendix 2 of this Report.

The summary of LOM schedule in yearly period is presented in Table ES-7 below.

Table ES-7: Summary of LOM Schedule

Mines	Year	UNIT	LOM Total	2018	2019	2020	2021	2022	2023	2024	2025	2026
Underground	CSC ROM	kt	2,394	-	-	245	413	415	416	404	378	124
	CSC GRADE	g/t	8.4	-	-	8.8	8.2	8.7	8.6	8.4	8.1	7.3
	CSC Gold Comtained	kt	20,093	-	-	2,167	3,380	3,632	3,571	3,385	3,055	903
	PSM ROM	kt	437	-	15	98	100	88	82	38	15	-
	PSM GRADE	g/t	7.2	-	9.2	7.5	7.0	6.8	7.1	7.5	6.4	-
	Gold Contained	kg	3,141	-	142.78	738	697	601	580	285	96	-
	Total ROM	kt	2,831	-	15	343	513	503	498	442	393	124
	Total GRADE	g/t	8.2	-	9.2	8.5	8.0	8.4	8.3	8.3	8.0	7.3
	Gold Contained	kg	23,233	-	143	2,905	4,077	4,233	4,151	3,670	3,151	903
Open pit	ROM	kt	429	-	-	-	-	-	-	32	97	299
	GRADE	g/t	4.6	-	-	-	-	-	-	4.0	3.8	4.9
	Gold Contained	kg	1,970	-	-	-	-	-	-	128	369	1,473
	WASTE	kt	4,120							600	1,169	2,351
	S/R	t/t	9.6							18.5	12.0	7.9
Total	**ROM**		**3,260**	**-**	**15**	**343**	**513**	**503**	**498**	**474**	**491**	**423**
	GRADE		**7.7**	**-**	**9.2**	**8.5**	**8.0**	**8.4**	**8.3**	**8.0**	**7.2**	**5.6**
	Gold Contained		**25,203**	**-**	**143**	**2,905**	**4,077**	**4,233**	**4,151**	**3,798**	**3,520**	**2,376**

The year for 2018 in the design is from July to December, 6 months.

Mineral Processing and Metallurgy

A series of metallurgical tests on various samples have been conducted. The finished tests include gravity separation, cyanidation leaching, flotation, roasting oxidation of concentrate, bacterial oxidation ("BIOX") of concentrate, cyanidation of flotation tailings, diagnostic leaching, ore crushability and grindability tests. Based on the results of these tests and Wilton's opinions, SRK and NERIN jointly studied the metallurgical production process and gold recovery, and designed the metallurgical plant. The designed processing flowsheet is as follows.

- Primary crusher, semi autogenous mill and ball mill ("SAB") constitute a comminution circuit for grinding the ore from maximum size of 500mm to P_{80}=75μm;
- Flotation process is applied to enrich the sulphide minerals into a gold concentrate;
- Roasting process is applied to oxidize the flotation concentrate for the successive gold extraction operation, and recover the SO_2 from sulphide mineral roasting to produce industrial sulphuric acid;
- Two separate carbon-in-leach ("CIL") processes are applied to extract gold from the flotation tailings and roasting calcine. Gold is leached out and absorbed into the active carbon;
- The gold loaded carbon is processed through desorption, electrowinning and carbon active regeneration. The deposited gold slime from electrowinning cells is dried, smelted, refining and casted to Doré gold bars ("DER");
- SO_2/air method is applied to process the CIL residue for cyanide detoxification; and
- The detoxified residue gravity to tailings storage facilities ("TSF").

The production facilities include ROM stockyard, primary crushing workshop, crushed ore stockyard, grinding and flotation workshop, CIL workshop, DER workshop, concentrate roasting workshop, sulfuric acid plant, TSF, laboratory, compressor station, reagent preparation workshop, maintenance workshop, etc. The designed throughput of the metallurgical plant is 495ktpta. It is scheduled to be constructed in two phases. Roasting workshop and sulfuric acid plant and roasting calcine CIL workshop are to be constructed in the second phase, and others are to be constructed in the first phase. The first-phase construction period is one and a half years, and the second-phase construction will be carried out after the production is started. The second-phase construction period will be two years. During the second phase of construction, the gold concentrate produced will be sold directly.

The designed flotation recovery rate is respective 81.50% and 20.0% for the underground primary ore and open cast oxide ore, the flotation concentrate grade is 30.00 g/t. The recovery rate of concentrate roasting-CIL is 93.00% and the CIL recovery of flotation tailings is 68.38% and 88.00% for primary ore and oxide ore, respectively. The recovery rate of DER process is 98.00%. The production plan within the life of mine is shown in Table ES-8.

Table ES-8: Metallurgical Production Scheme for LOM

Description	Unit	Total	Production Period							
			2019	2020	2021	2022	2023	2024	2025	2026
Floatation Feed	kt	3,260		359	495	495	495	495	495	426
Feed Grade	g/t	7.73		8.49	7.95	8.39	8.34	8.09	7.15	5.60
Gold contained in Feed		25,204		3,048	3,937	4,155	4,130	4,006	3,538	2,389
Floatation Recovery	%	76.69		81.50	81.50	81.50	81.50	80.00	74.78	43.25
Tonnage of Concentrate	kt	644		83	107	113	112	107	88	34
Gold Grade of Concentrate	g/t	30.00		30.00	30.00	30.00	30.00	30.00	30.00	30.00
Concentrate for Roasting	kt	229						100	95	34
Grades of Concentrate for Roasting	g/t Au	30.0						30.0	30.0	30.0
	% S	22.0						22.0	22.0	22.0
	% As	3.0						3.0	3.0	3.0
Tonnage of 98% Sulphuric Acid	kt	157.8						68.8	65.3	23.7
Tonnage of 98% As_2O_3	kt	9.3						4.0	3.8	1.4
Floatation Concentrate CIL-DER Recovery	%	91.1						91.14	91.14	91.14
Floatation Tailings CIL-DER Recovery	%	72.2		67.01	67.01	67.01	67.01	68.89	73.68	83.87
Gold from Roasting Concentrate by CIL-DER	kg	6,274						2,734	2,598	942
Gold from Flotation Tailings by CIL-DER	**kg**	**4,240**		**378**	**488**	**515**	**512**	**552**	**657**	**1,137**
Salable Gold Concentrate	**kt**	**415**		**83**	**107**	**113**	**112**	**-**	**-**	**-**
Gold Contained in Salable Concentrate	**kg**	**12,446**		**2,484**	**3,209**	**3,386**	**3,366**	**-**	**-**	**-**
Gold Doré Salable	**kg**	**10,514**		**378**	**488**	**515**	**512**	**3,286**	**3,255**	**2,079**
Total Gold Receovered (in bullion & concentrate)	**kg**	**22,959**		**2,862**	**3,697**	**3,901**	**3,878**	**3,286**	**3,255**	**2,079**

The feasibility designs for the metallurgical facilities are comprehensive and detailed with no principal defect. However, due to the gold refractory caused by complicated ores characteristics, the metallurgical process is relatively complicated. Although the process stated in this Report is technically stable and reliable and can achieve a high gold recovery rate, the capital and operating costs may still have some room to be refined. SRK recommends conducting supplementary process mineralogy researches to further analyse the reasons of refractory, in order to simplify and optimise the metallurgical process flowsheet, which would result reducing the capital and operating costs.

Safety

SRK has sighted the original Occupational Health and Safety ("OHS") officer appointment approval for the Ciemas Gold Project with its English translation. This approval was issued by the Department of Mining and Energy of the Regent of Sukabumi on 9 December 2011.

In addition, SRK reviewed some sections of the feasibility study reports with respect to the proposed OHS management measures for the project.

SRK notes that the project is still under construction, and therefore records of OHS statistics, such as the number and type of incident/accidents and associated injuries, have not yet been generated.

Environment, Social and Health

Environmental protection in Indonesia is governed by various laws, regulations and decrees, and non-compliance may result in fines and penalties and revocation of licenses and/or permits in extreme cases. Indonesia's Environment Law provides that an Environmental Impact Analysis (*Analisa*

Mengenai Dampak Lingkungan or AMDAL in Indonesian) is required for those activities which exploit natural resources and cause environmental pollution. An AMDAL (including an Environmental Impact Assessment (*Analisis Dampak Lingkungan* or ANDAL), an Environmental Management Plan (*Rencana Pengelolaan Lingkungan* or RKL), and an Environmental Monitoring Plan (*Rencana Pemantauan Lingkungan* or RPL)) was issued to the Ciemas Project in August 2010 and its approval by the Regent of Sukabumi is dated 16 August 2010. However, the proposed roasting and acid making system are not involved in the aforementioned AMDAL. The environmental impact assessment (AMDAL) for the Ciemas Project will be updated in due course to involve the roasting and acid making system. The management measures of As_2O_3 generated by the roasting system should be particularly studied in the updated AMDAL.

Some key environmental permits, which include Environmental License, Water Extraction Permit, Hazardous Waste (B3) Handling and Operations Permit, Hazardous Waste Location Permit, Hazardous Waste Transport Permit and Wastewater Disposal Permit, will be required as the project moves forward. Meanwhile, it must be pointed out that the permitting requirements should be considered case by case and may vary by the different stages of project development.

Based on the study, the major environmental risks for the development of the Ciemas Project comprise dust and gas emissions, noise emissions, water contamination, biodiversity degrade, soil erosion and cyanide management. The management measures to mitigate the adverse impacts proposed by the AMDAL are comprehensive and reasonable.

The operational Closure and Rehabilitation Plan has yet to be developed for the Ciemas Project. For the moment, total amount of Rp 500,000,000 has been deposited in the bank as a reclamation guarantee to local authority. The unit reclamation cost for the Ciemas Project is estimated to be Rp 10,000 per square meter. The estimated cost on mine closure is USD 3.4 million.

The public hearings, questionnaires dissemination and interviews were carried out to the public to understand the attitudes and expectations of the community to the project. At the time of this report completion, the Ciemas Project allocated USD 100,000 for the local villagers' relocation. Furthermore, 2% of gross profit will be spend on contribution to local responsibility. The shareholders show their concerns over the local labor recruitment, local community involvement, assist in and cooperation with surrounding communities and environmental protection. The development of the corporate social responsibility (CSR) and community development (CD), which identify the project's social responsibility and ensure the community engagement, is underway.

Capital Costs (CAPEX) and Operating Costs (OPEX)

The production capacity of the Ciemas Gold Project is designed as 495,000tpa of mining and ore processing (or 1,500t/d), as well as 65,000tpa flotation concentrate roasting and associated acid manufacture. According the construction plan by Wilton, the initial construction in 2018 will include the mining system and the ore processing system without considering the further processing of flotation concentrates, and the roasting system for further processing of the flotation concentrates as well as associated facilities, such as acid manufacture plant will be construction in 2023. Table ES-9 summarizes the initial capital expenditures for the project. Table ES-10 summarizes the sustaining CAPEX for the project.

Table ES-9: Summary of Initial CAPEX Required for the Ciemas Gold Project

Cost centres	Cost (1000 USD)	%
Mining	34,005.8	34.2%
Ore processing	19,090.7	19.2%
TSF	6,922.3	7.0%
Infrastructure	17,808.2	17.9%
Others	10,492.2	10.6%
Subtotal	88,319.0	88.9%

Contingency	11,039.9	11.1%
Total	**99,358.9**	**100.0%**

Table ES-10: Summary of Sustaining CAPEX Required for the Ciemas Gold Project (1000 USD)

Cost Centres	Development	Construction	Procurement	Installation	Other	Subtotal
Mining	7,269.1					7,269.1
Ore processing		628.1	2,151.9	540.6		3,320.6
Roasting		11,838.3	20,927.4	9,062.7		41,828.4
Infrstructure		3,125.8	1,043.1	283.1		4,452.0
Total Enginerring	**7,269.1**	**15,592.3**	**24,122.4**	**9,886.4**		**56,870.1**
Other costs					4,524.9	4,524.9
Engineering+Other	**7,269.1**	**15,592.3**	**24,122.4**	**9,886.4**	**4,524.9**	**61,395.0**
Contingency					7,674.4	7,674.4
Total Sustaining Capital	**7,269.1**	**15,592.3**	**24,122.4**	**9,886.4**	**12,199.3**	**69,069.4**

Note: Totals may not sum due to rounding

In addition to the CAPEX, there will be USD7 million as working capital for mining and processing which will be invested into the project in 2020, and USD 5 million as working capital for the roasting and acid manufacture in 2024, and the working capital will be recovered when closure of the project. It is estimated that the mine closure cost is USD 3.4 million.

The CAPEX, working capital, as well as mine closure cost will be invested into the project as scheduled in Table ES-11.

Table ES-11: Summary of CAPEX Investment Schedule for the Ciemas Gold Project (1000 USD)

Year	2018	2019	2020	2021	2022	2023	2024	2025	2026	Total
Mining	7,495.3	26,510.3	3,034.0	2,354.0	1,475.0	86.0				40,954.6
Ore processing	7,636.3	11,454.4				3,320.6				22,411.3
TSF	2,768.9	4,153.4								6,922.3
Infrastructure	7,123.3	10,684.9				4,452.0				22,260.2
Roasting and acid						41,828.4				41,828.4
Other cost	4,196.9	6,295.3				4,524.9				15,017.1
Contingency	3,652.6	7,387.3	379.3	294.3	184.4	6,776.5				18,674.2
Working Capital			7,000.0				5,000.0		(12,000.0)	-
Mine closure									3,400.0	3,400.0
Total	**32,873.2**	**66,485.6**	**10,413.3**	**2,648.3**	**1,659.4**	**60,988.4**	**5,000.0**		**(8,600.0)**	**171,468.1**

It is estimated that the annual direct operating expenses in the full production years will be USD14.45 million for mining, USD16.84 million for ore processing and smelter, USD2.61 million for roasting and acid making, USD0.392 million for public supportive workshops, USD8.21 million for G & A. Table ES-12 gives details.

Table ES-12: Average Direct Operating Expenses in Full Production years

Cost centre	Total Cost (1000 USD)	Unit cost (USD/t ore)
Mining	14,449	29.19
Processing and smelter	16,845	34.03
Roasting and acid making	2,611	5.27

Supportive infrastructure	392	0.79
General and administration	8,206	16.58
Total	**42,503**	**85.86**

It is proposed that the roasting and acid manufacture facility will be constructed later in 2023. From 2020 to 2023, the products of the project will be gold concentrates from flotation and gold dore from the leaching of flotation tailings. In the period, the operating expenses have been given in Table ES-13. The average operating cost is US$69.26/t ore.

Table ES-13: Average Direct Operating Cost in 2020-2023 (assumed full production)

Cost centre	Total Cost (1000 USD)	Unit cost (USD/t)
Mining	14,449	29.19
Processing and smelter	12,509	25.27
Supportive infrastructure	392	0.79
General and administration	6,935	14.01
Total	**34,285**	**69.26**

For the open pit operation, the mining and stripping will be contracted out, and the contractors will use their own machinery. The rates for mining and stripping, including transportation, are USD4.76/t ore, and USD1.47/t waste, respectively.

Economic Analysis

An economic analysis for Ciemas Gold Project is carried out using a discount cash flow approach on an after-tax basis. The internal rate of return (IRR) on total investment was calculated based on 100% equity financing. The Net Present Value (NPV) was calculated from the cash flow generated by the project based on a discount rate of 9%. The payback period based on the undiscounted annual cash flow of the project was also indicated as a financial measure. Furthermore a sensitivity analysis was also performed for the after-tax base case to assess the impact of variations of the project capital costs, operating costs and price of gold.

The economic analysis is on the purpose for Ore Reserve estimates only.

The general assumptions and parameters used for this analysis are summarized in Table ES-14 below.

Table ES-14：DCF Model Criteria

Description	Unit	Value
Volume		
Ore Reserve	kt	3,260
Au Grade	g/t	7.7
Processing Recovery Rate for Cons.	%	81.5
Processing Recovery Rate for Tailing	%	18.5
Conc. Roasting-CIL Recovery Rate (Incl. smelter)	%	90.8
Tailings CIL Recovery Rate（Incl. smelter）	%	68.7
Recovery Rate for Cons. Rost-CIL	%	74.0
Recovery Rate for Tailing CIL	%	12.7
Final Recovery Rate for ROM to Gold	%	86.7
Capacity and Schedule		
Mine Capacity	ktpa	from 400 to 511
Mill Capacity	ktpa	495
Mine Life	year	7
Construction	year	1.5

Description	Unit	Value
Volume		
Mine Economy		
Initial Capital	USD M	99.4
Sustaining Capital	USD M	68.7
Mine Closure	USD M	3.4
Working Capital	USD M	12.0
U/G Mining Cost	USD/t ROM	29.2
O/C Mining Cost	USD/t ROM	4.8
O/C Stripping Cost	USD/t rock	1.5
Processing and smelter Cost (Excluding Roasting)	USD/t ROM	25.3
Processing and smelter Cost (Including Roasting)	USD/t ROM	34.0
Roasting Cost	USD/t ROM	5.3
Supportive infrastructure Cost	USD/t ROM	0.8
G & A Cost Excl Roasting	USD/t ROM	14.0
G & A Cost Incl Roasting	USD/t ROM	16.6
Payable Rate for Cons. (before Roasting)	%	95
Refine Charge for Cons. (before Roasting)	USD/t Cons.	190
Royalty to Gold Revenue	%	3.75
Enterprise Income Tax	%	25

Table ES-15 below presented the projection of the DCF model of Ciemas Gold Project.

Table ES-15: Economic Analysis Summary

Item	Unit	Value
Base Case NPV @ 9%	USD M	180.3
Base Case IRR	%	43.0
Base Case Payback Years	Years from July 2018	3.4
NPV @ 4%	USD M	254.8
NPV @ 6%	USD M	221.8
NPV @ 8%	USD M	193.1
NPV @ 10%	USD M	168.1
NPV @ 12%	USD M	146.2
NPV @ 14%	USD M	126.9
NPV @ 16%	USD M	110.0

Table ES-16 below gives the results of NPVs compared to changes in CAPEX, OPEX, and forecast gold price.

Table ES-16: NPV (@9% discount rate) vs. CAPEX, OPEX and Gold Price (in USD Million)

Sensitivity	Revenue	OPEX	CAPEX
-25%	83.1	210.0	202.0
-20%	102.6	204.1	197.7
-15%	122.0	198.1	193.3
-10%	141.4	192.2	189.0
-5%	160.9	186.3	184.7
0%	180.3	180.3	180.3
5%	199.8	174.4	176.0
10%	219.2	168.4	171.6
15%	238.6	162.5	167.3

20%	258.1	156.5	163.0
25%	277.5	150.6	158.6

The changes in price of gold have the most significant effect on NPV.

SRK conducted a further sensitivity analysis on gold price to the Project and the results are presented in Table ES-17 below. The break-even price (NPV=0 at 9% discount) is around a change of -46.2% from the base scenario prices. On this basis SRK considers that the Project is economically feasible, and the conversion of the ore reserves is reasonable.

Table ES-17: Summary of Sensitivity on Gold Price to Project

Scenario	**Gold Price**	**NPV @9%**	**IRR**	**Payback Period**
Unit	USD/oz	USD M	%	Years
Base Case	1300-1220	180.3	43.0	3.4
Case -20%	1040-976	102.6	29.3	4.0
Case -10%	1170-1098	141.4	36.3	3.7
Case +10%	1430-1342	219.2	49.4	3.2
Case +20%	1560-1464	258.1	55.5	3.0
Case Break-even (-46.2%)	700-657	-	9.0	

Project Risk Analysis

Mining is a relatively high risk industry. In general, the risk may decrease from exploration and development to the production stage. The Ciemas Gold Project is an advanced exploration/development project with some historical open pit production, and risks exist in various areas. SRK considered various technical aspects which may affect the project's feasibility and future cash flow under the proposed production schedule, and conducted a qualitative risk analysis which has been summarised in the following table.

Table ES-18: Risk Assessment Table

Risk Source/Issue	Likelihood	Consequence	Risk Rating
Geology and Resource			
Lack of Significant Resource	Unlikely	Moderate	Low
Lack of Significant Reserve	Unlikely	Major	Medium
Unexpected Groundwater Ingress	Possible	Moderate	Medium
Mining			
Significant Production Shortfalls	Unlikely	Major	Medium
Low Production Pumping System Adequacy	Unlikely	Moderate	Low
Significant Geological Structure or Geotechnical Issues	Possible	Moderate	Medium
Excessive Surface Subsidence	Unlikely	Minor	Low
Poor Ground Conditions	Possible	Moderate	Medium
Poor Mine Plan	Possible	Moderate	Medium
Poor Stability of Backfilling System	Possible	Moderate	Medium
Ore Processing			
Lower Yields (output / raw ore)	Possible	Moderate	Medium
Lower Recovery	Possible	Moderate	Medium
High Production Cost	Likely	Moderate	Medium
Poor Plant Reliability	Unlikely	Moderate	Medium
Environmental and Social			
Land disturbance, rehabilitation and site closure	Possible	Moderate	Medium
Poor Water management (i.e. stormwater/surface water drainage – including any mine dewatering).	Possible	Moderate	Medium
Poor Waste rock stockpiling/ dumping management	Possible	Moderate	Medium
Land contamination (i.e. hydrocarbon storage and handling).	Possible	Moderate	Medium
Social aspects (i.e. local community interactions)	Possible	Moderate	Medium
Capital and Operating Costs			
Project Timing Delays	Possible	Moderate	Medium
Capital Cost Increases	Possible	Moderate	Medium
Operating Cost Underestimated	Possible	Moderate	Medium
Project Implementation			
Construction or Production Delay	Possible	Moderate	Medium

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Disclaimer

The opinions expressed in this Report have been based on the information supplied to SRK Consulting (China) Limited ("SRK") by PT. Wilton Wahana, Indonesia ("Wilton"). The opinions in this Report are provided in response to a specific request from Wilton to do so. SRK has exercised all due care in reviewing the supplied information. Whilst SRK has compared key supplied data with expected values, the accuracy of the results and conclusions from the review are entirely reliant on the accuracy and completeness of the supplied data. SRK does not accept responsibility for any errors or omissions in the supplied information and does not accept any consequential liability arising from commercial decisions or actions resulting from them.

1 Introduction and Scope of Report

In 2017, PT. Wilton Wahana Indonesia ("Wilton" or "the Company") engaged SRK Consulting (China) Limited ("SRK") to undertake a feasibility study to develop the mineral resources of the Ciemas Gold Project ("Ciemas"), and to provide Wilton, potential equity investors and possible future shareholders with an Independent Qualified Person's Report ("IQPR" or the "Report") compliance to JORC Code and to present a clear understanding of the project. Dr Anson Xu, the key author of this report, is an Australian Joint Ore Reserves Committee ("JORC") Code Competent Person ("CP") as well as Qualified Person ("QP") according to the Canadian professional designation.

The projects and operations at Ciemas are operated by Wilton. The IQPR may be included with documents to be submitted to the Singapore Exchange's ("SGX").

2 Background and Briefing

2.1 Background of the Project

Wilton is a lawfully registered corporation in Indonesia, focused on mineral development, mining and related commercial business. Wilton consolidated the previous exploration/mining tenements into mining concessions from 2008 to 2018.

SRK was commissioned by Wilton to review and report all relevant technical aspects of Wilton's exploration/mining properties in the Sukabumi Region, Republic of Indonesia. The mining concessions are currently wholly held by the Company.

Prior to this IQPR, SRK estimated the Mineral Resources and Ore Reserves for the Project as of 31 May 2013 and reported these Mineral Resources in accordance with the 2004 Edition of the JORC Code. This previous resource estimate by SRK was included in an independent qualified person's report (the "IQPR – 2013", full name: "Independent Qualified Person's Report for the Ciemas Gold Project, Ciemas, Sukabumi Region, Republic of Indonesia", June 2013) which forms part of the listing documents submitted to SGX in October 2013.

In 2014, an "Updated Resource Report for the Ciemas Gold Project in Sukabumi Region, Indonesia (effective date: 30 June 2014, the "Resource Report") was prepared by SRK, which has utilised data obtained from 16 additional drill holes. The report in 2014 updated Mineral Resource estimates in accordance with JORC Code (2012 Edition); whereas there were not any Ore Reserve stated by SRK then, due to the technical studies at the time were considered at a level lower than a "Preliminary Feasibility Study" as this term is defined in JORC Code (2012 Edition) as a minimum requirement for reporting Ore Reserves.

Since 2014, a total of 30 additional drill holes were completed for the purpose of comprehensive metallurgical tests on both oxidised and primary mineralisation, as well as in-fill exploration. Multiple additional exploration work, mining and metallurgical studies were conducted including topography, compilation mapping, trenching, geophysics, open pit and underground designes, metallurgical tests, and a processing plant design. A scoping study followed by a comprehensive feasibility study were completed.

2.2 Background of the Properties

The reviewed properties have been managed by several previous tenement holders, who have conducted prospecting and exploration works at various levels of detail from the 1980s to the present; the work still continues.

The licensed mining concession of the Ciemas project consists of the following major blocks: Pasir Manggu (West, Middle, and East), Cibatu, Sekolah, Cikadu, Cigombong, Cileuweung, Cibak, Cipancar, Cikole, Cipirit, Ciheulang and Japudali.

The tenement is currently in the stage of detailed exploration and pre-mining at Pasir Manggu West, Cikadu, Cibatu and Sekolah; and the Cibak – Cipancar zones has been preliminarily investigated and evaluated.

3 Program Objectives and Work Program

3.1 Purpose of the Report

The principal objectives of this Report is to provide Wilton and the potential equity investors and possible future shareholders and the SGX with an IQPR suitable for inclusion in documents that Wilton plans to submit to the SGX. The SRK report is proposed to provide the SGX and existing and potential shareholders of Wilton with an IQPR which provides an unbiased technical assessment of the risk and opportunities associated with the mining and processing assets of Wilton's Ciemas gold project.

3.2 Reporting Standard

This Report has been prepared as a "Qualified Person" report complying with the Listing Rules which referred to *Practice Note 4c, Disclosure Requirements For Mineral, Oil And Gas Companies,* of the SGX. The Report has also been prepared to the standard of an Independent Technical Assessment Report under the guidelines of the Valmin Code. The Valmin Code is the code adopted by the Australasian Institute of Mining and Metallurgy and incorporates the JORC Code for the reporting of Mineral Resources and Ore Reserves. The standard is binding upon all members of the Australasian Institute of Mining and Metallurgy ("AusIMM").

This Report is not a valuation report and does not express an opinion as to the value of mineral asset. Aspects reviewed in this Report do include product prices, socio-political issues, and environmental considerations; however, SRK does not express an opinion regarding the specific value of the assets and tenement involved.

3.3 Limitations Statement

SRK is not professionally qualified to opine upon and/or confirm that Wilton has exact percentage of ownership of its underlying tenements and/or has any unresolved legal matters relating to any transfer of ownership or associated fees and royalties. SRK has therefore assumed that there are no legal impediments regarding the existence of the relevant tenements and that Wilton has legal right to all underlying tenements as purported. Assessing the legal tenures and rights to the prospects of Wilton and/or any of its subsidiary companies are the responsibility of legal due diligence conducted by entities other than SRK.

3.4 Work Program

The work program involved five phases:

- Phase 1: Review of provided historical data and information, and a site visit to the Ciemas Gold Project near Ciemas, Sukabumi, Indonesia, in September 2017. Tasks include: inspection of the project sites, including general inspection of the prospects, drill-core storage, electricity and water supplies, current and proposed industrial sites, i.e. ore processing plant site, decline portals and mining industrial sites, tailings storage facility site, office and dormitory sites, etc.; discussion of issues with Wilton staff, collection of documents; and preliminary decision on the development scheme and method in different aspects;
- Phase 2: From September to October 2017, SRK proposed the development scheme and methods in different aspects with trade-off studies, and together with the Wilton personnel to determine the final development scheme and methods used in the feasibility study;
- Phase 3: From November 2017 to May 2018, SRK, together with China Nerin, a subcontractor by SRK, conducted the detailed study and design for the feasibility study of developing the deposits of Pasir Manggu West, Cibatu, Cikadu, and Sekolah, and submit a draft report for the review of the client.
- Phase 4: From June to July 2018, by considering the comments from the client, SRK finalized the feasibility study;

- Phase 5: From July to August 2018, SRK compiled the update of the Independent Qualified Person's Report to include the new findings of the feasibility study. This is a draft report, and will be finalized by considering comments and feedbacks from the client and other related third parties.

3.5 Project Team

The SRK team and their areas of responsibility are shown in Table 3-1.

Table 3-1: SRK Consultants, Title and Responsibility

SRK Personnel	Project Role
Dr Anson Xu	Corporate Consultant – chief compiler of updated report, and joint Competent/Qualified Person for Mineral Resources and Ore Reserves
Pengfei Xiao	Principal Consultant - Exploration, Resource Estimate and QA/QC Protocol and joint Competent Person for Mineral Resources
Yonggang Wu	Principal Consultant – Mining review for updating the report, joint Competent Person for Ore Reserves (open pit)
Falong Hu	Senior Consultant- Ore reserve conversion for updating report, joint Competent Person for Ore Reserves (underground)
Qiuji Huang	Principal Consultant – Review of Mining and Reserve Conversion and joint Competent Person for Ore Reserves
Lanliang Niu	Principal Consultant - Mineral Processing and metallurgic Assessment and joint Competent Person for Ore Reserves
Dr Yuanhai Li	Principal Consultant – Hydrogeology assessment
Nan Xue	Senior Consultant – Environmental and Social Assessment
Dr Yonglian Sun	Corporate Consultant- Internal Peer Review and Quality Control, geotechnical issues
Anne-Marie Ebbels	Principal Consultant- External Peer Review and Quality Control
Anthony Stepcich	Principal Consultant- External Peer Review and Quality Control – (cost and economic analysis)

Dr Anshun (Anson) Xu, *PhD, FAusIMM,* is a Corporate Consultant (Geology) who specialises in the exploration of mineral deposits. He has more than 30 years' experience in the exploration and development of various types of mineral deposits, including Cu-Ni sulphide deposits related to ultra-basic rocks, tungsten and tin deposits, diamond deposits, and especially deep expertise in various types of gold deposits, including vein-type, fracture-breccia zone type, alteration type, and Carlin type. He was responsible for the resource estimations of several diamond deposits and for reviews of resource estimations for several gold deposits. He recently completed for clients from both China and overseas several due diligence projects, including technical review projects, such as Canadian NI43-101 reports and Hong Kong Stock Exchange (HKEx) IPO technical reports. ***Dr Xu is the main Competent/Qualified Person and chief compiler of the updated Report.***

Pengfei Xiao, *MSc, MAusIMM,* is a Principal Consultant (Geology) with a specialty in mineral exploration through application of comprehensive geological and geophysical methods and whose expertise also includes resource modelling and estimation. He is familiar with both theory and practice in sampling, sample preparation, and chemical analysis. He has over 10 years' experience in the area of geology and resource evaluation and has been active in over 100 projects, including due diligence reviews, exploration design, data verification, and resource estimation. His experience pertains to precious metal (Au, Ag, and PGE), base metal (Cu, Ni, Pb, and Zn), and other metal deposits (Fe, Mn, V, Mo, and Co) and also includes a few non-metal projects (phosphorite, potash, and gypsum). ***Mr Xiao reviewed and assessed the geology including the resource of the project.***

Yonggang Wu, *MEng, MAusIMM,* is a Principal Consultant (Mining), joining SRK in 2007 after his graduation from the Jiangxi University of Science and Technology. He has acquired specialised knowledge of mining engineering and MineSight software and has been involved in a large number of projects to date. He has accumulated extensive experience in resource/reserve estimation, pit limit optimisation and design, underground-mining design, long-term production planning, and due diligence studies, with minerals including Au, Pb, Zn, Mn, Cu, Fe, fluorite, potassium salts, alum, and phosphorus among many others. Yonggang has expertise in geological and mining modelling and is proficient in using MineSight, AutoCAD, and other specialised software packages. ***Mr Wu conducted the pit optimization and conversion of ore reserves (for open pits) under the supervision of Dr Xu.***

Falong Hu, ***B.Eng, MBA, MAusIMM,*** is a Senior Consultant (Mining). He has a Bachelor's degree in Mining Engineering from Central South University. Before joining SRK he worked as an on-site and head office mining engineer at Sino Gold Mining Limited (which later merged with Eldorado Gold Corp.) and Silvercorp Metals Inc. He is familiar with underground mine production systems and has been involved in mine design, scheduling, and development; underground mining production; longhole blasting; rock mechanics; ventilation; back-fill; and cost accounting. He is also proficient in digital modelling using Gencom Surpac. ***Mr Hu did the mine modelling and conversion of ore reserves (for underground mines) under the supervision of Dr Xu.***

Qiuji Huang, ***BEng, FAusIMM, MCSM, MCGA,*** is a Principal Consultant (Mining). Prior to joining SRK, he was the technical department manager for a number of gold mines in southwest China, where he was responsible for mine development and mining design. He later joined the Gold Administration Bureau of Guangxi Province and the Guangxi Branch of National Gold, where he was in charge of review, purchasing, planning, and production management. Qiuji has more than 33 years of mining experience, including deposit development and planning, open-pit mining, underground mining, mine design, and consultation, with the commodities involved ranging from precious metals (Au, Ag) to non-ferrous metals (Cu, Zn, Pb, W, Mo), ferrous metals (Fe, Mn), and other metal deposits as well as non-metallic deposits formed under different conditions (such as U, K, S, coal, and stone). He also has experience with mine technology and review, mine construction, production testing, mine management, among other areas. Since joining SRK, Qiuji has worked in more than 20 countries—spanning Asia, Africa, and South America—and has been involved in many due diligence studies, including CNNC, CITIC DAMENG, CNMC, DAYE Non-Ferrous Metal Mining, and Hengshi Mining Investments, all of which have been listed successfully on the Hong Kong Stock Exchange. ***Mr Huang reviewed the mining section.***

Lanliang Niu, ***BEng, MAusIMM, MCAMRA,*** is a Principal Consultant (Processing) with SRK Consulting China Ltd. He has over 30 years' experience in processing testing and studies, production management and technical consultancy service. Lanliang is actively involved with the new development and application of processing technologies, facilities, and reagents and has received two national awards for his achievements in this area. Since joining SRK, he has been involved in hundreds of independent technical review projects for fund raising and acquisition and has accumulated profound experience on technical review of mining project. ***Mr Niu reviewed the mineral processing and metallurgical sections.***

Andy Li, ***PhD, MAusIMM,*** is a Principal Environmental Consultant with SRK Consulting China Ltd. Having graduated with a doctoral degree in Environmental Engineering from the Florida State University, he has over 12 years' experience in the environmental engineering field and has worked in various environmental projects in the USA, China, Mongolia, and a number of South Asian countries. He has particular expertise in environmental due diligence reviews, environmental compliance, and impact assessments for mining, mineral processing, refining, and smelting; in contaminated-site assessments and remedial design; in wetland and landfill rehabilitation; and in environmental-risk assessment. He also has extensive experience in water/wastewater treatment design, water distribution systems, and storm water management system design. ***Dr Li reviewed the hydrogeological, environmental and social aspects.***

Nan Xue, ***MSc, MAusIMM,*** is a Senior Consultant (Environmental) at SRK China. He holds a master's degree in Environmental Science from Nankai University, in Tianjin. He has eight years' experience in environmental impact assessment, environmental planning, environmental management, and environmental due diligence. He has been involved in a number of large EIA projects and pollution source surveys for SINOPEC as well as in the environmental-planning project funded by UNDP. He has particular expertise in construction project engineering analysis, pollution source calculation, and impact predictions. He also has an acute understanding of equator principles and International Finance Corporation environmental and social performance standards. After joining SRK, Nan has been involved in a number of IPO and due diligence projects in China, Laos, Russia, Mongolia, Philippines, and Indonesia; these projects include the Fuguiniao Mining project, Zijin Mining project, Hanking Mining project, and Future Bright Mining project. Mr Xue reviewed the environmental and social aspects.

Dr Yonglian Sun, *BEng, PhD, FAusIMM, FIEAust, CPEng,* is the Managing Director of SRK China and a Corporate Consultant (Geotech) with over 25 years' experience in geotechnical and mining engineering in five countries across four continents. He has extensive international mining experience with an emphasis on site investigation, analysis, and modelling of geotechnical issues in open pits, underground mines, and civil tunnels. He also possesses considerable experience in evaluating mining projects. In recent years, Yonglian has coordinated and led dozens of due diligence projects, most of which have been successfully listed in the Stock Exchange of Hong Kong Limited. ***Dr Sun reviewed the geotechnical aspect and also provided the internal peer review for the Report.***

Anne-Marie Ebbels, *BEng (Mining), MAusIMM (CP),* is a Principal Consultant (Mining) with SRK Australasia. Anne-Marie is a mining engineer with over 20 years' experience in mining operations and consultancy in Australia and overseas. Her expertise includes mine design, scheduling, drill and blast, economic modelling, supervision and contract management. Anne-Marie has significant practical experience in mine planning and scheduling using 5D Planner and EPS. Consulting experience includes scoping, pre-feasibility and feasibility studies, technical reviews, due diligence, economic modelling and site support. Anne-Marie has mining experience in open stoping, narrow vein mining, caving and drift and fill mining. Anne-Marie is a competent person for JORC and NI 43-101 Reporting and has completed numerous Ore Reserve reports for lead-zinc, copper and gold deposits. ***Ms Ebbels peer reviewed the feasibility study compiled in 2018.***

Anthony Stepcich, *BEng, MSc, FAusIMM (CP),* is a Principal Mining Engineer at SRK Australasia with over 20 years of mining industry experience, covering both underground and open-pit metalliferous mining, and open-pit coal mining. Anthony has postgraduate qualifications in finance and economics and specialises in open-pit design and scheduling as well as in project evaluation. ***Mr Stepcich peer reviewed the techno-economic model for the Project.***

3.6 Qualified Person Statement

As the author of portions of and chief compiler of the Report for Wilton on certain mineral properties in Sukabumi Region, Republic of Indonesia, I, Anshun (Anson) Xu, do hereby certify that:

- I am a Corporate Consultant in Geology and Mineral Resources, and the partner and division director of SRK Consulting (China) Limited ("SRK") with an office at:
  - B1205 COFCO Plaza, 8 Jianguomen Nei Dajie, Beijing, the People's Republic of China, 100005
  - Phone: 86-10-6511 1000
  - Fax:86-10-85120385
  - Email: axu@srk.cn

- I graduated with a Bachelor's degree in Geology of Mineral Deposits from Nanjing University, China (B.Sc.) in 1982, a Master's degree in Geology of Mineral Deposits from Chengdu University of Technology, China (M.Sc.) in 1988, and a Doctoral degree in Geology from University of Nebraska-Lincoln, USA (Ph.D.) in 1996.

  I have practiced my profession since 1982. From 1982 to 1990 I worked in teaching geochemistry and geology of ore deposits in Chengdu University of Technology. From 1990 to 1996, I worked in University of Nebraska-Lincoln in teaching and researching assistance; and from1996 to 2004 I worked in Canadian mining companies as chief geologist, and since 2005 I worked in mining consulting business in SRK. I worked in exploration management, resource estimates, ore reserve conversion and technical review and reporting, as well as feasibility study, for various types of mineral deposits, including gold, silver, iron, copper, nickel, cobalt, lead-zinc, diamond, bauxite, and others located in China, Canada, Mongolia, Kazakhstan, Indonesian, Philippines, North Korea, Congo (King), Cameron, Madagascar, and Peru, etc. I authored/co-authored several technical reports for IPO/RTO listing in Toronto Stock Exchange, Hong Kong Stock Exchange, and Singapore Stock Exchange.

- I am a Fellow of the Australasian Institute of Mining and Metallurgy (FAusIMM) (No. 224861) since 2005, and in a good standing.
- I have read the definition of "Competent Person" set out in the JORC Code and certify that by reason of my education, affiliation with a professional association and past relevant work experiences, I fulfil the requirements to be a "Competent Person" for the purposes of JORC.
- I have visited the Ciemas Gold Deposit during the period of 27 - 30 March 2013, 5-10 September 2017.
- I am the primary author being responsible for updating this technical report and the full content of this report.
- I have had the involvement with the Wilton's projects for preparing the previous technical reports, but I have no interest, nor do I expect to receive any interest, either directly or indirectly, in the Wilton's. Project, nor in the securities of Wilton and/or its subsidiary mining companies.
- I am not aware of any material fact or material change with respect to the subject matter of the Technical Report that is not reflected in the Technical Report, the omission to disclose which makes the Technical Report misleading.
- I am independent of the issuer applying all of the tests described in JORC.
- I have read the JORC and VALMIN Codes, and the Technical Report has been prepared in compliance with these codes.
- I am an independent Competent Person as this term is defined by the JORC Code and is also an independent Qualified Person as this term is defined by NI 43-101 and is used by the SGX.

Mr Pengfei Xiao, Mr Qiuji Huang, Mr. Yonggang Wu, Mr. Falong Hu, Mr. Lanliang Niu, Dr. Andy Li, Mr. Nan Xue, and Dr Yonglian Sun are also independent Competent Persons on resource, mining and reserve, ore processing, environmental and social issues and overall quality control. Their qualifications have been outlined in the short biographical noted above.

3.7 Statement of SRK's Independence

Neither SRK nor any of the authors of this Report have any material present or contingent interest in the outcome of this Report, nor do they have any pecuniary or other interest that could be reasonably regarded as being capable of affecting their independence or that of SRK.

SRK has no prior association with Wilton in regard to the mineral assets that are the subject of this IQPR. SRK has no beneficial interest in the outcome of the technical assessment being capable of affecting its independence.

SRK's fee for completing this Report is based on its normal professional daily rates plus reimbursement of incidental expenses. The payment of that professional fee is not contingent upon the outcome of the Report.

Neither SRK's staff nor any authors of this report have any direct or indirect interest in any assets which had been acquired, or disposed of by, or leased to any member of the Company or any of the Company or any of its subsidiaries within the two years immediately preceding the issue of this transaction.

Neither SRK nor any of the authors of this report have any shareholding, directly or indirectly, in any member of the Group or any right (whether legally enforceable or not) to subscribe for or to nominate persons to subscribe for securities in any member of the Group.

3.8 Representation

Wilton represented to SRK that full disclosure has been made of all material information and that, to the best of their knowledge and understanding, such information is complete, accurate, and true. SRK has no reason to doubt the representation.

3.9 Indemnities

As recommended by the VALMIN Code, Wilton has provided SRK with an indemnity under which SRK is to be compensated for any liability and/or any additional work or expenditure resulting from any additional work required:

- Which results from SRK's reliance on information provided by Wilton or due to Wilton not providing material information; or
- Which relates to any consequential extension workload through queries, questions, or public hearings arising from this Report.

3.10 Consents

SRK consents to this Report being included, in full, in documents that Wilton proposes to submit to the SGX, in the form and context in which the technical assessment is provided, and not for any other purpose.

SRK provides this consent on the basis that the technical assessments expressed in the Summary and in the individual sections of this Report are considered with, and not independently of, the information set out in the complete Report and the Cover Letter.

3.11 SRK's Experience

SRK Consulting is an independent, international consulting group with extensive experience in preparing independent technical reports for various stock exchanges around the world (see www.srk.com for a review). SRK is a one-stop consultancy offering specialist services to mining and exploration companies for the entire life cycle of a mining project, from exploration through to mine closure. Among SRK's more than 1,500 clients are most of the world's major and medium-sized metal and industrial mineral mining houses, exploration companies, banks, petroleum exploration companies, agribusiness companies, construction firms and government departments.

Formed in Johannesburg, South Africa, in 1974 SRK now employs more than 1,600 professionals internationally in 50 permanent offices on six continents. A broad range of internationally recognized associate consultants complements the core staff.

SRK Consulting employs leading specialists in each field of science and engineering. Its seamless integration of services, and global base, has made the company a world's leading practice in due diligence, feasibility studies, and confidential internal reviews.

The SRK Group's independence is ensured by the fact that it holds no equity in any project and that its ownership rests solely with its staff. This permits the SRK Group to provide its clients with conflict-free and objective recommendations on crucial judgment issues.

SRK Consulting has been active in China since 1999 and SRK Consulting (China) Ltd. was established in 2005. In January 2009, SRK China opened its branch office in Nanchang (capital of Jiangxi Province) focusing on mine design and pre-feasibility study projects. So far, we are mainly working on Chinese/Asian mining projects independently or together with SRK's other offices, mainly SRK Consulting Australasia. We have prepared dozens of independent technical reports on mining projects for various companies who acquired Chinese projects or completed public listings on stock exchanges with a summary of list as shown in Table 3-2.

Table 3-2: Recent Reports to HKEx by SRK

Company	Year	Nature of Transaction
Yanzhou Coal Limited (listed in HKEx)	2000	Sale of Jining III coal mine to the listed operating company
Chalco (Aluminum Corporation of China)	2001	Listing on HKEx and New York Stock Exchange
Fujian Zijin Gold Mining Group	2004	IPO Listing on HKEx
Lingbao Gold Limited	2005	IPO Listing on HKEx
Yue Da Holdings Limited (listed in HKEx)	2006	Acquisition of shareholding in mining projects in Yunnan, China
China Coal Energy Company Ltd (China Coal)	2006	IPO Listing on HKEx
Sino Gold Mining Limited	2007	Dual Listing on HKEx
Xinjiang Xinxin Mining Industry Co., Ltd	2007	IPO Listing on HKEx
Kiu Hung International Holding Limited	2008	Acquisition of shareholding in coal projects in Inner Mongolia, China
Hao Tian Resource Group Limited	2009	Very Substantial Acquisition of two coal mines in Inner Mongolia, China
Green Global Resources Holdings Ltd	2009	Acquisition of shareholding in one iron project in Mongolia
Ming Fung Jewellery Group Holdings Ltd	2009	Acquisition of shareholding in gold project in Inner Mongolia, China
Continental Holdings Limited	2009	Acquisition of a gold project in Henan, China
North Mining Shares Company Limited	2009	Acquisition of a molybdenum mining project in Shaanxi, China
CNNC International Ltd	2010	Acquisition of an uranium mine in Africa
Sino Prosper Mineral Products Ltd	2010	Acquisition of shareholdings in one gold project in Inner Mongolia, China
New Times Energy Corporation Ltd	2010	Acquisition of shareholding in gold projects in Hebei, China
United Company RUSAL Limited	2010	IPO Listing on HKEx
Citic Dameng Holdings Limited	2010	IPO Listing on HKEx
China Hanking Holdings Limited	2011	IPO Listing on HKEx
China Daye Non-Ferrous Metal Mining Limited	2012	Very Substantial Acquisition on HKEx
China Nonferrous Mining Corporation Limited	2012	IPO Listing on HKEx
Hengshi Mining Investments Limited	2013	IPO Listing on HKEx
Future Bright Mining Holdings Limited	2014	IPO Listing on HKEx
King Stone Energy Group Limited	2014	Acquisition of Shareholding in silver mines in Fujian, China
Agritrade International Pte LTD	2015	Acquisition of Shareholding in one coal mine in Indonesia
China Unienergy Group Limited	2016	IPO Listing on HKEx

3.12 Forward-Looking Statements

Estimates of resources, reserves, and mine production are inherently forward-looking statements, which being projections of future performance will necessarily differ from the actual performance. The errors in such projections result from the inherent uncertainties in the interpretation of geologic data, in variations in the execution of mining and processing plans, in the inability to meet construction and production schedules due to many factors including weather, availability of necessary equipment and supplies, fluctuating prices, ability of the workforce to maintain equipment, and changes in regulations or the regulatory climate.

The possible sources of error in the forward-looking statements are addressed in more detail in the appropriate sections of this report. Also provided in the report are comments on the areas of concern inherent in the different areas of the mining and processing operations.

4 Regional Description

4.1 Regional Location and Access

Administratively, the Ciemas deposit area is located in the Jampang Kulon area, in the southwestern part of the Sukabumi Region, West Java Province, Republic of Indonesia, about 200 km south of Jakarta.

An expressway connects Jakarta and the city of Bogor (55 km), from where a secondary paved road leads through Sukabumi to the coastal city of Pelabuhan Ratu, from where access to the mine and exploration area is provided by 45 km of paved asphalt road. Generally, access to the area is convenient. However, the road deteriorates as it approaches the mine. Figure 4-1 shows the regional and local location of the project area.

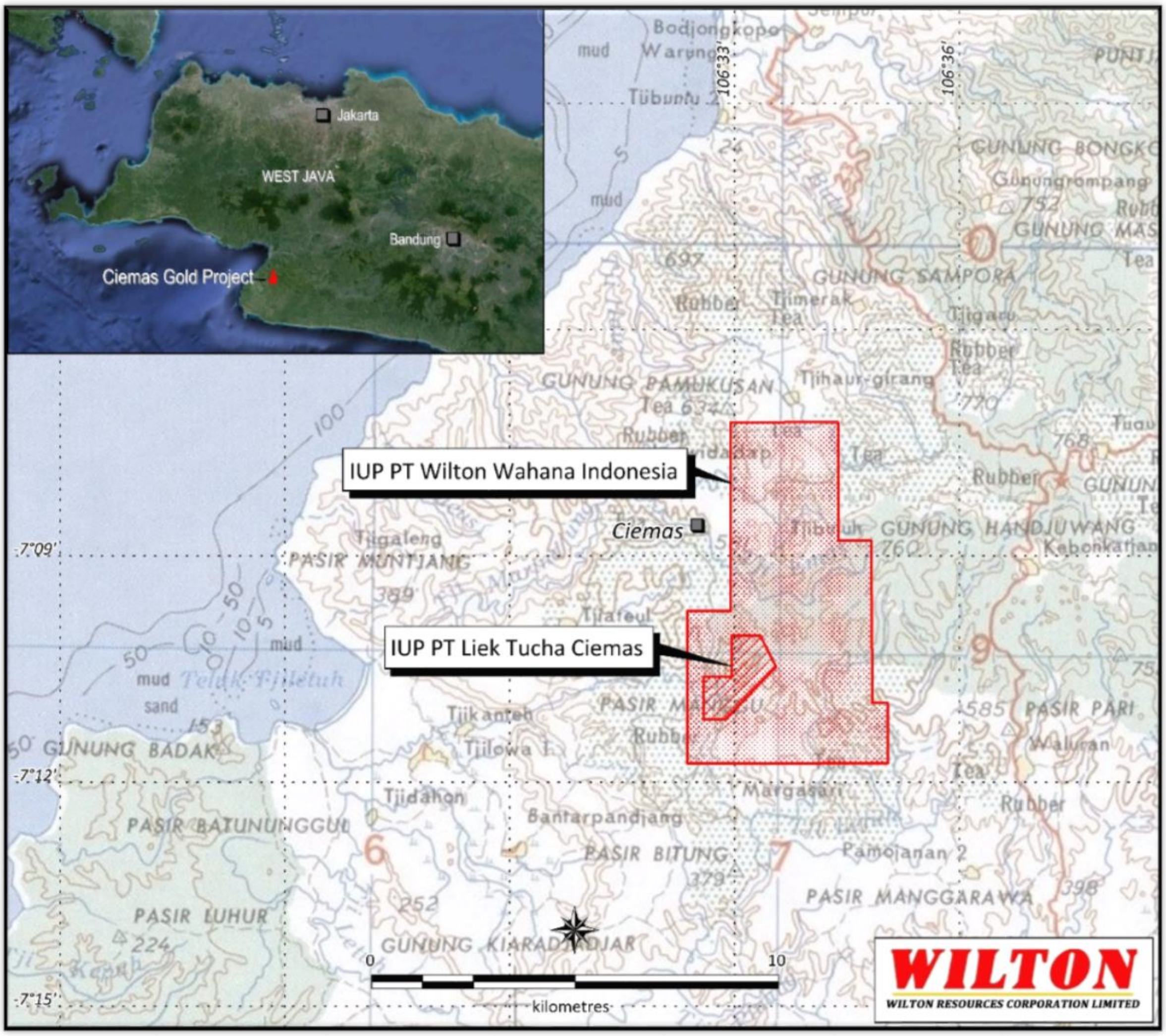


Figure 4-1: Project Location in Sukabumi Region, Indonesia

4.2 Topography and Climate

The landform of the exploration and mining area is represented by an undulating terrain with elevations varying from 379 to 760 m above sea level ("ASL"), generally with the lower parts in the southern areas.

The typical monsoon tropical climate is characteristic of the West Java province; the year has two seasons, dry and rainy. The temperature is stable year round, remaining between 18° and 28°C day and night. Precipitation is nearly 4,000 mm per annum, mostly concentrated between November and April, which is the rainy season.

Water resources are abundant and the level of groundwater is high. Most of the ore bodies are located below the groundwater table. Sukabumi has a tropical monsoon climate, with hot weather, thick soil layers, and dense vegetation.

4.3 Regional Economy and Infrastructure

The project is located in an impoverished mountainous area. The local economy is based mainly on agriculture. Main crops include rice, bananas, corn, and papayas, and plantations of cloves, rubber, and tea are also common.

Presently the power supply is via the local grid; generators are another major source of electricity. A large-scale power station and port project are under construction in Pelabuhan Ratu, about 12 km in a straight line from the mine site.

The water supply is sufficient due to the extremely well-developed river system and high levels of precipitation; water pools and elevated tanks are available on the mine site.

Wilton is one of the few mining enterprises in the Ciemas area; in some places local people pan gold from strongly altered volcanic rock outcrops and soils.

The Indonesian government is focused on attracting investment and increasing employment opportunities. Wilton intends to recruit a majority of project employees from the local population.

5 Operational Licences and Permits

5.1 Business Licences

SRK has sighted two original business licenses, one for PT. Wilton Wahana Indonesia and one for PT. Liek Tucha Ciemas ("Liek Tucha"). SRK has also sighted an original supporting document with its translation indicating that the Company owns 95% of Liek Tucha. Details of the business licences for the Ciemas Project are presented in Table 5-1.

Table 5-1: Business Licences

Business Licence No.	Issued To	Issued By	Issue Date	Expiry Date	Business Activities	Type of Goods/Service
00363/P-01/1.824.271	Pt. Wilton Wahana Indonesia	Industry and Trade Service, Jakarta Special Capital Region Province	3-Apr-13	/	Trading	Electronics, Mine (Coal, Mineral), Telecommunication
503.17/3106/380/10-22/PM.Herr-BPMPT/2013	Pt. Liek Tucha Ciemas	Integrated permit service agency, Government of Sukabumi Regency	12-Feb-13	/	Trade of Goods	Mining of Gold

SRK should note that according to the new regulation obligated in Indonesia at the time of this FS Report, a business license no longer must have an expiry date, as long as the company's business is legally active, therefore the business licences listed above for this Project are automatically extended and valid for life.

5.2 Mining Licences

Indonesian national law on mining, *Mineral and Coal Mining (No.4 of 2009)* (the "Mining Law"), allows the issue of mining licences under the following three categories:

- **Mining Business Licence** – called an *Izin Usaha Pertambangan* ("IUP") in Indonesian, a general mining licence issued to specific companies conducting mining business activities within a Commercial Mining Business Area – a mining area for larger scale mining, called a *Wilayah Usaha Pertambangan* ("WUP") mining area.
- **Special Mining Business Licence** – *Izin Usaha Pertambangan Khusus* ("IUPK"), a licence issued to specific companies conducting mining business activities within a specific State Reserve Area – a mining area reserved for the national strategic interest, called a *Wilayah Pencadangan Negara* ("WPN") mining area.
- **People's Mining Licence** – *Izin Pertambangan Rakyat* ("IPR"), a licence granted only to Indonesian citizens/invertors conducting mining business of a limited size and investment, within a People's Mining Area – a mining area for small scale local mining, called a *Wilayah Pertambangan Rakyat* ("WPR") mining area.

Two IUPs have been issued for the Ciemas Project, one for the Company and the other for P.T.Liek Tucha Ciemas. SRK has sighted these two original IUPs with their respective English translations. The details of these IUPs are summarised in Table 5-2, and the approximate mining areas are depicted in Table 5-2.

Table 5-2: Ciemas Project IUPs

IUP No.	Issued To	Issued By	Issue Date	Expiry Date [1]	Area (km^2)	Mining Type
503.8/7797-bppt/2011	Pt. Wilton Wahana Indonesia	Integrated Licensing Services Board Administration of Sukabumi District	5-Oct-11	7-Sep-30	28.785	Construction, production, transportation, and sale, as well as processing and purification (gold mine)
503.8/3106-bppt/2012	Pt. Liek Tucha Ciemas	Integrated Licensing Services Board Administration of Sukabumi District	8-May-12	4-Jan-28	2.00	Construction, production, transportation, and sale, as well as processing and purification (gold mine)

[1] Can be extended 2 times (twice) based on mining commodity pursuant to Law No. 4 of 2009.

SRK notes that the common standard conditions for the Ciemas Project IUPs include the following key technical items:

- The companies have the right to implement the project's "Production Operation" which is defined as including "construction, production, processing, purification, and transportation and sales".
- The companies have the right to utilise the general facilities and infrastructure for IUP Production Operation activity.
- The companies must appoint a "head of technical mine" (mining technical manager) responsible for the IUP production operation, and the mining environmental, health and safety management.
- The companies must submit the initial annual project Work Program and Budgets (called *Rencana Kerja dan Anggaran Belanja* or "RKAB" in Indonesian) to the Head of the Sukabumi District not more than 60 (sixty) working days after the issuance of the IUP. The follow up RKABs are to be submitted in November of each year.
- The companies must submit a "reclamation plan" and "post mining plan" (no dates are provided).
- The reclamation warranty (rehabilitation guarantee) is to be assigned before commencement of production.
- The mining security closure (post-mine guarantee) must be reserved.
- The companies must submit the Mine Closure Plan (*Rencana Penutupan Tambang* or "RPT") two years before the end of production activities.
- The companies must provide the agreed-upon compensation to the "rights holder of the land and forest enforcement" that has been disturbed by IUP production operation.
- The companies are required to construct all relevant project related infrastructure, including transport (ports, railways, roads), communications, power/water supply facilities, and accommodation and social support facilities (including waste treatment facilities).

It has also noticed that as per the regulation of the Indonesian government "Implementation of Mineral And Coal Mining Business Activities" No. 23 of 2010, and amended by No. 24 of 2012, 21, Chapter IX Article 97 (1) stating that " Mining permit holders and Special Mining Permit holders in the scope of foreign investment must upon 5 (five) years of production divest shares in stages", such that in the tenth year at least 51 % (fifth-one percent) of their shares shall be owned by the Indonesia participants. This could impact on the shareholding structure/status of the Project.

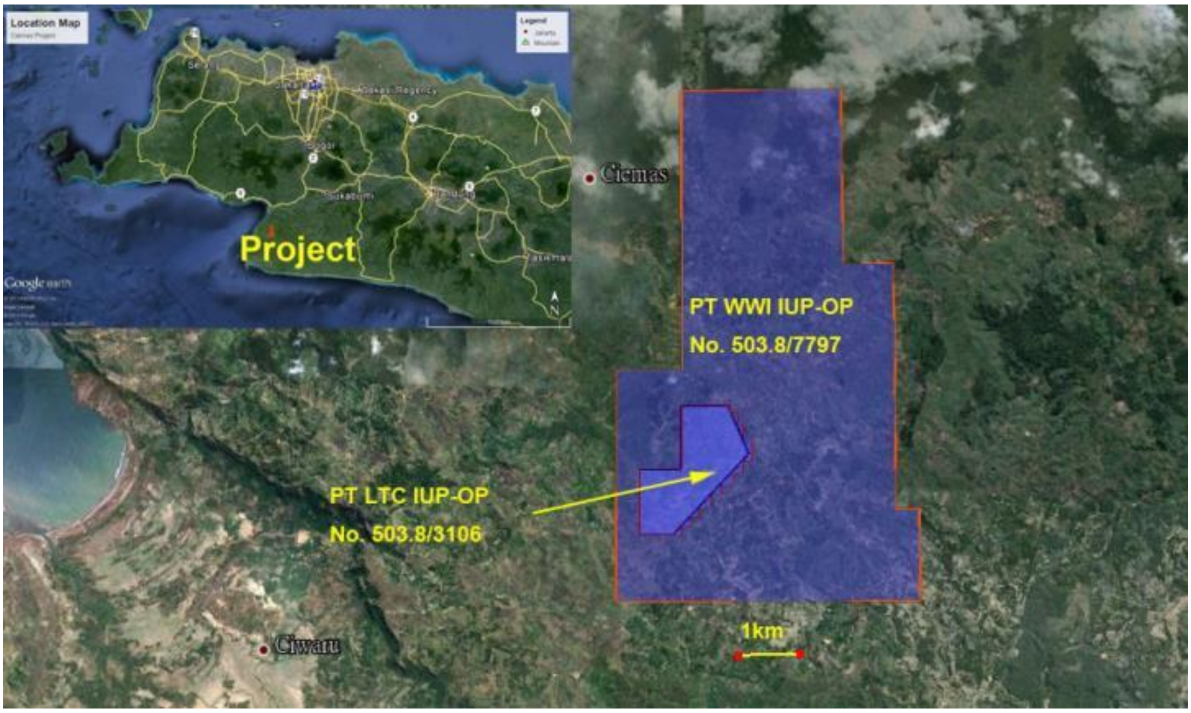


Figure 5-1: Tenure Information

5.3 Land Purchase Agreements

SRK has sighted the original land access/compensation agreements for the Ciemas project and was also provided with a list of land access/compensation agreements created by the Company. According to this list, the Company has obtained land access rights of approximately 100 hectares ("ha") from the local residents in the Pasir Manggu and Cileuweung gold bearing zone areas during past five years.

6 Geological Settings and Mineralization

6.1 Geological Settings

The Ciemas Project is situated within a volcanic polymetallic metallogenic belt of gold ("Au"), lead ("Pb"), zinc ("Zn"), and copper ("Cu"), in Ciletah Bay, West Java, Indonesia. Tectonically it is located at the southern margin of Sundaland, which is the continental core of southeast ("SE") Asia formed by the accretion of blocks to the Eurasian margin, and was assembled by the time of the Late Triassic (Figure 6-1).

The Ciemas gold deposit is hosted by a late Eocene to early Miocene volcanic rock belt. The belt is composed mainly of volcanic breccias and mostly covered by Quaternary eluvium and alluvium as well as a post-mineralisation tuff blanket up to 20 m thick. Volcanic breccias, tuffs, and andesite are widely distributed in the Project area.

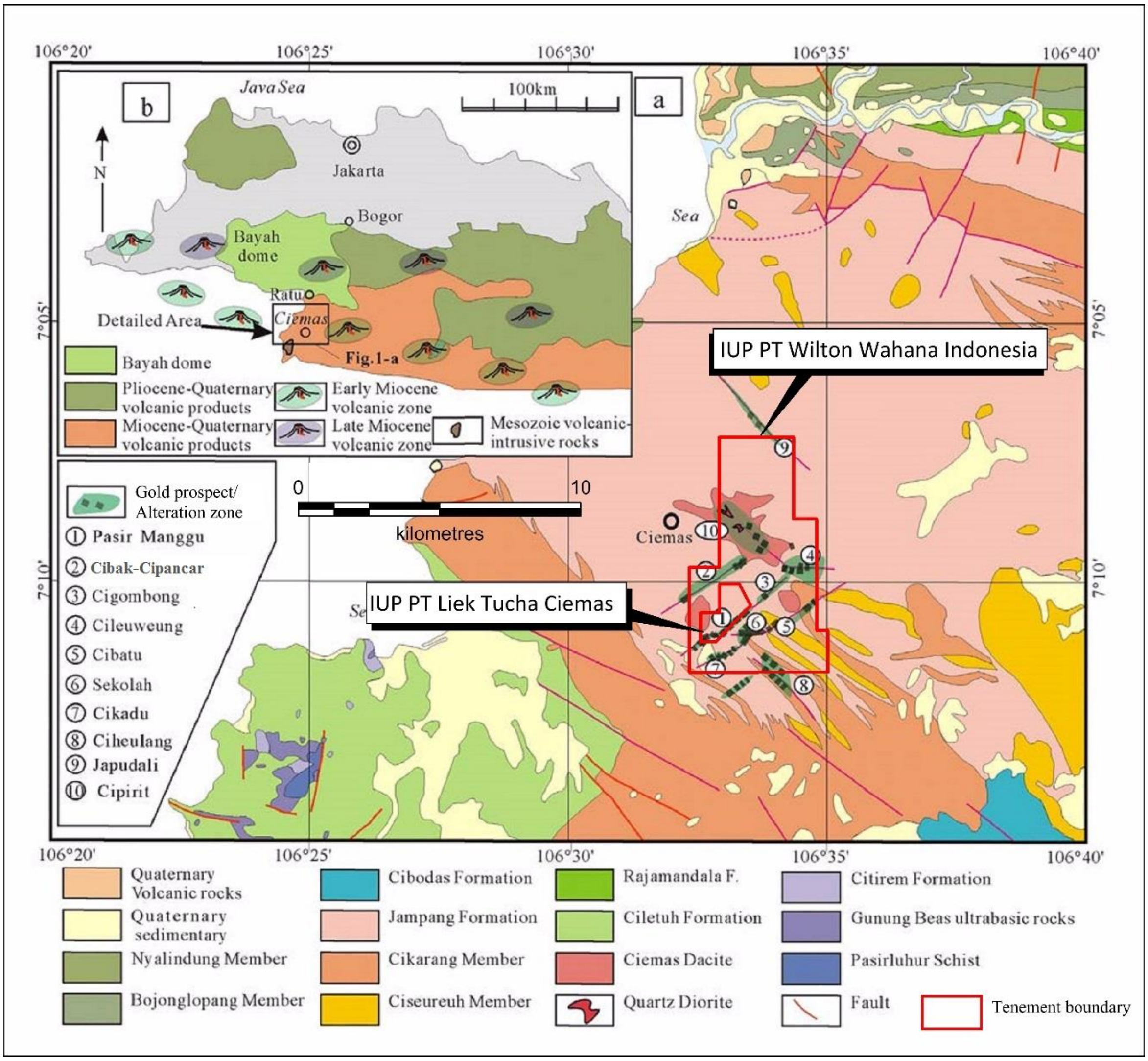


Figure 6-1: Geological Setting and Gold Mineralised Zones of the Ciemas Gold Project

Modified from Zhengwei Zhang and others: "The trinity pattern of Au deposits with porphyry, quartz–sulphide vein and structurally-controlled alteration rocks in Ciemas, West Java, Indonesia", link: http://dx.doi.org/10.1016/j.oregeorev.2014.07.003

*(a) Geological map showing the distribution of gold deposits/occurrences in the Ciemas area in West Java, Indonesia.

*(b) Map of West Java showing the Ciemas Project hosted by the late Eocene to early Miocene volcanic rock belt.

Relevant geological investigation suggests that the genesis of gold deposits in Ciemas is closely related to the magmatic hydrothermal activity whereby Miocene quartz diorite porphyrite intruded into andesite and dacite, from the perspective of mineralisation-forming space and time.

The regional geology of southwest Java is controlled by Miocene volcanic activities. The Project is located at the south end of the Sumendala fault, in the Jampang Kulon area of West Java. Sumendala is an important controlling factor for hydrothermal mineralisation and volcanic activities in the region.

Regionally, two sets of faults and/or fractures are developed, striking northeast ("NE") and northwest ("NW") (refer to Figure 6-1). The extensions of these faults/fractures vary from some one hundred metres to several kilometres, with the widths generally varying from 1 m to 20 m. These faults/fractures are the primary structures controlling the mineralisation and mineralisation-bearing zones in this area. Folding mainly consists in the Ciemas syncline with a NE axial direction.

Structural analysis indicates that the mineralisation-bearing faults represent three stages of tectonic activity. Early activity in the extensional fault is shown in the stockwork and filling structure. The middle stage activity is indicated by compressional faults with shear alterations consisting of tectonic schist and fracture breccias, and the late activity is represented by extensional faults with gold-bearing fractured zones with chalcedony–quartz veins, silicification, pyritisation, and carbonatization.

In addition to gold mineralised zones discovered in Ciemas, an Au-Pb-Zn polymetallic deposit is found in Cikondang, about 60km northeast of the Ciemas tenement. Three types of gold ore (or mineralised rocks) were distinguished in Cikongdang and they can be described as quartz veins, structurally altered rock, and quartz-dacite porphyry ores (or mineralised rocks), which are similar to the mineralisation discovered in Ciemas.

6.2 Mineralisation Characteristics

The structures in the Ciemas Project area are consistent with the regional structures, and are dominated by NE and NW faults and/or fractures. Within these structure zones, chalcedony-quartz veins are intermingled, often showing boudinage along strike and down dip.

The gold mineralisation at Ciemas is related to different fault stages of dominant structures and tension zones. These structure zones could be secondary fractures related to the Sumendala fault. The volcano mouth and associated dacite (usually presented as quartz-dacite porphyry) intrusion also provides favourable geological conditions for mineralisation.

The Ciemas gold mineralisation is hosted in quartz veins or structurally altered rocks with tectonic breccia, or in quartz porphyry. Mineralisation is predominantly related to NE-SW and NW-SE veins with the extensions varying from some 100 m to about 1,000 m; and the width of the mineralised bodies generally varies from 1 m up to about 15 m.

A number of gold mineralised zones have been defined by the exploration conducted in the Project area within an area of approximately 10 km^2 in the central part of the Ciemas tenement (IUP 503.8/7797). A simplified geological map for the major mineralised zones defined in the Project is shown in Figure 6-2.

Mineralised rocks have been identified as porphyry, quartz–sulphide veins, and structure-controlled alteration rocks. The mineralisation types of all major gold mineralised zones which have been discovered in the Project are classified as follows:

- Four mineralised zones, Pasir Manggu, Cigombong, Cileuweung, and Cibak – Cipancar, are of the quartz vein type;
- The gold mineralisation at Cikadu, Sekolah, Cibatu, Ciheulang, and Japudali is of the structurally controlled alteration type; and
- Cipirit, Ciaro and Cibuluh are related to the quartz porphyry intrusive type.

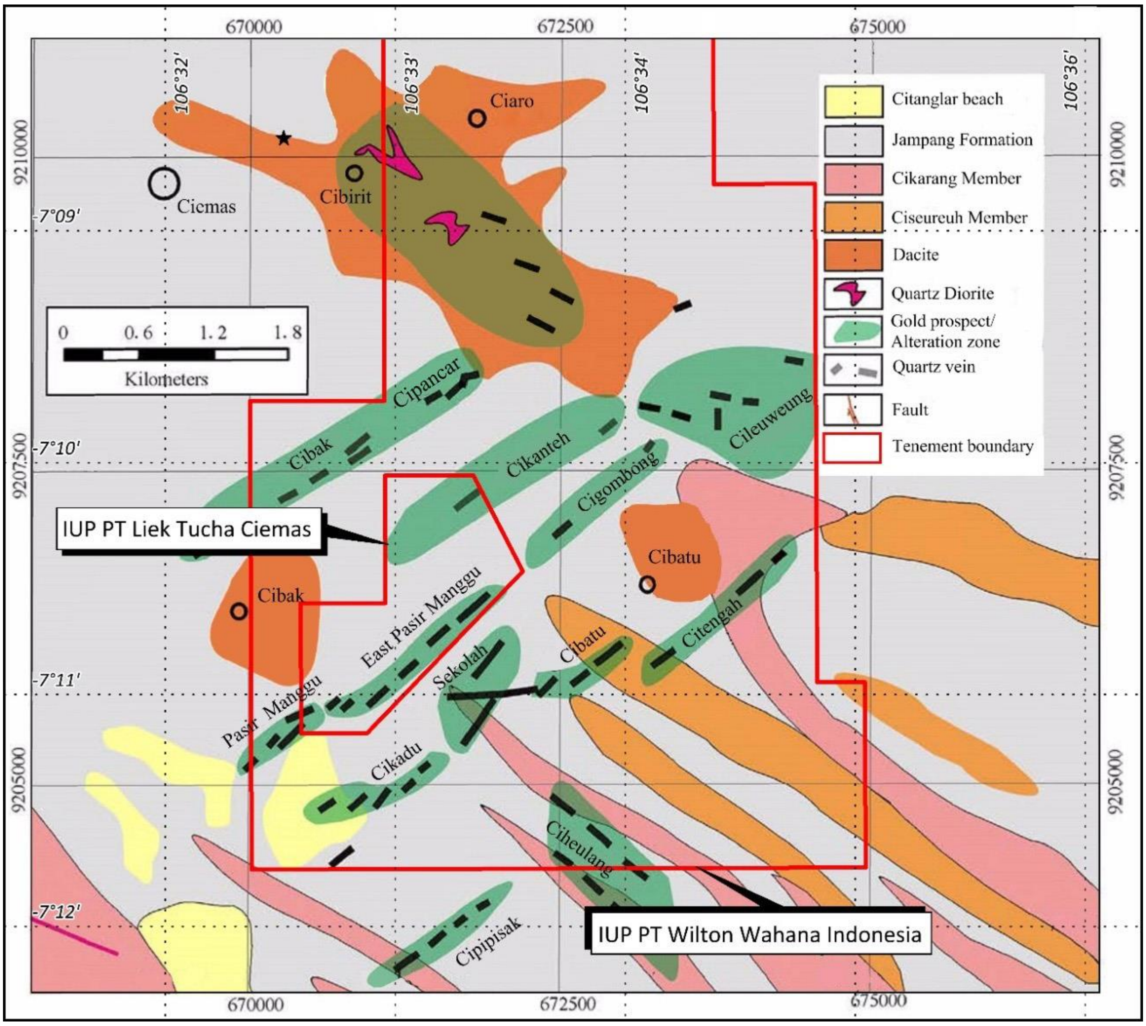


Figure 6-2: Simplified Geological Map of the Major Mineralised Zones in Ciemas Project

*Modified from original maps courtesy of Professor Zhengwei Zhang et al., 2014 and Jonathan 2007.

The interpretation of four main prospects ("4 Prospects"), namely Pasir Manggu, Cikadu, Sekolah and Cibatu mineralised zones, are presented in Figure 6-3. Their mineralisation characteristics are given in Table 6-1 and a more detailed description of the mineralisation can be found in SRK reports for Wilton's Ciemas Project published previously, i.e. IQPR – 2013 and Updated Resource Report in 2014.

In 2016, SRK issued a report titled "*Independent Qualified Person's Report of Cibak and Cipancar Prospects at Ciemas Gold Project in Republic of Indonesia*" ("Cibak and Cipancar Report in 2016"). The two prospects were interpreted basing on shallow well (shaft) data, as shown in Figure 6-4.

Typical cross sections of gold mineralisation interceptions in the quartz vein type mineralisation zones and the structurally altered rock type mineralisation zones are shown in Figure 6-5. More examples of cross sections can be found in the IQPR prepared by SRK in 2013.

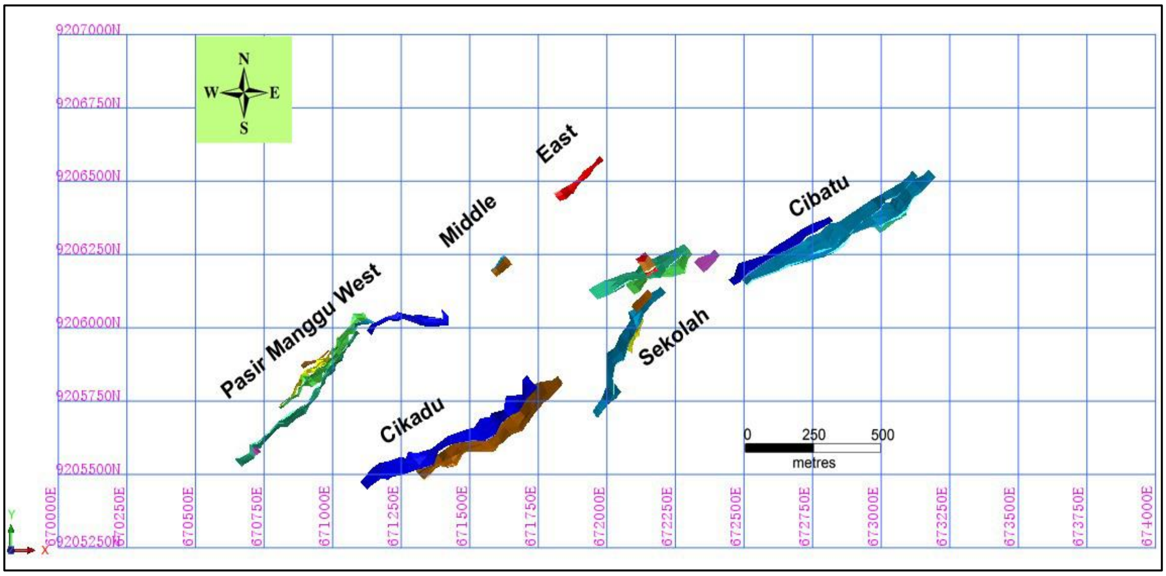


Figure 6-3: Geological Interpretation of Mineralised Zones at the Four Prospect Areas

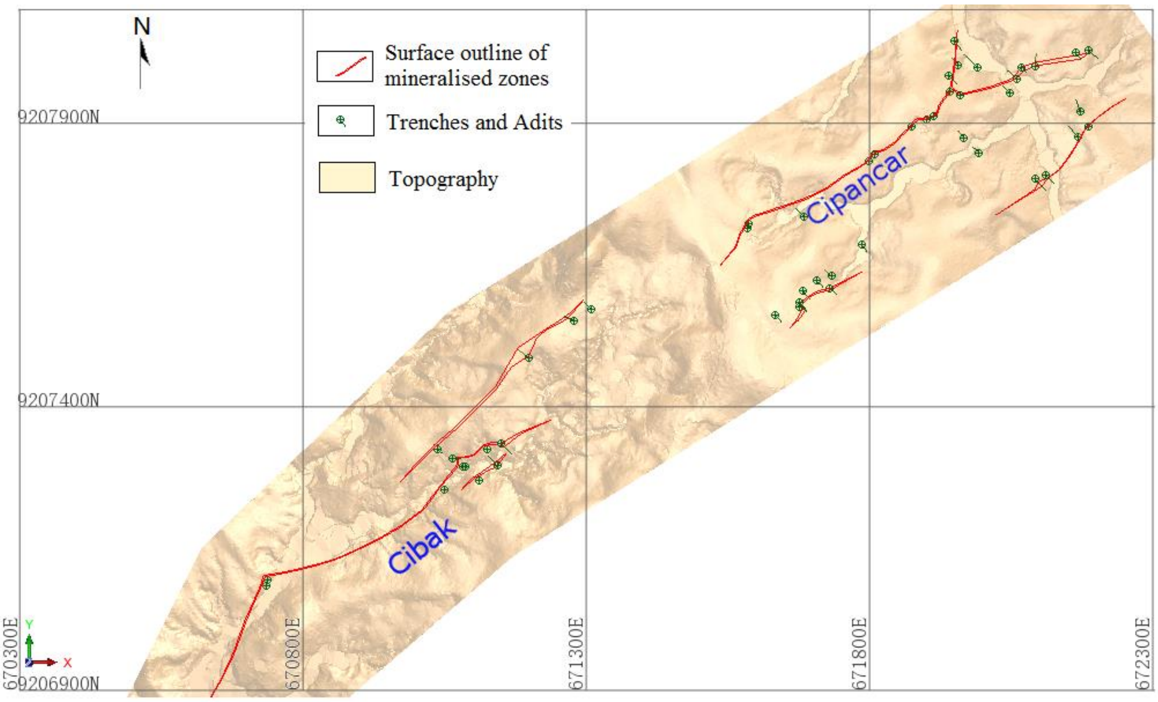


Figure 6-4: Geological Interpretation of Mineralized Zones at Cibak and Cipancar Areas

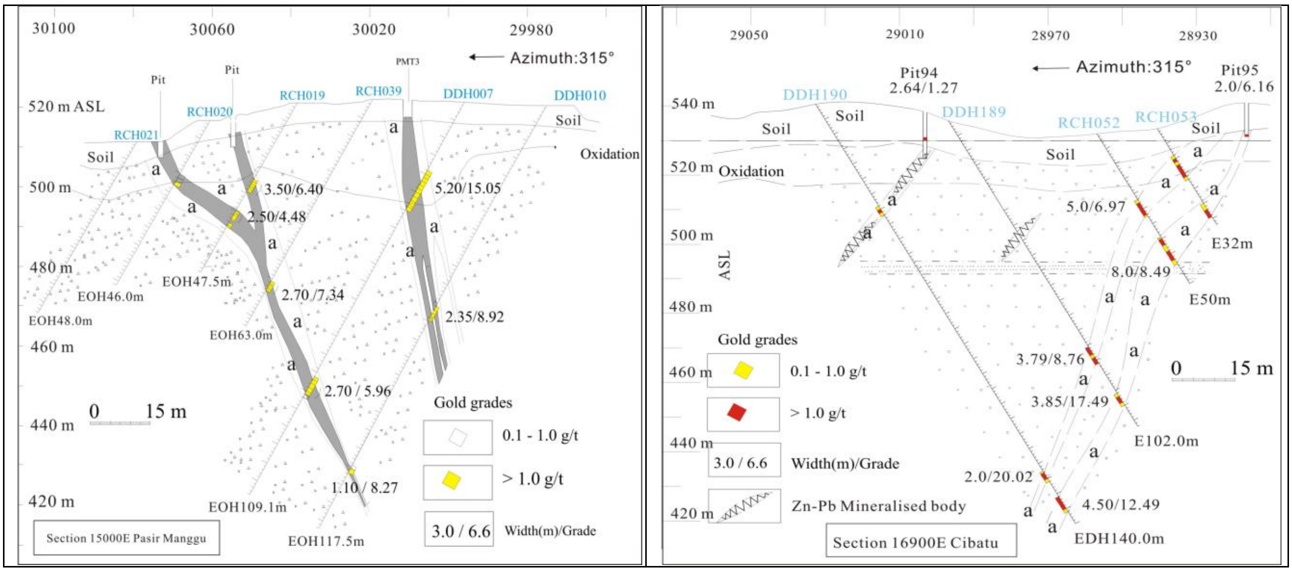


Figure 6-5: Typical Cross Section Showing Gold Mineralisation Interception in Quartz Vein (Left) and Structurally Altered (Right) Mineralised Zone

*Modified after Professor Zhengwei Zhang et al., 2014 and Jonathan 2007.

Table 6-1: Mineralisation Characteristics at Pasir Manggu, Cikadu, Sekolah, and Cibatu

Mineralised Zone and Location	Mineralisation Type and Controlling Structures	Mineralised Body (Vein) Characteristics	Structure and Texture of Ore (Mineralisation)	Natural Type of Mineralised Rocks, Minerals Association and Grade	Host Rock and Altered Wall Rock
Pasir Manngu central - southwest of the Ciemas licence area	Dominated by quartz veins, controlled by faults striking NE (about 45º).	A total of 10 gold mineralised veins including 4 main veins are defined in the Zone, overall striking NE, dipping SE at 70º - 80º. Strike length about 800 m at maximum with barren gap (not well explored), down dip extension generally 60 – 120 m; thickness 1 – 10 m.	Fine grained xenomorphic, panidiomorphic - hypautomorphic granular, and poikilitic textures, and in disseminated, fine stockwork, sparsely filling disseminated, and occasionally bulky structures.	Oxidised and sulphide ores, oxidation zone generally at depths of 0 - 30 m, 5 - 10 m wide. Mainly pyrite, arsenopyrite, and pyrrhotite. Au grade of sulphide zone varies from 0.5 to 226 g/t, average about 6 g/t.	Jampang Formation andesitic lava, breccia, and volcano-clastic rock. Hanging wall and foot wall in andesitic volcanic breccia with strong clay alteration, alteration belt thickness generally 2 – 15 m.

Mineralised Zone and Location	Mineralisation Type and Controlling Structures	Mineralised Body (Vein) Characteristics	Structure and Texture of Ore (Mineralisation)	Natural Type of Mineralised Rocks, Minerals Association and Grade	Host Rock and Altered Wall Rock
Cikadu About 1 km directly southeast of Pasir Manggu West	Dominated by structurally altered rocks and occasionally quartz veins, hosted in faults striking NE - NNE.	Two (2) major mineralised veins defined in the Zone, overall striking NE – NNE (about 55º), dipping NW –NNW at 60º - 75º. Extension along strike about 700 m, down dip extension to 150 m below surface, thickness 2 – 10 m.	Granular crystalloblastic, cataclastic, and lepidoblastic textures. Mylonitic, schistose, and some mesh-vein structures. Assemblages of galena-native gold-quartz-pyrite.	Oxidised and sulphide ores, oxidation zone generally at depths of 0 - 30 m, 5 - 10 m wide. Mainly pyrite, arsenopyrite, sphalerite, galena, and pyrrhotite. Au grade of sulphide zone varying from 0.5 to 82 g/t, average about 9 g/t.	Hanging wall and foot wall within andesitic volcanic breccia, alteration with mainly chloritization and pyritisation as well as silicification, carbonation and epidotization. Alteration belt thickness generally 2 – 20 m.
Sekolah About 200 m directly abutting Cikadu to the northeast	Dominated by structurally altered rocks and occasionally quartz veins, hosted in faults striking NNE (about 35º) – NEE (about 60º).	A total of 8 mineralised veins defined in the Zone, overall striking NNE – NEE, dipping NNW – NWW at 60º - 75º. Extension along strike about 500 m, down dip extension to 150 m below surface, thickness 2 – 10 m.	Granular crystalloblastic, cataclastic, and lepidoblastic textures. Mylonitic, schistose, and some mesh-vein structures. Assemblages of galena-native gold-quartz-pyrite.	Oxidised and sulphide ores, oxidation zone generally at depths of 0 - 30 m, 5 - 10 m wide. Mainly pyrite, arsenopyrite, sphalerite, galena, and pyrrhotite. Au grade of sulphide zone varying from 0.5 to 58 g/t, average about 9 g/t.	Hanging wall and foot wall within andesitic volcanic breccia, alteration with mainly chloritization and pyritisation as well as silicification, carbonation and epidotization. Alteration belt thickness generally 2 – 20 m.
Cibatu About 200 m directly abutting Sekolah to the northeast	Dominated by structurally altered rocks and occasionally quartz veins, hosted in faults striking NE - NNE.	Three major mineralised veins defined in the Zone, overall strike NE – NNE (about 55º), dip to NW –NNW, dip angle 60º - 75º. Extension along strike about 800 m, down dip extension to 150 m below surface, thickness 2 – 10 m.	Granules crystalloblastic texture, cataclastic texture and lepidoblastic texture. Mylonitic structure, schistose structure, partly mesh-vein structure. Assemblages of galena-native gold-quartz-pyrite.	Oxidised and sulphide ores, oxidation zone generally at depths of 0 - 30 m, 5 - 10 m wide. Mainly pyrite, arsenopyrite, sphalerite, galena, and pyrrhotite. Au grade of sulphide zone varying from 0.5 to 78 g/t, average about 8 g/t.	Hanging wall and foot wall within andesitic volcanic breccia, alteration with mainly chloritization and pyritisation as well as silicification, carbonation and epidotization. Alteration belt thickness generally 2 – 20 m.

Contents in this table are referenced to SRK, 2013 and Professor Zhengwei Zhang, 2014.

7 Exploration and Data Quality

7.1 Summary

The exploration history and data were detailed in SRK previous reports in 2013, 2014 and 2016. The quality of data used for resource estimates in 2014 and in 2016 were assessed by SRK. Details of the previous data can be referred to

- IQPR in 2013,
- Updated Resource Report (for the 4 Prospects) in 2014 (with JORC Table 1 in Appendix A of that report); and
- Cibak and Cipancar Report in 2016 (with JORC Table 1 in Appendix 2 of that report).

For Pasir Manggu, Cikadu, Sekolah and Cibatu prospects, the main changes since 2014 are:

- A total of 30 diamond drill holes ("DDH") with aggregate length of 1,756.7 m were completed additional to the data reported in 2014. The data have well verified and collaborated the previous data and have been incorporated into integrated database for resource update.
- The topography and surface geological mapping were further updated.
- Additional metallurgical tests and studies have been done.
- More detailed interpretation on oxide and fresh zones were completed according to the integrated database.

The additional 30 DDHs were designed for the main purpose of metallurgical tests and secondarily for the in-fill resource exploration. The holes were drilled with standard tubes and were opened at a size of PQ and fished at HQ. These holes were drilled at a dip angle varying from -65° to -15°, for the main purpose of intersecting oxidised zones in the shallow part.

The 30 DDHs have good representativity of the mineralised zones. There are 8 in Pasir Manggu, 9 in Cikadu, 6 in Sekolah and 7 in Cibatu.

Samples were taken following a standard procedure same with the exploration in 2012 and 2013. Sample logging, photograph, splitting, and security were well performed in accordance with general industry practice. Samples for each mineralised interval were prepared and analysed in Intertek laboratory in Jakarta and reanalysed in Australian Minmet Metallurgical Laboratories Pty. Ltd. ("AMML") prior to metallurgical tests.

SRK has reviewed the information of drill hole collar, survey and samples, and validated the data through a site visit checking the drill hole collar and sealing status, reviewing the original copies of laboratory results, core and sample photos, as well as inspecting the remaining samples (coarse rejects and pulps).

In this Report there is no additional exploration data to be disclosed for Cibak and Cipancar prospects. There are no material changes on the Mineral Resources reported in 2016, except minor volume and grade adjustment made during the process of model refining in 2018, and the changes are considered less than 1% for both volume and grade.

7.2 Pasir Manggu, Cikadu, Sekolah and Cibatu

A detailed description of the historical exploration carried out in the Ciemas Project area can be found in the IQPR prepared by SRK in June 2013. In addition to the database reported in the IQPR 2013, a total of 30 drillholes with a total length of 1,756.7 m completed in 2013 has been incorporated into the latest resource estimate reported in this Report.

SRK assessed the historical data compiled by Wilton or other consultants on behalf of Wilton in 2012 and 2013. A verification drilling program was completed following SRK's recommendation and the drilling and sampling was performed following standard procedures in gold mineral exploration. Based on the data review and verification results, SRK is of the opinion that the integrated database is adequate for Mineral Resource estimates of Pasir Manggu, Cikadu, Sekolah and Cibatu.

Details of the exploration and sampling techniques summarised in the form of JORC Code Table 1 have been published by Wilton in previous technical reports.

7.2.1 Sampling Techniques

Samples from the Project were collected mainly from diamond drilling holes ("DDHs"), reverse circulation holes (RCH), trenches and pits. The compiled exploration database for Pasir Manggu, Cikadu, Sekolah and Cibatu has been reviewed in detail. For other properties of this Project, exploration is represented by trenching and pitting but these data are insufficient for a JORC Code compliant resource review/estimation. The delineation of mineralised bodies in the Ciemas Project is based primarily on the drilling results. As the historical pitting and trenching data are incomplete, the resource estimation in this Report only involves the DDH and RCH drilling.

Core sampling, RC chips, and channel sampling comprised the primary sampling methods, of which only the data derived from diamond core and RC drillings were used for Mineral Resource estimates. The surface channel (trench) sample data and percussion drilling data were used for the geological interpretation and delineation of resource domains (wireframe).

The sampling grids were generally 20 m × 20 m (in Pasir Manggu), 40 m × 40 m, and 80 m × 80 m. Simplified layouts of exploration work conducted in the four property areas.

Most of the DDHs were drilled with a dip angle of 60°. Drill cores were split into two halves and the sample length was approximately 1 m. RC chips at the mineralised sections were sampled at an average interval of 1 m. Channel samples were collected from trenches and pits. The channel sample length was 1 m.

Core recovery rates of historical drilling conducted by Parry, Terrex and Meekatharra were unknown because of a lack of data. The reprinted historical DDH and RCH log books recorded the lithology and sample intervals as well as coordinates but there was no information about recovery. Original drill hole logging sheets were found for only a few historical DDHs and SRK noticed the manuscripts recorded core recoveries generally above 85%. Except for some core residuals, there are no cores available for recalculating the historical drill sample recoveries.

For the new drilling program, conducted since 2012, the measurements of cores and footage (length) drilled in each run were recorded in the drilling logs and were reviewed by both PT ASI and SRK site geologists. In general, the core recovery of the drilling program conducted by Wilton is high, averaging 95%. The average recovery of mineralised intervals is higher than this number, rating 98%.

The gold mineralisation is related to breccia and fractured zones, as well as structurally controlled alteration rocks. The mineralised intervals are sometimes fractured but this does not imply a low recovery rate. Core recovery and assay grades are not correlated, as SRK observed.

7.2.1.1 Core Sampling

Drill cores were either HQ-size with core diameter of 63.5 mm (including its variation of triple tube HQ3-size with 61.1 mm diameter core) or an NQ-size with core diameter of 47.6 mm. In the recent Wilton drilling campaign from 2011 to 2013, all holes were drilled with an HQ3-size diameter (61.1 mm) and, during SRK's site visit in 2012, some drill core residuals from historical drilling programs with NQ-size diameter were observed. The core diameter is considered adequate for splitting and sampling.

In Pasir Manggu, a total of 911 core samples with an average length of 0.94 m were taken from 88 diamond drill holes with a cumulative drilled length of 7,164 m. In Cikadu, Sekolah, and Cibatu, a total of 2,228 core samples with average length of 0.97 m were taken from 147 DDHs with a cumulative drilled length of 13,613 m.

7.2.1.2 RC Sampling

RC samples were taken from chips, with each single sample weighing 2.5–5.0 kg. Overweight samples were split into two halves after being mixed evenly. In Pasir Manggu West, a total of 769 samples with an average length of 1 m were taken from 64 reverse circulation holes with a cumulative drilled length of 3,294 m. In Cikadu, Sekolah and Cibatu, a total of 443 chip samples with an average length of 0.98 m were taken from 42 RCHs with a cumulative drilled length of 2,011 m.

7.2.1.3 Channel Sampling

In Pasir Manggu West, a total of approximately 450 samples with average length of 0.90 m were taken from 16 trenches and pits. Trenches and pits excavated in Cikadu, Sekolah and Cibatu were also assessed separately during the geological interpretation and resource domain delineation. Due to the difficulty of verifying the original surface sampling and assaying accuracy, SRK did not use the channel samples in the Mineral Resource estimates for the four properties.

7.2.2 Data Verification

Verification and infill diamond drilling program was performed in 2012 and 2013 (drilling was completed by the end of 2012 and some sampling and assaying was finished in 2013) near the previous boreholes to verify their data. The verification review included geological logs, collar and down-hole surveys, and assay comparisons with the previous mineral resource database. The details of data verification were addressed in previous reports and are summarised below.

The additional diamond boreholes were drilled at Pasir Manggu West, Cibatu, Cikadu and Sekolah deposits by PT Sugihjaya Tata Lestari (Sugihjaya) from March 2012 to December 2012 and supervised by SRK. A total of 24 boreholes were drilled, six each at Pasir Manggu, Cikadu and Sekolah and Cibatu. The verification drill holes were deployed along exploration lines across the strikes of the mineralized veins/bodies. The samples were used to verify mineralisation continuity and to compare with the previous exploration result along the exploration lines; they were also intended to test the potential for an extension of the resources at these property areas.

The drillhole locations were surveyed before the drilling commenced and re-surveyed after drilling. Down-hole surveys were completed using a Proshot microscope probe. All drill cores were photographed and logged by field geologists. After geological logging, each drilling sample was split by an alloy cutter along the core's long axis. One half of the core was put in a sample bag with a unique sample number plate, and the other half was replaced in the core box and kept in the core storage shed. The sample was also photographed, and sample sheets were filled out.

The core samples were prepared and assayed in Intertek's Jakarta laboratory with insertion of internal coarse blanks and standards. The field blanks were made of quartz and contained less than 0.005 grams per tonne (g/t) Au, which is the lower detectable limit for the fire assay method used for the Ciemas gold analysis. The inserted standards were pulps made of certified reference materials (CRM). Both the blanks and standards were inserted into routine samples at an approximate frequency of 1:20. External coarse blanks and quarter core duplicates were inserted at an approximate frequency of 1:20. There were no external pulp duplicates inserted. In the last round of assaying in the 2012 drilling campaign, a series of suitable external CRMs sourced from OREAS in Melbourne were inserted into the sample set at a ratio of 1:20. These were selected based on ore type and expected grade of sample and a set of four separate CRMs were used depending on the material intersected in drilling. Generally the results were very close to the certified value.

The basic assay method for these samples used was an FA50 fire assay, assaying 50 g of fine pulps with a lower detection limit of 0.005 g/t Au. When the gold value exceeded 50 g/t, gravimetric fire assays were used to determine the higher gold grade.

SRK has reviewed and compared the data derived from verification and in-fill drilling programme and has concluded that the historical data was well verified with both sample grades, intersection width of the veins as well as overall geological continuity. The most recent 30 DDHs have also proved that the integrated database is acceptable for Mineral Resource estimate.

7.2.3 Data Quality

The Ciemas Gold Project has been explored and evaluated in stages by various companies and consultants. Historical data was not appropriately passed on during the changes of owners and exploration stages. Data compilation and integration was performed by Wilton with its technical consultants prior to SRK's review. The previous samples were assayed by laboratories Kep Seksi Kimia Mineral, Inchcape Testing Service, and PT Inchcape Utama Service. SRK sighted part of the original laboratory sample results for the historical exploration (i.e. exploration conducted before 2008) but no detailed indications were available regarding assaying methodology or QA/QC procedures. To evaluate the reliability and accuracy of the historical sampling and assays, Wilton conducted verification drilling following SRK's recommendations made in March 2012.

Drillhole collars, down-hole surveys, and sample data of 88 DDHs with a cumulative length of 7,164 m and 64 RCHs with a cumulative length of 3,295 m at Pasir Manggu have been incorporated into the exploration database. The compiled database also contains 147 DDHs with a cumulative length of 13,613 m and 42 RCHs with a cumulative length of 2,011 m, drilled at Cikadu, Sekolah and Cibatu.

Prior to the 2012 verification drilling, Wilton staff worked with an experienced research geologist, Professor Zhengwei Zhang, to re-assess the quality of the historical data using data compilation and some validation trenching and pitting conducted by Wilton from 2009 to 2011. SRK inspected a number of drill collars and surface trenches on site and reviewed drill logs. Drilling, logging, bulk density testing, sampling procedures, and data quality aspects were discussed and reviewed with Wilton staff.

SRK considers that it is not reasonably possible to trace and validate the quality of the exploration conducted by previous owners Parry, Terrex, and Meekatharra in terms of samples, drill cores, and duplicates, except by assessing the inherited database, documents and maps, which reveal abundant exploration works conducted in the Project area apparently of high quality, although performed prior to the publication and wide adoption of the JORC Code.

Based on the verification and in-fill drill sample assay results and SRK's check samples taken in 2012 (detailed in the 2013 IQPR), SRK believes there are relatively continuous mineralised bodies existing in the Pasir Manggu, Cikadu, Sekolah and Cibatu gold deposits. Although there are some differences in gold grades in the section figures, SRK is of the view that the discrepancy is within an acceptable range for the type of quartz vein and structurally altered rock gold deposits found in the Ciemas Project.

As a consequence, SRK considers that the verified and integrated database can be reasonably used for the Mineral Resource estimates for the four properties of Pasir Manggu, Cikadu, Sekolah and Cibatu. The missing information is reported on a transparent 'if not, why not' basis following JORC Code Table 1 (JORC, 2012).

7.3 Cibak and Cipancar

The details of exploration in Cibak and Cipancar are presented in the report issued in 2016.

Pt Prihaditama was engaged by Wilton to conduct a resource survey on the Cibak and Cipancar area using Geophysics surveying method. IP & Resistivity techniques were utilized to locate any anomaly (mineralization zone) beneath the investigated area and estimate potential mineralization quantity.

The survey was conducted on a stretch of 470m line perpendicular to the predicted ore body vein. There are 48 electrodes at 10m interval along the line where Measurements are taken. A total of nine lines were conducted at a parallel interval of 200m across the entire Cibak and Cipancar Prospects. The resulting anomaly model can be made to a depth of about 60m.

The data acquired at the Cibak and Cipancar Prospects is comprised of historical data from 33 trenches by Terrex Resources during 1992 to 1994 and Meekatharra Minerals during 1996 to 1998, and the data from 31 shafts acquired by Wilton recently.

The historical trenches were not investigated on site. The data of historical trenches were sorted by PT Asia Sejati Indonesia ("PT ASI") commissioned by Wilton, and were provided to SRK for mineral resource estimation.

The Wilton's miners dug vertical shafts, sampled using continuous rock-chips, and measured the sample locations using band tape. The location of the shafts were surveyed using hand held GPS by Wilton's geologist. All samples from Wilton's shafts were packed by plastic bags with numbering, and were sent to PT Intertek based in Jakarta for analysis. A synoptic logging was also made by the Wilton's geologist.

Documents provided by the Company presents the sampling method on site during the previous exploration campaigns. Rock samples were picked up from a representative main vein body and the both side alteration zone (halo) on the hanging wall and footwall of the veins. Samples were collected across the mineralised body and alteration zone, and channel sampling method is applied perpendicular the strike line. The sampling procedure is as follows.

For shafts in 10 to 20 meters depth, channel sampling method was not possible to be applied for the safety reason. Continuous grab sampling was an alternative way to collect the samples.

Trenching was required in addition to discover minerals. It was also to secure data concerning the ore body conditions such as depth, physical characteristics, country rocks, direction and gradients etc.

The horizontal direction of trenching in a location was defined according to approximate strike direction of the adjacent outcrops in the concerned location. Trenching was strike of the ore body, thus it will intersect the available stratification. The trenching was between 20 – 30 meter lengths. With 2 meter wide and 2-4 meters deep.

Soil samples were picked up by manual digging to horizon B. Each sample was packed in plastic bag approximately 2,500 grams.

8 Mineral Resource Estimation

8.1 Introduction

This section describes the Mineral Resource estimates for the Project. Due to there is little change on the Cibak and Cipancar prospects, this section has not included details for them.

The Mineral Resource Statement presented herein represents the evaluation of gold Mineral Resources prepared for the Ciemas Project in accordance with the JORC Code 2012 Edition.

The Mineral Resource model prepared by SRK is based on 205 DDHs and 106 RCHs drilled from 1990 to 2014. This section describes the resource estimation methodology and summarises the key assumptions considered by SRK. In the opinion of SRK, the resource evaluation reported herein is a reasonable representation of the global Mineral Resources defined in the Ciemas Gold Project at the current level of sampling. The Mineral Resources have been estimated in conformity with generally accepted "Estimation of Mineral Resource and Ore Reserves Best Practices" guidelines and are reported in accordance with the JORC Code (2012). SRK followed the techniques and procedures set out in the JORC Code Table 1 as presented in Appendix 1 during the resource evaluation and preparation of this Report.

In accordance with the JORC Code (2012), Mineral Resources have reasonable prospects of eventual economic extraction. They are not "Ore Reserves" and do not have demonstrated economic viability. There is no certainty that all or any part of the Mineral Resource will be converted into Ore Reserve.

The database used to estimate the Ciemas Project gold Mineral Resources was audited by SRK. SRK is of the opinion that the current drilling information is sufficiently reliable to interpret with confidence the boundaries for gold mineralisation and that the assay data is reliable enough to support Mineral Resource estimation.

Three dimensional (3D) explicit modelling in Surpac version 6.5 was applied to create mineralisation boundaries used to constrain the estimation volume. Surpac was also used to construct the geological solids, prepare assay data for geostatistical analysis, conduct variography, construct the block model, estimate metal grades, and tabulate Mineral Resources.

Considering that the type of mineralisation in Pasir Manggu is predominantly of the quartz sulphide vein type and is different from the structurally altered rock type of gold mineralisation dominant at Cikadu, Sekolah and Cibatu, SRK separated the Pasir Manggu model from the other three, which is consistent with SRK's previous resource estimate for the Project in 2013.

8.2 Resource Estimation Procedure

The resource evaluation methodology involved the following steps:

- Database compilation and verification;
- Reviewing the geological wireframes and block models produced by SRK in 2014;
- Updating geological and resource domains;
- Data conditioning (compositing and capping) for geostatistical analysis and variography;
- Updating the block model and performing grade interpolation;
- Resource classification and validation;
- Assessment of "reasonable prospects for economic extraction" and selection of appropriate cut-off grades; and
- Preparation of the Mineral Resource Statement.

8.3 Resource Database

A resource database has been compiled for the Pasir Manggu, Cikadu, Sekolah and Cibatu properties. Prior to SRK's resource evaluation of the Ciemas Project, initially performed in March 2012, a dataset for the resource evaluation was constructed with data from diamond drilling, surface channelling, and RC drilling, each consisting of collar, survey, and sample information as a basic requirement for geological interpretation and resource estimation. Other data also included cross-sections spaced 20–40 m apart in each property area.

Historical data were reviewed prior to being incorporated into the database. Some data reflected in documents and section maps were further checked with relevant logging and sample records; and parts of the incomplete historical data were rejected. A digitised database for exploration of the Pasir Manggu, Cibatu, Cikadu and Sekolah deposits was prepared using the available cross-section maps and sample sheets.

SRK reviewed the database compiled for the resource estimation of Pasir Manggu, Cibatu, Cikadu and Sekolah, as provided by Wilton, and performed random checks of the database against the cross-section maps and drillhole layouts. The reviewed sample data for each deposit are shown in Table 8-1.

Table 8-1: Sample Data Used for Geological Modelling and Resource Estimation

Deposit	DDH		RCH		Trench/Pit	
	Hole	Sample	Hole	Sample	Hole	Sample
Pasir Manggu	88	911	64	769	16	23
Cikadu, Sekolah and Cibatu	147	2,228	42	443	101	850
Total (four deposits)	**235**	**3,139**	**106**	**1,212**	**117**	**873**

Note: trench and pit samples are not used for resource estimation, only for geological interpretation.

The database used for the Pasir Manggu resource estimation comprises sample data derived from 88 DDHs and 64 RCHs completed by Parry, Terrex, and Wilton. The deposits of Cikadu, Sekolah and Cibatu (C-S-C) are grouped in a line from southwest to northeast. These properties share a similar metallogenic background and are structurally altered gold deposits hosted in the same fracture zone. The database used for the resource estimates of C-S-C comprises sample data derived from 147 DDHs and 42 RCHs, of which 107 DDHs and 42 RCHs were completed by Parry and Terrex, and 30 DDHs were drilled by Wilton in 2012 and 2013.

As with the Pasir Manggu database, there were a few minor errors in the C-S-C database, such as incorrect survey and sample intervals, but most of these errors appeared to be simple typing mistakes and were checked and revised manually in Surpac. The topography for the whole area was resurveyed in 2012 and the Universal Transverse Mercator (UTM) grid was adopted to locate the historical borehole collars.

After reviewing the combined exploration data, including an additional 18 verification DDHs completed in the C-S-C zones in 2012, SRK is of the opinion that the integrated database supports a reasonable resource estimate. Generally, drilling at the C-S-C is laid out following grids of 40 m × 40 m and the sections are deployed with azimuth 135°.

Wilton completed a topographic survey and specific gravity measurements in 2012 following SRK's advice. The UTM projection was adopted in the survey and previous local coordinates were converted to UTM.

Figure 8-1 and Figure 8-2 show plan views of the drilling layout at Pasir Manggu and C-S-C, respectively.

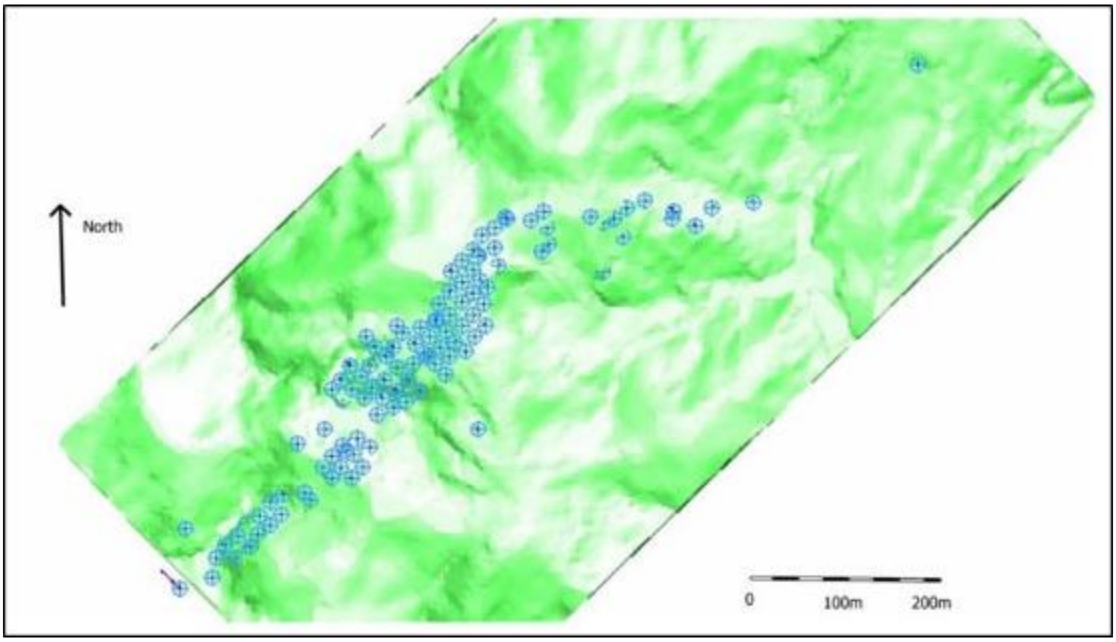


Figure 8-1: Plan View with Drilling Layout and Topography at Pasir Manggu West

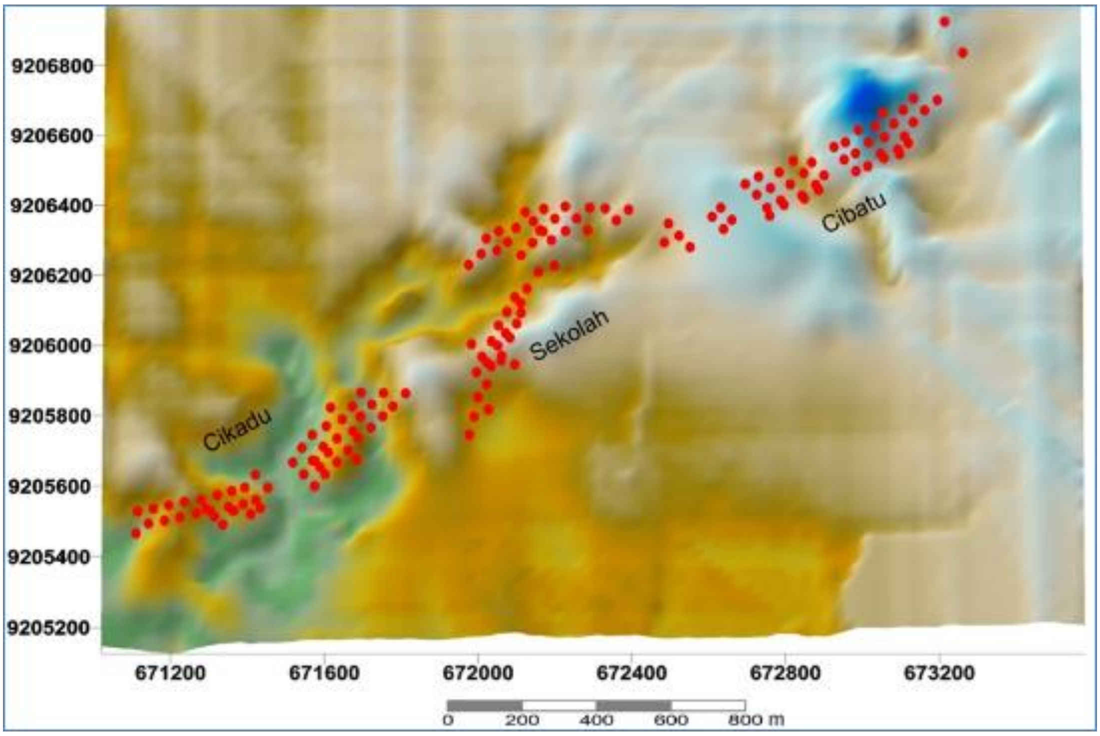


Figure 8-2: Plan View with Drilling Layout and Topography at C-S-C

The oxidation zones are considered to extend to about 30 m below the surface. The part of the mineralised bodies located below this level was classified as the primary (fresh ore) zone. In 2013, SRK created a model of the topographic surface, oxidation, and supergene zones, and the model has been refined according to new data, as shown in Figure 8-5.

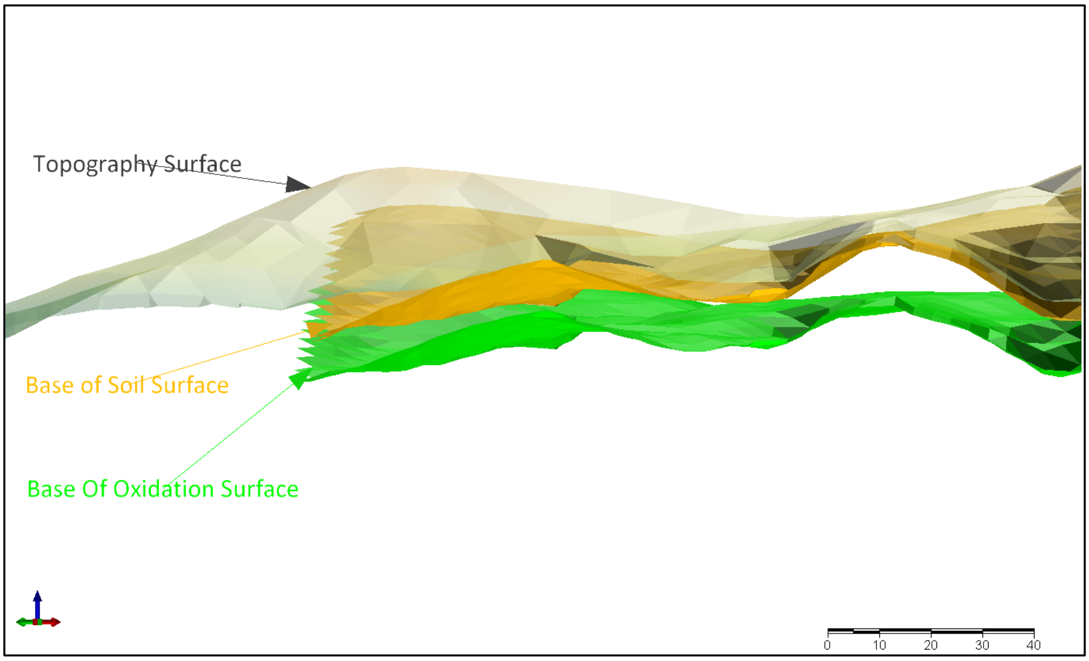


Figure 8-3: Updated Model of Topography, Soil and Oxidation Surfaces

8.4 Solid Body Modelling

8.4.1 Pasir Manggu

Solid wireframes of the mineralised zones at Pasir Manggu have been updated according to recent geological findings. Changes in the earlier wireframes, made based on a cut-off grade of 0.5 g/t Au in 2013, include the following:

- Refining geological interpretations based on upgraded geological understanding from recent works such as the study of oxidised zones and in-depth studies of the continuity of major mineralised zones;
- Reconsidering the geological continuity of small mineralised veins and refining the delineation of resource domains; and
- Updating the wireframes of mineralised zones (mineralisation domains) using a cut-off grade of 0.8 g/t Au.

An updated wireframe of mineralised zones at Pasir Manggu is shown in Figure 8-4. A total of ten mineralised bodies/veins are modelled, among which four are major mineralised bodies and six are small bodies. Generally the gold veins extend from the southwestern corner of Pasir Manggu West, striking NE toward Pasir Manggu Middle and East. The main area of interest in Pasir Manggu West contains four gold veins (Veins #1, #2, #3, and #4 as shown in Figure 8-4) and some of their splits (#1-b, #2-b, and #3-b).

The mineralised zones at Pasir Manngu generally strike NE and dip SE with dip angles of 70º - 80º. Strike lengths reach about 800 m with barren gaps (not well explored) and down dip extensions generally from 60 to 120 m with thicknesses of 1 – 10 m.

There are not enough DDH results to show a consistent and continuous mineralisation extending NE and connecting the main veins as mentioned above, but RCH results and surface evidence suggests that those small veins, such as the northeasternmost, #6, and two parallel veins, #5-1 and #5-2, situated between #6 and the main zone in the southwest, possibly reflect some continuity of gold mineralization.

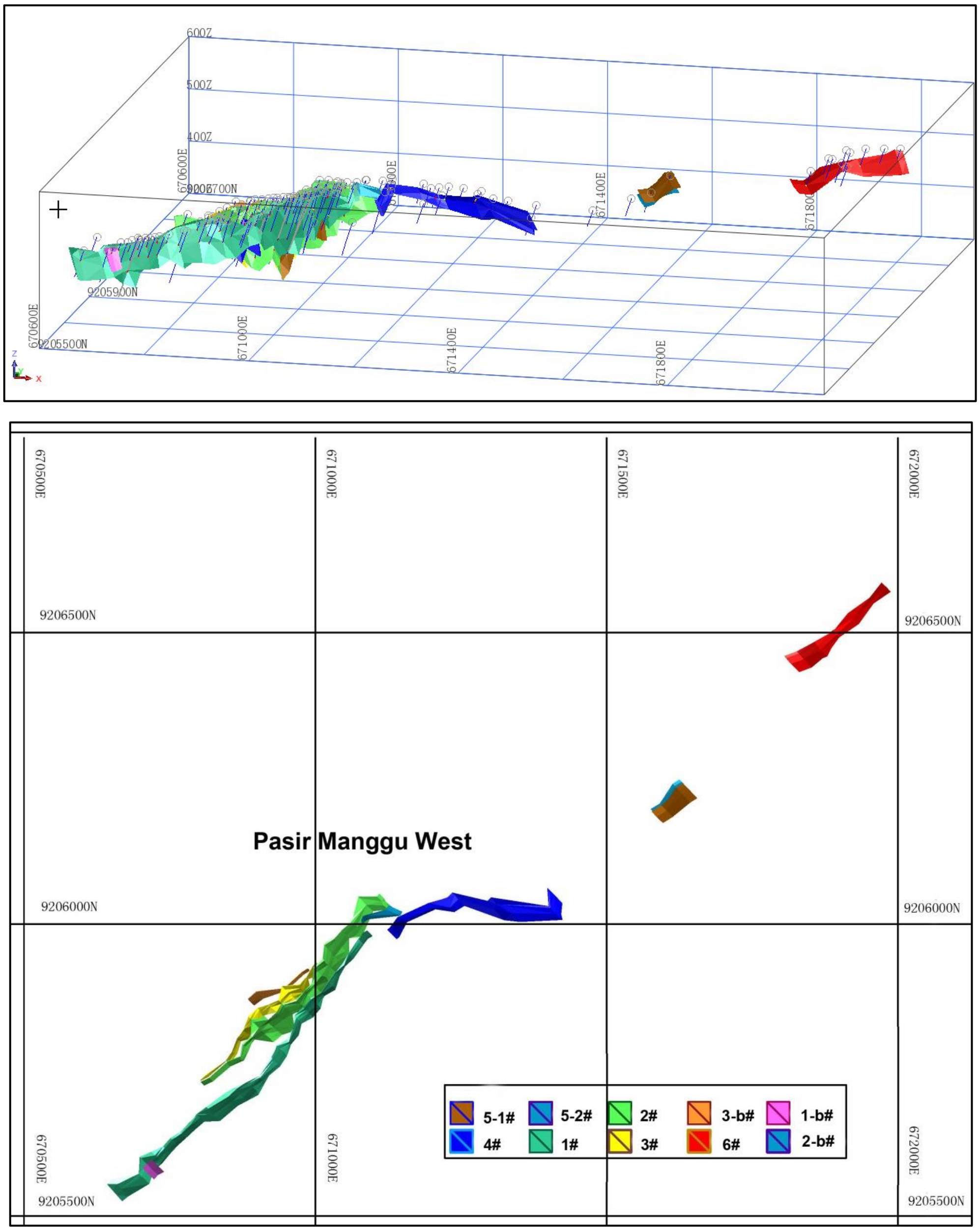


Figure 8-4: 3D Solid Wireframe and Drill Interceptions of Pasir Manggu Mineralised Zones

8.4.2 Cikadu, Sekolah and Cibatu

The updated solid model for the C-S-C properties used a total of 10 additional holes drilled in 2012. Wireframing was performed based on a cut-off grade of 0.8 g/t Au.

The updated wireframe of mineralised zones at C-S-C is shown in Figure 8-5.

Two major mineralised veins are defined in Cikadu. Overall the mineralised zones strike NE – NNE at about 55º and dip NW – NNW at 60º - 75º. The veins extend about 700 m along the strike, and their down dip extensions reach 150 m below the surface with thicknesses of 2 – 10 m.

A total of eight mineralised veins were defined in Sekolah, striking NNE – ENE and dipping NNW – WNW at 60º – 75º. They extend about 500 m along the strike, with down dip extensions of up to 150 m below the surface, and thicknesses of 2 – 10 m.

Three major mineralised veins were defined in Cibatu, overall striking NE – NNE at about 55º and dipping NW – NNW at 60º - 75º. They extend about 800 m along the strike, with down dip extensions to 150 m below the surface and thicknesses of 2 – 10 m.

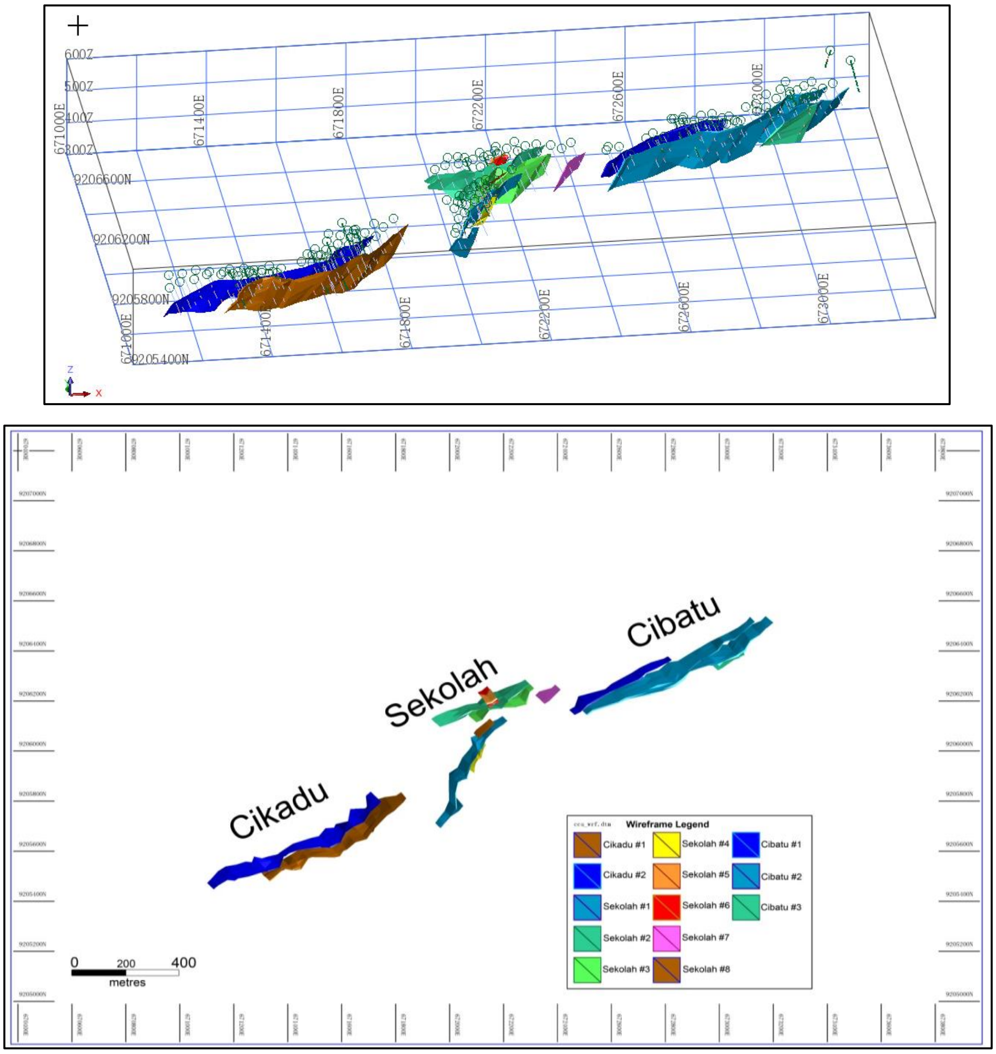


Figure 8-5: 3D Solid Wireframe and Drill Interceptions of C-S-C Mineralised Zones

8.5 Specific Gravity

SRK found that records of the ore density samples from previous exploration were not available, and in the previous resource estimation an overall density of 2.65 tonnes per cubic metre (t/m^3) was used as an assumption. Following SRK's suggestion, a total of 45 specific gravity (SG) samples were collected from the Pasir Manggu West deposit on 4 April 2012 including 15 oxidized ore samples, 15 mixed ore samples, and 15 primary ore samples. They were sent to PT Zhongye Mineral Resources Exploration Development (Zhongye) for analysis using sealing wax densitometry.

Another batch of bulk density samples was collected and analysed for the Cikadu, Sekolah and Cibatu zones in 2012. The tests show that the average density value for the fresh mineralised cores is approximately 2.7 t/m^3.

An SG value of 2.7 t/m^3 was used in the resource estimation. An insufficient number of SG measurements are available for the oxidised zone, although SRK notes that the Company has collected samples from the recently-drilled 30 shallow drillholes to acquire an adequate number of measurements.

8.6 Compositing

Prior to statistical analysis, the samples were composited into equal length composites to provide a constant sample volume. Actual sample lengths from the four properties average 1.0 m.

Based on the sample length statistics (Figure 8-6 and Figure 8-7), the 1.0 m length was considered appropriate for compositing. All data from the Surpac database containing the flagged raw sample intervals were composited to 1.0 m downhole lengths, with a minimum length of 75 cm required to create a composite for grade interpolation. Composites with lengths less than 75 cm within the mineralised zones were also created but coded with a different number and they were not used for grade interpolation.

SRK applied 1.0 m composites within the domains for all subsequent statistical and geostatistical analyses and grade interpolations.

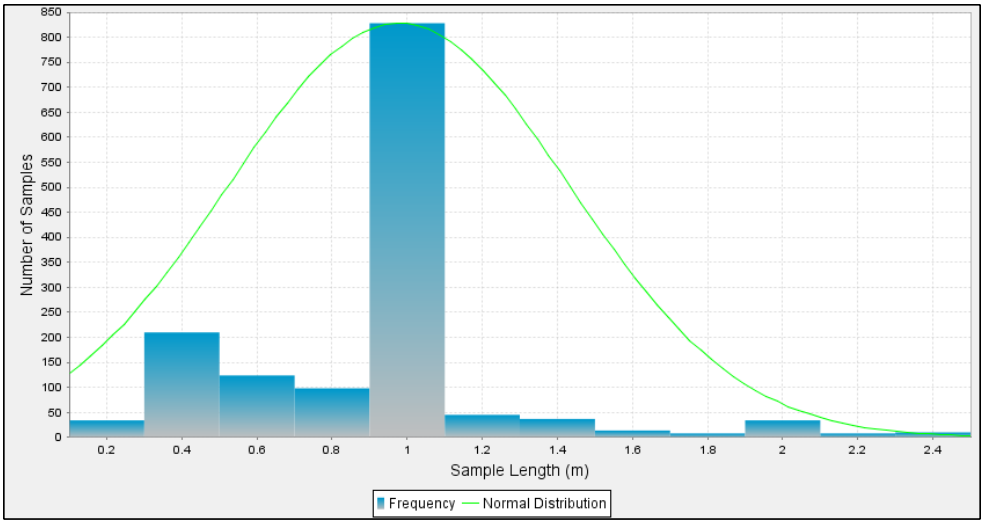


Figure 8-6: Sample Lengths at Pasir Manggu

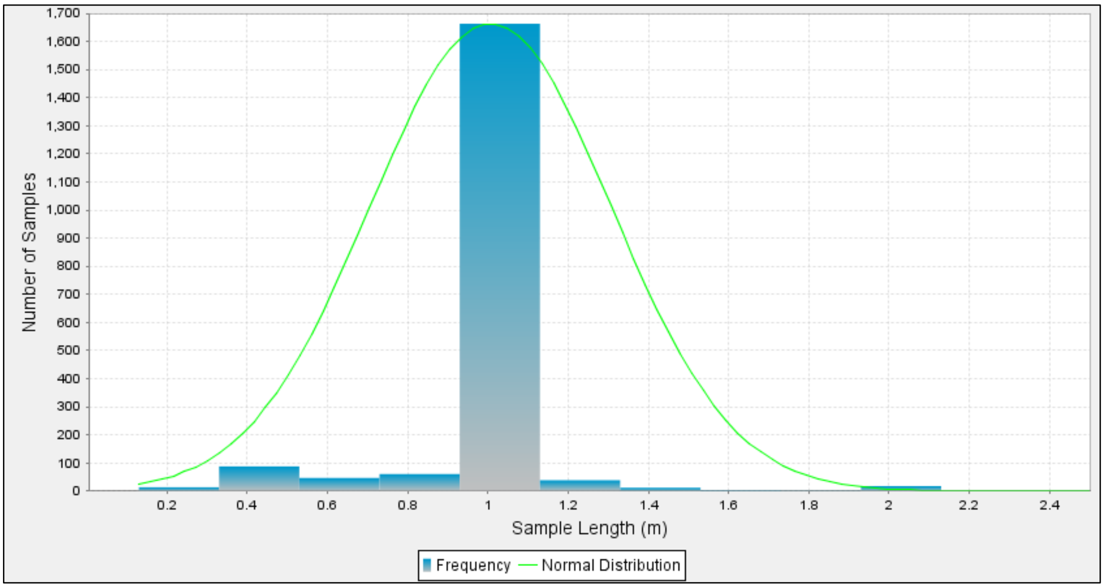


Figure 8-7: Sample Lengths at C-S-C

8.7 Evaluation of Outliers

Top cut analysis for gold was conducted based on statistics for all domains, prior to block model grade interpolation. Top cut analysis is undertaken to assess the influence extreme grades have on the sample population. Although the extreme grades are real, these outliers may result an overstatement of the block grades in some parts of the deposit if left uncut. In order to avoid any disproportionate influence of random, anomalously high grade assays on the resource average grade, SRK performed grade capping after studying the histogram of composite assays for each mineralised zone. Figure 8-8, Figure 8-9, Figure 8-10 and Figure 8-11 show the distributions of the composite assays at Pasir Manggu and C-S-C.

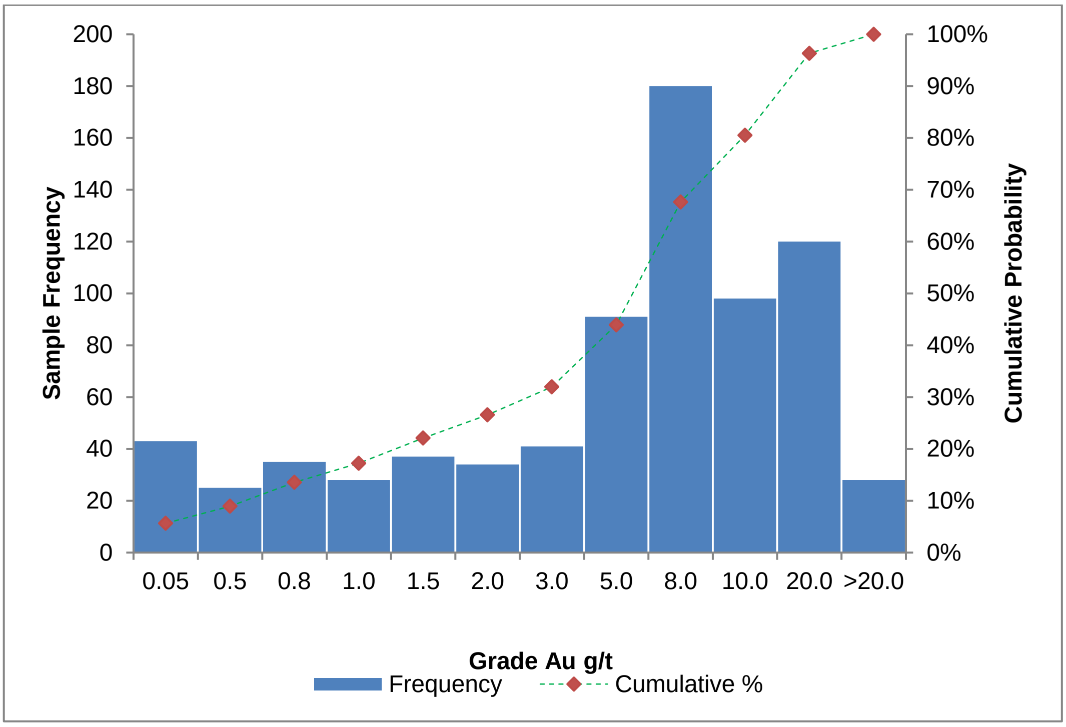


Figure 8-8: Distribution of Composite Assay Grades at Pasir Manggu

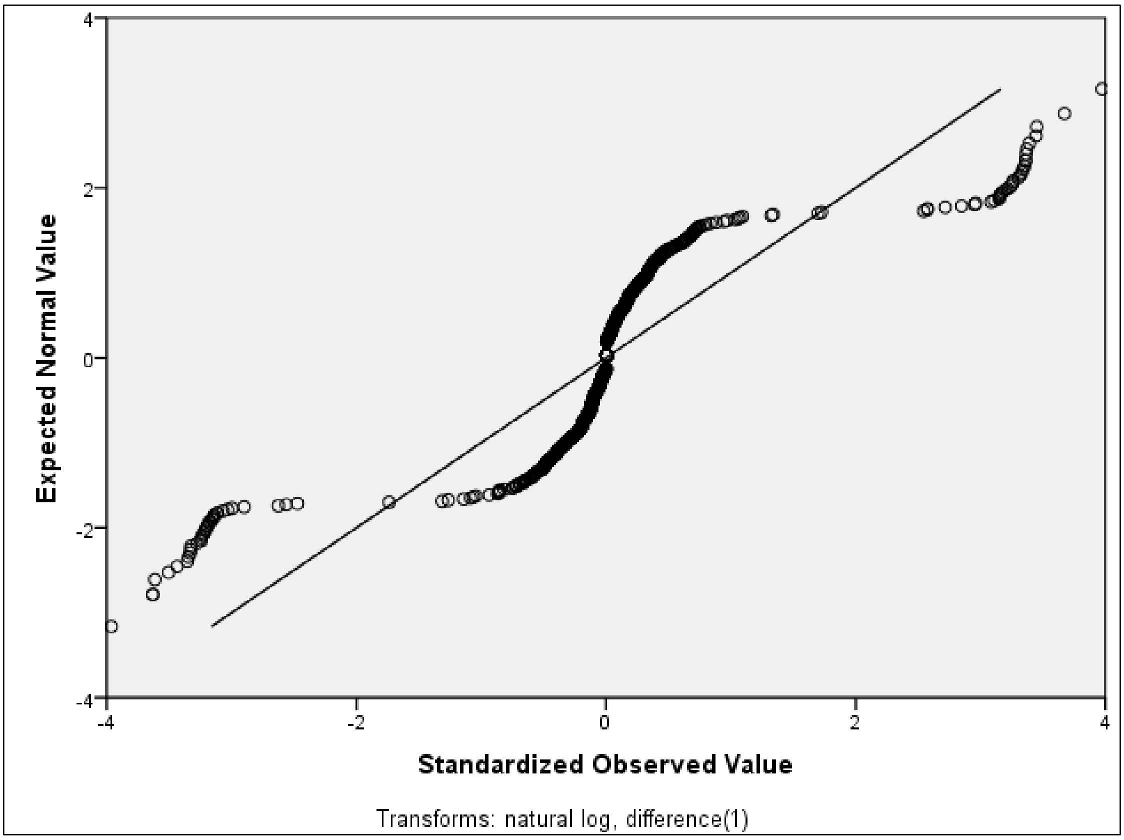


Figure 8-9: Quartile-Quartile Plot of Composite Assays at Pasir Manggu

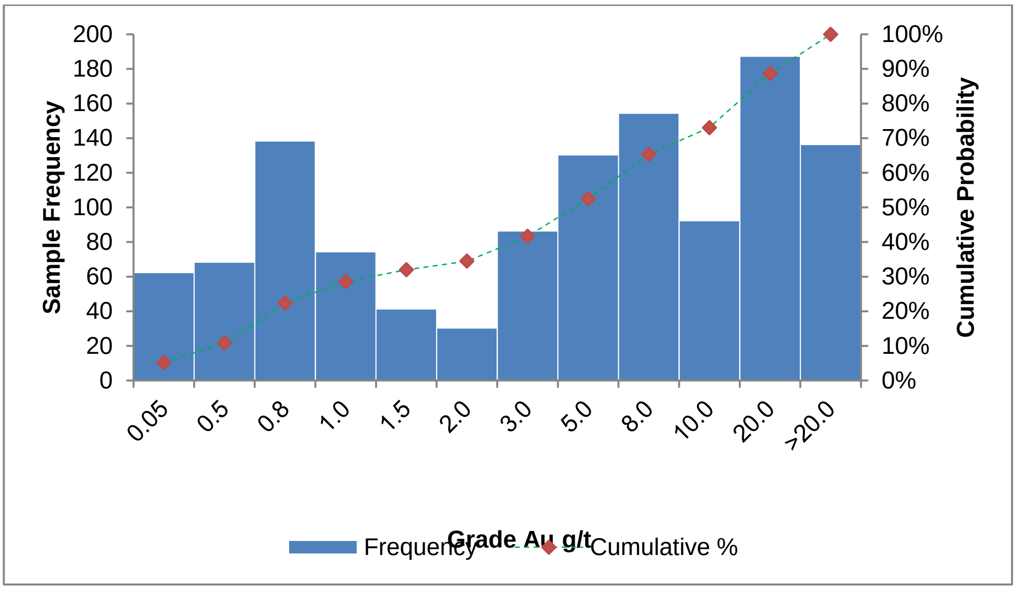


Figure 8-10: Distribution of Composite Assay Grades at C-S-C Areas

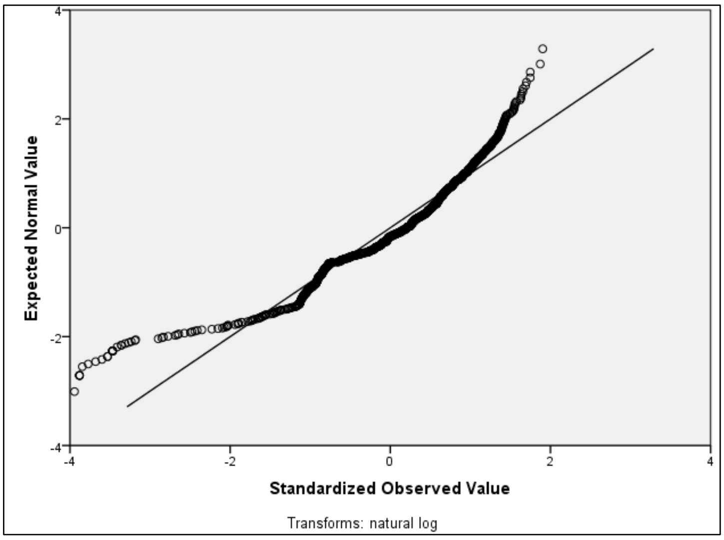


Figure 8-11: Quartile-Quartile Plot of Composite Assays at C-S-C Area

Gold grades in the raw database of Pasir Manggu vary from 0 to 226.0 g/t. Within the wireframes of outlined veins, the maximum grade after compositing is 108.4 g/t Au and the length-weighted maximum grade of composites is 79.5 g/t Au. Basic statistics show that the average grade of raw samples within the modelled mineralized zones at Pasir Manggu is about 6.8 g/t and the grade at the 97.5th percentile is 24.6 g/t Au. Grades were capped at 40 g/t Au at Pasir Manggu. The basic statistics of sample and composite assays for Pasir Manggu are shown in Table 8-2.

Table 8-2: Sample and Composite Grades for Pasir Manggu

Item	Raw Samples within Mineralised Zones	Length-weighted Composites before Grade Capping	Length-weighted Composites after Grade Capping
Number of samples	735	664	664
Minimum value (g/t)	0.0	0.0	0.0
Maximum value (g/t)	226.0	79.5	40.0
25th Percentile	1.2	1.3	1.3
50th Percentile (median)	5.1	5.1	5.1
75th Percentile	9.0	8.5	8.5
95th Percentile	17.2	15.3	15.3
97.5th Percentile	25.6	24.6	24.6
99th Percentile	46.6	33.9	33.9
Mean (g/t)	6.8	6.4	6.3
Variance (g/t)	130.2	48.3	40.7
Standard Deviation (g/t)	11.4	6.9	6.4
Coefficient of variation	1.7	1.1	1.0
Skewness	11.3	3.6	2.4
Kurtosis	196.4	27.0	11.1

Note: The compared raw samples and composites are both within the outlined mineralised zones, and samples outside the wireframes are exclusive. Composites with a length less than 75 cm are not taken in account.

For C-S-C, sample compositing was performed with drill intersections at each modelled mineralised zone. Gold grades in the raw database for all samples vary from 0 to 82.1 g/t, with a mean grade of 6.2 g/t. Within the modelled wireframes the sample grades vary from 0 to 82.1 g/t with a mean grade of 8.2 g/t Au. Grade capping was applied to each body based on composite statistics of each mineralised body. The values of outliers were screened based on the 97.5th percentile of composites at each body (Table 8-3).

Table 8-3: Statistics of Length-weighted Composite Grades and Grade Capping at C-S-C

Mineralised Body	Number of Composites	Minimum Value (g/t)	Maximum Value (g/t)		Mean Grade (g/t)	
			Before Capping	After Capping *	Before Capping	After Capping
Cikadu #1	285	0.0	82.1	45.0	9.4	9.0
Cikadu #2	174	0.0	54.4	36.0	8.3	7.9
Sekolah #1 and #1-a	167	0.0	44.8	32.0	8.5	7.9
Sekolah #2	91	0.2	58.4	34.0	9.3	9.0
Sekolah #3	103	0.0	35.3	34.0	9.2	9.0
Sekolah #4	26	0.0	37.1	33.0	8.3	8.1
Sekolah #5	23	0.0	28.2	20.0	3.5	3.2
Sekolah #6	41	0.0	31.6	25.0	5.1	4.9
Sekolah #7	less than twenty composites					
Sekolah #8	less than twenty composites					
Cibatu #1	44	0.3	8.4	8.4	1.9	1.9
Cibatu #2	319	0.0	78.0	42.0	8.6	8.3
Cibatu #3	41	0.3	46.1	45.0	10.3	10.2

Mineral Resources in Sekolah mineralized bodies #5, #6, #7, and #8 were not estimated due to the insufficient number of drillhole intersections. The values used for grade capping are equal to the maximum values after capping at each mineralised zone.

8.8 Statistical Analysis and Variography

8.8.1 Pasir Manggu

Geostatistical analysis was conducted for major gold mineralised zones within the wireframe built using a cut-off grade of 0.8 g/t. The variogram map function in Surpac was used to confirm the parameters of the variography and search ellipsoid to be used in the grade interpolation. Semi-variograms were calculated within the mineralised zones.

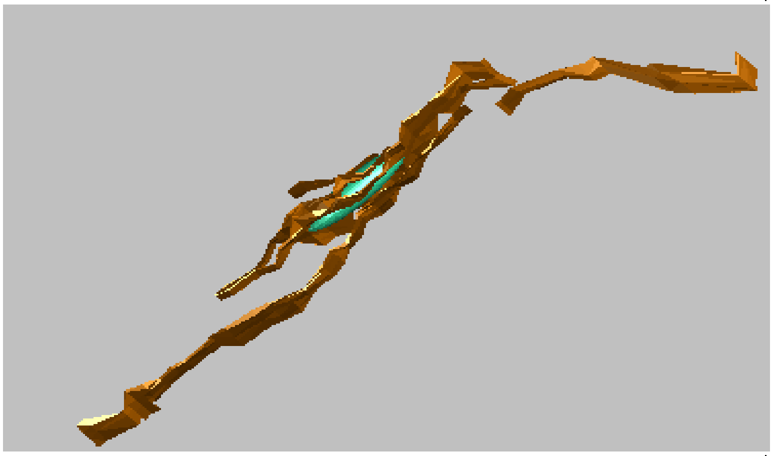
For the small mineralised zones, SRK has concluded that there is insufficient data to construct reasonable variograms. Therefore, the parameters used for major mineralised zones are used for the other small domains.

Ordinary kriging was used for the resource estimation. Variogram modelling was performed and the main parameters used for grade interpolation are shown in Figure 8-12 and the variogram models for the major, semi-major and minor axes are shown in Figure 8-13. The anisotropy of major / semi and major / minor indicate the directionally dependent at 2.5:1 and 5.0:1 in distance, respectively.

Prior to the grade interpolation, SRK validated the variogram models produced by Surpac. For each data point, a kriged grade can be calculated and compared with the measured grade. In order to be considered appropriate the following conditions should be satisfied:

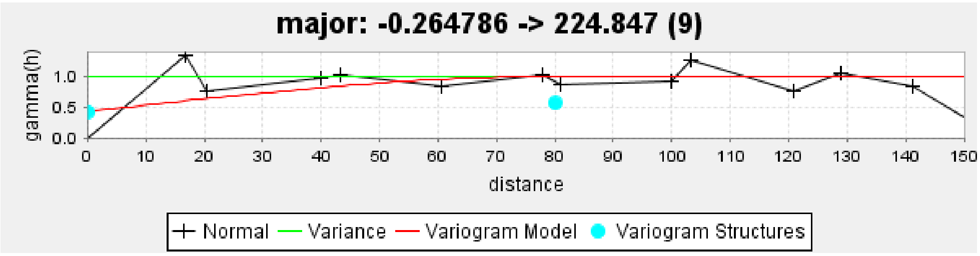
- The average error should be close to zero; and
- The variance of the errors should be close to the average predicted kriging variation.

Based on the analysis shown in Table 8-4 it is clear that the mean of the actual kriging errors is very close to zero, and the percentage of the kriging errors within two standard deviations of the mean is close to 95%, indicating that the spread of kriging errors is within a preferred range; therefore, the variogram models used are appropriate for the data set used.



Ellipsoid Parameters	
Bearing	225
Dip angle	0
Tilt angle	-70
Anisotropy - major/semi	2.5
Anisotropy - major/minor	5.0
Spherical Variogram Model	
Nugget effect	0.43
Sill - Structure	0.57
Range	80

Figure 8-12: Variogram Parameters for Ordinary Kriging



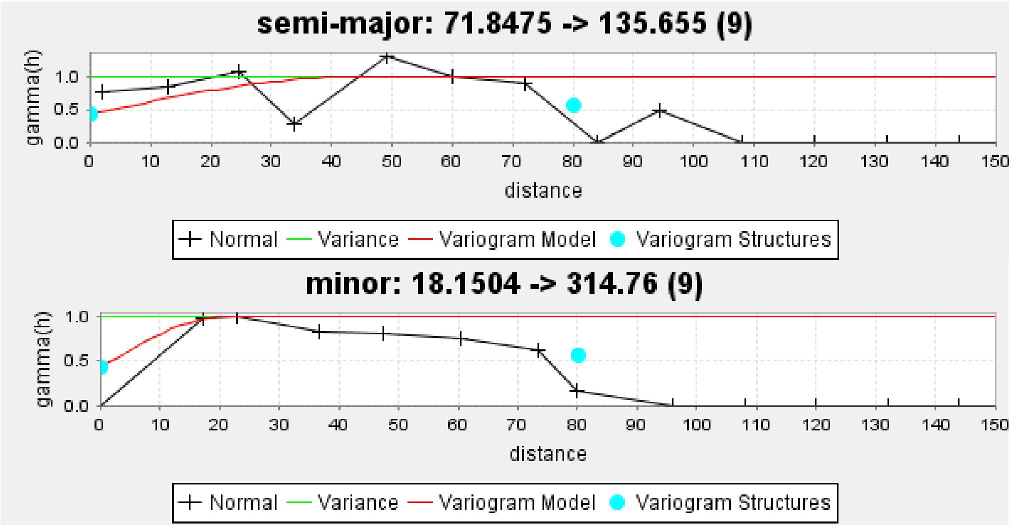


Figure 8-13: Variography for Pasir Manggu

Table 8-4: Kriging Errors for Variography Model Validation

Item	Value
Mean	0.0004
Variance	0.0721
No. of assays	664
Average kriged variance	0.478
Errors within 2 Standard Deviations	93.7%

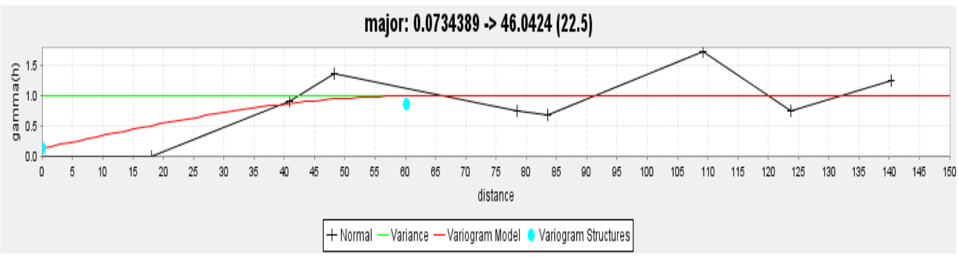
8.8.2 Cikadu, Sekolah and Cibatu

Variograms were generated for mineralised bodies Cikadu #1 and Cibatu #2, and containing 265 and 297 composites, respectively. Ordinary kriging was applied based on the calculated variograms, as shown in Table 8-5, Figure 8-14 and Figure 8-15. As with the variography model validation, SRK has validated the two spherical variogram models.

Table 8-5: Spherical Variograms Used for Ordinary Kriging

Body	Variogram Model	Sill - Structure	Nugget	Range	Bearing	Plunge	Dip Angle	Major /Semi	Major /Minor
Cikadu #1	Spherical	0.87	0.13	60	46	0	-60	1.5	3.1
Cibatu #2	Spherical	0.88	0.12	60	30	0	-65	2.7	3.9

Note: The anisotropy of major / semi and major / minor indicate the ratio of directionally dependent in distance.



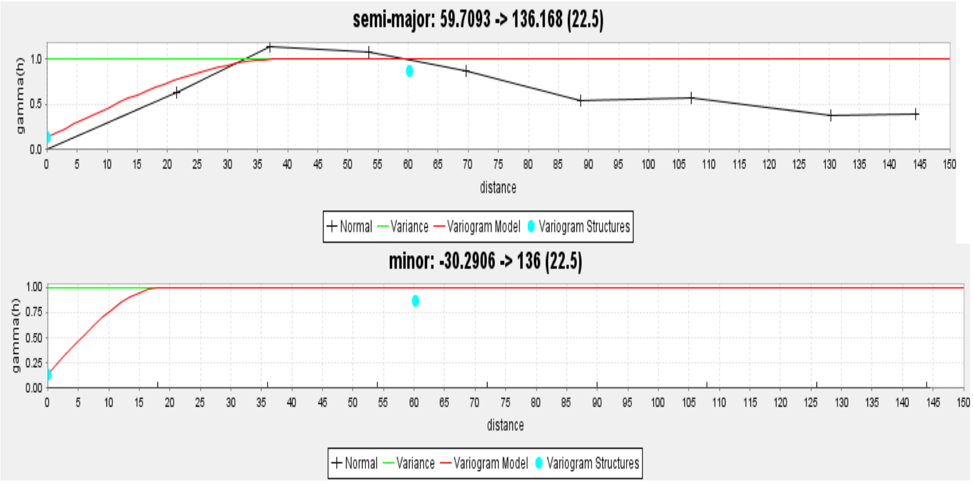


Figure 8-14: Variography for Mineralised Zone Cikadu #1

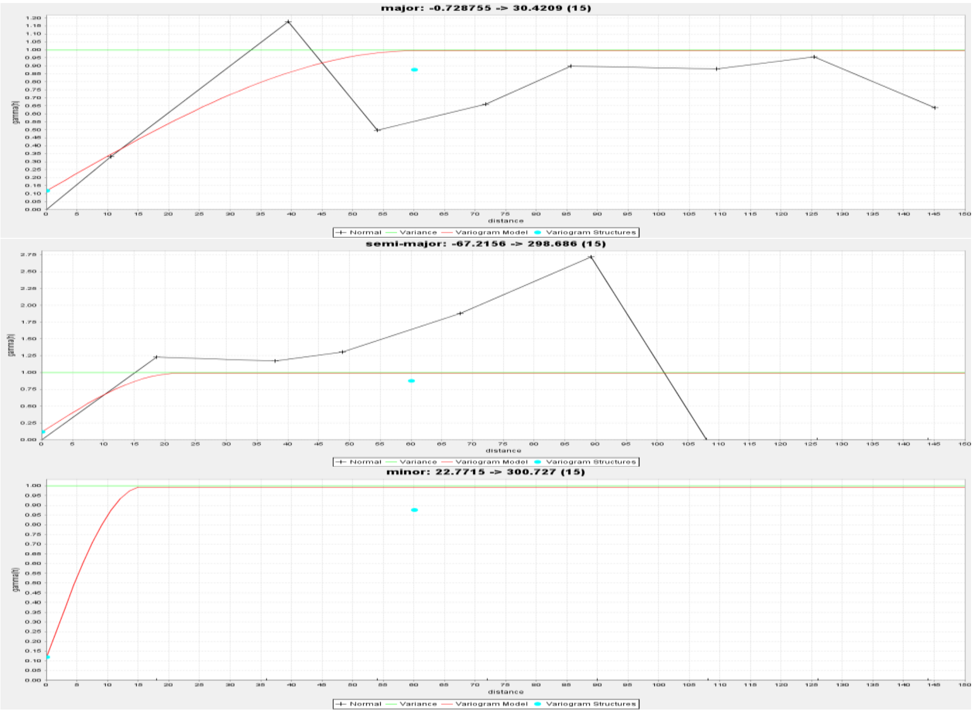


Figure 8-15: Variography for Mineralised Zone Cibatu #2

The Inverse Distance Weighted (IDW) method (square) was applied in the grade interpolation for Cikadu #2, Sekolah #1, #2, #3 and #4, and Cibatu #1 and #3. The anisotropy was studied by comparing the geometry of the mineralised zones and the parameters for search ellipsoids are given in Table 8-6.

Table 8-6: Anisotropic Parameters for IDW

Body	Bearing	Plunge	Dip Angle	Major / Semi	Major / Minor
Cikadu #2	50	0	-65	2.6	3.5
Sekolah #1	35	0	-70	2.7	3.5
Sekolah #2	55	0	-65	2.6	4.2
Sekolah #3	60	0	-65	2.6	4.0
Sekolah #4	30	0	-65	2.7	4.0
Cibatu #1	50	0	-60	2.7	4.5
Cibatu #3	50	0	-60	2.7	3.5

8.9 Block Model and Grade Estimation

8.9.1 Pasir Manggu

A block model was created based on the distribution and range of the mineralized veins. A total of 11,004 blocks are included with minimal block size of 1 m (Y axis, northing) by 5 m (X axis, easting) by 2.5 m (Z axis, elevation). A summary of the block model is shown in Table 8-7. Grade interpolation is constrained by the solid 3D wireframe model of mineralized veins and the surface topography.

Table 8-7: Block Model Summary for Pasir Manggu

Block Model Geometry

Min Coordinates	Y	9205520	X	670640	Z	400
Max Coordinates	Y	9206590	X	672000	Z	540
User block Size	Y	2	X	10	Z	5
Min. block Size	Y	1	X	5	Z	2.5
Rotation	Bearing	0	Dip	0	Plunge	0

Block Summary

Total No. Blocks	11004
Storage Efficiency %	99.93

Grade estimation was performed in three rounds, applying the variography model (search ellipsoid), with a maximum search distance of (1) 100 m to estimate all blocks constrained in the defined mineralised zones, (2) 60 m to apply the variography within its calculated range, and (3) 30 m to improve the grade interpolation in certain local zones.

8.9.2 Cikadu, Sekolah and Cibatu

A block model was set up for the C-S-C resource estimation, and the prototype is shown in Table 8-8. The block model was used for all mineralised domains and was constrained below the topography as surveyed by Wilton on 30 April 2012. No material changes have occurred to the surveyed topography since the date of the survey.

Table 8-8: Block Model Summary for Cikadu, Sekolah, and Cibatu

Block Model Geometry

Min Coordinates	Y	9205400	X	671100	Z	320
Max Coordinates	Y	9206700	X	673300	Z	540
User block Size	Y	10	X	10	Z	5
Min. block Size	Y	5	X	5	Z	2.5
Rotation	Bearing	0	Dip	0	Plunge	0

Block Summary

Total No. Blocks	135047
Storage Efficiency %	98.65

Grade interpolation was constrained within the modelled wireframes of the mineralised bodies following two rounds of search passes. Samples outside the interpreted solids were excluded from the grade estimation. The first search pass, with a maximum distance of 100 m constrained within the mineralised bodies, was employed to estimate the Inferred Resource blocks; and the second search pass, with a maximum distance of 50 m, was used for more confident grade estimations on potential Indicated Resource blocks.

8.10 Model Validation and Sensitivity

SRK validated the block model to confirm the reasonableness of the estimation parameters and estimation result using the following methods:

- Visual validation of block grades against drill hole grades;
- Swath plots; and
- Statistical validation of the mean composite grades versus block estimates.

Visual validation from both 3D plan views and cross-sectional and longitudinal views revealed that the estimated block grades generally honour the drilling results. An example of the visual validation is shown in Figure 8-16.

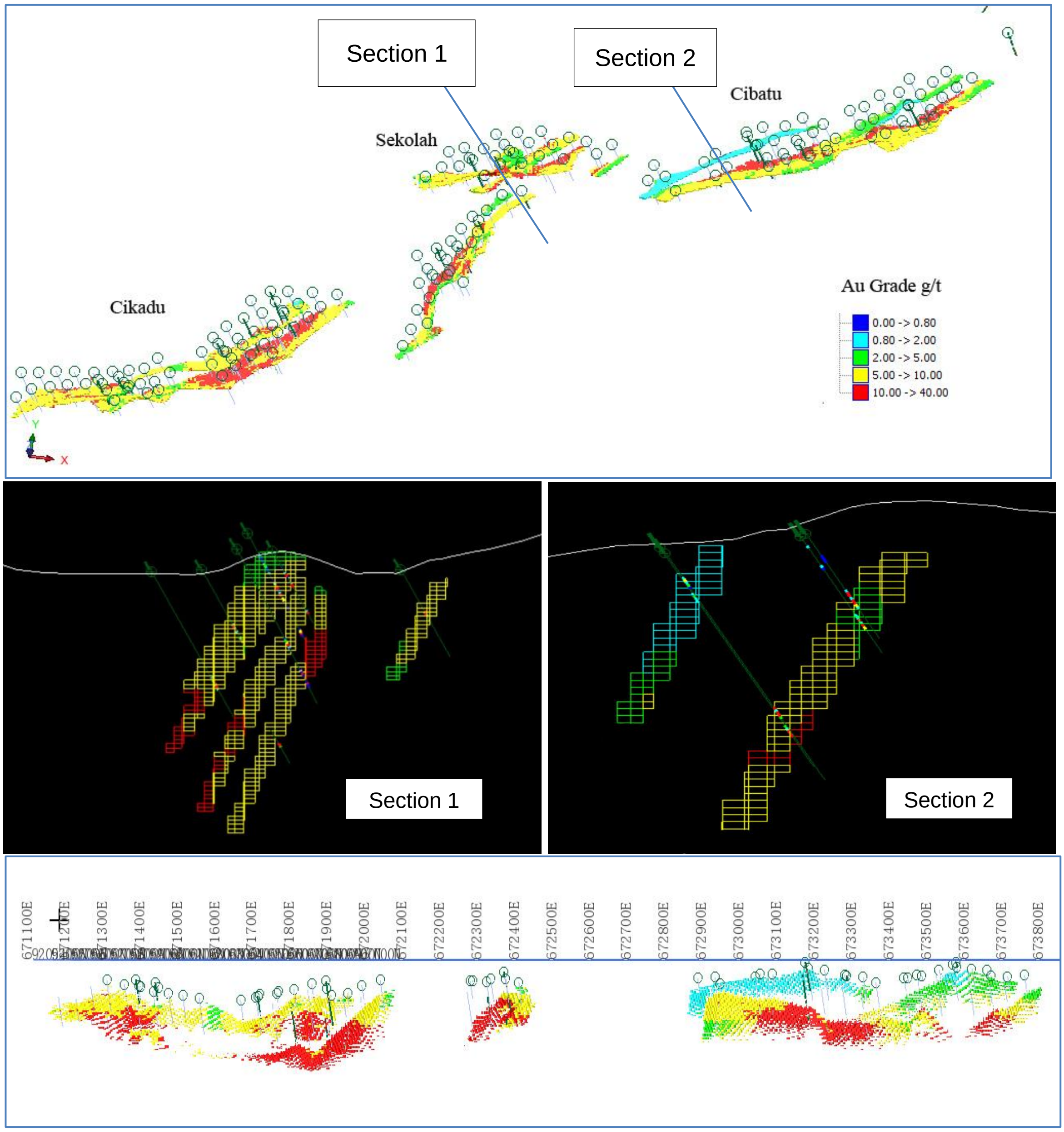


Figure 8-16: Visual Validation of the Estimated Resource Block with Drill Interceptions at C-S-C Zones

Note: top – plan view, middle – cross section; bottom – long section.

Swath plots were generated to compare the model grade and tonnage with the drill hole grades and the number of samples over various directions, i.e. per bench, along E-W lines, and along N-S lines. The swath plots indicate a good correlation between the drillhole assays and estimated grade models.

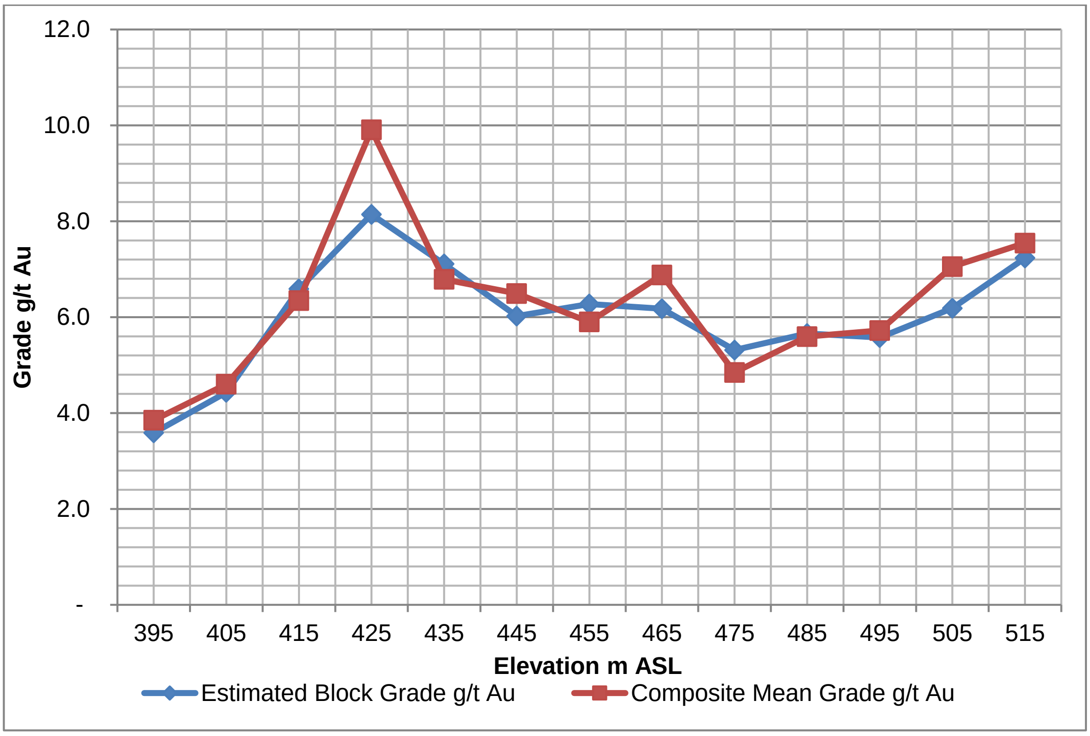


Figure 8-17: An Example of Swath Plot Along Vertical Direction at Pasir Manggu

The estimated block grades were compared with the composite mean grades in the same mineralised zone/vein. The comparison suggests that the estimated block mean grades are close to the composite mean grades, with relative differences within the range of 5%, as shown in Table 8-9.

Table 8-9: Comparison between Block Mean Grades and Composites Mean Grades

Mineralised Zone	Composites Mean After Grade Capping (g/t)	Block Mean (g/t)	% Difference	Absolute Difference (g/t)
Pasir Manggu	6.3	6.3	0.0	0.0
Cikadu	8.5	8.9	4.6	0.4
Sekolah	8.8	9.0	2.2	0.2
Cibatu	8.4	8.7	3.5	0.3

8.11 Mineral Resource Classification

Block model quantities and grade estimates for the Ciemas Gold Project were classified according to the JORC Code by Mr Pengfei Xiao, MAusIMM under the guidance and supervision of Dr Anshun Xu, FAusIMM, an appropriate independent Competent Person as defined by the JORC Code.

Mineral Resource classification is typically a subjective concept. Industry best practice suggests that resource classification should consider the confidence in the geological continuity of the mineralised structures, the quality and quantity of exploration data supporting the estimates, and the geostatistical confidence in the tonnage and grade estimates. Appropriate classification criteria should aim at integrating these concepts to delineate regular areas at similar resource classification levels.

SRK is satisfied that the geological modelling presented in this report represents the current geological information and knowledge. The locations of the samples and the assay data are sufficiently reliable to support resource evaluation. The sampling information was acquired primarily by diamond and RC drilling. To define the Mineral Resource, SRK assumed a cut-off grade of 1 g/t Au. The following guidelines were applied to the resource classification by SRK:

- Mineral resources are categorized on the basis of geological confidence derived from different exploration data (DDHs, RCHs, and surface trenches/pits used for geological interpretation).

- Exploration grids are frequently referenced in the classification of resource categories.

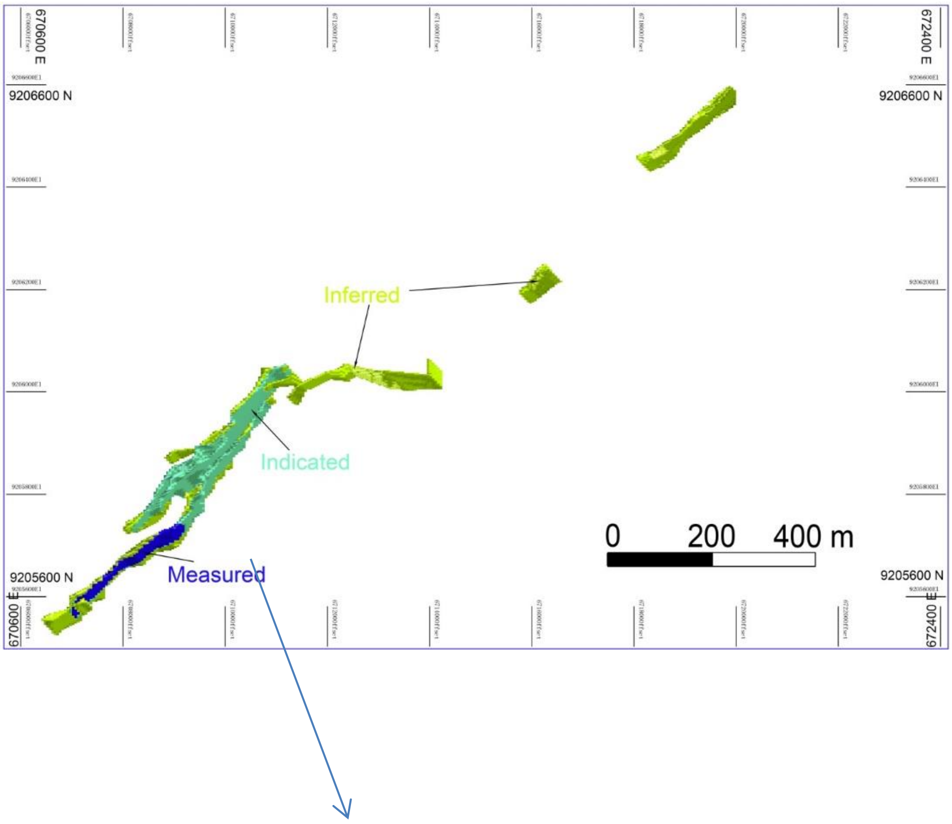
8.11.1 Pasir Manggu

For the Pasir Manggu property, Measured Resources are defined within a basic DDH grid of 20 m × 20 m and the average distance of grade interpolation in a Measured block is limited within 25 m. No RCH data was used in the estimation of Measured Resources. Measured Resources are only assigned to a part of Vein #1, which is defined with a high density of DDHs.

Indicated Resources are assigned to blocks within a basic DDH grid of 40 m × 40 m. The maximum ellipsoid searching distance for Indicated blocks is 50 m. No RCH data was used in the Indicated Resource estimation. Veins #2, #3, and #4 are partly assigned as Indicated Resources.

Within the delineated mineralised veins, Inferred Resources are estimated based on the geological extrapolation from Measured and Indicated Resources and the supplementary data derived from RCHs. The sectional extrapolation of mineralized veins from drill control is generally 10 m – 20 m down dip. All veins except #1, #2 and #3 are categorized as Inferred Resources.

A longitudinal projected view of the resource categorisation at two main veins is shown in Figure 8-18. Subsequent to this process, SRK smoothed the boundaries of each category to remove block irregularities.



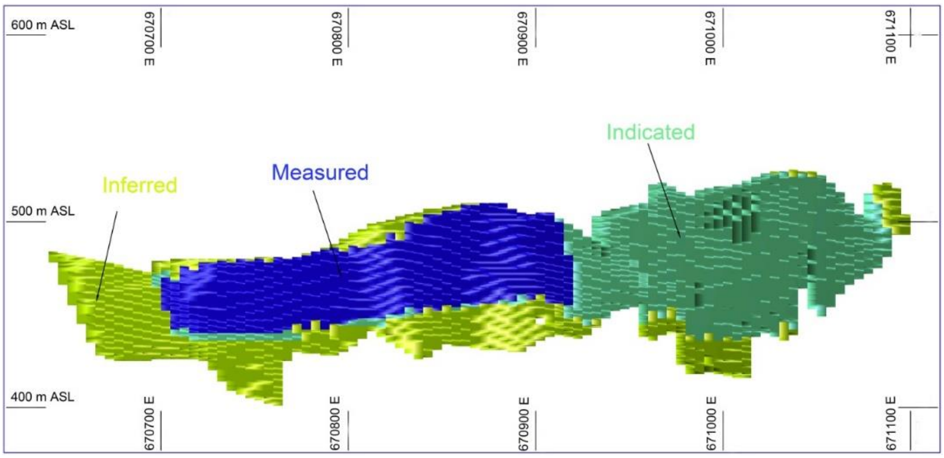


Figure 8-18: Resource Categorisations of Pasir Manggu Mineralised Veins

8.11.2 Cikadu, Sekolah and Cibatu

Grade interpolation was constrained within the modelled wireframes of the mineralised bodies following two rounds of search passes. Samples outside the interpreted solids were excluded from the grade estimation. The first search pass, with a maximum distance of 100 m constrained within the mineralised bodies, was employed to estimate the Inferred Resource blocks, and the second search pass, with a maximum distance of 50 m, was used for more confident grade estimations on potential Indicated Resource blocks.

The categorisation of Mineral Resource for the C-S-C properties, as defined in JORC Code 2012 Edition, was performed based on geological confidence derived predominately from data density. Of all the mineralized veins, six veins, namely Cikadu #1 and #2, Sekolah #1, #2, and #3, and Cibatu #1, were intersected by drillholes laid out on a grid of 40 m by 40 m, and the others were interpreted from a sparser drilling grid. SRK considered that Indicated Resources could be appropriately assigned to those estimated blocks:

- With at least two drillholes located within 40 m and no more than 40 m from the nearest informing samples;
- Constrained within Cikadu #1, #2, Sekolah #1, #2, #3 and/or Cibatu #1; and
- Estimated in the second search pass.

All other estimated blocks were categorised as Inferred Resources. Figure 8-19 shows the resource categorisation for the C-S-C properties prior to smoothing of the blocks to remove irregularities.

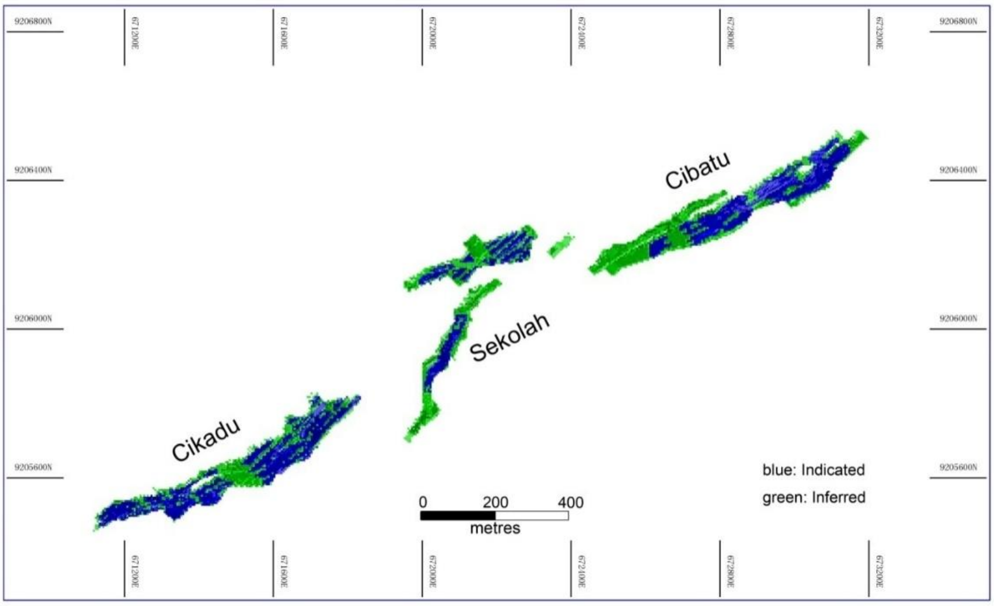


Figure 8-19: Resource Categorization of the C-S-C Zones in the Planar

8.12 Mineral Resource Statement

The JORC Code defines a Mineral Resource as

"(A) concentration or occurrence of diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics, and continuity of a Mineral Resource are known, estimated, or interpreted from specific geological evidence and knowledge".

The "reasonable prospects for economic extraction" requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade, taking into account extraction scenarios and processing recovery rates. In order to meet this requirement, SRK considers that major portions of the Ciemas Gold Project are amenable for open pit extraction followed by underground mining.

The assumptions for cut-off grade selection were considered based on experience and benchmarking against similar projects, as well as previous studies completed of the Ciemas Project. SRK is of the opinion that a gold grade of 1.0 g/t is an appropriate cut-off grade for the Mineral Resource statement for the 4 Prospects (2.5 g/t Au for Cibak and Cipancar) at this stage, assuming a gold price of around US$1,300/oz.

Table 8-10 gives the Mineral Resource Statement for the Ciemas Project. As of 30 June 2018 and under a cut-off grade of 1.0 g/t Au, the Project contains about 3.42 million tonnes (Mt) of Measured and Indicated Resources with an average grade of 8.6 g/t Au, in addition to 2.56 Mt of Inferred Resources averaging 6.5 g/t Au.

Table 8-10: Mineral Resource Statement, Ciemas Gold Project, as of 30 June, 2018

Property	Type	Category	Resource (kt)	Au (g/t)	Au (kg)
Pasir Manggu West	Oxide	Indicated	109	7.2	783
		Inferred	36	5.6	200
	Fresh	Measured	100	7.3	731
		Indicated	380	7.3	2,776
		Inferred	206	4.7	975

Property	Type	Category	Resource (kt)	Au (g/t)	Au (kg)
Cikadu	Oxide	Indicated	81	6.2	496
		Inferred	20	6.9	134
	Fresh	Indicated	1,008	9.1	9,126
		Inferred	280	9.7	2,718
Sekolah	Oxide	Indicated	89	5.8	510
		Inferred	128	4.9	621
	Fresh	Indicated	612	9.6	5,869
		Inferred	326	8.3	2,689
Cibatu	Oxide	Indicated	129	6.2	794
		Inferred	78	3.0	233
	Fresh	Indicated	907	9.1	8,216
		Inferred	377	7.8	2,951
4 Prospects Total	Oxide	Indicated	407	6.3	2,583
		Inferred	261	4.5	1,188
	Fresh	Measured + Indicated	3,007	8.9	26,718
		Inferred	1,188	7.9	9,332
	Oxide + Fresh	**Measured + Indicated**	**3,415**	**8.6**	**29,301**
		Inferred	**1,449**	**7.3**	**10,520**
Cibak	Oxide + Fresh	Inferred	660	5.6	3,717
Cipancar	Oxide + Fresh	Inferred	450	5.6	2,520
Cibak & Cipancar Total	**Oxide + Fresh**	**Inferred**	**1,110**	**5.6**	**6,237**
4 Prospects + Cibak & Cipancar Total	**Oxide + Fresh**	**Measured + Indicated**	**3,415**	**8.6**	**29,301**
	Oxide + Fresh	**Inferred**	**2,559**	**6.5**	**16,757**

Note: Mineral Resources are not Ore Reserves and do not have demonstrated economic viability. All figures are rounded to reflect the relative accuracy of the estimate. All composites have been capped where appropriate.

Figures for Au metal in this table are estimated based on the resource tonnages and grades, and do not represent the exact amount of extractable metal for this Project. They should be treated differently from the expected production of gold bullion.

The information in this Report which relates to Mineral Resource estimates is based on information compiled by Dr Anson Xu, and Mr Pengfei Xiao, employees of SRK Consulting China Ltd. Dr Xu, FAusIMM, and Mr Xiao, MAusIMM, have sufficient experience relevant to the style of mineralisation and type of deposit under consideration and to the activity which they are undertaking to qualify as Competent Persons as defined in the 2012 Edition of the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves. Dr Xu and Mr Xiao consent to the reporting of this information in the form and context in which it appears.

Grade sensitivity analysis suggests that the Ciemas Gold Project is not sensitive to cut-off grades ranging from 0 g/t up to 3 g/t, which reflects the characteristics of gold deposits with narrow quartz veins and high-grade nuggets in the structural alteration zones.

8.13 Comparison with Recent Resource Estimates

The Mineral Resource estimate for the 4 Prospects (Pasir Manggu, Cikadu, Sekolah and Cibatu) presented in this Report is an update of the recent publically released one that was prepared as of 30 June 2014, in compliance with the JORC Code 2012 Edition.

Resource tonnage in this Report is increased by approximately 375 kt (0.375Mt) of Measured and Indicated Resources in the 4 Prospects, resulting an increment of approximately 2.56 tonnes of gold contained. The changes are also reflected by a reduction of about 151 kt (0.151 Mt) of Inferred Resources with about 1.71 tonnes of gold contained.

The changes of Mineral Resource estimates for the 4 Prospects are predominately due to additional drilling results incorporated into the current estimates. Changes also include the refining of the previous resource model parameters, which results in some adjustment of the grade estimation.

The Mineral Resource estimate for the Cibak and Cipancar prospects presented in this Report is has followed the resource model and estimates that were dated as at 31 August 2016 and published in February 2017. There are not any material changes to the recent statement for Cibak and Cipancar prospects expect a few discrepancies due to the figure rounding.

Table 8-11: Resource Estimates Comparison – SRK, 2018 and 2014/2016

Property	Category	Model Report as of 30 June 2018			Model Report as of 30 June 2014		
		Resource (kt)	Au (g/t)	Au (kg)	Resource (kt)	Au (g/t)	Au (kg)
Pasir Manggu	Measured	100	7.3	731	120	7.3	870
	Indicated	489	7.3	3,559	450	7.5	3,390
	Inferred	242	4.9	1,174	270	3.8	1,030
Cikadu	Indicated	1,089	8.8	9,622	1,100	9.1	9,970
	Inferred	299	9.5	2,852	360	8.4	3,040
Sekolah	Indicated	700	9.1	6,379	710	9.2	6,520
	Inferred	453	7.3	3,310	300	8.6	2,580
Cibatu	Indicated	1,036	8.7	9,010	660	9.1	5,990
	Inferred	455	7.0	3,184	670	8.3	5,580
Sub-total (4 Prospects)	**Measured**	**100**	**7.3**	**731**	**120**	**7.3**	**870**
	Indicated	**3,315**	**8.6**	**28,570**	**2,920**	**8.9**	**25,870**
	Measured+Indicated	**3,415**	**8.6**	**29,301**	**3,040**	**8.8**	**26,740**
	Inferred	**1,449**	**7.3**	**10,520**	**1,600**	**7.6**	**12,230**

Property	Category	Model Report as of 30 June 2018			Model Report as of 31 August 2016		
		Resource (kt)	Au (g/t)	Au (kg)	Resource (kt)	Au (g/t)	Au (kg)
Cibak	Inferred	660	5.6	3,717	663	5.6	3,712
Cipancar	Inferred	450	5.6	2,520	451	5.6	2,516
Sub-total (Cibak and Cipancar)	**Inferred**	1,110	5.6	6,237	1,114	5.6	6,228

Note: discrepancies may occure due to figure rounding.

9 Mining Assessment

The Ciemas Gold Project is primarily designed with underground mining stopes and back-filling systems for the fresh zones and secondarily with open pit extractions for the oxidised zones at shallow surface.

The open-pit mining is designed to start at the last three years of the mining schedule when the underground mining capacity decreases. The open pit production will support to maintain a total throughput of 1,500 tonnes ore for processing plant per day.

The safety pillar leaved at surface is about twenty meters thick, and it has been assumed that the cut and fill method will be applied to underground mining operation. Usually, there will be void space occurrence in a stope below the surface safety pillar after several years shrinking of fill body. Therefore, filling the stope again is required to support safety operation of open pit mining.

9.1 Geotechnical and Hydrogeological Conditions

9.1.1 Geotechnical Condition

The local lithology of the belt is formed mainly of volcanic breccia (VBR) and mostly covered by quaternary eluvium and alluvium as well as post-mineralisation tuff blanket up to 20 m thick. Volcanic breccia, tuffs, and andesite are widely distributed in the project area.

The orebodies are generally quartz veins, or structurally altered rocks with tectonic breccia, or in quartz dacite porphyry. It is estimated that the orebodies are typically 3.0-4.0 m thick, dipping predominantly 75° to the south-east for PSM, dipping 60° to the north-west for SEK south, and dipping 60-65° to the north-north-west for SEK2 north and CKD.

Data acquired from verification drill holes (DDH1003, DDH1021, DDH1031, and DDH1041) indicate that wall rocks are generally composed of volcanic breccia in the prospect areas of Pasir Manggu, Cikadu, and Cibatu-Sekolah. The mechanical properties of intact rock were measured on a few specimens using Point Load Tests (Is50), Uniaxial Compressive Strength (UCS) tests and triaxial tests. According to the Darmawan 2012 Report and Golder 2016 Report, the specimens were sampled from relatively shallow rocks for CBT, processing plant (PP) and TSF, and mostly from foot walls for PSM, SEK and CKD. UCS values range from medium strong (31.3 MPa) to very strong (68.2 MPa). Due to the diversity of sampling location and depth, it is estimated that the average UCS value of 46.2 MPa with a range between medium strong and very strong could represent overall intact rock strength for the Ciemas Gold Project, which is considered applicable in mining geotechnical stability analyses at a minimum. It appears that the hanging walls and footwalls are medium strong to very strong. The following paragraphs provide the analysis principles and suggestions for the underground mining from the geotechnical perspectives.

Rock Mass Classification is the process of placing a rock mass into groups or classes on defined relationships (Bieniawski, 1989) and assigning a unique description (or number) to it, on the basis of similar properties/characteristics, so that the behaviour of the rock mass can be predicted. The Rock Mass Rating (RMR) scheme uses five classification parameters: strength of the intact rock material, Rock Quality Designation (RQD), spacing of joints, condition of joints and groundwater conditions. The numerical value of the RMR value ranges from 0 to a maximum of 100. For the Ciemas Gold Project, the total RMR values of VBR is 57–80, indicating the rock mass quality is FAIR to GOOD.

For the Ciemas Gold Project, given the VBR rock mass is FAIR to GOOD in quality, corresponding local supports may be required. Based on support measures with the RMR system (after Bieniawski 1989), it is suggested that:

- for Good Rock (RMR = 61–80), locally 3 m long bolts in crown, spaced 2.5 m with occasional wire mesh, and 50 mm shotcrete in crown where required
- for Fair Rock (RMR = 41–60), 4-m long systematic bolts, spaced 1.5–2 m in crown and walls with wire mesh in crown, and 50–100 mm shotcrete in crown and 30 mm in sides

- for the worst-case scenario when rock mass quality is Poor (RMR = 21–40), the support measures could be 4–5 m long systematic bolts, spaced 1–1.5 m in crown and walls with wire mesh; shotcrete of 100–150 mm in crown and 100 mm in sides; and steel set with light to medium ribs spaced 1.5 m where required
- for stope walls, rock bolting should be into the immediate hanging walls and foot walls.

9.1.2 Hydrogeology

According to the regional hydrogeological map of Indonesia, Sheet III Ujungkulon and IV Sukabumi (Java) with a scale of 1: 250,000 generated by Soetrisno. S in 1985, the project area is in a poor groundwater productivity area with very low groundwater yield or without exploitable groundwater. Two Chinese design institutes, Henan Metallurgical Planning and Design Institute and Shandong Gold Group Yantai Design and Research Institute, independently estimated the maximum underground inflow for West Pasir Manggu, which were 260 m^3/d (3.0 l/s) and 600 m^3/d (6.9 l/s) respectively. However, backup calculations and related parameters with assumptions were not provided. It seems that the design institutes both assumed the inflow to the underground mining was not significant.

In 2012, PT. Wilton Wahana Indonesia carried out follow-up detailed exploration work by conducting borehole activities in several portions of the project area:

- West Pasir Manggu (6 boreholes, terminated up to a depth of 150 m)
- Sekolah (1 borehole, terminated at a depth of 95 m)
- Cikadu (1 borehole, terminated at a depth of 120.10 m)
- Cibatu (1 borehole, terminated at a depth of 70.40 m).

Based on the boring logs, PT. Wilton Wahana Indonesia established the following 3D views shown in Figure 9-1 below:

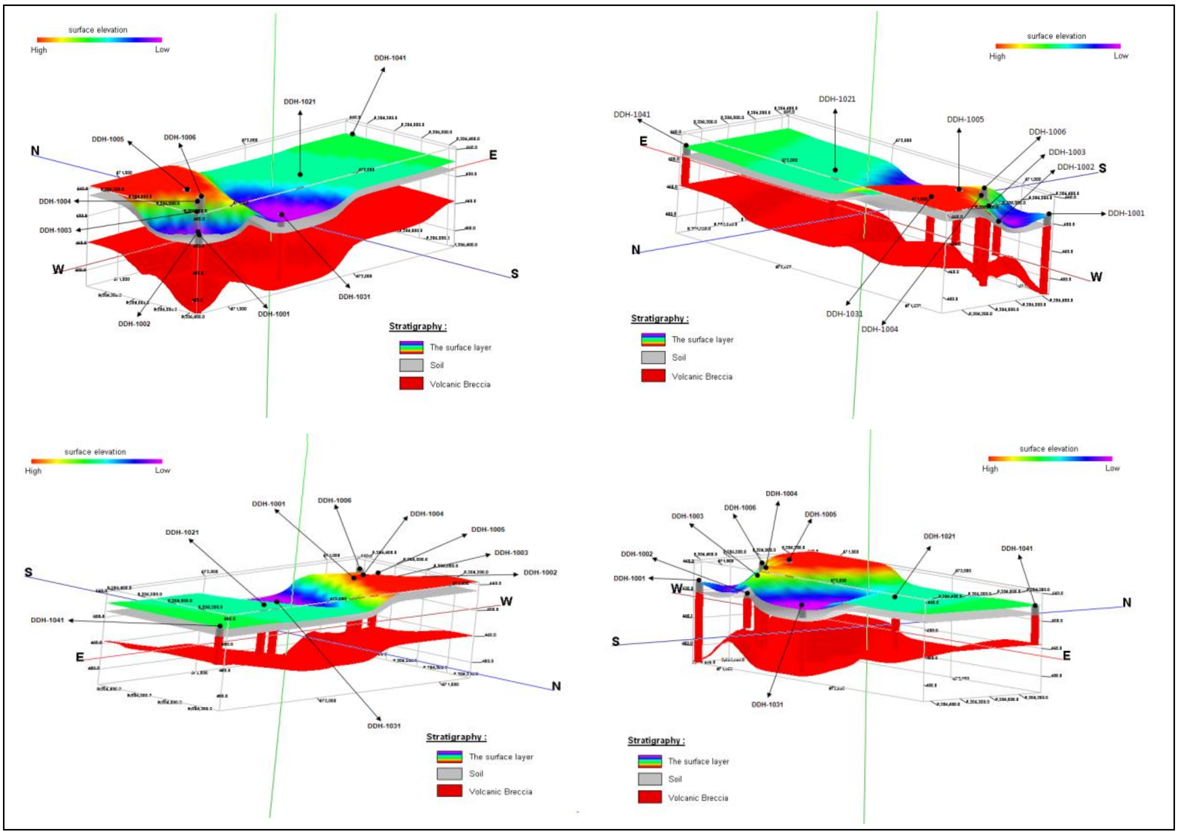


Figure 9-1: 3D Views of SW, NW, NE, and NW Geological Layers

The general site stratum consists of three layers, which are soil layer, saprolite (weathering transition) layer and bedrock (volcanic breccia) layer in descending order from the surface. Since no in situ hydrogeology tests were conducted, site-specific parameters including hydraulic conductivity and transmissivity were not estimated and a site-specific hydrogeological model with inflow, outflow, and other boundary conditions was not established during this study.

In May 2016, Golder conducted limited hydrogeological investigation for the proposed processing plant and TSF area, and a total of 14 bore holes were drilled up to 24 m deep below the ground surface. No further in situ hydraulic tests were conducted in the saprolite layer and slightly weathered to fresh volcanic breccia beneath the saprolite layer down to 150 m below the surface during this investigation. However, Golder estimated the hydraulic conductivities in these two layers based on its knowledge in other volcanic terrains in Indonesia. Average permeability of the saprolite layer is estimated to be about 10^{-5} to 10^{-6} m/s, and the average permeability of the volcanic breccia beneath the saprolite layer is likely to be about 10^{-7} to 10^{-9} m/s. The thicknesses of the saprolite layer and the volcanic breccia layer were not estimated for underground mining purposes.

Golder also conducted a preliminary hydrogeological assessment. It is likely that groundwater is recharged by rainfall infiltration within a preferential surface area and moves downwards under steep hydraulic gradients and is then deflected laterally, most likely in the weathered breccia and saprolite zone. Groundwater will likely either continue to move downward in the fracture systems in the volcanic breccia or ore-bearing zones, or discharge to perennial streams and/or springs. As such, much of the immediate recharge may re-emerge in a relatively short distance through shallow groundwater flow systems, discharging both through springs and as baseflow into creeks. Golder also considers that only a small portion of rainfall recharge is expected to continue moving downwards through the volcanic breccia and to recharge the regional groundwater flow system in the underground mining area. However, SRK is of the view that pumping tests and packer tests should be conducted systematically in the proposed underground mining area to estimate the permeability and groundwater yield rates for dewatering system design purposes. Meanwhile, the inflow to the underground workings is estimated by Dupuit Equation based on the available hydrogeological information.

9.2 Mine Design Criteria

Based on the geographical distribution, geological exploration level and resources/reserves conditions, four vein group sections in the south west portion of the deposit are included in the scope of this study, they are the Pasir Manggu West (PSW), the Cikadu (CKD), the Sekolah (SEK) and the Cibatu (CBT) sections.

The four vein groups are mainly distributed in the elevation of 330–530 m ASL, and outcrop on the surface. To ensure the safety of mining operation, orebodies within 20 m of the surface (mainly oxide ore) are designed as crown pillars, which will not be recovered underground. Open cast is considered for recovering part of these resource, which is about 15 m from the surface. The remnant 5 m will be retained as a safety pillar.

The underground mine design scope is as follows:

- PSW: from 510 to 410 m ASL
- CKD, SEK, and CBT: from 520 to 360 m ASL.
- The mine design criteria is presented in Table 9-1 below.

Table 9-1: Mine Design Criteria Summary

Item	Unit	CSC	PSM
Ore Reserve	t	2,394,000	437,000
Reserve grade	g/t	8.4	7.2
Gold contained	kg	20,093	3,141
COG	g/t	3	3

Item	Unit	CSC	PSM
Swell factor	factor	1.6	1.6
Specific gravity of waste rock and ore	t/m^3	2.7	2.7
Mining method		MCAF	TCAF
Stope geometry (length by height)	m	100 by 20	100 by 40
Level height	m	20	40
Vein average width	m	4	3
Stope quantity	no.	123	33
Mining recovery factor	%	88.8	92.5
Dilution factor	%	8.6	7.6
Stope development / stope mass	Standard m/kt	19	5
Mine access method		Trackless decline	Incline shaft
Horizontal level haulage method		Truck	Railway locomotive and tramcar
Rock mucking method		LHD	Rock loader
Total vertical/decline/incline development	m	3,101	505
Total of horizontal development	m	9,610	1,677
Mining productivity	t/d per stope	100	60
Production capacity daily	t/d	1200	300
Production capacity annually	kt/a	396	99
Mine life	years	8.5	7
Mine life including: construction	years	1.5	1
ramp-up	years	1	1
full capacity	years	4	3
reducing	years	2	2
Working scheme	hour/shift/day	8/3/330	8/3/330
Development volume for construction	X 1000 m^3	70.4	8.1

The open-pit mining is planned to begin at the end (the last few years) of underground mining to maintain a throughput of 1,500tpa ore for processing plant.

9.3 Stopes and Stoping Method

The cut and fill stoping method is proposed for risk reduction and operational safety assurance. Mechanized overhand cut and fill method (MCAF) is applied in CKD, SEK and CBT sections. Traditional overhand cut and fill method (TCAF) is proposed in the PSM section. This method matches the mine access method and railway transportation.

9.3.1 Mechanized Cut and Fill

Stopes Layout

The stope is arranged every 100 m along the vein. The width is the thickness of the ore vein, which is 3 to 4 m on average. The vertical height of the CKD, SEK and CBT complex is 20 m. The stope layouts are shown in Figure 9-2.

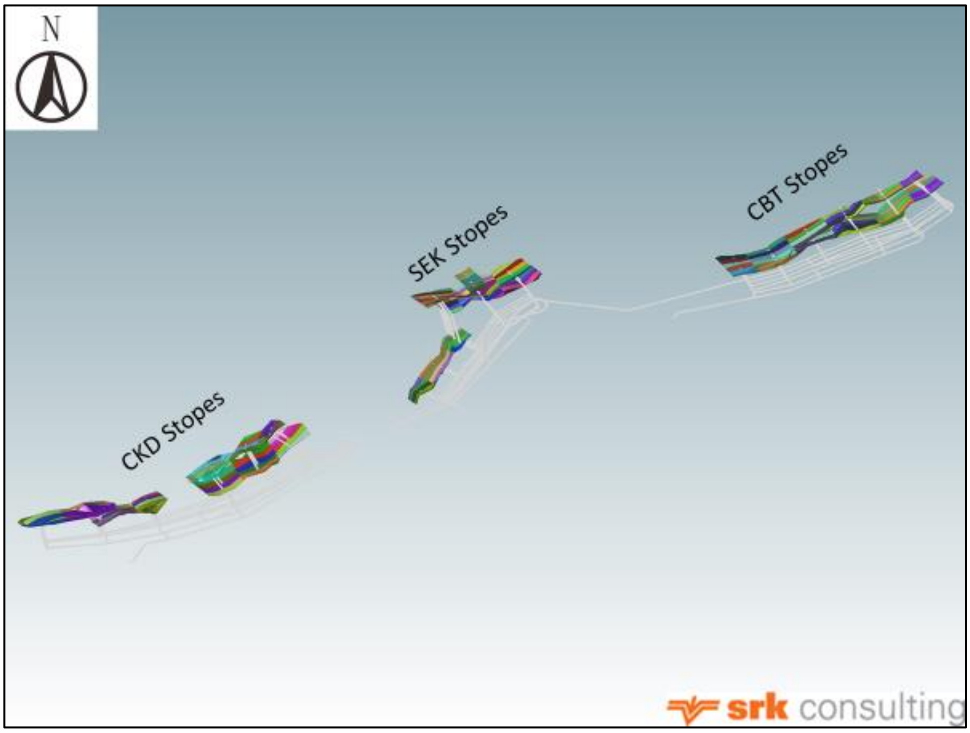


Figure 9-2: Stopes Layout of CKD, SEK & CBT Complex

Stope Arrangement

The standard stope structure and engineering are presented in Figure 9-3.

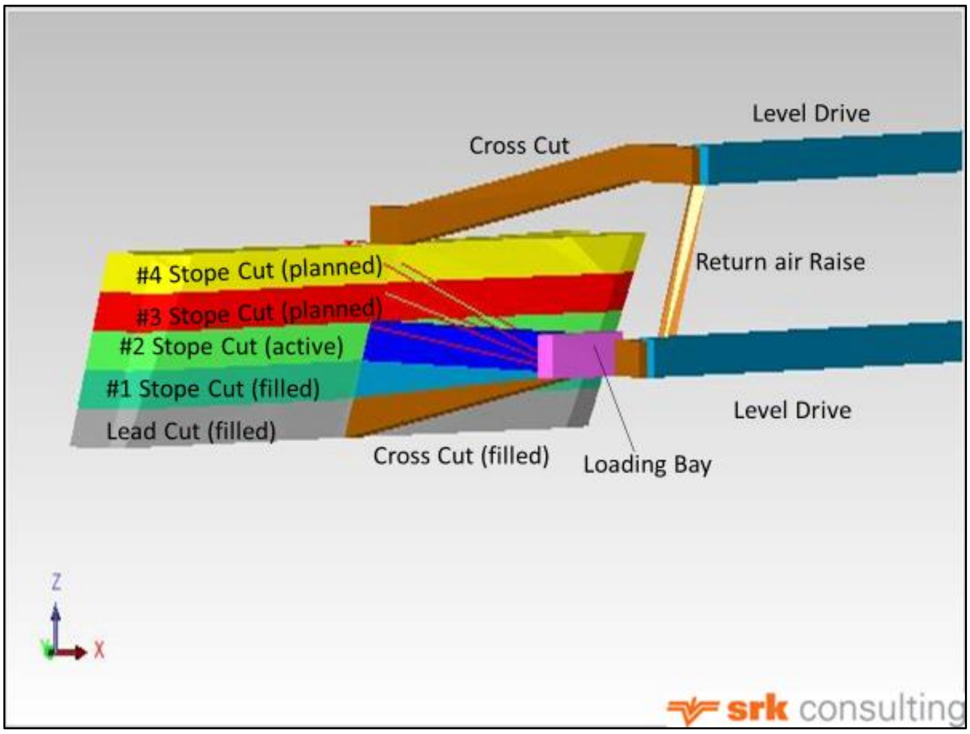


Figure 9-3: MCAF Stope Structure of CKD, SEK and CBT Complex

The stope preparation engineering includes:

Cross Cut (xcut): provides the access from level drive to stope, of about 100m length. It is generally in the middle of the stope, which drives two headings of each stope cut.

The cross cut is the stope lead cut access to initiate the attack ramps for the subsequent lift to stope cuts. It is excavated approximately 65 m at -14% grade, subsequent attack ramps develop upwards to a maximum grade of +14%.

Attack ramp: a set of four drives connect lateral drive and stope cuts. The first (#1) attack ramp is at -7% grade from the end of the lateral drive, aiming #1 cut which is the first lift slice from the lead cut. The second (#2) attack ramp has a minor downward grade (-0.3%) towards the level drive, facilitating drainage out of the stope area. The third (#3) attack ramp inclines at 7% grade. The fourth (#4) is inclined at 14% grade.

Return Air Raise (RAR) and Return Air Way (RAW): a raise located beside the cross cut connecting the production level to the upper level. Return Air Way is the short drift connected to the cross cut and RAR. Fresh air goes from cross cut to active headings, then returns by RAR to upper level.

Loading bay: a chamber located beside the cross cut used for load, haul and dump ("LHD") loading of rock into trucks.

9.3.2 Traditional Cut and Fill

Stopes Layout

The stope is arranged every 100 m along the vein. The width is the thickness of the ore vein, which is 3 to 4 m on average. The height of the PSM section is 40 m. The stope layouts are shown in Figure 9-4.

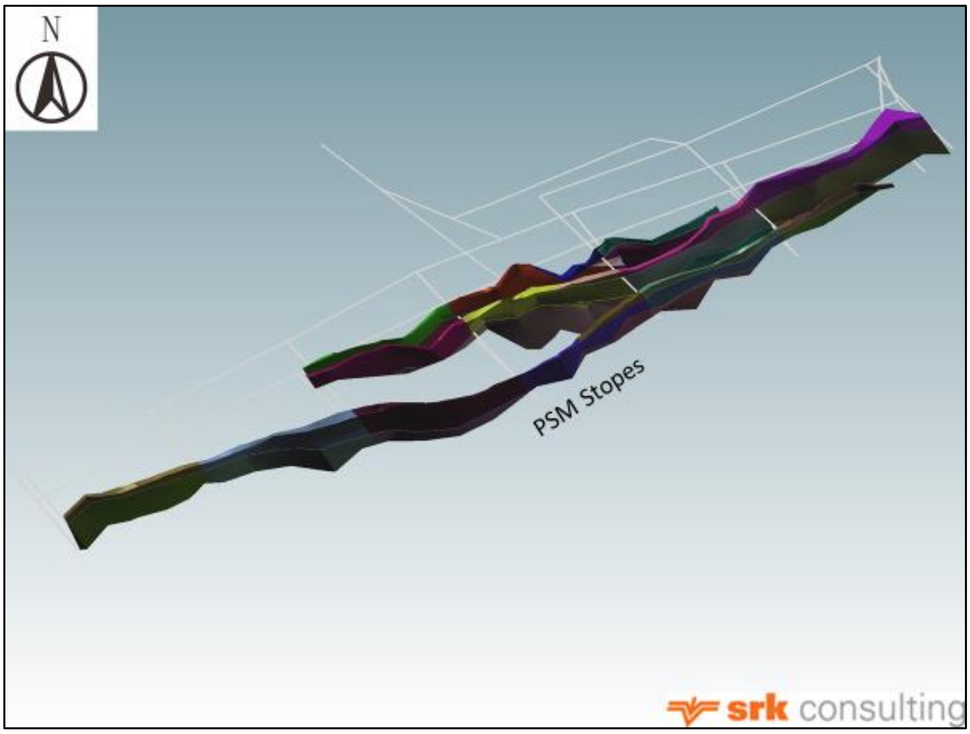


Figure 9-4: Stopes Layout of PSM Section

Stope Structure

The typical stoping arrangement including development is presented in Figure 9-5.

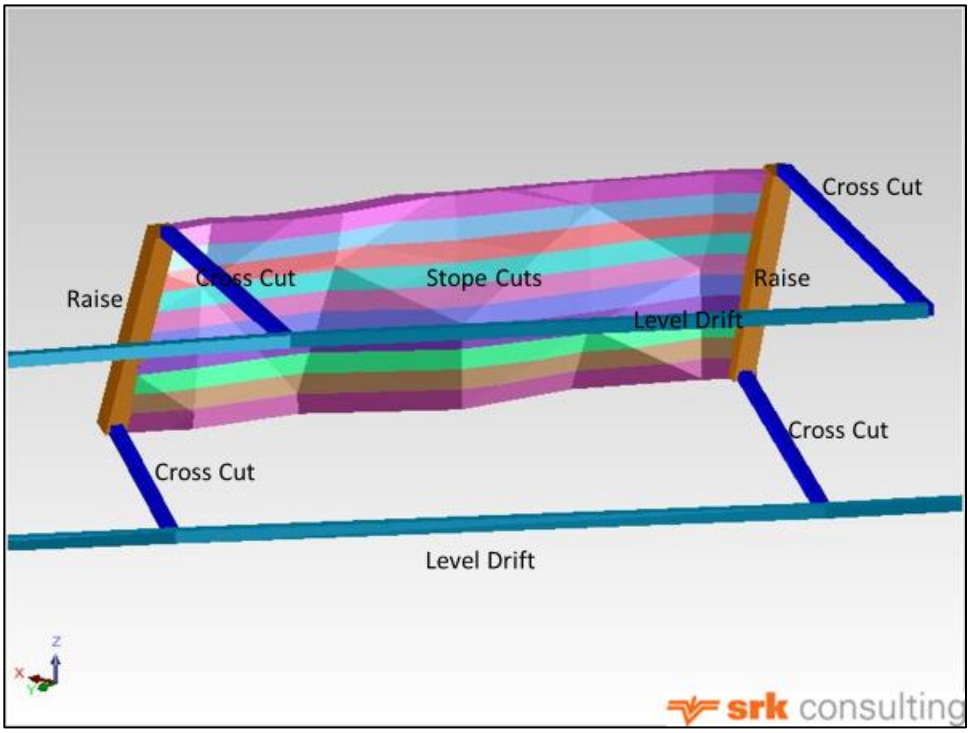


Figure 9-5: TCAF Stope Structure of PSM Section

The stope layout includes:

Cross Cut (xcut): there are two cross cuts providing the access from level drive to stope, of about 80 m length each. They are arranged at the end of the stope, and are shared by two stopes. The cross cut has a minor down grade (-0.3%) towards the level drift, facilitating drainage away from the stope.

Raise: a raise is situated at the end of the cross cut within the stope. A stope has two raises, the same as x-cuts. One is used for personnel and for water drainage, the other is for ore removal (lower part) and ventilation (upper part).

9.3.3 Recovery and Dilution

Based on the mining method and the stopes, the estimated recovery and dilution rates are presented in Table 9-2 below.

Table 9-2: Summary of Recovery and Dilution

Section	Mining Method	Recovery Rate	Dilution Rate
CSC	MCAF	88.8%	8.6%
PSM	TCAF	92.5%	7.6%

9.4 Primary and Secondary Access

A dual decline access system is proposed in the CKD, SEK and CBT complex, and an incline shaft system is proposed in the PSM section.

9.4.1 CKD, SEK and CBT Complex

The dual declines provide access to the underground workings and production areas. There are two portals into the mines: one is the production decline providing access to the mine production in CKD, which is nearest to the plant, and the other acts as the logistics access in CBT.

The ore is mucked from the stoping area into the trucks at the loading bay in the level access for each stoping block. The trucks haul the rock to the surface via the level drive and decline. The access system layout of CKD, SEK and CBT is shown in Figure 9-6 below.

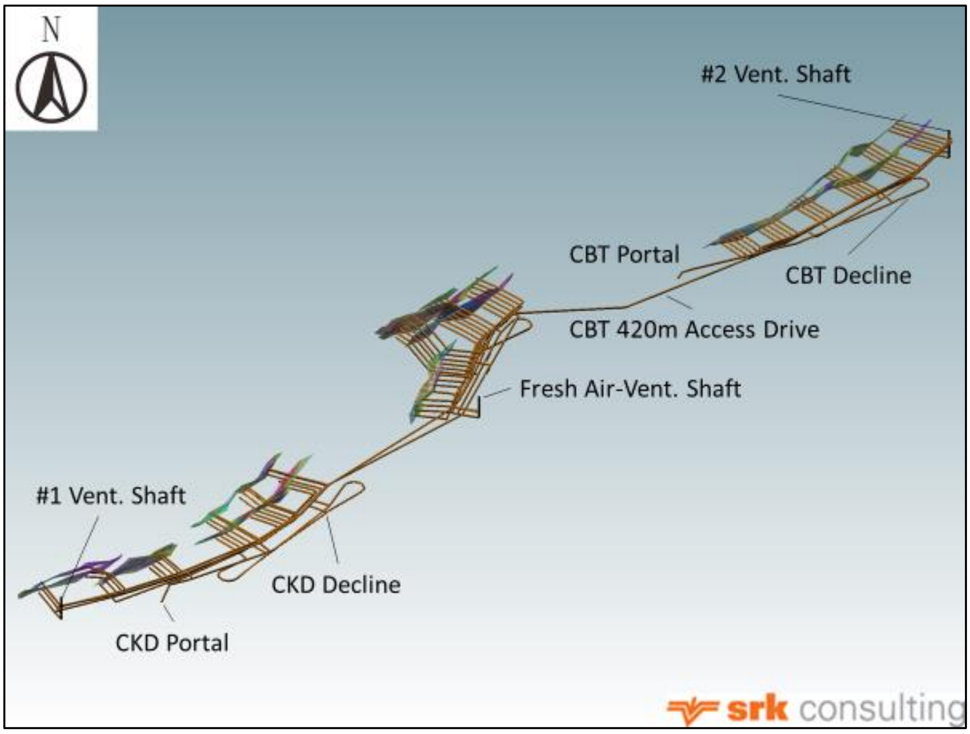


Figure 9-6: Access System Layout of CKD, SEK & CBT

Main decline: is used as general access for ore and waste transporting laterally by truck, as well as transporting material, equipment and personnel. Another usage is for the auxiliary air intake of about 90 m^3/s. The gradient of the straight-line sections is 1:7 and that of the curved sections is flat. A walkway is set on the same side as a ditch. A 20 m flat segment is placed every 140 to 150 m, which provides the access to the level drive. A re-muck bay of 15 m is mined in the middle of each flat segment during the decline development to provide a loading point for the development material to be loaded into trucks. The re-muck bay will become the access to the level drives as the development schedule advances.

Logistics decline: is used as personnel and material access to the CBT area and as the secondary means of egress. The parameters are the same as the main decline except the for cross section which is the same as the level drive.

Level drive: is the main horizontal haulage access route from the production area to the decline for ore, waste, material, equipment and personnel, as well as the fresh air and power.

The level spacing is 20 m. The number of levels for each section is different, but the elevations are the same.

- CKD has six levels, which are 460, 440, 420, 400, 380 and 360 m.
- SEK has six levels, which are 480, 460, 440, 420, 400 and 380 m.
- CBT has five levels, which are 500, 480, 460, 440 and 420 m.

Fresh Air-Vent shaft: is the main intake air raise of $120m^3/s$, and also serves as the secondary means of egress. The shaft will also be used as a conduit for, backfill pipes, water supply, drainage, compressed air supply and power supply. The entrance is about 515 m ASL in SEK section and its bottom is at 440 m ASL. The connection drives are located 460 m and 440 m to the relevant level drives of SEK section.

The shaft's diameter is φ4.0 m. It is equipped with a ladder compartment with landing at 5 m intervals.

Ventilation Shaft: there are 2 ventilation shafts in CKD, SEK and CBT complex: one (#1 Vent. or CKD Vent) is located in the south-west of CKD, and the other (#2 Vent. or CBT Vent) is in the north-east of CBT. Their diameter is the same, at φ3.5 m.

The estimated life of mine access development and the proposed excavation dimensions are summarized in Table 9-3 below.

Table 9-3: Mine Access Development LOM Length and Dimension of CKD, SEK & CBT

Section	Engineering	Cross Section	Quantity	Total length	Volume	Tonnage
		W by H		M	m3	t
CKD	Decline	4 x 4	1	1,109	16,416	44,324
CKD	Access Drive 460	3x 3	1	345	2,897	7,822
CKD	#1 Vent. Shaft	φ3.5	1	80	769	2,077
CKD	Level Drive	3.5 x 3.5	6	3,284	37,439	101,085
SEK	Decline_440	4 x 4	1	542	8,015	21,642
SEK	Incline_440	4 x 4	1	303	4,482	12,102
SEK	Access Drive_440	4 x 4	1	376	5,561	15,014
SEK	Backfill & Air Shaft	φ4	1	75	942	2,543
SEK	Level Drive	3.5 x 3.5	6	1,969	22,450	60,616
CBT	Decline	3.5 x 3.5	1	897	10,226	27,610
CBT	Access Drive_420	4 x 4	1	484	7,166	19,347
CBT	#2 Vent. Shaft	φ3.5	1	95	914	2,467
CBT	Level Drive	3.5 x 3.5	6	3,272	37,301	100,714
Total Vertical/Declined			7	3,101	41,765	112,764
Total Horizontal			21	9,610	111,038	299,802

9.4.2 PSM Section

A single incline shaft is used to access the PSM section. The incline shaft is an extension of the existing development, the existing development requires some stripping to increase its dimensions.

The ore is cleared by electric scraper to the ore pass raise, loaded into tramcars by a rock loader in the cross cut, then eight marshalled tramcars are hoisted by an electric locomotive (3t) to the incline shaft station. The hoister gets the tramcars to surface via incline shaft.

The waste rock is loaded directly from the heading into tramcars, and the shares the same route as ore to the surface.

The payload of each train is 8.35t, and the length of the train is approximately 16m.

The mine access system layout is shown in Figure 9-7 below.

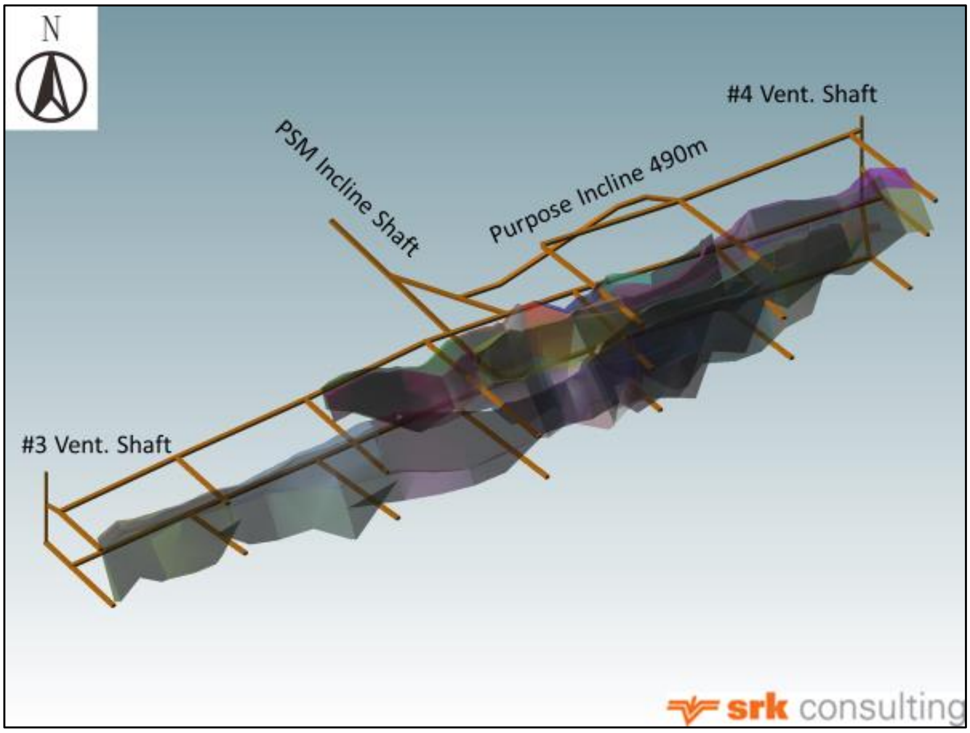


Figure 9-7: Access System Layout of PSM Section

Main Incline shaft: It serves rock hoisting, material, equipment and personnel transportation requirements via a rail-hoist system.

The shaft portal is located in the middle of the footwall side of the PSM West section. The elevation of it is 474m, the shaft section is 2.3 × 2.4m, and the slope is 25°. It has been advanced down approximately 80 m. The incline shaft needs to be extended to the 410m level, and the length will be extended to 170 m. The shaft section should be stripped to 3 × 2.8 m.

It utilizes a single-drum and single-hook hoisting system. The hoister room area is about 8.5×9m^2 which includes a 5t chain lifter for maintenance. There is a total of three loading and unloading points at the portal, 450 m level and 410 m level respectively.

Steel rails of 30kg/m are laid in the incline shaft, with a gauge of 600 mm. The incline shaft is equipped with normally-closed anti-derailing devices, upper backstops, car stoppers and other safety devices.

Purpose incline shaft 490 m: provides an access from the 450 m level to the 490 m level. A temporary winch and rails are equipped during the operation of the 490m level, as the rock is transported downwards.

Level drift: is the main horizontal haulage access from the production area to the incline shaft for ore, waste, material, equipment, men, as well as fresh air and power.

The level height is 40 m, and 3 level drifts are divided as 490 m, 450 m and 410 m.

Level haulage uses electric locomotives. The narrow gauge track of the levels uses 22kg/m steel rails with the gauge of 600 mm, the track slope of 3～5‰ and a minimum curvature radius of 12 m.

Ventilation Shaft: there are two ventilation shafts in the PSM section: one (#3 Vent) is located in the south-west of the PSM, and the other (#4 Vent) is in the north-east of this section. Their diameter is the same, at φ2.5m.

#3 Vent is mainly responsible for transferring a quantity of 30m^3/s exhausted air from the western part of the production work. It is opening at 485 m ASL, its bottom is at the 410 m level and a connection drift is at the 450 m level.

#4 Vent serves the eastern part of the production work with a quantity of 30m^3/s exhausted air. Its opening is at 505 m ASL and it finishes at 410 m, connected to each level.

The estimated life of mine development length and the proposed excavation dimensions are summarized in Table 9-4 below.

Table 9-4: PSM LOM Mine Access Development Parameters

Item	Cross Section	Quantity	Total length	Volume	Tonnage
	W by H		m	m^3	t
Incline Shaft	3 x 2.8	1	170	1,329	3,589
#3 Vent. Shaft	φ2.5	1	75	368	994
#4 Vent. Shaft	φ2.5	1	95	466	1,258
Access Drift_450	5 x 2.8	1	85	1,146	3,094
Temporary Incline_490	2.2 x 2	1	165	676	1,825
Level Drift	2.2 x 2	3	1,592	6,526	17,620
Total Vertical/inclined		4	505	2,839	7,666
Total Horizontal		4	1,677	7,672	20,714

9.4.3 Development

According to the description of rock mechanics characteristics in Section 4, the surface dislocation range or subsidence cone is proposed as 65° to hanging walls, 70° to footwalls, and 70° to the endings of orebodies. The access development is placed outside these cones.

The dimensions of drives are dependent on the equipment requirements. An arched tunnel cross section is proposed.

The development parameters of each type is summarized in Table 9-5 below.

Table 9-5: Development Parameters Summary

Type	Engineering	Cross Section		Ground Support Mode *		
		W by H	Area (m^2)	Portal Segment	General	Special
Access of Trackless	Decline & Main Access Drive	4 x 4	14.8	1	4	7
	CKD Access Drive 460	3 x 3	8.4	N/A	4	7
	Level Drive & Logistics Decline	3.5 x 3.5	11.4	1	4	7
	#1 & #2 Vent. Shaft	φ3.5	9.6	2	5	7
	Backfill & Air Shaft	φ4	12.6	2	6	7
Access of Railway	Incline Shaft	3 x 2.8	7.8	1	4	7
	Access Drift_450	5 x 2.8	13.5	N/A	5+7	8
	Temporary Incline_490	2.2 x 2	4.1	N/A	5	7
	Level Drift	2.2 x 2	4.1	N/A	4	7
	#3 & #4 Vent. Shaft	φ2.5	9.6	3	5	7
Stope Dev of MCAF	Cross cut	3.5 x 3.5	12.0	N/A	7	7
	Attack ramp #1 to #4	3.5 x 4.5	15.0	N/A	7	7
	RAR & RAW	2 x 2	4.0	N/A	N/A	7
	Loading bay	3.5 x 4.5	15.0	N/A	4+7	7
Stope Dev of TCAF	Cross cut	2 x 2	3.8	N/A	7	7
	RAR	2 x 1.5	3.0	N/A	N/A	7

* Ground Support Mode:

Support 1	Ferroconcrete 30cm thickness
Support 2	Ferroconcrete 50cm thickness
Support 3	Ferroconcrete 40cm thickness
Support 4	Shotcrete 10cm thickness, at least 25mpa UCS (28Days)
Support 5	Shotcrete 15cm thickness, at least 25mpa UCS (28Days)
Support 6	Shotcrete 30cm thickness, at least 25mpa UCS (28Days)
Support 7	Split set or resin bolts
Support 8	Wire mesh

Note: "Genral" refers to the main part of the tunnel / ramp; and "Special" denotes some particular location of the engineering, which are occasionally revealed during the developing progress.

9.5 Open Pit Mining

9.5.1 Mining Techniques

The safety pillar leaved at surface is about twenty meters thick, and it has been assumed that the cut and fill method (or MCAF) will be applied to underground mining operation. Usually, there will be void space occurrence in a stope below the surface safety pillar after several years shrinking of fill body. So, filling the stope again is required to support safety operation of open pit mining.

Mining operation in open pits is to be outsourced to the contractor. The contractor will bring small size equipment to supplement the owner's equipment to move approximately 2,832kt materials throughout the producing years.

The mine operation will involve just two procedures of loading and hauling. No drilling and blasting is necessary due to the materials to be moved are soft rocks or weathered soils. The hydraulic shovel proposed by SRK is $1.0m^3$. The trucks are same as that of underground operation, which has a nominated loading capacity of 15t.

Mining starts at the top bench, then is driven down to the pit base bench by bench. No pushback is proposed due to short life of open pit mining. The active bench is 2m high with a bench face angle of 60 degrees. Each four benches will be vertically stacked to an 8m high bench at the final pit location with a bench face angle of 55 degrees and a berm width of 3m. Both the width of pit ramp and the minimum pit base are 8m.

9.5.2 Open Pit Optimization

GEOVIA Whittle (Version 4.5.5) is applied to pit optimization. GEOVIA Surpac (Version 6.8) is used to final pit design and reporting.

Only the rock with a name of "ore" coded in the software is treated as economic materials, while the other mineral resources are treated as waste materials. Nested pit shells were produced based on various revenue factors, which range from 0.01 to 100, to study the potential final pit. Summary of key parameters is presented in Table 9-6.

Pit economics and tonnage were analysed and shown in figures from Figure 9-8 to Figure 9-11. Due to a short service life, which is less than one year when calculated with production capacity of 495ktpa (1,500tpd) ore, just the net value rather than the net present value (NPV) of pits were estimated with a discount rate of 0 and the capital costs of 0. SRK would like to propose the potential final pit based on the maximum net value rather than NPV, as the net value is independent of long-term schedule, while the NPV is closely related to the long-term schedule, especially the detailed year end maps. Usually, the pit with maximum NPV will be reached earlier than the pit with maximum net value. But this will be a negligible flaw due to short service life.

Table 9-6: Summary of Key Parameters to Pit Optimization

Parameters	Unit	Value	Description
Overall slope angle	degrees	45	
Mining cost	USD/t feed	1.00	
Mining recovery rate	%	100	SMU was applied
Mining dilution rate	%	0	SMU was applied
Processing cost	USD/t feed	63.21	
Processing recovery rate	%	80	
Selling price	USD per troy oz	1,230	
Waste Specific Gravity	t/m^3	2.6	
Waste Grade	g/t Au	0	

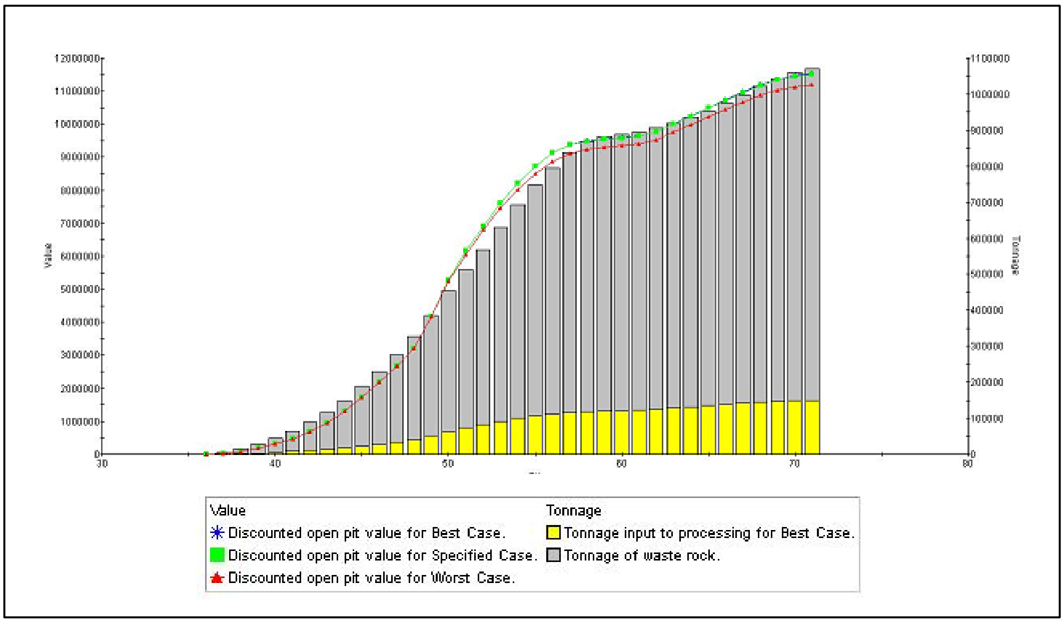


Figure 9-8: Pit Economics and Tonnage in PSM

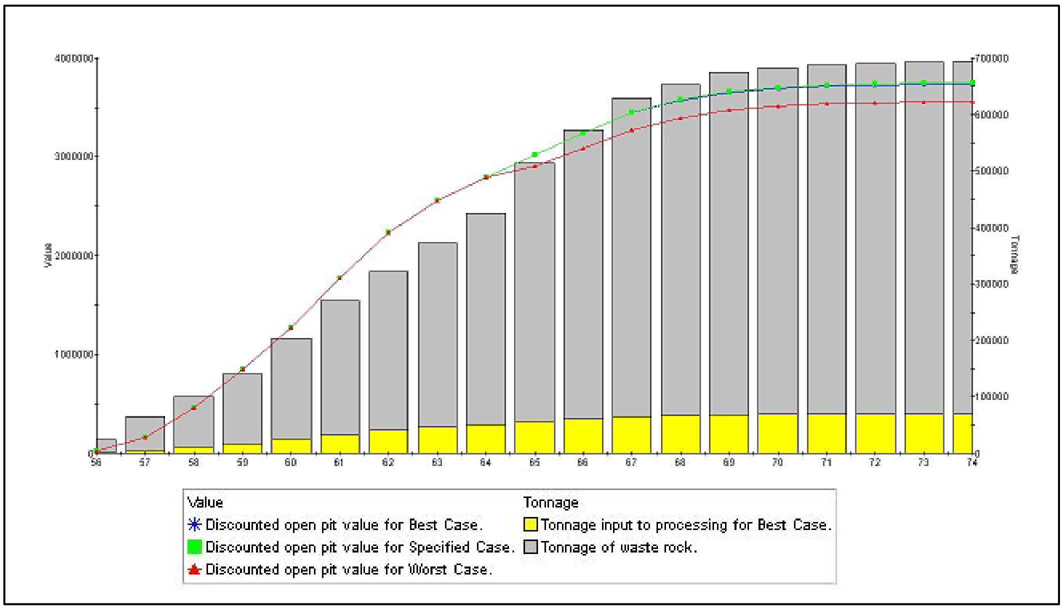


Figure 9-9: Pit Economics and Tonnage in CKD

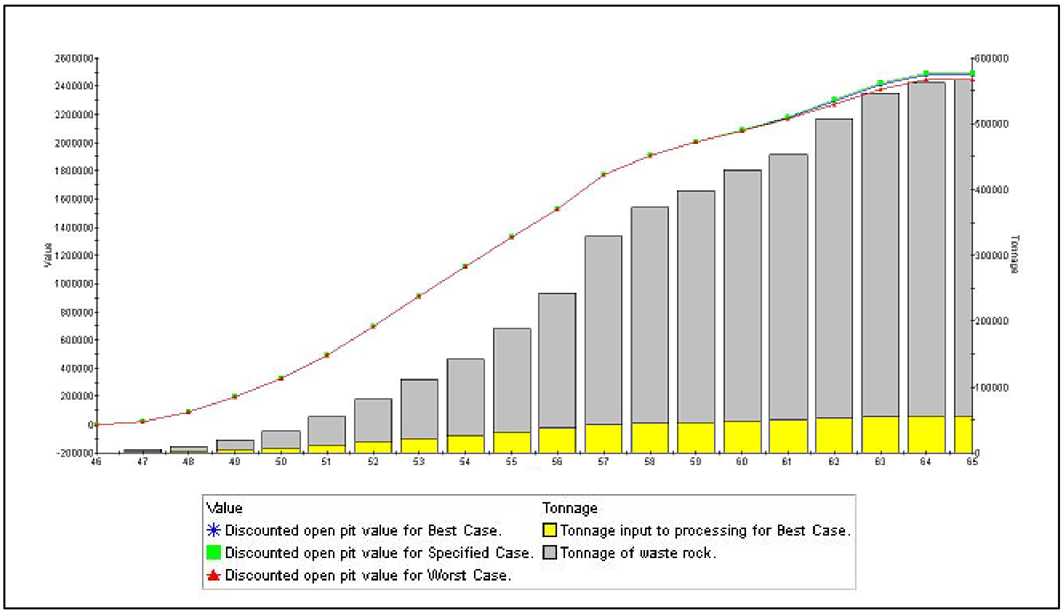


Figure 9-10: Pit Economics and Tonnage in SEK

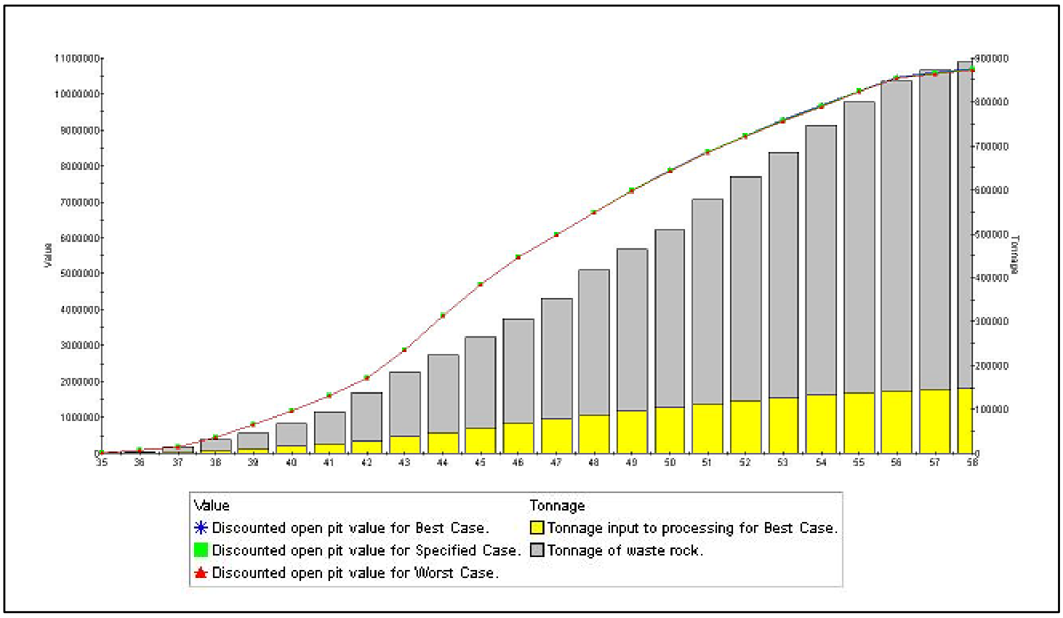


Figure 9-11: Pit Economics and Tonnage in CBT

9.5.3 Final Pit Design

SRK used Surpac to conduct a series of simulations for open-pit optimizations, from small pits to large pits for each mining section. For each simulation, the ore and waste tonnages, and economic values of the pit have been derived.

Summary of open pit design parameters are shown in Table 9-7. A total of 7 pits were designed, which are shown in Figure 9-12. Summary of pit inventory is presented in Table 9-8. Summary of pit properties is shown in Table 9-9.

Although the potential ore tonnages in pits B1, C2 and C3 are very few and the strip ratios are very large, SRK opines that the global strip ratio is about 9.60 t/t, which is an economically viable value.

Table 9-7: Summary of Open Pit Design Parameters

Parameters	Unit	Value	Comment
Bench Height	meters	2.0	A small bench height to avoid large dilution
Stacking Bench Height	meters	8	Four benches are vertically stacked to form a large one at final pit location
Bench Face Angle	degrees	55	
Final Slope Angle	degrees	<= 45	It is dictated by the bench face angle and the berm width.
Berm Width	m	3.0	
Pit Ramp Width	m	8.0	Single
Pit Ramp Gradient	%	10	
Minimum Pit Base Width	m	8.0	Equal to pit ramp width

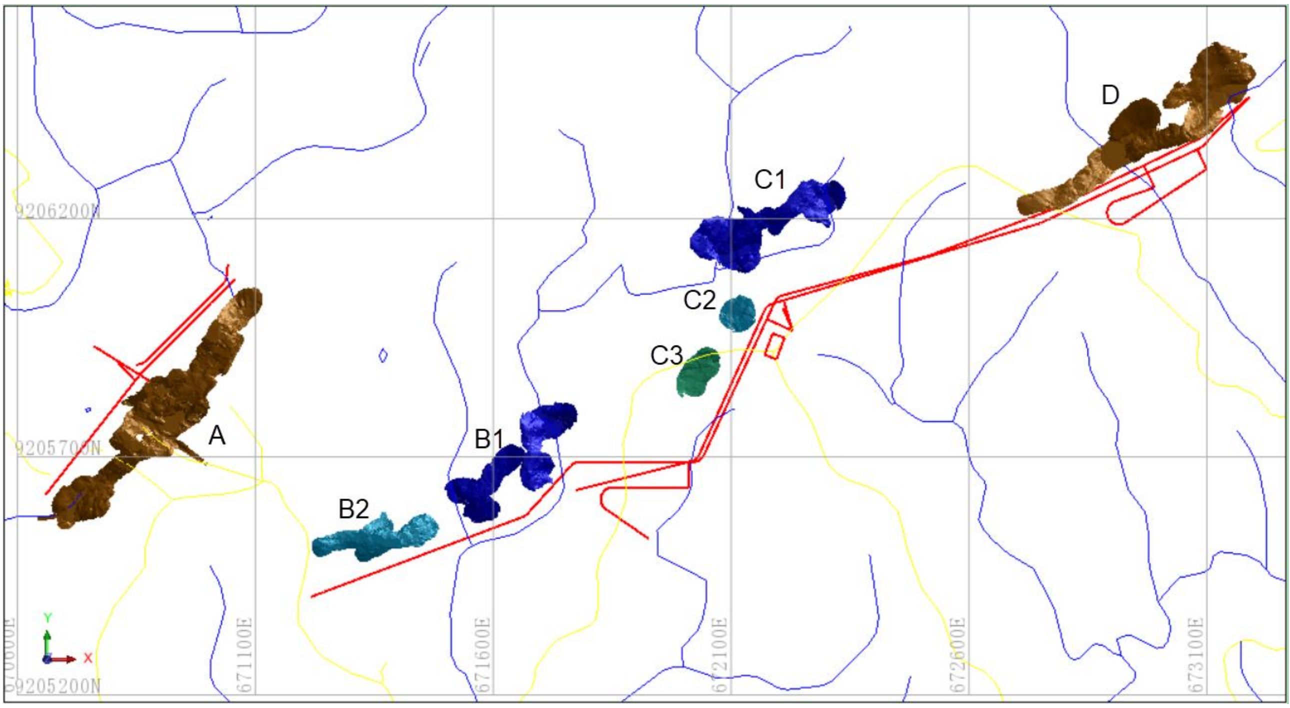


Figure 9-12: Top View of Final Open Pits

Table 9-8: Summary of Pit Inventory

Pit	Rock	MAT	Tonnage (kt)	Au (g/t)	Au (kg)
A	ore	indm	150	5.0	757
	wst	inds	82	0.5	40
	wst	infm	12	5.2	62
	wst	infs	7	0.5	4
	wst	wst	1,190	-	-
B1	ore	indm	32	4.0	128
	wst	inds	26	0.5	12
	wst	infm	4	4.8	17
	wst	infs	2	0.6	1
	wst	wst	568	-	-

Pit	Rock	MAT	Tonnage (kt)	Au (g/t)	Au (kg)
B2	ore	indm	41	4.0	161
	wst	inds	28	0.5	14
	wst	infm	1	4.0	3
	wst	infs	1	0.4	0
	wst	wst	288	-	-
C1	ore	indm	47	3.8	178
	wst	inds	33	0.5	16
	wst	infm	14	3.5	49
	wst	infs	9	0.5	5
	wst	wst	565	-	-
C2	ore	indm	2	3.0	5
	wst	inds	3	0.5	1
	wst	infm	-	-	-
	wst	infs	-	-	-
	wst	wst	93	-	-
C3	ore	indm	8	3.2	24
	wst	inds	7	0.5	4
	wst	infm	2	3.7	8
	wst	infs	1	0.6	0
	wst	wst	124	-	-
D	ore	indm	149	4.8	716
	wst	inds	76	0.5	36
	wst	infm	1	4.0	6
	wst	infs	1	0.5	0
	wst	wst	980	-	-

Table 9-9: Summary of Pit Properties

Item	Unit	Pit A	Pit B1	Pit B2	Pit C1	Pit C2	Pit C3	Pit D
E-W Size	m	474	281	269	329	79	94	503
N-S Size	m	508	255	107	199	79	107	364
Vertical Size	m	76	45	43	42	28	29	50
Pit Top Elevation	m ASL	534	501	507	514	514	513	536
Pit Base Elevation	m ASL	458	458	464	472	486	484	486
Ore Quantity	kt	150	32	41	47	2	8	149
Ore Grade	g/t Au	5.0	4.0	4.0	3.8	3.0	3.2	4.8
Ore Content	kg Au	757	128	161	178	5	24	716
Waste Quantity	kt	1,292	600	317	621	96	135	1,059
Strip Ratio	t/t	8.59	18.50	7.82	13.09	55.14	17.88	7.10

9.6 Support Systems

9.6.1 Ventilation System

Based on the deposit strike length and the access system, the central intake and two-wing exhaust strategy is proposed for both the CKD, SEK and CBT complex, and the PSM section.

The fresh air flows through fresh air shafts and declines, flows over the work headings, then is exhausted by the fans located in #1 and/or #2 ventilation shaft. The fresh air shafts are exhaust ventilation shafts during the construction period.

In the PSM section, there is a similar air flow pattern from the incline shaft to the work headings and is then exhausted via #3 and/or #4 ventilation shaft.

CKD, SEK and CBT Complex

The proposed fresh air volume is 210m^3/s for this complex. The proposed ventilation network and the air volume are presented in Table 9-10 below.

Table 9-10: Ventilation Network and Air Volume for CKD, SEK, & CBT

Item	Unit	Volume	Velocity (m/s)
Fresh air volume of decline	m^3/s	90	6.1
Fresh air volume of vent shaft	m^3/s	120	9.6
Exhausted air volume of #1 vent. shaft	m^3/s	110	11.4
Pressure of #1 vent. shaft	Pa	1780	
Exhausted air volume of #2 vent. shaft	m^3/s	100	10.4
Pressure of #2 vent. shaft	Pa	1700	

The ventilation facilities and fans are located next to the portal of ventilation shaft on surface. Based on the required air volume, and assuming a working efficiency of 80%, the proposed ventilation facilities and fans are summarised as follows:

#1 Ventilation Shaft: equipped with a Φ2.6m counter-rotating fan and two AC motors with power of 250 kW and voltage of 380 V, using a variable frequency control system with a working efficiency of ~ 80%. As it can reversibly return air, the inverted ventilation rate is greater than 60%. Therefore, the fan station does not need an air-reversing tunnel.

#2 Ventilation Shaft: equipped with a Φ2.5m counter-rotating fan and two AC motors with power of 180kW and voltage of 380V. The other parameters are the same as #1 ventilation shaft.

PSM section

The proposed fresh air volume is 60m3/s for this section. The proposed ventilation network and the air volume are presented in Table 9-11 below.

Table 9-11: Ventilation Network and Air Volume for PSM

Item	Unit	Value	Velocity (m/s)
Fresh air volume of incline shaft	m^3/s	60	7.7
Exhausted air volume of #3 vent. shaft	m^3/s	30	6.1
Pressure of #3 vent. Shaft	Pa	270	

Exhausted air volume of #4 vent. Shaft	m^3/s	30	6.1
Pressure of #4 vent. shaft	Pa	360	

The ventilation facilities and fans are lacated next to the portal of ventilation shaft on surface. Based on the required air volume, and assuming a working efficiency of 80%, the proposed ventilation facilities and fans for both ventilation shafts are the same:

#3 & #4 Ventilation Shaft: equipped with a Φ1.3m counter-rotating fan and two AC motors with power of 55kW and voltage of 380V, using a variable frequency control system with a working efficiency of ~ 80%. As it can reversibly return air, the inverted ventilation rate is greater than 60%. Therefore, the fan station does not require an air-reversing tunnel. The area of the fan room is 6×8 m^2.

Secondary Ventilation

Secondary fans of 5.5 kW or 11 kW will be used for the work headings and places with ventilation difficulties and for air volume adjustment and auxiliary ventilation.

9.6.2 Backfill System

The principal method of backfilling at the project is using cemented tailings. The use of cemented tailings backfill has the potential to lower the storage volume required for the tailing storage facility (TSF).

The tailing is pumped from the processing plant to the backfill plant vertical sand silos. The cemented tailing is prepared as 68–70% slurry in the backfill plant at the top of the vent shaft in the SEK section. Due to the elevation and the length of ore bodies, the backfill material could not reach the stopes without pumping so the slurry is pumped underground via pipes in vent shafts or via surface pipes to CBT or PSM sections.

a) Backfill capacity

As the mining method require all voids to be backfilled, the volume of backfill required is the same as the mining capacity of 1,500 t/d, which is approximately 560 m^3/d.

The sedimentation ratio is assumed as 25% and the backfill material loss rate is 10%. The daily backfill material is about 780 m^3/d and thus the proposed backfill station capacity is 800 m^3/d.

b) Backfill material

The backfill material strength is assumed based on knowledge of similar mining practices and cut dimensions. No backfill material was available for testing at the time. The assumed parameters should be confirmed once production commences. The material parameters proposed are presented in Table 9-12.

Table 9-12: Backfill Material Parameter

Item	Unit	Parameter
Mining capacity	t/d	1500
Ore density	t/m3	2.7
Tailings dry density	t/m3	2.65
Cement density	t/m3	3.14
Void space	m3/d	556

Material loss rate		15%			
Sedimentation ratio		25%			
Backfill capacity	m3/d	799			
Material type		I	II	III	Total
Type percentage	%	12%	13%	75%	100%
Filling amount	m3/d				
Slurry concentration (mass)	%	70%	68%	68%	
Cement percentage of solid mass		20%	11%	5%	
Slurry density	t/m3	1.80	1.75	1.74	
Daily material consumption					
Cement	t/d	24.25	13.77	35.32	73
Tailings	t/d	96.99	111.43	671.16	880
Water	t/d	51.96	58.92	332.46	443

The life of mine backfill material requirement is summarised in Table 9-13 below.

Table 9-13: LOM Backfill Volume Estimate

Section	Unit	Type I Total	Type II Total	Type III Total	Total
PSM	m3	615	830	4,702	6,147
CKD, SEK, and CBT	m3	70,694	77,198	437,456	585,349
LOM total	m3	71,309	78,028	442,159	591,495

The tailing production capacity is approximately 1,170 t/d at a yield of gold concentrate of 22%, which could satisfy the backfill requirements of 880 t/d on average

c) Stowing gradient (N=L/H)

The CKD, SEK and CBT complex has a long distance longitudinal of about 2,300 m, though the terrain is relatively gentle at an average 500m elevation. The PSM section is much shorter and the LOM requirement is small also. Thus, one backfill plant shared by both mining sections is proposed, located in SEK section at 520 m ASL.

The stowing gradient estimate shows that it hardly meets the gravity flow requirement of slurry which, using rule-of-thumb data is less than 4 or 5. The stowing gradient of levels is presented in Table 9-14 below.

Table 9-14: Stowing Gradient of Levels

Section	Level（m）	Cumulative Altitude Difference（m）	Maximum Horizontal Distance（m）	Minimum Horizontal Distance（m）	Maximum Stowing Gradient	Minimum Stowing Gradient
CBT	500	20	1400	900	71.0	46.0
CBT	420	100	1400	900	15.0	10.0
CKD	460	60	1500	400	26.0	7.7
CKD	360	160	1200	500	8.5	4.1
SEK	500	20	600	200	31.0	11.0
SEK	380	140	650	400	5.6	3.9
PSM	490	30	1600	1000	54.3	34.3

PSM	410	110	1600	1000	15.5	10.1

d) Backfill plant and Facilities

The backfill facilities mainly include backfill plant and transmission pipe lines. The tailing from the processing plant is pumped to the sand storage bins in the backfill plant. The slurry is prepared by the agitation station in the backfill plant to meet the backfill specifications, then pumped to the underground through the vent shaft and the filling pipeline, and finally transported to the stopes for filling through the return air raise and cross cut/attack ramp. The pipeline for the PSM section is laid along the surface road and goes underground through #4 ventilation shaft.

The daily required slurry amount for stopes is about 780m^3/d. The daily filling time is 12 hours. The required filling material preparation capacity is 780/12 = 65 m^3/h.

Backfill is an important part of the mining cycle and to ensure that when there is an imbalance between for mining and filling, two 80 m^3/h backfill systems are required: a primary system and a secondary system that is on standby for daily filling.

The backfill plant requires a deep cone thickener and two sets of cement bins. The filling station area is 27 × 28 m^2. The deep cone thickener diameter is 10 m, side wall height is 8 m, and daily processing capacity is about 800 m^3. The effective storage capacity of cement bins is 250 m^3. Each cement bin requires a φ250 × 3000 mm micro powder scale to feed the agitation tank.

During filling operations, the tailings from the processing plant are pumped to the deep cone thickener for thickening, the bottom fraction flows into the high concentration agitation bucket, and is then blended with cement in the required ratio. After complete agitation through the φ2000×2100 mm high concentration agitation bucket to prepare the tailings slurry to a concentration of 70%, the slurry is pumped to the underground void by industrial pumps. Two sets of filling industrial pumps are utilized, a primary and a standby. The technical parameters of the flow are 100 m^3/h, the maximum outlet pressure is 14 Mpa, with a motor power of 320 kW.

Measuring instruments and computerized automatic adjustment control systems for the quality, concentration, flow of tailings, cement and filling slurry from the filling material preparation system, are included in the installation.

9.6.3 Compressed Air

The surface air compressor station is located near the vent shaft in SEK, for the supply of demand in the CKD, SEK and CBT complex, as well as the PSM section. The main underground compressed air consumption equipment are jack leg drills and shotcrete machines, with total air consumption of 157 m^3/min, as presented in Table 9-15 below.

Table 9-15: Main Air Consumption Equipment

Equipment	Type	Quantity (set)	Unit Air Volume (m^3/min)	Working Air Pressure (MPa)	Total Air Volume (m3/min)
Jack leg drills	YT-30	14	5	0.5	70
Jack leg drills	YSP45	8	5	0.5	40
Shotcrete machine	PZ-5	4	8		32
Others			15		15
Total					157

Considering the simultaneous operation of the compressed air equipment, pipe leakage and other factors, the maximum air consumption of the entire mine is calculated to be 174.6 m^3/min (at a pressure of 0.75 MPa).

Based on the maximum underground production compressed air consumption, the air compressor station requires four screw air compressors (three working and one on standby), for which the technical parameters are as follows:

- compressed air volume of 60.2 m^3/min, and
- pressure of 0.75MPa.

A singular system is planned for the emergency compressed air production. The compressed air for production stops under emergencies, but a single air compressor can meet the emergency air requirement. Each air compressor utilises a 300kW, 380V motor.

The main compressed air reticulation is from the air compressor station from the surface to the underground by the SEK air shaft and the PSM inclined shaft. The specification of the seamless steel pipe is φ273 × 7 mm, and the underground level air pipe is φ219 × 6mm seamless steel pipe.

9.6.4 Power Supply

The project electricity is from Pelabuhan Ratu, which is approximately 40 km away from the site. The power supply system study battery limit starts from the high-voltage outlet cabinet (cable head) of the general step-down substation at the site, excluding the external power supply and the general step-down substation.

The standard voltage of Indonesia national low-medium voltage is 20 kV, 380/220 V, AC voltage frequency of 50 Hz.

This chapter includes the electric transformer and distribution, electric drive, energy saving requirements which are matched to the mining engineering needs:

- Power distribution system for the mining site;
- 20/0.4 kV substation for the mining area; and
- Electric drive and energy saving for electric power mining equipment.

The demarcation point is the 20kV general step-down substations 20kV high-voltage outlet cabinet, which is in the processing plant.

The surface substation and distribution station are summary in Table 9-16 below.

Table 9-16: Summary of Surface Substation and Distribution Station

Substation and Distribution Station Name	Transformer Capacity（kVA）	Substation and Distribution Station Name	Transformer Capacity（kVA）
#1 Vent. 20/0.4kV substation and distribution station	2×800	Backfill plant 20/0.4kV substation and distribution station	1×800
Surface air compressor room 20/0.4kV substation and distribution station	2×1250	#3 Vent. 20/0.4kV substation and distribution station	2×200
#2 Vent. 20/0.4kV substation and distribution station	2×630	Incline hoist 20/0.4kV substation and distribution station	1×400
Maintenance workshop 20/0.4kV substation and distribution station	1×125	#4 Vent. 20/0.4kV substation and distribution station	2×200
Decline portal 20/0.4kV substation and distribution station	1×125		

The underground substation and distribution station are summarised in Table 9-17 below.

Table 9-17: Summary of underground Substation and Distribution Station

Substation and Distribution Station Name (CKD, SEK & CBT Complex)	Transformer Capacity（kVA）	Substation and Distribution Station Name (PSM)	Transformer Capacity（kVA）
Level 440 pump room 20/0.4 kV substation and distribution station	2×315	Level 410 pump room 20/0.4 kV substation and distribution station	2×160
Level 360 pump room 20/0.4 kV substation and distribution station	2×315	Level 450 20/0.4 kV substation and distribution station (mining)	1×250
Level 440 20/0.4 kV substation and distribution station (mining)	1×400		

9.6.5 Water Supply

The operational water is sourced from mine water underground and Cimas River 6km from the site, and the potable water is sourced from the groundwater wells. The underground production fresh water demand volume of CKD, SEK and CBT complex is 400 tpd, which is supplied by the surface high pond. The fresh water supply pipeline is shared with the emergency system water supply pipeline and leads to underground production water consumption sites and emergency sites along the ventilation shaft.

The underground production water volume requirement for the PSM Section is 100 tpd, which is supplied by the surface high pond. The production water supply pipeline is shared with the emergency system water supply pipeline and leads to underground production water consumption sites and emergency sites along the inclined shaft.

To meet the requirements for a firefighting water supply, water supply pipe size for the main haulage drive and machine maintenance chambers underground should be not less than φ108×4 mm.

According to the mining requirements, the minimum water consumption of the underground emergency system water supply is about 600 L/d. In the event of an emergency, potable water will connect with the potable water supply system in the mining area, and the pipeline of underground water supply system will be utilized, and lead to the underground stope work place and personnel evacuation site. A set of 3 direct links and valves shall be installed every 200 to 300 m on the water supply pipelines of the intake ventilation shaft in main levels and a set of tee joint and valves, and a drinking water purification and filtration device will be installed on the water supply pipeline at the emergency personnel evacuation site.

9.6.6 Drainage

Underground water is mainly from rainfall and surface water infiltration recharge. Calculation shows that the daily normal and maximum underground water inflows in the CKD, SEK and CBT complex are expected to be 1,600 m3/d and 2,400 m3/d respectively, and the daily normal and maximum underground water inflows in the PSM Section are expected 450 m3/d and 675 m3/d respectively.

Backfill dewatering is also considered during the calculation of the dewatering volume. The calculation is presented in Table 9-18 below.

Table 9-18: Underground Dewatering Volume Calculation

Section	Normal Water Inflow	Max Water Inflow	Production Water	Filling Water	Normal Dewatering Volume	Max Dewatering Volume
	(m^3/d)	(m^3/d)	(m^3/d)	(m^3/d)	(m^3/d)	(m^3/d)
CKD, SEK & CBT	1600	2400	400	410	2410	3210
PSM	450	675	100	140	690	915

CKD, SEK and CBT Section utilises relayed drainage, with underground waste water being discharged to the surface through a ventilation shaft. The pump station and sump are positioned in Level 440 near the ventilation shaft during the construction period and, in the future, a pump station and sump will be constructed near Level 360. According to calculations, the pumping head and volume required of the dewatering pump during the capital construction period and the later period are approximately: 114m, 121 m3/h (normal) and 161 m3/h (max). Four multistage pumps are planned in the pump room. When the water inflow is normal, two pumps work at the same time, with one for standby and one for maintenance. When the water inflow is maximum, three pumps work at the same time, the other pump are for standby or for maintenance. Pump technical performance is as follows: the flow is 85 m3/h, the lift is 135 m, and the electrical machinery power is 55 kW.

Two φ159 mm × 7 mm seamless steel pipes are provided inside the ventilation shaft as the dewatering pipes. During normal dewatering, one pipe is required and the other is standby. During maximum dewatering, both pipes are required. The water pump room area is 20 × 3.5 m^2, and with no lifting beam included the pump room, a simple portable tripod will be required to lift equipment for maintenance.

The PSM Section uses one-section drainage. Underground waste water is discharged to the surface through the incline shaft, with the pump station and sump installed in Level 410 near the inclined shaft during the construction period. According to the calculations, the pumping head and volume required by the dewatering pump are approximately 98 m, 35 m^3/h (normal) and 46 m^3/h (max), and three multistage pumps are used in the pump room. When the water inflow is normal, one pump required, one for standby and one for maintenance. When the water inflow is at a maximum, one pump is for work and the others are for standby or for maintenance. Pump technical performance as follows: the flow is 48 m3/h, the lift is 100 m, and the electrical machinery power is 30 kW.

Two φ108 mm × 5 mm seamless steel pipes are provided inside the inclined shaft as the dewatering pipes. During normal dewatering, one pipe is required, and one for standby. During maximum dewatering, both pipes are used for work. The water pump room area is 16 × 3.4 m^2 and with no lifting beam in the pump room, a simple portable tripod will be required to lift equipment for maintenance.

The dewatering system includes the water pump chamber, transformer chamber, internal (or external) water tank, pipe chute, absorbing well and water distribution lane. The transformer chamber and water pump chamber are provided with a waterproof door and grille door. The total effective volume of the water tanks is calculated according to the normal water inflow of 6 to 8 hours. The pipe platform is more than 7 m higher than the ground of the shaft inset. The ground level of the water pump chamber is 0.5 m higher than the track surface of the underground car yard roadway in the access, and 0.3 m lower than the transformer chamber floor. The pump chamber and substation chamber use cast-in-place concrete supports, the water tanks use shotcrete supports, and both have a concrete strength grade of C25.

9.6.7 Communication

The communication design for the mine site includes a telephone communication system, computer network and wiring system, integrated wiring system, automatic fire alarm system, comprehensive underground communications platform, underground communication system, underground personnel positioning system, underground video surveillance system and line laying of the site.

9.6.8 Workshop and Warehousing

According to the mining schedule, there is one maintenance workshop for the mobile equipment at the mine site; and there is one maintenance workshop for periodical inspection and maintenance of electric locomotives and tramcars.

Spare parts and consumable are dependent on outsourcing and collaboration with original equipment manufacturers (OEMs) for maintenance requirements.

Mobile Equipment Workshop

There are nine 15t trucks, two 20 t trucks and eight light vehicles in the mine. Through calculation, there should be two repair sites, two maintenance sites and one parts storage site in the workshop.

The workshop is located at the portal of the decline of CKD and consists of a main maintenance area and an auxiliary area. The main area is repair and parts storage area, and the auxiliary area is spare parts warehouse, duty room and bathroom.

The area of the vehicle repair workshop is 540 m^2 (length×width: 30 m×18 m). The workshop has two areas, and the main area width is 12 m, and the rail height is 8 m. There is 1 LD electric single beam crane, Gn=5 t, S=10.5 m, H=9 m. The auxiliary area is 180 m2 (length×width: 30 m×6 m), and the back height is 4.5m.

The main equipment in the workshop is as follows:

- HDS801D high pressure washer,
- TQD-3 automobile electric universal test platform,
- Oil cylinder separate-installed machine;
- Large wheel balancer, 510 mm wheel hub width, andΦ660 mm diameter;
- Ultrasonic cleaner, cleaning tank size: 1500×1000×1000 mm;
- Vulcanization tyre repair machine, 360 mm width; and
- CR-DC48B fast charger, battery voltage 48 V.

Maintenance Workshop

There are two 3 t electric locomotives in the mine. Two drive-through repair lines are laid in the workshop; one is for electric locomotives, and the other is for tramcar repairs. The gauge of the two repair lines is 600 mm. Two inspection pits are set in the electric locomotives repair line.

The maintenance workshop is positioned near the PSM incline shaft portal, and is mainlyfor electric locomotive disassembly and assembly, benchwork repairs, rivet weld repairs of frames and carriages, and electrical equipment maintenance.

The area of the maintenance workshop is 540m^2 (length×width: 45 m×12 m) and the rail level is 6m. There is 1 LD electric single beam crane, Gn=5t, S=10.5 m, H=6 m.

The main equipment in the maintenance workshop is as follows:

- Z4120 bench drilling machine;
- M3215J bench grinder; and
- BX6-250 AC arc welder.

9.6.9 Fuel Station

There is one fuel station, which is on the surface, providing a refueling service for the transportation vehicles. There is one explosion-proof separation refueling device, storing 0# conventional diesel (class B). The storage capacity is 30 m^3, and the storage cycle is 7 days.

The technical indexes of 0# conventional diesel are shown in Table 9-19 below.

Table 9-19: Quality Indexes of Conventional Diesel

Item	0# Diesel
Use system	All year
Kinematic viscosity（20℃）mm^2/s	3.0～8.0
Flashing point℃ no lower than	55
Solidifying point ℃ no higher than	0
Moisture（%） no more than	Trace
Sulphur content（mg/kg）no more than	50

Diesel is transported from a fuel truck, and a fuel pump is to unload the fuel. Light diesel in the fuel truck is pumped through an airtight pipeline system to the fuel tank in the skid-mounted refueling device for storage.

The skid-mounted refueling device is a ground mobile refueling equipment, with a fire-proof and explosion-proof oil storage tank, refueling machine, and automatic extinguisher. The refueling device has safety and anti-explosion, energy conservation and environment protection features, economy and practicability, reduced space requirements, easy installation, and portability, and will not explode when encountering open fire, gunshot, welding, lightning strike, static electricity and accident.

9.7 Mine Equipment

The main mining mobile equipment is summarised in Table 9-20 below. The equipment summary is the quantity required for full production. The staggered purchase plan is designed so that the initial construction machinery is all purchased and installed in 2019, and mobile production equipment are staggered over 3 years, 30% in 2018, 50% in 2019 and 20% in 2020.

Table 9-20: Summary of Main Mine Equipment

Item	Capacity/Model	Total Quantity	Active Quantity
Jack leg	YT-30	28	14
Jack leg	YT-45	16	8
Electric scraper	2JP-15	6	3
Rock loader	Z-30	2	1
Truck	15t	9	7
Locomotive	P=3t, DC250v	2	1
Tramcar	0.7m3	11	8
LHD	3t	8	6
Shotcrete	PZ-5	6	4
Light vehicle (pick up)	Diesel	8	6
Secondary fan	11kw	10	8
Secondary fan	5.5kw	4	3

Inclined shaft development is proposed in the PSM Section with small scale production (300 tpd). Electric scrapers and rock loaders are employed as mucking equipment. An electric locomotive and tramcars are proposed for the lateral transportation. The main parameters of the incline shaft hoisting are presented in Table 9-21.

Table 9-21: Main Technical Parameter of Incline Shaft Hoisting

No.	Item	Parameter or Description
1	Hoisting method	Single-hook trip hoisting
2	Incline shaft dip angle	25°
3	Hoisting dip length	About 175 m
4	Winch specification	φ1.6×1.2 m
	Diameter of hoist drum	φ1.6 m
	Maximum hoisting speed	3.14 m/s
5	Electromotor	Variable-frequency AC motor
	Power	130 kW
	Rotational speed	750 rpm
	Voltage	380 V
6	Tramcar group	
	Model	0.7 m^3
	Quantity	4
	Dead weight	4×0.71 t
	Payload	4×0.93 t
7	Mancar group	
	Type	Incline shaft mancar for 15 people
	Quantity	One head car and one trailer
	Dead weight	Head car of 1.756 t; trailer of 1.903 t
	Maximum lifting number of people per time	2×15
8	Hoisting rope	
	Quantity	4
	Diameter	Φ20 mm
	Tensile strength	1770 Mpa
	Aggregate breaking force of ropes	285.3 kN
9	safety factor of rope	
	Material hoisting	8.35 > 7.5
	Personnel hoisting	10.31 > 9

9.8 Life-of-Mine (LOM) Schedule

The working system of the mine site is 330 days per year, 3 shifts per day, and 8 hours per shift. A total production capacity of 1,500 tpd for the mine and mill plant is proposed. SRK reviewed the proposed capacity and verifies that it is achievable.

9.8.1 Underground Mining

The proposed stope productivity is 100 tpd and 60 tpd for MCAF and TCAF mining methods respectively.

The work cycle of development mining is similar to stope mining. The shotcrete method for ground support is recommended, with the shotcrete works running in parallel during development but with a 20 m lag behind the active heading. The proposed single heading advance rates of main type development drives are as follows:

- Decline and level drive: 100 m per month;

- Raise: 60 m per month;
- Sink shaft: 60 m per month;
- Incline shaft: 60 m per month.

The initial mining areas are: PSM 410m level stopes, CKD 440m level stopes, SEK 460m level stopes, and CBT 420m level stopes for each section.

The summary of the underground mines annual LOM schedule is presented in Table 9-22 below.

Table 9-22: Summary of Underground Mines LOM Schedule

YEAR	UNIT	LOM Total	2018	2019	2020	2021	2022	2023	2024	2025	2026
CSC ROM	kt	2,394	-	-	245	413	415	416	404	378	124
CSC GRADE	g/t	8.4	-	-	8.8	8.2	8.7	8.6	8.4	8.1	7.3
CSC Gold Contained	kt	20,093	-	-	2,167	3,380	3,632	3,571	3,385	3,055	903
PSM ROM	kt	437	-	15	98	100	88	82	38	15	-
PSM GRADE	g/t	7.2	-	9.2	7.5	7.0	6.8	7.1	7.5	6.4	-
Gold Contained	kg	3,141	-	142.78	738	697	601	580	285	96	-
Total ROM	kt	2,831	-	15	343	513	503	498	442	393	124
Total GRADE	g/t	8.2	-	9.2	8.5	8.0	8.4	8.3	8.3	8.0	7.3
Gold Contained	kg	23,233	-	143	2,905	4,077	4,233	4,151	3,670	3,151	903
Capital Dev	km	15.8	1.8	5.8	3.2	2.1	1.5	1.4	0.0	-	-
Operating Dev	km	50.8	-	-	13.8	13.2	12.7	9.7	1.4	-	-
Total Dev	km	66.6	1.8	5.8	17.0	15.3	14.2	11.2	1.4	-	-

The year for 2018 is from July to December, 6 months.

The capital development is for the primary and secondary accesses. The operating development is for the stope developments such as cross cut, raise and cut and fill ramps.

9.8.2 Open Pit Mining

The mining operation starts in Cikadu and ends in Cibatu, as both them are located near to the proposed location of processing plant.

The operation is expected to last three years. Life of mine schedule for the open pit mining is presented in Table 9-23.

Considering the short life of mine, just bench by bench mining is proposed.

Table 9-23: Summary of Life of Mine Schedule – Open Pit

Section	Pit	Item	Unit	Tonnage (kt)	2024	2025	2026
Pasir Manggu West	A	Ore Tonnage	kt	150	-	-	150
		Waste Tonnage	kt	1,292	-	-	1,292
		Au Grade	g/t	5.0	-	-	5.0
		Au Content	kg	757	-	-	757
		Strip Ratio	t/t	8.59	-	-	8.59
Cikadu	B1	Ore Tonnage	kt	32	32	-	-
		Waste Tonnage	kt	600	600	-	-

Section	Pit	Item	Unit	Tonnage (kt)	2024	2025	2026
		Au Grade	g/t	4.0	4.0	-	-
		Au Content	kg	128	128	-	-
		Strip Ratio	t/t	18.50	18.50	-	-
	B2	Ore Tonnage	kt	41	-	41	-
		Waste Tonnage	kt	317	-	317	-
		Au Grade	g/t	4.0	-	4.0	-
		Au Content	kg	161	-	161	-
		Strip Ratio	t/t	7.82	-	7.82	-
Sekolah	C1	Ore Tonnage	kt	47		47	-
		Waste Tonnage	kt	621		621	-
		Au Grade	g/t	3.8	-	3.8	-
		Au Content	kg	178		178	-
		Strip Ratio	t/t	13.09	-	13.09	-
	C2	Ore Tonnage	kt	2		2	-
		Waste Tonnage	kt	96		96	-
		Au Grade	g/t	3.0	-	3.0	-
		Au Content	kg	5		5	-
		Strip Ratio	t/t	55.14	-	55.14	-
	C3	Ore Tonnage	kt	8		8	-
		Waste Tonnage	kt	135		135	-
		Au Grade	g/t	3.2	-	3.2	-
		Au Content	kg	24		24	-
		Strip Ratio	t/t	17.88	-	17.88	-
Cibatu	D	Ore Tonnage	kt	149			149
		Waste Tonnage	kt	1,059			1,059
		Au Grade	g/t	4.8	-	-	4.8
		Au Content	kg	716			716
		Strip Ratio	t/t	7.10	-	-	7.10
Total		Ore Tonnage	kt	429	32	97	299
		Waste Tonnage	kt	4,120	600	1,169	2,351
		Au Grade	g/t	4.6	4.0	3.8	4.9
		Au Content	kg	1,970	128	369	1,473
		Strip Ratio	t/t	9.60	18.50	12.02	7.85

10 Ore Reserve Estimation

The purpose of this section is to summarize the Ore Reserve estimate. Based on the relevant modifying factors studied in this report, the Ore Reserves for the Project are estimated using Surpac (Version 6.6) and Mineshed (Version 9.0). The estimated Ore Reserves were reported in accordance with the JORC Code (2012 Edition).

10.1 Break-even Cut-off Grade (BCOG)

The following formula is applied to calculate BCOG of Au for run of mine (ROM).

$$\mathrm{A} = \frac{\mathrm{Cm} + \mathrm{Cp} + \mathrm{Cb} + \mathrm{Cs} + \mathrm{Cg}}{\mathrm{P} \times \mathrm{R}}$$

Parameters that are applied to this formula are presented in Table 10-1.

Table 10-1: Calculation of Break-even Cut Off Grade

Item	Unit	Value	Description
A	g/t	3.02/1.50	ROM cut-off grade for underground/open pit mining
Cm	USD/t ROM	32.78	Total mining cost
Cp	USD/t ROM	34.03	Total processing cost
Cb	USD/t ROM	11.81	Baking and acid making cost
Cs	USD/t ROM	0.79	Supportive infrastructure cost
Cg	USD/t ROM	16.58	Total G & A cost
R	%	80	Total Recovery for processing and smelter
P	USD / g	39.55	Forecast gold price, equivalent to USD 1,230 per ounce

The BCOG is estimated to be 3.02g/t Au, which is rounded to 3.0 g/t. SRK is of the view that material that has an Au grade more than 3.0 g/t can be mined economically, and reserves at BCOG will have positive revenues. Therefore, SRK uses 3.0g/t as the BCOG of Ore Reserve estimation.

Please note that the BCOG were calculated based on technical and economic assumptions described in Section 9. These assumptions may change in the future, which will affect the BCOG calculation, which will impact the ore reserve estimate.

With respect to the open pit mining, the marginal cut-off grade was calculated and rounding to 1.5g/t based on the parameters which are addressed in details in Section 9.5. Just the marginal costs are considered to calculate this cut-off, which includes processing, baking and acid making. The marginal cut-off grade is applied to feed ore, rather than in-situ mineral resources.

10.2 Selection of Mining Unit

10.2.1 Underground Mining

SRK has completed the Resource Estimate for the Project using Surpac (Version 6.6) software. The model has a user block size of 4 m×4 m×2 m (X×Y×Z).

The designed mining methods is cut and fill (CAF) mining method with two sub-types of mechanized CAF and traditional CAF with different mining equipment. One cut is designed 4 m high and around

3–4 m wide. So, an SMU size of 4 m×4 m×2 m (X×Y×Z) along each direction is suitable for this project.

10.2.2 Open Pit Mining

SRK has finished the resource estimate for the Project using a user block size of 8m×8m×4m (X×Y×Z). The block size that applied to the resource estimate is not appropriate to be used directly to estimate the Ore Reserves. The key reason is that the economically viable mining areas are generated based on a whole block's economics in theory, which means that larger the block size, rougher the mining pit, especially for a deposit with thickness which has no similar order of magnitude as that of block size.

The selective mining unit (SMU) is 2m*2m*2m (X*Y*Z) in size, which is the minimum block to separate ore blocks from waste blocks. Although applying of SMU is optional at feasibility study stage, SRK prefers it to applying a predefined ore losses and mining dilution.

10.3 Block Model

10.3.1 Underground Mining

Model limits for the underground mining are shown in Figure 10-1.

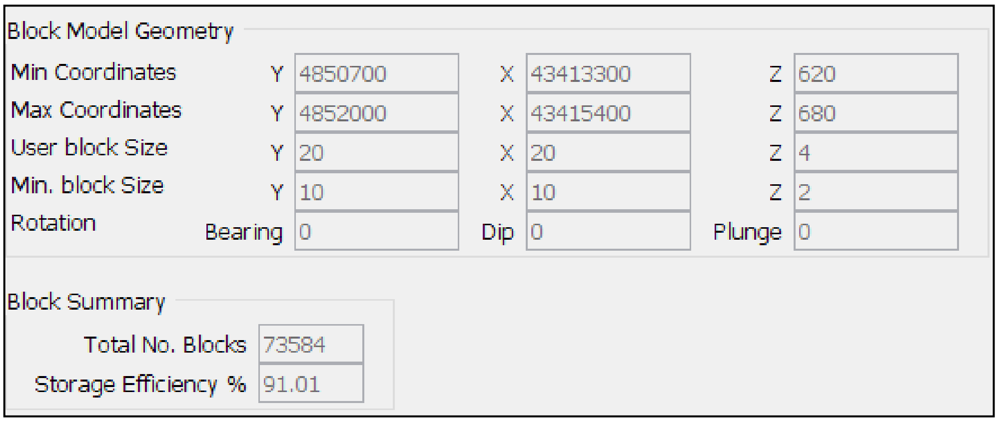


Figure 10-1: Ore Reserve Model Limits

10.3.2 Open Pit Mining

SRK has finished the resource estimate for the Project using a user block size of 8m×8m×4m (X×Y×Z). Usually, the block size that applied to the resource estimate is not appropriate to be used directly to estimate the Ore Reserves. The key reason is that the economically viable mining areas are generated based on a whole block's economics in theory, which means that larger the block size, rougher the mining pit, especially for a deposit with thickness which has no similar order of magnitude as that of block size.

The resource block model was re-blocked to generate reserve block model to optimize potential open pit shells, and some additional attributes are defined and assigned values. Model limits are shown in Table 10-2. Summary of key attributes in reserve block model attributes is presented in Table 10-3. Material definition is shown in Table 10-4.

The selective mining unit (SMU) is 2m*2m*2m (X*Y*Z) in size, which is the minimum block to separate ore blocks from waste blocks. Although applying of SMU is optional at feasibility study stage, SRK prefers it to applying a predefined ore losses and mining dilution.

Table 10-2: Model Limits of Reserve Block Model

Type	Y	X	Z
Minimum	9205000	670000	300
Maximum	9207000	674000	600
SMU Size	2	2	2
Rotation	0	0	0

Table 10-3: Summary of Key Attributes in Reserve Block Model

Attribute	Description
au	Gold grade, which is directly coming from resource block model
aud	Gold grade of SMU, which is weighted by volume and specific gravity
cat	Mineral resource code, which is 2 for Indicated, 3 for Inferred and 0 for others
domain	Domain name
domain_pct	Domain partial
mat	Material name, which is to report pit inventory.
pit	Pit shell number
rock	Rock type required in Whittle
sg	Specific gravity, which is directly coming from resource block model
sgd	Specific gravity of SMU, which is weighted by volume
topo_pct	SMU partial below topography
type	Oxide or fresh

Table 10-4: MAT and ROCK Definition

ROCK	MAT	TOPO_PCT	CAT	AU	Description
ore	indm	> 0	2	>= 1.5	Marginally economic Indicated
wst	inds	> 0	2	0.0~1.5	Sub-economic Indicated
wst	infm	> 0	3	>= 1.5	Marginally economic Inferred
wst	infs	> 0	3	0.0~1.5	Sub-economic Inferred
wst	wst	> 0	0	0.0	Waste block
air	air	0	0	0	Air block

10.4 Modifying Factors

The modifying factors in relation to Mining are described in Section 9.2 of Mining Design Criteria. Mining targets and objectives are described in the Stopes and Stoping Method section (Section 9.3). The LOM schedule is described in the LOM Schedule section (Section 9.8). The mining recovery and dilution have been estimated for both underground and open pit.

The calculation of break-even cut-off grade and details of the assumptions are presented in Section 10.1.

A detailed description of the Modifying Factors is presented in Section 4 of JORC Table 1 in Appendix 1. Other Modifying Factors for the Ore Reserve estimation include:

- Processing recovery rate for concentrate: 81.5% and the overall recovery rate for ROM to gold dore: 86.7%, as suggested by the metallurgical study.
- The project cost study addressed in Section 17 of this Report.

- No apparent and inherent issues observed in the environmental and social aspects; and the related costs in environmental sectors such as mine closure have been considered in the cost study.
- Infrastructure considerations has been reflected in cost study, and
- Market studies: gold price of USD 1,230 per ounce has been used.

10.5 Ore Reserve Classification

The economically mineable part of both Measured and Indicated Mineral Resources was converted into Probable Ore Reserves, in compliance with the JORC Code 2012 Edition. No Proved Ore Reserves were classified in this Project.

10.6 Ore Reserve Statement

Table 10-5 summarises the Ore Reserves estimate for the Project, based on the Mineral Resource model and includes modifying factors, as at 31 March 2018. The reported Ore Reserve is within the limits of Wilton's proposed mining license area.

At a BCOG of 3.0g/t Au and including dilution material, the Project has 2.8 million tonnes of Probable Ore Reserves, averaging 8.2g/t Au or contains 23.2 t of gold inclusion. Please note that the reported Ore Reserves are included in the Mineral Resource.

Table 10-5: Summary of Underground Ore Reserves, as of 30 June, 2018

Section	Category	Reserve	Grade	Gold
		kt	g/t	kg
CKD	Probable	913	8.3	7,560
SEK	Probable	622	8.5	5,303
CBT	Probable	859	8.4	7,229
PSM	Probable	437	7.2	3,141
Total	Probable	2,831	8.2	23,233

Note:
All figures are rounded to reflect the relative accuracy of the estimate.
The information in this report that relates to Ore Reserve conversion is based on information compiled by Mr Falong Hu, MAusIMM, and Mr Qiuji Huang, FAusIMM, employees of SRK Consulting (China) Ltd. Both Mr. Huang and Mr. Hu have sufficient experience relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking to qualify as Competent Persons as defined in the 2012 Edition of the "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves". Mr. Huang supervised the work of Mr Hu. Mr. Huang and Mr. Hu consent to the reporting of this information in the form and context in which it appears.

For the open pit Ore Reserve estimate, at a BCOG of 1.5g/t Au and including dilution material, the open pit mine has 429,000 tonnes of Probable Ore Reserves averaging 4.6g/t Au and contains 1,970 kg gold metal, as presented in Table 10-6 below.

Table 10-6: Summary of Open Pit Ore Reserves, as of 30 June, 2018

Section	Category	Reserve	Grade	Gold
		kt	g/t	kg
CKD	Probable	73	4.0	290
SEK	Probable	57	3.6	208
CBT	Probable	149	4.8	716
PSM	Probable	150	5.0	757
Total	Probable	429	4.6	1,970

Note:
All figures are rounded to reflect the relative accuracy of the estimate.
The information in this report which relates to Ore Reserve conversion is based on information compiled by Mr Yonggang Wu, MAusIMM, and Mr Qiuji Huang, FAusIMM, employees of SRK Consulting (China) Ltd. Both Mr. Huang and Mr. Wu have sufficient experience relevant to the style of mineralisation and type of deposit under consideration and to the activity which he is undertaking to qualify as Competent Persons as defined in the 2012 Edition of the "Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves". Mr. Huang supervised the work of Mr Wu. Mr. Huang and Mr. Wu consent to the reporting of this information in the form and context in which it appears.
Mining recovery rate and mining dilution rate were not predefined but were calculated based on the SMU. The statistics for mining recovery rate and mining dilution rate after Ore Reserves convert output values of about 88% and 31%, respectively. These two huge rates are mainly caused by the extremely thin orebodies, which is averaging at about 1~2 meters.

Table 10-7 below presented the total Ore Reserve including underground and open pit mines.

Table 10-7: Summary of Total Ore Reserves, as of 30 June, 2018

Section	Category	Reserve	Grade	Gold
		kt	g/t	kg
CKD	Probable	986	8.0	7,850
SEK	Probable	679	8.1	5,511
CBT	Probable	1,008	7.9	7,945
PSM	Probable	587	6.6	3,898
Total	Probable	3,260	7.7	25,203

10.7 Potential Impacts to Ore Reserve Estimation

As in the case for most mining projects, the extent to which the estimate of Ore Reserves may be affected by mining, processing and smelter conditions, infrastructure, permitting, market conditions, and other factors could vary from major gains to total losses of ore reserves. There are no issues known to the Competent Person of this section expected to materially affect the Ore Reserve estimate.

11 Mineral Processing and Metallurgical Tests

11.1 Introduction

Wilton has entrusted metallurgical institutes conducted a lot of metallurgical tests on various samples from Ciemas gold deposits.

Australian Minmet Metallurgical Laboratories Pty. Ltd. ("AMML") reported its testwork in *Metallurgical Testwork on Ciemas Gold and Silver Deposits* in April 2015. Cyanidation leach, gravity separation and flotation tests were carried out on various composite samples.

A *Preliminary Metallurgical Review* of AMML's testwork was completed by PT. Geoservices. This review was to provide guidance on further metallurgical tests required for the feasibility study, sample selection, potential process flowsheet options and development of the project. The results of the metallurgical testwork can be summarised by the following statements: The Ciemas ores have been tested for their amenability to cyanidation agitation and responses to flotation techniques for the recovery of gold. The tests were largely qualitative in nature and were performed on samples reporting a wide range of gold head grades from four different deposits in the Ciemas area. The ores demonstrated predominantly refractory gold characteristics with poor to little recovery by standard cyanidation agitation. Flotation responses were deemed to be encouraging with gold recoveries above 90% to a rougher concentrate. The bulk flotation concentrate will require pre- oxidation prior to cyanide leaching.

Further metallurgical testwork was recommended by PT. Geoservices to be performed on composites which represent proposed mine and mill feed gold grades and further confirm flotation responses and cyanide leaching of flotation tails. The oxidation of high grade gold flotation concentrates and subsequent cyanidation leaching is a crucial circuit that requires investigation. Following this recommendation, PT. Geoservices conducted a *Characterization Tests* to figure out the gold deportment, and a set of Comminution Tests to determine the crushability and grindability of Ciemas Ores. Metallurgical Response and Optimization Test was carried out continuing AMML' s testwork. Tests of diagnostic leaching, flotation, gravity separation, flotation of gravity tailings, CIL of flotation tailings, flotation concentrate roasting and CIL and flotation concentrate bacterial oxidation and CIL were conducted.

Based on the test results and an *Oxidation Options* study, PT. Geoservices recommended the flowsheet of "flotation – roasting – CIL" to be optimum, and deduced the gold recovery will be 92%.

11.2 AMML's Metallurgical Test

11.2.1 Introduction

According to the degree of oxidation, the mineralization of the Ciemas deposit can be divided into three types: oxidized ore, mixed ore, and primary ore. AMML's test work was completed in two stages:

- Stage 1: Leachability screening – Standard carbon-in-leach process was applied on 52 interval composites collected from 168 drill core individual intervals of 30 DDH as per orebodies. 8 composite samples were composited from the 52 interval composites as per mineralization type and leachability for Stage 2 optimization testing.
- Stage 2: Composites testing – detailed investigation to optimize the processing routes and reagent consumptions in order to determine a suitable flowsheet and technology.

11.2.2 CIL Test on Interval Composites

Stage 1 test work was comprised of leachability (cyanidation) screening tests to classify mineralization types prior to composite preparation for Stage 2 test work. It's should be noted that the intervals with head grades of less than 0.5 g/t Au were excluded from compositing and leachability screening tests in Stage 1.

Under a nominal grinding fineness of P_{80}=75µm, standard carbon-in-leach ("CIL") cyanidation tests were conducted by rolling bottle on 52 interval composites to determine their leachability. A summary of Au and Ag dissolutions classified by ore type is shown in Table 11-1, and the details are listed in Table 11-2.

Table 11-1: Summary of CIL Test Results by Ore Type, AMML

Classification	Au Dissolution %			Ag Dissolution %		
	Range	*Average**	*Feed Grade Range (g/t)*	*Range*	*Average**	*Feed Grade Range (g/t)*
Oxide	17.6-96.4	71.2	0.07-28.7	31.4-97.8	82.6	0.1-29.8
Mix	3.2-85.5	19.5	0.01-15.3	25.1-78.4	42.4	0.2-77.3
Primary	11.8-85.5	39.9	0.01-10.8	44.6-89.3	54.1	0.1-33.9

*Note *: Average values were weighted based on weight of Interval composite. Represents the effective dissolution for the entire body of ore type intervals tested.*

Table 11-2: CIL Test Results of Interval Composites, AMML

No	Test	Location	Interval (m)	Ore Type	P_{80} (mm)	Au Balance			Reagent Consumption		
						Head Assay (g/t)	Calc Assay (g/t)	Dissolution (%)	Lime (kg/t)	NaCN (kg/t)	Residual NaCN (%)
1	L40	Cibatu	0.0-1.25	Oxide	50	0.53	0.56	96.4	12.6	1.78	0.112
2	L39	Cibatu	10.9-15.1	Oxide	35	1.57	1.50	75.3	9.99	1.00	0.134
3	L42	Cibatu	15.5-17.4	Oxide	39	2.40	2.35	49.8	16.0	2.03	0.054
4	L37	Cikadu	11.812.9	Oxide	89	0.71	0.68	75.1	4.08	0.64	0.145
5	L35	Pasir Manggu	16.8-17.7	Oxide	38	1.25	1.36	89.7	10.9	1.15	0.142
6	L31	Pasir Manggu	18.1-20.3, 21.3-21.75	Oxide	43	1.53	1.71	90.1	10.2	1.92	0.068
7	L32	Pasir Manggu	20.3-21.3	Oxide	49	5.29	5.64	86.5	11.5	2.74	0.056
8	L33	Pasir Manggu	20.6-22.3	Oxide	44	1.13	1.28	89.8	8.22	0.64	0.140
9	L26	Pasir Manggu	21.75-23.6	Oxide	39	1.64	1.86	84.4	10.7	1.74	0.068
10	L27	Pasir Manggu	27.1-28.6	Oxide	43	1.08	1.19	17.6	9.07	2.37	0.070
11	L29	Pasir Manggu	28.5-28.85	Oxide	44	3.53	4.56	25.2	11.3	2.70	0.072
12	L28	Pasir Manggu	29.05-31.6	Oxide	60	6.60	6.74	18.4	10.9	2.44	0.052
13	L34	Pasir Manggu	31.0-33.0	Oxide	107	28.7	28.4	96.1	6.06	1.38	0.100
14	L36	Pasir Manggu	38.4-38.8	Oxide	106	6.74	6.43	79.0	6.79	1.19	0.112
15	L38	Sekolah	34.85-36.3	Oxide	69	0.29	0.35	91.4	4.95	0.62	0.146
16	L21	Sekolah	36.7-38.2	Oxide	47	0.07	0.14	64.1	4.77	0.86	0.138
17	L41	Sekolah	44.2-47.3	Oxide	39	0.86	0.89	83.2	7.59	0.90	0.140
18	L23	Sekolah	65.3-68.3	Oxide	42	0.94	0.84	57.2	7.64	1.16	0.106
19	L30	Pasir Manggu	28.85-37.6	Oxide, mix	42	0.07	0.15	80.3	11.1	1.74	0.086
20	L14	Cibatu	16.5-17.9	Mix	27	0.32	0.35	17.5	1.89	1.20	0.138
21	L15	Cibatu	17.9-18.3	Mix	64	9.80	9.64	18.8	2.75	0.99	0.114
22	L16	Cibatu	183-19.4	Mix	50	12.2	11.0	10.0	1.50	1.24	0.110
23	L17	Cibatu	19.4-20.3	Mix	54	5.25	5.20	33.0	0.57	2.00	0.065
24	L18	Cibatu	20.3-21.0	Mix	50	4.22	4.09	32.0	0.48	2.16	0.052
25	L19	Cibatu	21.0-22.0	Mix	61	3.65	3.52	52.0	0.80	1.76	0.076
26	L20	Cibatu	22.0-23.0	Mix	67	6.28	6.04	5.5	1.07	1.83	0.070
27	L11	Cibatu	342-36.8	Mix	60	1.24	1.23	13.9	5.84	1.21	0.114
28	L13	Cikadu	33.8-34.7	Mix	57	1.84	1.99	39.2	3.27	1.34	0.112
29	L6	Cikadu	49.1-50.0	Mix	50	0.92	0.95	15.0	20.8	1.89	0.086
30	L2	Cikadu	71.4-74.4	Mix	78	1.04	1.09	9.6	3.25	0.79	0.138
31	L46	Pasir Manggu	102.0-106.0	Mix	77	7.95	7.84	6.7	8.63	2.38	0.102
32	L47	Pasir Manggu	106.0 - 107.0	Mix	81	0.03	0.07	84.8	3.85	1.65	0.084
33	L48	Pasir Manggu	108.0- 109.0	Mix	73	0.01	0.07	85.5	4.99	0.65	0.148
34	L45	Pasir Manggu	118.0-120.0	Mix	83	0.63	0.75	31.9	6.61	1.93	0.068
35	L50	Pasir Manggu	148.0-151.0	Mix	113	5.70	5.85	3.5	4.55	0.64	0.144
36	L49	Pasir Manggu	56.0-59.0	Mix	63	1.35	1.36	12.6	10.5	1.76	0.076
37	L43	Pasir Manggu	80.0-85.0	Mix	64	6.48	6.39	11.1	7.06	1.95	0.064
38	L44	Pasir Manggu	87.0-88.0	Mix	85	2.75	2.66	8.4	7.12	136.00	0.102
39	L51	Pasir Manggu Timur	*Not specified*	Mix	91	15.3	15.2	46.0	3.86	3.74	0.056
40	L7	Sekolah	76.8-35.9	Mix	81	1.60	1.69	6.9	2.60	1.46	0.094
41	L1	Cikadu	55.9-62.4	Mix, primary	60	1.03	1.33	30.2	3.72	1.62	0.084
42	L12	Sekolah	106.8-122.8	Mix, primary	80	0.97	1.10	20.2	1.51	1.07	0.120
43	L8	Cibatu	49.85-56.25	Primary	43	1.06	0.94	71.4	2.15	1.16	0.114
44	L3	Cikadu	40.8-42.3	Primary	54	4.28	4.12	55.6	22.9	2.28	0.040
45	L4	Cikadu	42.8-43.8	Primary	48	0.44	0.53	58.4	33.4	2.38	0.064
46	L5	Cikadu	43.8-44.3	Primary	56	6.86	6.73	11.8	21.8	2.30	0.062
47	L9	Cikadu	55.45-56.9	Primary	41	0.67	0.74	39.2	20.2	1.81	0.086
48	L10	Cikadu	56.9-57.8	Primary	32	0.01	0.07	85.5	15.1	2.48	0.066
49	L24	Pasir Manggu	223-23.3	Primary	51	3.16	3.16	77.2	11.5	2.22	0.040
50	L25	Pasir Manggu	23.3-24.1	Primary	44	0.74	0.82	41.3	12.1	2.27	0.054
51	L52	Pasir Manggu Barat	Not specified	Primary	99	10.8	10.8	27.7	1.93	2.52	0.034
52	L22	Sekolah	51.8-53.3	Primary	73	0.36	0.43	21.1	13.7	0.88	0.132

11.2.3 CIL Test on Composite Samples

Stage 2 tests were conducted on the following 8 composite samples which were made of the 52 interval composites based on ore type and cyanide leachability. Resulting composite assays are shown in Table 11-3.

- Oxide ore type composites
    - Ox1 low grade oxide ore type with high dissolution
    - Ox2 high grade oxide ore type with high-moderate dissolution
    - Ox3 low grade oxide ore type with low-moderate dissolution
- Primary and mixed ore type composites
    - MP1 low grade mixed ore type with low dissolution
    - MP2 low grade mixed/primary ore type with moderate dissolution
    - MP3 high grade mixed ore type with low dissolution
    - MP4 high grade primary ore type with low dissolution
    - MP5 high grade mixed/primary ore type with moderate dissolution

Table 11-3: Ore Type Composite Assay, AMML

Composite	Au (g/t)	Ag (g/t)	As (%)	Fe (%)	Cu (%)	Pb (%)	Zn (%)	S (%)
Ox1	1.24	6.00	0.236	5.51	0.019	0.014	0.012	0.65
Ox2	5.49	8.00	0.255	5.98	0.030	0.125	0.040	1.26
Ox3	1.82	5.00	0.433	6.24	0.017	0.043	0.025	3.79
Mp1	1.39	2.50	0.219	5.71	0.014	0.048	0.058	3.15
Mp2	1.02	4.20	0.178	3.08	0.008	0.025	0.036	1.96
Mp3	5.50	21.0	0.679	5.99	0.035	0.648	0.560	5.53
Mp4	10.8	30.0	0.411	5.09	0.085	0.104	0.413	5.74
Mp5	11.1	11.0	1.14	5.68	0.056	0.020	0.020	4.55

Standard CIL process was applied on different grind fineness. The tests results are summarized in Table 11-4, and the profile of leach kinetics is shown in Figure 11-1, which indicates that the effect of grind fineness is not obvious for Au leaching for oxidized ore, while the Au leaching rate increase when the sulphide ore is ground to finer, i.e. the grind fineness is higher. The Au leaching rate for oxidized ore is obviously higher than that of the sulphide ore. Leachable gold has a rapid dissolution rate generally. The composites contained a fairly significant refractory component.

Though the Au leaching rates for OX1 and OX2 are over 85%, the parameters for all the rest composite samples are really poor, especially for MP3, of which the Au leaching rate is less than 8%. Generally speaking, the Au recovery rates obtained through cyanidation are not favourable. Diagnostic leach test was conducted for 8 composite samples and the test result is shown in Table 11-5. The Diagnostic leach test is designed to reveal the deportment of the gold in the ore minerals. The overall test is performed in a way liberating the gold in a series of steps under different chemical conditions and thus discloses the location of the gold in the ore sample. The test results show that the refractory gold is mainly occurred in the arsenopyrite, and then in the carbonate mineral, and a small amount is encapsulated in the quartz. Therefore, the extraction of gold from arsenopyrite should be the focus of metallurgical experiments.

Table 11-4: CIL Test Result of Composite Samples, AMML

Composite	P_{80} (μm)	Au Balance			Ag Balance			Reagent Consumption		
		Head Assay (g/t)	Calc Assay (g/t)	Dissolution (%)	Head Assay (g/t)	Calc Assay (g/t)	Dissolution (%)	Lime (kg/t)	NaCN (kg/t)	Residual NaCN (%)
0x1	75	1.24	1.27	84.2	6.00	5.93	96.6	8.90	2.33	0.030
	38	1.24	1.21	86.0	6.00	6.62	99.6	9.31	2.32	0.048
	20*	1.24	1.24	87.9	6.00	6.00	96.7	9.15	2.34	0.046
0x2	75	5.49	5.26	84.6	8.00	7.82	85.9	6.85	2.36	0.032
	38	5.49	5.69	85.8	8.00	6.87	86.9	7.19	2.51	0.028
	20*	5.49	4.96	85.7	8.00	7.71	88.3	7.24	2.44	0.040
0x3	75	1.82	1.75	29.7	5.00	6.41	73.5	8.65	2.86	0.038
	38	1.82	1.79	37.6	5.00	6.15	75.4	8.47	2.87	0.046
	20*	1.82	2.15	48.0	5.00	5.88	77.9	8.60	3.06	0.046
MP1	75	1.39	1.80	41.1	2.50	3.16	33.5	1.67	2.01	0.024
	38	1.39	2.84	60.9	2.50	3.39	44.0	1.90	2.08	0.026
	20*	1.39	1.20	10.7	2.50	2.93	42.0	1.98	2.74	0.036
MP2	75	1.02	1.17	37.4	4.20	4.32	53.7	2.48	2.45	0.050
	38	1.02	1.15	38.4	4.20	4.40	56.8	2.65	2.60	0.056
	20*	1.02	1.15	47.2	4.20	4.89	63.2	2.61	3.08	0.046
MP3	75	5.50	5.53	5.31	21.0	21.8	37.5	2.54	2.70	0.038
	38	5.50	5.54	6.70	21.0	20.1	47.2	2.51	2.94	0.034
	20*	5.50	5.44	7.50	21.0	20.8	50.0	2.80	3.09	0.046
MP4	75	10.8	11.0	25.8	30.0	33.4	43.4	2.47	3.26	0.030
	38	10.8	10.5	30.4	30.0	32.5	50.1	1.91	3.60	0.028
	20*	10.8	10.4	33.5	30.0	32.4	56.2	2.29	3.72	0.036
MP5	75	11.1	10.8	44.7	11.0	11.2	78.5	13.7	3.49	0.048
	38	11.1	11.1	46.7	11.0	11.4	79.0	12.6	4.56	0.034
	20*	11.1	10.8	48.8	11.0	10.9	81.2	12.58	4.06	0.036

Note*: Grind times were extrapolated from trial grind data curves. As a result, the P_{80} value of 20μm is approximate only

Table 11-5: Diagnostic Leach Test Result – Gold Deportment, AMML

Comp	Au Head Assay (g/t)	Au Deportment %					Ag Head Assay (g/t)	Ag Deportment %				
		CN Soluble[1]	HCl Liberated[2]	HNO_3 Liberated[3]	Aqua Regia Liberated[4]	Residue[5]		CN Soluble[1]	HCl Liberated[2]	HNO_3 Liberated[3]	Aqua Regia Liberated[4]	Residue[5]
0X1	1.24	83.3	11.5	4.4	0.3	0.6	6.00	94.7	3.5	1.0	0.4	0.4
0X2	5.49	83.7	5.2	10.3	0.0	0.8	8.00	85.2	8.1	4.3	1.2	1.2
0X3	1.82	27.4	9.8	56.6	0.2	6.0	5.00	66.7	11.0	14.4	4.0	4.0
MP1	1.10	6.4	2.7	85.8	0.3	4.7	2.00	29.2	24.2	33.9	6.4	6.4
MP2	0.97	34.8	8.0	49.8	0.4	6.9	4.00	50.2	11.4	25.8	6.3	6.3
MP3	5.50	5.2	1.8	80.4	0.1	12.6	21.0	36.5	22.0	36.8	3.2	1.6
MP4	10.8	26.9	10.4	58.7	0.1	4.0	30.0	43.6	5.1	42.8	2.1	6.3
MP5	11.1	41.8	14.8	40.9	0.0	2.5	11.0	75.3	17.5	5.9	0.6	0.6

[1] Inferred as free milling Au/Ag
[2] Inferred as carbonate-locked Au/Ag
[3] Inferred as arsenopyrite-locked Au/Ag
[4] Inferred as pyrite-locked Au/Ag
[5] Inferred as silicate-locked Au/Ag

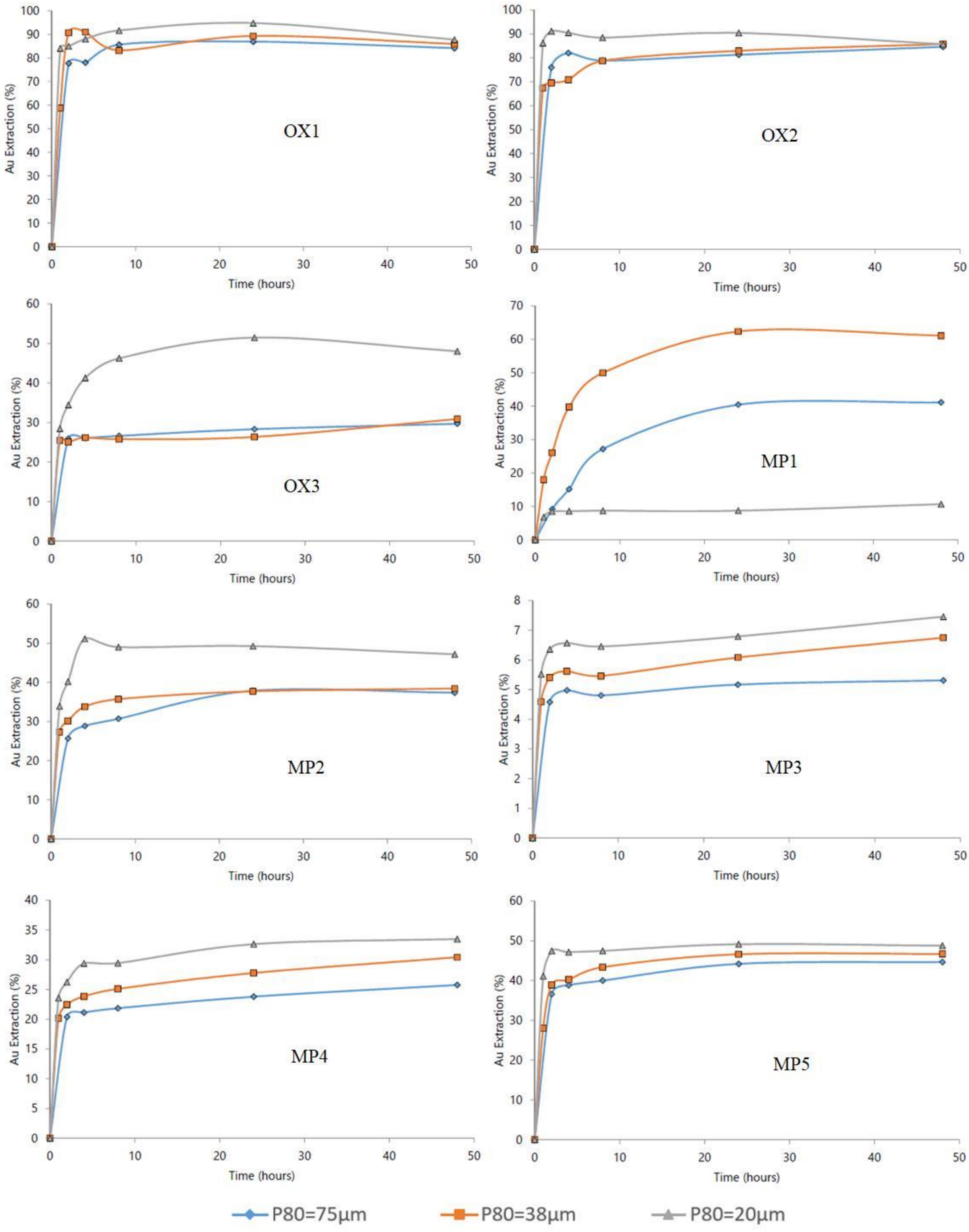


Figure 11-1: Kinetic Leach Dissolution Profiles of Composite Samples, AMML

11.2.4 Gravity Test on Composite Samples

Composite sample was ground to P_{80}=75µm and fed to Knelson concentrator in single pass. The Knelson concentrate was mercury amalgamated to extracted the free gold. Mercury amalgamation, amalgam tail and Knelson tail were dried, weighed and assayed. SRK calculated the Knelson concentrate grade and gold recovery as listed in Table 11-6. The gold recovery of sulphide ore is higher than that of oxide ore, but the concentrate grade is low, indicating gravity is not much applicable to process the ore of Ciemas deposit.

Table 11-6: Gravity Test Result, AMML

Comp	Product	Wt (%)	Au (g/t)	Au Dist'n (%)	Ag (g/)	Ag Dist'n (%)
OX1	Concentrate	5.10	8.49	26.1	13.2	11.8
	Knelson Tail	94.9	1.27	73.9	5.3	88.2
	Cal'c Head	100.0	1.66	100.0	5.7	100.0
OX2	Concentrate	5.20	9.51	8.5	16.6	11.7
	Knelson Tail	94.8	5.75	91.5	6.9	88.3
	Cal'c Head	100.0	5.84	100.0	7.4	100.0
OX3	Concentrate	6.84	8.94	32.0	18.1	20.0
	Knelson Tail	93.16	1.40	68.0	5.3	80.0
	Cal'c Head	100.0	1.91	100.0	6.2	100.0
MP1	Concentrate	5.92	15.8	57.4	17.7	35.8
	Knelson Tail	94.1	0.74	42.6	2.0	64.2
	Cal'c Head	100.0	1.63	100.0	2.9	100.0
MP2	Concentrate	5.20	9.83	40.9	18.0	22.0
	Knelson Tail	94.8	0.78	59.2	3.5	78.0
	Cal'c Head	100.0	1.25	100.0	4.3	100.0
MP3	Concentrate	7.80	31.3	43.4	97.9	37.0
	Knelson Tail	92.2	3.43	56.6	14.1	63.0
	Cal'c Head	100.0	5.62	100.0	20.6	100.0
MP4	Concentrate	7.72	49.5	35.1	125.0	28.5
	Knelson Tail	92.3	7.68	64.9	26.2	71.5
	Cal'c Head	100.0	10.9	100.0	33.8	100.0
MP5	Concentrate	6.75	38.2	21.1	28.2	17.0
	Knelson Tail	93.3	10.4	78.9	10.2	83.0
	Cal'c Head	100.0	12.2	100.0	11.5	100.0

11.2.5 Flotation Test on Composite Samples

The particle sizes of P80=106µm, 75µm, 53µm were selected for the flotation test, and the cupric sulfate, the Potassium Amyl Xanthate ("PAX") and the Interfroth 50 are respectively used as the activator, the collector and the foaming agent. The flotation test process comprises of 4 to 5 roughers of open circuit, and the coarse concentrates of each section are mixed to be the final flotation concentrate. The results of the flotation test are shown in Table 11-7, indicating that the selected particle size has a great influence on the flotation results. When the ore sample has a grind size of P80=75µm, the recovery rate of gold is higher. However, as for the MP1, MP3 and MP4, the recovery rate is over 90%, while the rest samples obtain poor Au recovery rates. In general, the flotation also failed to achieve a satisfactory gold recovery rate.

Table 11-7: Flotation Test Results, AMML

Composite	P_{80} µm	Au Balance			Ag Balance		
		Head Assay g/t	Concentrate g/t	Recovery %	Head Assay g/t	Concentrate g/t	Recovery %
Ox1	106	1.24	1.81	30.4	6	19.2	72.9
	75		2.02	26.2		20.1	60.6
	53		1.79	35.1		15.2	72.8
Ox2	106	5.49	10.90	32.6	8	24.1	61.1
	75		11.90	34.5		28.8	59.8
	53		10.40	38.8		24.4	67.0
Ox3	106	1.82	6.08	56.8	5	19.8	57.0
	75		7.06	62.0		20.9	56.8
	53		4.88	66.9		14.4	61.4
MP1	106	1.39	6.54	92.3	2	14.8	74.9
	75		8.86	95.1		14.8	74.6
	53		5.23	86.7		12.0	75.0
MP2	106	1.02	4.84	74.4	4	19.4	66.1
	75		5.14	73.9		19.3	66.4
	53		4.75	75.3		16.7	68.9
MP3	106	5.50	23.30	88.9	21	89.7	92.3
	75		22.80	90.6		83.6	92.2
	53		20.40	81.5		76.4	91.8
MP4	106	10.80	66.30	91.6	30	196.1	94.5
	75		59.50	93.1		191.0	96.5
	53		55.60	93.9		159.8	96.2
MP5	106	11.10	48.80	62.8	11	59.4	83.7
	75		46.10	66.4		60.8	85.4
	53		42.30	65.8		50.5	84.7

11.3 PT. Geoservices' Metallurgical Test

11.3.1 Composite Samples

PT. Geoservices - Minerals Division, a geometallurgical laboratory in Indonesia, conducted a set of metallurgical tests and comminution tests on samples from Ciemas deposit.

A *Characterization Test Report* dated November 2015， reported the diagnostic leach tests on five orebody composites and comminution tests on two composites. The composites were prepared based on the four deposits, and respectively consisted of 30 intervals for Cibatu, 42 intervals for Cikadu, 14 intervals for PasirManggu, and 8 intervals for Sekolah. The samples taken for each composite were selected based upon gold grade and deposit. The gold grade was selected based on what may be expected in the ROM ores. A range from 1-30g/t thus including the low-grade halo of dilution expected in the mining operation. The fifth composite was prepared based on very high grade (>30g/t) gold assays selected from 6 intervals blend of the deposits and was labelled High Grade.

The composites were separated into five deposits initially for the diagnostic testing to detect any differences in ore mineralogy between the deposits. Following the diagnostic tests, five composites were to be combined and mixed in preparation for the Response Test Program. The chemical components of composites are shown in Table 11-8, and minerals components detected by x-diffraction are shown in Table 11-9.

Table 11-8: Chemical Components of the Test Composites, PT. Geoservices

COMPOSITE	Au	Ag	As	Cu	Fe	Pb	Sb	Zn	TC*	OC*	TS*	SS*
	g/t	g/t	%	g/t	%	g/t	g/t	g/t	%	%	%	%
CIBATU	6.32	5.0	0.102	200	4.76	484	24	730	1.22	0.45	2.23	1.10
CIKADU	9.58	12.0	0.234	133	5.92	77	55	236	0.15	0.11	3.80	2.20
PASIR MANGGU	3.30	6.6	0.600	210	7.02	102	71	68	0.06	0.04	2.55	1.46
SEKOLA	1.85	5.5	0.416	230	6.99	371	32	599	0.25	0.17	2.94	2.07
HIGH GRADE	25.88	9.8	0.071	220	3.92	1,967	41	7,741	2.17	0.78	4.04	3.52

Note*: TC inferred as Total Carbon; OC inferred as Organic Carbon;
TS inferred as Total Sulphur; SS inferred as Sulphide Sulphur.

Table 11-9: Mineral Components of the Test Composites, PT. Geoservices

X-RD Phase Name		CIBATU	CIKADU	PASIR MANGGU	SEKOLAH	HIGH GRADE
Mineral Group	Mineral	Wt%	Wt%	Wt%	Wt%	Wt%
Calcite Group	Calcite, $CaCO_3$	10.0	2.2		1.6	20.0
Mica Group	Muscovite, $H_4K_2(Al,Fe)_6Si_6O_{24}$	21.8	22.0	18.5	30.0	19.6
Pyrite Group	Pyrite, FeS_2	2.1	3.7	3.2	2.8	2.7
Chlorite Group	Clinochlore, $Al_2Mg_5Si_3O_{10}(OH)_8$	15.1	17.3	4.8	10.2	8.4
Grossite	Grossite, $CaAl_4O_7$	4.0	6.1	5.4	4.2	2.9
Quartz	Quartz, SiO_2	39.0	28.5	43.8	37.5	26.8
Montmorillonite-Vermicullite Group	Montmorillonite, $CaMg_2AlSi_4 (OH)_2 \cdot H_2O$	2.5	2.5	9.9	2.3	7.0
Diaspore Group	Goethite, $Fe_2O_3 \cdot H_2$ 0			5.5	3.4	
Zeolite Group	PhHIipsite, $K_{0.8}Na_{0.7}Ca_{0.7}Al_{2.8}Si_{5.1}O_{16} \cdot 6.4H_2O$	5.5	8.8	7.2	7.0	12.8
Anatase	Anatase, TiO_2		1.3	1.7	1.0	
Plagioclase Group	Albite, $NaAlSi_3O_8$		7.5			

11.3.2 Diagnostic Leach Test

The results of the diagnostic test are summarized in Table 11-10. The following comments are presented after the review of the diagnostic leach tests:

- The diagnostic leach tests are believed to have been successfully performed due to the final residue grades reporting in the range of 0.15 to 0.25g/t Au range. The Ciemas residues are within this common range of residues normally found in a diagnostic leach test.
- The High Grade composite reported a very high cyanide leachability. This composite also reported low proportion of gold associated with the pyrite and arsenopyrite minerals. This is consistent with the head assays and also the low Arsenic levels. The gold in this sample is far more liberated and not locked in solid solution within the pyrite or arsenopyrite;
- There are significant amounts of gold locked in the sulphides or pyrite and arsenopyrite in the other four composites. The Cibatu, Cikadu, PasirManggu and Sekolah composites reported the following proportions of gold locked in the sulphides – 25.7%, 58.6%, 39.3%, and 52.7%;

- The diagnostic tests were performed at fine grinds of 80% passing 38um and therefor grind size cannot be inferred as the reason for the low cyanide leach recoveries;
- Three of the composites, Cibatu and Cikadu and High Grade were tested at different grinds. The Cibatu and Cikadu composites reported only marginally increased leach recoveries at the finer grind and therefore confirms that ultrafine grinding is not be viable process option. This was also observed in the AMML test report;
- The tests have identified the amount of gold leached by cyanide and adsorbed onto naturally occurring carbonaceous material (organic carbon) present in the ores. This was reported in all composites tested at varying degrees. The Cikadu and PasirManggu reported 18.7% and 24.7% respectively as gold leached, but subsequently adsorbed onto the organic carbon in the sample, which means 50% and 44% of the gold successfully liberated and leached by cyanide, was lost to organic carbon in these tests, also known as "preg-robbing". The magnitude of the preg-robbing is significant and will need to be considered when designing the process plant;
- The composites prepared were not prepared based on high Sulphur grades but more on average mine grades with some variation and all reported preg-robbing.

Table 11-10: Summary of Diagnostic Leach Test, PT. Geoservices

Composite	Grind Fineness (P_{80})	Au Deportment										Diag Leach Calculated Head Grade
		Cyanide SolubleAu						Roasted & Aqua Regia Digest		Residue Fire Assay		
		Direct Cyanide Leach		Elution Test		Total Cyanide Soluble Au						
		CN Soluble		Preg Robbed				Pyrite Locked		Silica Locked		
	μm	%	g/t	%	g/t	%	g/t	%	g/t	%	g/t	g/t
CIBATU	38	68.8	5.12	2.3	0.17	71.1	5.30	25.7	1.91	3.3	0.25	7.45
CIKADU	38	18.4	0.63	18.7	0.64	37.1	1.26	58.6	1.99	4.3	0.15	3.40
PASIR MANGGU	38	31.4	0.56	24.7	0.44	56.2	0.99	39.3	0.70	4.5	0.08	1.77
SEKOLAH	53	28.1	0.39	5.0	0.07	33.1	0.46	52.7	0.73	14.2	0.20	1.39
HIGH GRADE	53	87.9	26.8	1.19	0.36	89.1	27.1	9.8	2.98	1.1	0.33	30.45

The response to cyanide leach had been conducted in two different grind sizes. The summary results are as Table 11-11. There is no indication of promising recovery to direct cyanide leach of the samples. The results confirm the diagnostic leach test.

Table 11-11: Direct Cyanidation Recovery at Different Grinded Size

Composite	P_{80}=106μm		P_{80}=38μm	
	Au %	Ag %	Au %	Ag %
CIBATU	67.4	40.7	12.8	34.1
CIKADU	5.7	2.1	4.7	3.1
PASIR MANGGU	21.6	1.2		
SEKOLAH	26.3	17.0		
HIGH GRADE	64.1	40.2	57.0	2.2

11.3.3 Flotation Test

Flotation tests were conducted on the sample of five composites combination, using several combinations of reagents such as PAX only, PAX plus $CuSO_4$ as surface modifier, SIBX (Sodium Iso Butyl Xanthate) plus $CuSO_4$. 4-stage rougher flotation circuit is applied under a nominal grind size P_{80}=75µm. The tests results are shown in Table 11-12. The PAX only - Sulphur recovery is the highest which indicate the best selectivity. However, the gold grade and recovery rate are not high.

Table 11-12: Flotation Test Results, PT. Geoservices

Products	Mass Recovery (%)	Grade (g/t)				Metal Recovery (%)				Note
		Au	Ag	As %	S_total %	Au	Ag	As	S_total	
Conc 1+2	3.7	12.0	29.0	0.37	11.90	5.8	14.7	7.8	15.0	P_{80}=63µm
Conc 3	4.7	6.0	33.0	0.41	12.10	3.7	21.2	11.0	19.3	
Conc 4	9.3	15.5	13.0	0.30	6.16	18.8	16.6	15.7	19.5	PAX
Tail	82.4	6.7	4.2	0.14	1.66	71.9	47.4	65.6	46.5	
Calc Head	100.0	7.7	7.3	0.18	2.94	100.0	100.0	100.0	100.0	
Conc 1+2	3.7	6.9	37.0	0.46	11.80	2.2	14.9	7.9	13.1	P_{80}=70µm
Conc 3	9.9	11.0	28.0	0.42	9.87	9.2	30.1	19.2	29.3	
Conc 4	9.9	21.0	18.0	0.42	7.77	17.6	19.4	19.2	23.0	PAX + CuSO4
Tail	76.6	11.0	4.3	0.15	1.52	71.2	35.8	53.7	34.9	
Calc Head	100.0	11.8	9.2	0.22	3.34	100.0	100.0	100.0	100.0	
Conc 1+2	5.5	5.0	18.0	0.27	6.50	3.9	11.8	7.1	11.3	P_{80}=70µm
Conc 3	6.7	6.0	21.0	0.37	8.78	5.6	16.8	11.7	18.6	
Conc 4	15.8	12.0	13.0	0.32	5.71	26.7	24.5	23.7	28.6	SIBX + CuSO4
Tail	72.1	6.3	5.5	0.17	1.83	63.9	47.2	57.5	41.8	
Calc Head	100.0	7.1	8.4	0.21	3.16	100.0	100.0	100.0	100.0	

11.3.4 GRG Test

Gravity separation test on combined composite by Knelson Concentrator ("KC") to concentrate gravity recoverable gold ("GRG") was conducted after a *sighter test*, which indicated a potential high GRG recovery rate may be reached. The GRG test result is shown in Table 11-13, indicating a gold grade about 376g/t and 66.5% gold recovery in concentrate under the grind size P_{80}=75µm.

Table 11-13: GRG Test Result, PT. Geoservices

Products	Wt (%)	Grade (g/t)			Recovery (%)		
		Au	Ag	As (%)	Au	Ag	As
Concentrate	1.5	375.99	57.9	1.1	66.5	7.2	8.5
Tail	98.5	2.83	11.2	0.2	33.5	92.8	91.5
Calc'd Head	100.0	8.31	11.8	0.2	100.0	100.0	100.0
Head Assay		15.16					

11.3.5 Flotation Test of KC Tail

The flotation tests were designed to confirm response and screen the best reagent combination. The main reagent PAX and SIBX combination are being reviewed. The flotation is followed by cleaning (flotation of the flotation concentrate) test. Bulk test flotation is conducted with the optimum condition, using bulk Knelson tail (KC-Tail). Flotation test was also carried out on a new composite. The result is shown in Table 11-14, indicating very poor flotation performance.

Table 11-14: Flotation Test Result of KC tail, PT. Geoservices

Products	Wt (%)	Grade (g/t)				Metal Recovery (%)			
		Au	Ag	As (%)	S_total (%)	Au	Ag	As	S_total
Combined Composite KC Tail									
Conc 1	1.8	3.87	31.0	0.22	5.9	3.5	8.9	2.2	5.0
Conc 2	2.1	4.61	33.0	0.27	8.6	5.0	11.3	3.3	8.6
Conc 3	5.9	5.49	24.0	0.30	6.8	16.4	22.7	9.7	18.8
Conc 4	5.2	4.71	20.0	0.33	6.4	12.4	16.6	9.5	15.6
Tail	85.0	1.45	2.95	0.16	1.3	62.7	40.4	75.4	52.0
Calc'd Feed	100.0	1.97	6.21	0.18	2.1	100.0	100.0	100.0	100.0
New Composite									
Conc 1	12.4	2.73	12.0	0.39	4.2	9.8	24.6	12.0	13.1
Conc 2	2.9	8.21	27.0	0.50	11.2	6.9	12.9	3.6	8.2
Conc 3	2.0	9.70	22.0	0.64	15.1	5.5	7.1	3.1	7.5
Conc 4	1.6	10.63	18.0	0.75	16.6	5.1	4.9	3.1	6.9
Tail	81.2	3.09	3.76	0.39	3.1	72.8	50.5	78.3	64.4
Calc'd Feed	100.0	3.45	6.04	0.40	4.0	100.0	100.0	100.0	100.0

11.3.6 Cyanide Leach Test on Flotation Tail

Flotation produces concentrate and tailing. The test is to confirm the amenability of the gold in the tailing to cyanide leach. The standard CIL process was applied on two samples of flotation. The test result is summarized in Table 11-15. The gold recovery is about 62% while silver is about 54% at 24 hours leach. The standard 48 hours leaching time is confirmed in improving the gold recovery to 67% and silver to 62%.

Table 11-15: Test Results of Flotation Tails

CIL Time	Calc'd Feed Grade (g/t)		Residue Grade (g/t)		Extraction (%)	
	Au	Ag	Au	Ag	Au	Ag
24hr	2.25	1.74	0.86	0.80	61.6	53.8
48hr	2.87	3.33	0.95	1.27	66.9	61.9

11.3.7 Roasting and Cyanidation Test of Flotation Concentrate

After a set of roasting condition optimization tests, two stage roasting at temperature 550℃ and 850℃ was adopted. The cyanide leach test of roasted of flotation concentrate was done at the original particle size distribution. It is reported 76.8 % Au recovery at 24 hours cyanide leaching. Longer roasting time and standard 48 hours cyanide leaching would improve the Au recovery to 90 %, provided the Calcine is being grinded to P_{80}=75μm before leaching. The results are summarized in Table 11-16.

Table 11-16: CIL Test Results on Roasting Calcine of Flotation Concentrate

CIL Time	Calc'd Feed Grade (g/t)		Residue Grade (g/t)		Extraction (%)	
	Au	Ag	Au	Ag	Au	Ag
24hr	5.26	9.81	1.22	7.10	76.8	27.6
48hr	6.78	10.88	0.7	8.04	89.7	26.1

11.3.8 BIOX and Cyanidation Test of Flotation Concentrate

The summary of the bacterial oxidation ("BIOX") process following by cyanide leaching is shown in Table 11-17. The BIOX processes indicate a gold recovery improvement from 84 to 91%. KC tail that was being used, has lower sulphur content compare to the direct flotation tail. It is also identified that bacteria A is the best, comparing to BSS and Mixed of the A and BSS.

Table 11-17: CIL Test Result on BIOX Flotation Concentrate

Extraction Processes	Cumulative Au Extracted (%)		
	Test 1	Test 2	Average
Direct Cyanidation	85.8	82.3	84.03
BIOX-A + Cyanidation	90.0	91.5	90.77
BIOX-BSS + Cyanidation	84.0	84.4	84.21
BIOX-Mixed culture + Cyanidation	85.9	87.0	86.46

11.3.9 Detoxification Test

SO_2/air method known as INCO method is widely used in the treatment of detoxification of cyanide-containing waste water/tailings. From the detoxification test using sodium metabisulphite ("SMBS"), the weak acid dissolvable ("WAD") and the total cyanide can be detoxified to the below government of Indonesia threshold limit after 360 minutes or about 6 hours. The detail of the test can be seen in Table 11-18.

Table 11-18: Detoxification Test Result

Item	Unit	Reaction Time (minute)				
		0	60	120	240	360
pH		8.84	8.6	8.86	8.74	8.64
Free CN	mg/L	75.8	44	12.2	2.4	0.7
WAD CN (Assayed)	mg/L	81	NA	NA	NA	1.7
Total CN (Assayed)	mg/L	86	NA	NA	NA	3.4
Cu	mg/L	23.47	NA	NA	NA	0.1
Ni	mg/L	<0.1	NA	NA	NA	<0.1
Fe	mg/L	36.91	NA	NA	NA	10.0
Zn	mg/L	14.28	NA	NA	NA	<0.2

11.4 PT. Geoservices' Comminution Test

A series of comminution tests including JK Drop-Weight test and/or SMC test on two composites were conducted in Cikarang laboratory of PT. Geoservices. The test data was provided to JKTech Australia to analysis and/or predict SAG/autogenous mill performance by JKSimMet software. A series of Bond Work Indices were tested to consummate the comminution parameters.

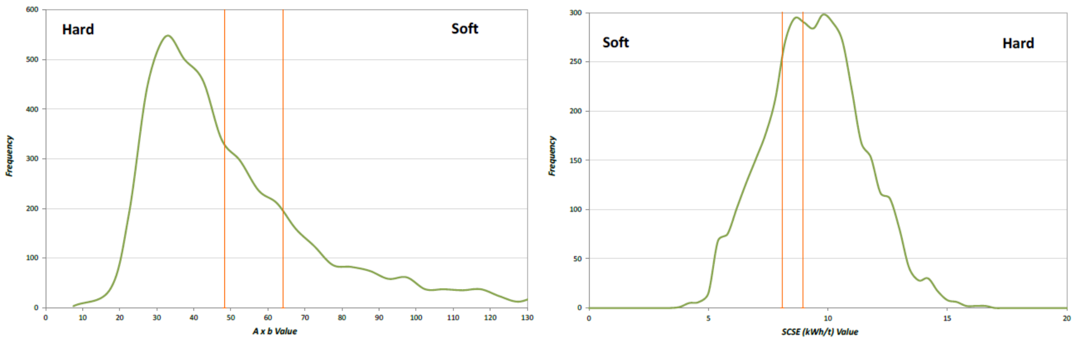
The results of the comminution tests will provide information for the selection and design of the comminution circuit in the Ciemas processing plant. The comminution composites were prepared based purely on depth whereby the intervals selected were of mine grade and separated out according to the following two categories – 0 to 80m & 80 to 150m depth down the hole. There was insufficient core sample available to prepare four distinct composites according to deposit. The two composites prepared based upon depth in the hole are therefore a blend of the four deposits.

The comminution test results are summarized in Table 11-19.

Table 11-19: Comminution Test Results and Parameters

Comminution Parameters			Comp.1 (0.00-80.00m)	Comp.2 (80.00-150.00m)
Drop-weight/SMC test results and derived Parameters	Drop-weight Index (DWi) (kWh/m^3)	Value	5.40	3.66
		%	34	15
	Mi (kWh/t)	M_{ia}	16.8	13.9
		M_{ih}	11.9	9.1
		M_{ic}	6.2	4.7
	Specfic Gravity		2.6	2.3
	A*b	Value	48.4	64.2
		Rank %	45.4	27.1
	ta	Value	0.48	0.71
		Rank %	45.9	25.0
	SAG Circuit Specific Energy (SCSE) (kWh/t)	Value	8.97	8.09
		Rank %	41.7	25.7
Bond Work Indices	Crushing Work Index (CWi) (kWh/t)		2.2	8.5
	Abrasion Index (Ai)		0.14	0.07
	Abrasion Index, Passing at 13.2mm		83.7	84.4
	Bond Rod Mill Work Index (BRWi) (kWh/t)		12.50	12.27
	Bond Ball Mill Work Index (BBWi) (kWh/t)		18.90	17.07

The Drop-Weight/SMC results and derived parameters are measures of resistance to impact breakage, which required to be processed by JKSimMet software. With the comparison to Drop-Weight database and SMC database, the SMC results/parameters indicate the two tested composites are in "soft" category as shown in Figure 11-2, which report the Ciemas ores to have moderate grinding energy requirements.



Frequency Distribution of A*b in the JKTech Database   Frequency Distribution of SCSE in the JKTech Database

Figure 11-2: Frequency of SMC parameters in the JKTech Database

Crushing Work Index (CWi) – also known as the Impact Work Index is an expression of the materials crushability, and is the energy required to accomplish a given crushing operation. The CWi values of the two Ciemas composites fall within the "very soft" classification (<10) and therefore highlights the low crushing energy requirements of the Ciemas ores.

Abrasion Index (Ai) – which is determined by the Bond Abrasion Test, is used to determine steel media and liner wear in crushers, rod mills, and ball mills. The two Ciemas composites prepared reported low abrasion indices which indicate low liner and grinding media consumption in the comminution circuits.

The BRWi&BBWi are measures of the resistance of the material to grinding. The indices are used to determine the grinding power required for a given throughput of material under ball mill grinding conditions. The work index is the power required to grind a short tonne of ore from a given feed size to a given product size. The two Ciemas composites prepared reported very similar Rod mill and Ball mill work indices. The grinding energy requirements of the Ciemas ores are deemed moderate in magnitude.

11.5 Conclusion and Recommendation

- With a high arsenic content, the Ciemas ore demonstrated refractory gold characteristics. Tests of gravity separation, cyanide leaching, diagnostic leaching, flotation, flotation concentrate roasting, bacterial oxidation - cyanide leaching, flotation tailings cyanide leaching were carried out. The test results showed that it was difficult to obtain a satisfactory gold recovery rate by a single processing and smelting method, and a combination method must be adopted.
- Although gravity separation could achieve good recovery results as an assistant method, but it can be exceeded by flotation method. Therefore, in the case of using the flotation method to recover gold, it is better not to apply gravity separation .
- A flowsheet of "Flotation + Concentrate Roasting and CIL + Flotation Tailings CIL" recommended by PT. Geoservices is technically reasonable, because it can adapt to changes in ore properties and achieve high gold recovery rate. However, economically, the flowsheet is more complicated, and the investment and operating costs are relatively high.
- In order to direct optimizing the metallurgical parameters and simplifying the flowsheet, SRK proposes that supplementary detailed process mineralogical study should be carried out to further analyse the types of gold minerals and their occurrence, the types of major metal minerals and their occurrence, and the reasons for refractory.
- It is recommended to collect representative samples and conduct a "Chemical Pre-oxidation + CIL" test to explore the feasibility of a simplified flowsheet.

12 Metallurgical Plant Design and Recovery

12.1 Introduction

SRK and NERIN conducted a feasibility study on the development of the Ciemas gold mine, and adopted the flowsheet of "Flotation + Concentrate Roasting and CIL + Tailings CIL" to design the processing and smelting engineering and facilities.

The designed metallurgical complex consists of a flotation plant, a concentrate roasting and sulphric acid plant, a flotation tailing CIL and roasting calcine CIL plant, a loaded carbon desorption-electrowinning-regeneration and refining plant ("DER"), a tailings storage facility, and associated laboratory and other auxiliary facilities. The metallurgical plant is designed with a nominal capacity of 495,000tpa for a design (nominal) rate of 1,500tpd, based on an overall availability of 90.4% with a 7-year life-of-mine. The plant is designed to operate 365 days per year, 24 hours per day. The crushing circuit is designed with a mechanical equipment availability of 67.8%.

The feasibility study proposes to construct the facilities of the metallurgical plant and associated facilities in two stages. In Stage One, the workshops for crushing, milling and flotation will be built, together with the CIL plant for flotation tailings, and the workshops for gold desorption, refining and carbon activity regeneration and tailings storage facilities. The construction period is 18 months for Stage One.

In Stage Two, the roasting workshops for flotation concentrate, associated acid manufacture, and the calcine CIL plant for the roasted concentrate will be constructed. The construction period of Stage Two is one year. Prior to the completion of Stage Two, the final products of the project will be flotation concentrate and the gold doré. The throughput of the roasting plant and associated CIL plant is 100,000tpa matching with the production of the flotation concentrate. The final product after Stage Two facilities have come into production include gold doré and associated sulphuric acid.

12.2 Metallurgical Flowsheet

The flowsheet is described as below. The simplified process flowsheets are as Figure 12-1 and Figure 12-2.

- **Primary crushing:** A dump pocket, heavy-duty plate ore feeder and jaw crusher in open circuit crushed the ROM (-500mm) down to minus 60mm. The crushed material is delivered into the intermediate bin by 1# belt conveyor.
- **Grinding:** ore in the intermediate bin is fed by 2 vibration feeders onto 2# belt conveyor and then directed into the SAG mill grinding the ore size down to 80% minus 1500μm. Material from the SAG mill is fed a set of hydro-cyclones which forms a closed circuit with a ball mill. The underflow of the cyclones gravities to a ball mill for secondary grinding. The discharge of the ball mill is pumped back into the cyclones. The overflow of the cyclones has a particle size of 80% minus 75μm, gravities to flotation circuit.
- **Flotation:** the cyclones overflow is conditioned with flotation reagents in an agitating tank, and then flows through a bank of flotation cells which consists of one roughing, two scavenging and two cleaning operations, producing a gold concentrate and a tailing for successive processing.
- **Concentrate dehydration:** flotation gold concentrate is pumped into a thickener in a diameter of ¢18 m. The thickener's underflow is then pumped into a box type filter press. The filtered concentrate is for sale or roasting before and after the roasting plant put into operation.
- **CIL of flotation tailing:** the flotation tailing is delivered by a slurry pump into a thickener to enhance the concentration up to 40%. The thickener bottom flow is regulated for pH=10

in a agitation tank, and then pumped into six CIL tanks connected in series. The first CIL tank is a pre-leaching tank. The gold-loaded carbon from the secondary CIL tank is hydraulically delivered onto a gold-loaded carbon recovery screen. The gold-loaded carbon remaining on the screen is directed into the DER operation. Fresh active carbon is injected from the last CIL tank and operates against the direction of the pulp. CIL tailings is directed by gravity onto the safety screen to recover fine carbon. Tailings from the screen is directed by gravity into the detoxification cycle where detoxification and inner plant backwatering is performed before the residue is delivered into the TSF.

The main process parameters for CIL gold extraction are given below:

- ✓ CIL pulp concentration: 40％;
- ✓ Leaching time: 48h;
- ✓ NaCN consumption: 2kg/t ore; and
- ✓ Gold-loaded carbon grade: 3.5Kg/t carbon.

- **Concentrate Roasting Oxidation**: two-stage fluid-bed roasting process is designed to oxidize the concentrate for gold liberation. The first stage roasting temperature is 700℃ and the second stage is 900℃. The roasting calcine is sent into a buffer bin for regrinding after cooling. The smoke of the furnaces is dedusted while recovering heat for cogeneration. Arsenic is oxidized to As_2O_3 during the roasting and settled in the flue pipe after dedusting and cooling. The As_2O_3 is collected periodically as a by-product, known as white arsenic. The fume after recovering As_2O_3 contains SO_2, converted onto SO_3 by going through vanadium catalyst bed two times. The SO_3 is absorbed by sulphric acid two times producing industrial sulphric acid with a concentration of 98%.
- **Roasting calcine regrinding:** roasting calcine from the buffer bin is fed out by a vibration feeder onto 3# belt conveyor and then into a ball mill for regrinding. Discharge from the ball mill is classified by a set of classifying hydro-cyclones. The underflow of the cyclones is returned to the ball mill, and overflow is pumped into the calcine CIL circuit.
- **Roasting calcine CIL:** the reground calcine pulp is pumped to a pre-leaching thickener to enhance the concentration to 40%. The bottom flow is regulated in a pH tank, and then pumped into six CIL tanks connected in series. The first CIL tank is a pre-leaching tank. The gold-loaded carbon from the secondary CIL tank is hydraulically delivered onto a gold-loaded carbon recovery screen. The gold-loaded carbon remaining on the screen is directed into the DER circuit. Active carbon is injected from the last CIL tank and operates against the direction of the pulp. CIL tailing is directed by gravity onto the safety screen to recover fine carbon. Tailings from the screen is directed by gravity into the detoxification cycle where detoxification and inner-plant thickening and backwatering are performed before the product is delivered into the TSF.
- **Cyanide Detoxification:** detoxification of the CIL residues via the SO_2/air process with the addition of copper sulphate, to produce tailings with a target of <1 ppm CN WAD (Weak Acid Dissociable) and the disposal of detoxified tailings in the TSF.
- **Gold extraction (DER process):** Zadra process is selected in the FS to extract gold from loaded carbon. After gold-loaded carbon is pickled, it is desorbed in an elution column to produce a gold-rich solution. The pregnant solution is filtered and heat-exchanged before it is pumped into the pregnant solution tank, from where it flows into an electrowinning cell. The resultant lean solution is returned to the desorption cycle after adding NaOH and NaCN. Cathode gold of electrowinning (sludge production) is then filtered, dried, and smelted to produce the final product - gold bullion or gold doré. The striped carbon is delivered into the carbon regeneration kiln to remove organic contaminants for reclaiming its activity. The regenerated carbon is then returned into the CIL cycle.

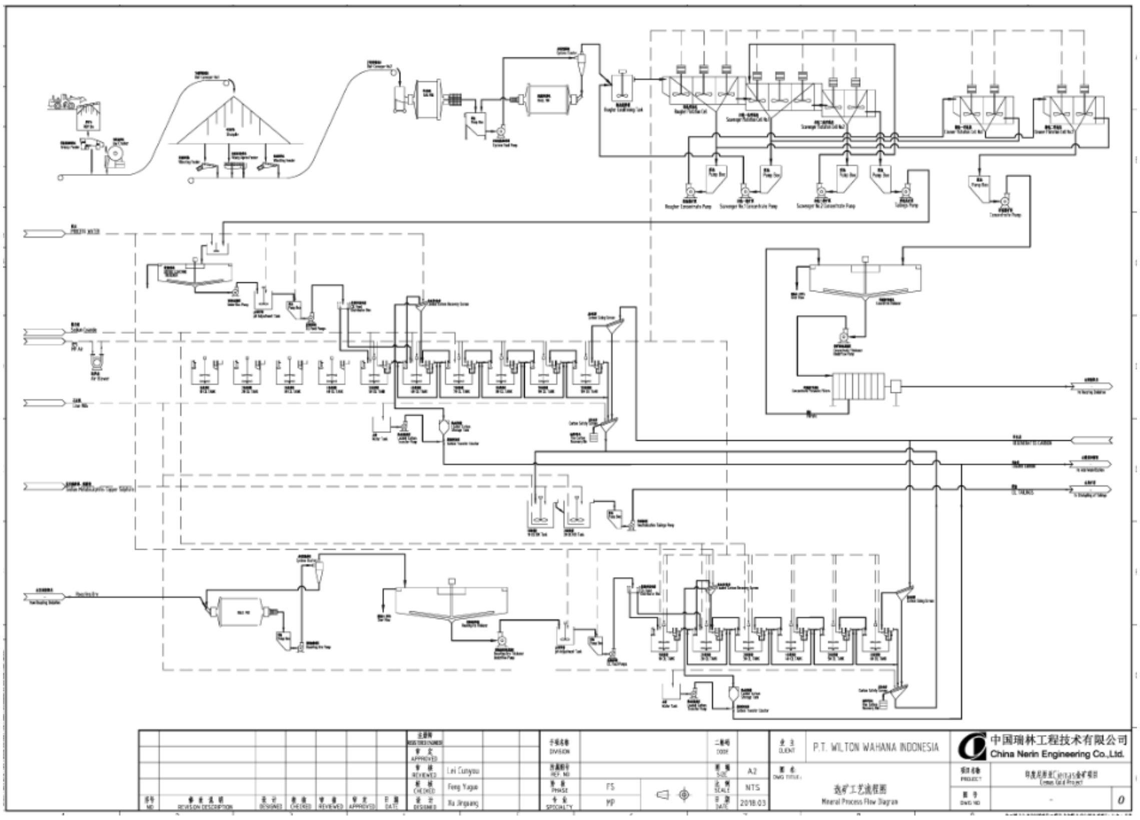


Figure 12-1: Simplified Flowsheet of Flotation and CILs

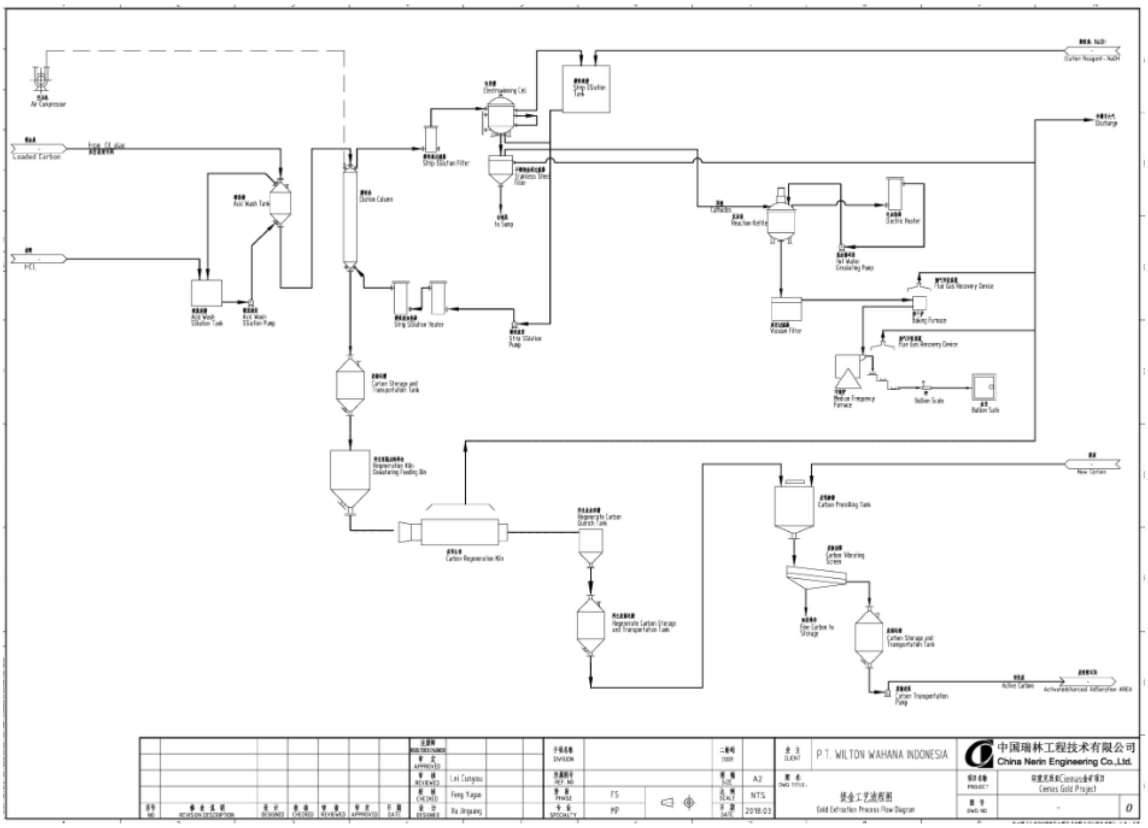


Figure 12-2: Simplified DER Flowsheet

12.3 Designed Metallurgical Recovery

Metallurgical Production schedule and recovery index of LOM is as Table 12-1.

Table 12-1: Metallurgical Production Schedule of LOM

Description	Unit	Total	Production Period						
			2020	2021	2022	2023	2024	2025	2026
Flotation Feed	kt	3,260	359	495	495	495	495	495	426
Feed Grade	g/t	7.73	8.49	7.95	8.39	8.34	8.09	7.15	5.60
Gold contained in Feed	kg	25,204	3,048	3,937	4,155	4,130	4,006	3,538	2,389
Flotation Recovery	%	76.69	81.50	81.50	81.50	81.50	80.00	74.78	43.25
Tonnage of Concentrate	kt	644	83	107	113	112	107	88	34
Gold Grade of Concentrate	g/t	30.00	30.00	30.00	30.00	30.00	30.00	30.00	30.00
Concentrate for Roasting	kt	229					100	95	34
Grades of Concentrate for Roasting	g/t Au	30.00					30.00	30.00	30.00
	% S	22.00					22.00	22.00	22.00
	% As	3.00					3.00	3.00	3.00
Tonnage of 98% Sulphuric Acid	kt	157.8					68.8	65.3	23.7
Tonnage of 98% As2O3	kt	9.3					4.0	3.8	1.4
Flotation Concentrate CIL-DER Recovery	%	91.14					91.14	91.14	91.14
Flotation Tailings CIL-DER Recovery	%	72.17	67.01	67.01	67.01	67.01	68.89	73.68	83.87
Gold from Roasting Concentrate by CIL-DER	kg	6,274					2,734	2,598	942
Gold from Flotation Tailings by CIL-DER	**kg**	**4,240**	**378**	**488**	**515**	**512**	**552**	**657**	**1,137**
Salable Gold Concentrate	**kt**	**415**	**83**	**107**	**113**	**112**			
Gold Contained in Salable Concentrate	**kg**	**12,446**	**2,484**	**3,209**	**3,386**	**3,366**			
Gold Doré Salable	**kg**	**10,514**	**378**	**488**	**515**	**512**	**3,286**	**3,255**	**2,079**
Total Gold Recovered (in bullion & concentrate)	**kg**	**22,959**	**2,862**	**3,697**	**3,901**	**3,878**	**3,286**	**3,255**	**2,079**

12.4 Metallurgical Equipment

Master metallurgical equipment list in Table 12-2 below.

Table 12-2: Master Metallurgical Equipment List

№	Operation	Equipment	Specification	Quantity
1	ROM Feeding	Heavy Duty Plate Feeder	HPF1060(S)	1
2	Primary Crushing	Jaw Crusher	JC1000	1
3	SAG Mill Feeding	Vibrating Feeder	GZG90-4	4
4	1st Grinding	SAG Mill	Φ8.5×5.2m	1
5	2nd Grinding	Ball Mill (Overflow)	Φ3.2×6.0m	1
6	Classifying for ROM Grinding	Hydro Cyclone Set	Φ350×8	1
7	Trash Removal	Vibrating Screen	ZKR1848	1
8	Roasting Calcine Grinding	Ball Mill (Overflow)	Φ2.4×4.0m	1
9	Classifying for Calcine Grinding	Hydro Cyclone Set	Φ350×4	1

10	Pulp Conditioning for Flotation	Agitation Tank	⊄2.5×2.5m	3
11	Flotation of Roughing and Scavenging	Flotation Cells	KYFⅡ-16	7
12	Flotation of Cleaning	Flotation Cells	KYFⅡ-4	4
13	Flotation Concentrate Dewatering	Thickener	Φ18m	1
14	Flotation Concentrate Dewatering	Filter Press	XMZG300/1500	2
15	Flotation Concentrate Roasting	Fluid Roaster	Φ7.8×20m	1
16	Oxidation Roasting	Fluid Roaster	Φ13.4×20m	1
17	Flotation Tailing Thickening	Thickener	Φ35m	1
18	Roasting Calcine Thickening	Thickener	Φ15m	1
19	Flotation Tailing CIL	Agitation Tank	Φ11×12m	6
20	Roasting Calcine CIL	Agitation Tank	Φ8.5×9.0m	6
21	Detoxification	Agitation Tank	Φ8.5×9.0m	2
22	Loaded Carbon Elution, Electrowinning	Integrated Line	3t per lot	1
23	Carbon Regeneration	Rotary Kiln	TZY-800,3t/d	1
24	Gold Smelting and Refining	Medium Frequency Furnace	KGPS-160	1

12.5 Tailings Storage Facilities

The tailings storage facility ("TSF") is located in a natural valley in the north-east of the metallurgical plant, which is adjacent to the TSF. The valley faces mountains on three sides, and the terrain is high in the north and low in the south, with a small catchment area of about 0.23km^2. By damming the valley mouth, it can form a storage capacity of 1,217,400m^3. So damming on this spot can create an ideal TSF.

The tailings dam uses a one-time damming method with roller-compacted earth-rock materials from the TSF and nearby. This not only reduces the cost of the dam body, but also increases the storage capacity of the TSF. The crest elevation of the tailings dam is 486.0m, the crest width is 6.0m, the maximum dam height is 32.0m (the foundation cleaning depth of the dam foundation is temporarily considered as 2.0m), and the axis of the tailings dam is about 270.0m. The slope of the upstream dam surface of the tailings dam is 1:2.5, and that of the downstream is 1:2.75. Berms of 3m width are set up at the 470m elevation of the upstream and downstream slopes.

In order to ensure that the TSF water does not infiltrate into the ground, a comprehensive anti-seepage treatment for the TSF area is tentatively scheduled at this stage. The seepage prevention materials use 2.0mm HDPE impermeable membranes, and the membranes are protected by 400g/m2 geotextile underneath.

Drainage prisms are set at the elevation of 470.0m downstream. The top width of the drainage prism is 3.0m. The upstream slope of the prism is 1:1.0, and the downstream slope is 1:1.8. The non-woven geotextile of 400g/m^2 is laid on the upstream slope of the prism as the reverse filter layer. The embedded depth of the geotextile in the dam base and abutment shall not be less than 0.5m, and it shall be densely packed with earth materials. To protect the geotextile from being damaged by the rock mound, a cushion layer consisting of 20cm-thick grit and 25cm-thick gravel shall be laid underneath.

In order to prevent the erosion of the abutment from the slope rainwater, abutment intercepting ditches should be designed along the junction slope of the downstream slope of the tailings dam and the slopes on both sides of the bank. The abutment drainage ditch has a rectangular cross-section with a size of 0.5m×0.5m, and adopts an M10 cement mortar block stone structure. In order to minimize the rainfall flowing into the TSF area from the slope, flood intercepting ditches shall be set up around the TSF to divert wastewater away from clean water. The discharge clearance cross section of the ditch adopts a trapezoidal cross section. The bottom width is b=1.0m, the depth is h=1.5m, and the slope ratio of the two side slopes is 1:1. At the same time, a flood spillway is set up on the left abutment, which adopts a broad-crested weir shape for water discharging. The broad-crested weir is 6m wide and 10m long. The inlet of the overflow weir is connected to a diversion canal. The canal is 22.0m long and 6.0m wide. The overflow weir is gradually reduced to a discharge chute of 4.0m wide through a transition section, which is 15.0m long.

The TSF adopts an overall anti-seepage scheme and the flood discharging uses a spillway in the shoulder of the dam. The tailings slurry is to be discharged into the TSF at the opposite end of the dam. An 8m×5m×3m steel plate floating pontoon is to be installed in the TSF with two sets of self-priming pumps. When one is in use, the other will be on standby as a back-up. The recycled water is pumped to a 700m^3 backwater head tank through an approximately 1000m-long welded steel pipe. The water then flows to the plant for reuse. In order to prevent the tailings water from seepage to the downstream from the dam body, a seepage collection tank with 10.0m×20.0m×3.0m is installed at the downstream of the tailings dam to collect the leaked tailings water, from where the water will be pumped to the head tank in the metallurgical plant.

12.6 Conclusion and Recommendation

- The feasibility study carried out a comprehensive design of the metallurgical plant. Based on the existing metallurgical test results, the "Flotation-Roasting-CIL" flowsheet and corresponding recovery rate indexes were formulated. There is no serious defect in the plant design. Due to the complexity of the process flowsheet which includes operations of flotation, roasting, acid making and CIL, it is going to well adapt the changes of ore characteristics and reach high gold recovery, but the capital cost and operating cost will be also relatively high. In order to simplify and optimise the flowsheet and lower the capital and operating costs in future, SRK recommends supplementary mineralogical and metallurgical researches before the plant construction.
- The flotation tailings and concentrate roasted product are designed subject to CIL separately, which is feasible, while a simplified scheme is to combine the two materials for CIL together. It is recommended to supplement cyanide leaching tests on the mixed materials of flotation tailings and concentrate roasted product.
- It is feasible to build a one-time completed dam with roller compacted earth-rock for the TSF. As the entire TSF will be laid with HDPE membrane to prevent seepage with the use of pontoon for backwater, and spillway is used to prevent flooding on the abutment, it is still worth discussing whether the dam body needs gravel drainage prism or not, since the construction cost of it is high, but the effect is limited.

13 Occupational Health and Safety

13.1 Project Safety Assessment and Approvals

SRK has sighted the original Occupational Health and Safety ("OHS") officer appointment approval with its English translation for the Ciemas Gold Project. This approval was issued by the Department of Mining and Energy of the Regency of Sukabumi on 9 December 2011. The certified OHS officer ensures the safety of the site and employees.

13.2 Occupational Health and Safety Management and Observations

SRK has not sighted the OHS management system/procedures and records for the current Ciemas Gold Project. However, SRK notes that the Feasibility Study Reports on gold mining, gold processing plant, and the Feasibility Study reports provide the following summary with respect to the proposed OHS management measures for the project:

- Occupational safety and health administration;
- Occupational safety and health training;
- Organisation of an occupational health and safety fund;
- Side slope protection measures;
- Safety mining, blasting and transportation procedures and guidance;
- Debris flow prevention measures;
- Electric shock and lightening stroke prevention measures;
- Fire prevention measures;
- Dust and noise prevention measures;
- Placing of safety and hazard signage;
- Provision of personal protection equipment ("PPE") to all relevant employees;
- Regular medical and physical checks for the employees;
- Operational safety guidance for equipment; and
- Mechanical maintenance safety guidance.

The Company is in the process to develop a comprehensive OHS system to include these components above, and they will be implemented during the operational stage.

13.3 Historical Occupational Health and Safety Records

SRK notes that the project is still under construction and therefore records of OHS statistics, such as the number and type of incident/accidents and associated injuries, have yet to be generated.

14 Infrastructure and Facilities

14.1 Water Supply and Drainage System

14.1.1 Water Supply

The requirements of water consumption, water quality and water pressure of mining, processing and roasting acid making are listed in Table 14-1.

Table 14-1: Water Consumption (Unit: m^3/d)

No.	Item	Total Water Consumption	Water Consumption				Elevation of Water Consuming Point (m)	Remarks
			New Water	Dewatering	Circulating water	Domestic Water		
1	Mining area(including filling station and compressor station)	2,814	554	100	2,160			
2	Processing plant	8,012	644	3,528	3,840			Repeating utilization rate of water 92%
3	Roasting acid making area (including dust collection station and power station)	57,532	3340		54,192			Repeating utilization rate of water 94.2%
4	Domestic water	200				200		
5	Unforeseen demand	762	762					Account for 16.1% of the production new water
6	Total	69,320	5,300	3,628	60,192	200		

Production (domestic) new water is supplied by the owner to the project's 1,500m^3 elevated tank. The new water will come from Ciemas River near the Ciemas Town, which is about 6 km from the project. According to the Company, the water use permit is in the process of being obtained.

14.1.2 Drainage

The normal discharge volume of Cikadu, Sekolah and Cibatu sections is 2,410 m^3/d, and the maximum discharge is 3,210 m^3/d. The water quality is good. At this stage, direct discharge is proposed, but in the future production, the water may be recycled for production based on its quantity and quality.

The normal drainage volume of the west section of Pasir Manggu is 690 m^3/d, and the maximum discharge is 915 m^3/d. The water quality is good. At this stage, direct discharge is proposed but in the future, the water may be recycled for production based on its quantity and quality.

14.2 Power Supply and Distribution System

Generally there is no concern in power supply for the Project as sufficient capacity exists at the Pelabuhan Ratu power generation plant which is available for Wilton.

14.2.1 External Power

The Project is designed to powered by electricity from Ciemas township, about 6 km away from the mine site. The power supply system study commences from the high-voltage outlet cabinet (cable head) of the general step-down transformer substation in the mine site, and excludes the external power supply and general step-down transformer substation.

Indonesia's national MV and LV standard voltage are 20kV and 380/220V. The AC voltage frequency is 50 Hz.

14.2.2 Power Loads Estimate

According to production process requirements, the drainage pump, ventilator and air compressor in the pit have the first level loads. The working capacity is 1,442 kW and the maximum motor power of the ventilator is 2×220 kW. The intermediate fan of the roasting furnace, booster pump, boiler circulating pump, electric boiler feed pump, emergency air compressor and other equipment in the roasting acid making area have the first level loads. The working capacity is 325.5 kW and the maximum motor power of the boiler circulating pump is 110 kW.

Other main production equipment has second level load. The auxiliary production facilities such as office buildings, laboratories and machine repair workshops are of third class loads.

The power load estimate results can be found in Table 14-2.

Table 14-2: Power Load Estimate Results

Items		Mining Area, Processing Area, and Roasting and Acid Manufacturing Area
Installed Load (kW)		21,724
Working Capacity (kW)		18,282
Power Load Estimate	Active Power (kW)	11,826
	Reactive Power (kvar)	4,741
	Apparent Power (kVA)	12,741
	Power Factor	Above 0.92
Total Power (k-kwh)		86,942.4

14.2.3 Power Supply and Distribution System in Infrastructure Area

The preliminary estimation of the voltage grade of power distribution in the mine site is:

- Voltage of power supply circuit in the mine site: 20 kV;
- Low voltage power distribution voltage: 380/220 V;
- Frequency: 50 Hz.

(1) 20 kV power Distribution system in the mining area

The mining areas of Cikadu, Sekolah and Cibatu sections are powered by two 20 kV overhead lines (A1/A2) which are introduced from different 20 kV different bus sections of the general step-down transformer substation in the mining area. The A1/A2 double circuit 20kV overhead lines are laid along 1# ventilation fan – motor vehicle repair workshop-ramp entrance (automobile repairing workshop)-filling station- air compressor room-2# ventilation fan.

The mining area of Pasir Manggu west section is powered by two 20 kV overhead lines (B1/B2) which are introduced from 20 kV different bus sections of general step-down transformer substation in the mining area. The B1/B2 double circuit 20 kV overhead lines are laid along 3# ventilation fan – hoist room of incline-4# ventilation fan.

(2) 20 kV power distribution system in the processing area

According to the processing flowsheet, workshop division, load size and distribution, a 20kV distribution substation is installed in the grinding workshop of the processing plant. The processing plant is powered by a 20 kV cable line which is connected from different 20 kV bus sections of the general step-down transformer substation. The 20 kV power distribution system in the processing plant uses sectionalized single-bus configuration, and a bus tie circuit breaker. The two circuits of power lines work simultaneously but operate independently. When one circuit becomes faulty, the other circuit can meet the running conditions of all power loads in this area.

(3) 20 kV power distribution system in the roasting acid making area

According to the processing flowsheet, workshop division, load size and distribution, a 20kV distribution substation is installed in the main control building of the roasting acid-making area. The roasting acid-making area is powered by a 20 kV cable line which is connected from different 20 kV bus sections of the general step-down transformer substation. The 20 kV power distribution system in the roasting acid-making area uses sectionalized single-bus configuration, and a bus tie circuit breaker is included. The two circuits of power lines work simultaneously, butoperate independently. When one circuit becomes faulty, the other circuit can meet the running conditions of all power loads in this area.

In order to make full use of the waste heat generated in production, a 20 kV/1500 kW waste heat power station is built in the roasting acid-making area. The waste heat power station is connected to the second section of the 20kV bus line of 20 kV distribution substation in the main control building of the roasting acid-making area.

14.2.4 Electrical Parameters

Electrical parameters (power and electrical equipment voltage) can be found in Table 14-3.

Table 14-3: Electrical Parameters

Power Supply in the Mine Site	20kV，3 phase，50Hz，
Electric Equipment Voltage	380/220 V（neutral grounding），TN-S system
Varying-speed Alternating-Current Motor	Determined by speed adjustment plan
Motor (250kW and above)	AC 20kV，3 phase，50Hz
Motor (Below 250kW)	AC 380V，3 phase，50Hz

14.3 Communication

The Project is designed to be divided into mining area, processing area, roasting acid manufacturing area and infrastructure. Infrastructure is described in this section. For details of the remaining areas, please find the corresponding parts in communication section.

The infrastructure area mainly consists of office buildings and dormitories.

The following communication systems have been designed and detailed in the Feasibility Study Report.

- Telephone Communication System;
- Computer Network System;
- Integrated Cabling System;
- Cable Television System;
- Video Surveillance System and

- Automatic Fire Alarm System.

14.4 Emergency Diesel Power Station

The emergency diesel power station is designed to be located in the master step-down substation in the mine area. It provides the first level load of power supply for the entire mining area during any power failures of the external network.

The scale of the power station is determined in consideration of the load capacity and load nature, the product specifications of current domestic diesel generator set, and the stability and reliability of power supply.

The power station is designed to be equipped with two 1,500 kw diesel generator sets, and the rated output power of the power station is 3,000 kW.

Fuel and Diesel will be transported by truck. Each of the power stations is equipped with a 5m^3 buried oil storage tank. When the oil tank is full, it can ensure the continuous operation of the work unit for about 8 to 10 hours. Tank refuelling can be provided by a tank truck or the general tank depot of the mine.

The diesel in the storage tank is pumped into the daily oil tank via oil pump, from where the diesel flows to the generator through pipelines.

The emergency diesel power station is an independent plant, in which the diesel generators are arranged. The daily oil tanks are arranged in separate compartments, and the buried oil storage tanks are installed outdoors.

The size of the diesel power station building is 18 m × 9 m with a storey height of 5.0 m.

The machine room air compressor station of the emergency diesel power station is designed to be equipped with ventilation devices. The emergency diesel power station will be equipped with ventilation devices between the oil tanks.

14.5 Buildings

The buildings for this Project were considered in following areas.

- Building Materials and Construction
- Fire Control and Evacuation of Buildings
- Safety Protection

General Building Construction includes:

(1) Wall body and wall surface:
Block walls of 240 mm thickness are used as the outer walls of the steel structure plant below 1.2 m, and coloured steel plates or anticorrosive plates are used as the wall surface above 1.2 m. The inner and outer wall surfaces of the block walls use cement sand plaster, and the surfaces are painted with wall coating.

(2) Floor:
The standard workshops use fine aggregate concrete flooring, and the distribution room uses high-grade cement mortar flooring.

(3) Doors and windows:

The windows use single-glass plastic steel windows or FRP daylighting bands, and the doors are plastic steel doors and steel doors. The bottom is open and a ventilating skylight is set on the ridge for workshops with ventilation requirements. FRP daylighting bands are used for the large-span roofing.

(4) Roofing:

The roofing of the steel structure plant uses single-layer profiled steel sheet. The building roofs generally allow the free fall of water, while those with greater heights have organized drainage with UPVC pipes as the downpipes.

(5) Steel ladders are made using steel grating treads.

(6) For steel railings and handrails, please refer to the China Standard Atlas.

14.6 General Layout

This Project is designed to be a large-scale mining, processing, smelting and chemical joint venture. The mining capacity is 1,500 t/d of raw ore by underground mining. The target mining sections designed are the four mining sections of Cikadu, Sekolah, Cibatu, and Pasir Manggu West. The production capacity of Cikadu, Sekolah and Cibatu is 1,200 t/d and that of the Pasir Manggu is 300 t/d.

The underground mine uses the cut and fill mining method. The zone development, ramp development and inclined shaft development methods are used for different sections of the mine.

The raw ore mined will produce gold concentrate through a series of processes including coarsely crushing, semi-autogenous grinding, ball-milling, classification, and one-roughing two-scavenging and two-cleaning flotation. The gold concentrate is dewatered by two-stage thickening and filter pressing, followed by roasting oxidation, roasting calcine regrinding and CIL treatments. The flotation tailings are treated by CIL too, prior to which it will be thickened. The gold-loaded carbon produced from the CIL processes of the flotation tailings and roasting calcine will be treated by DER process to produce the final product of alloy gold (3.34 t/a) and the by-product of flue gas acid production, which is 98% H_2SO_4 (100 kt/a).

The capital construction period of the mine is 2 years, and the service life of the mine (LOM) is 7 years.

According to the production requirements, the mine needs to build underground acess, waste dumps, tailings storage facilities (TSF), mining industrial sites, ore processing industrial sites, tailings facilities and other industrial sites, as well as management and living areas, sewage treatment systems and auxiliary facilities such as water supply and power supply facilities.

The site selection plans for this project mainly consist of the site selections of the processing, smelting and chemical industrial site, the waste dump site, and the TSF site.

The project is located in a tropical hilly area with complex terrain conditions. There are criss-cross valleys on the sites and the local relative height difference can reach more than 20m. At the same time, the site traffic conditions are poor, and the roads within the sites must be re-planned. The surface industrial facilities at the site should be rationally arranged within the available land around the mining operation sites, by comprehensively considering factors such as topography, climate, logistics, technical feasibility and economic rationality.

Figure 14-1 is a simplified map of the General Layout for the Project according to the Feasibility Study.

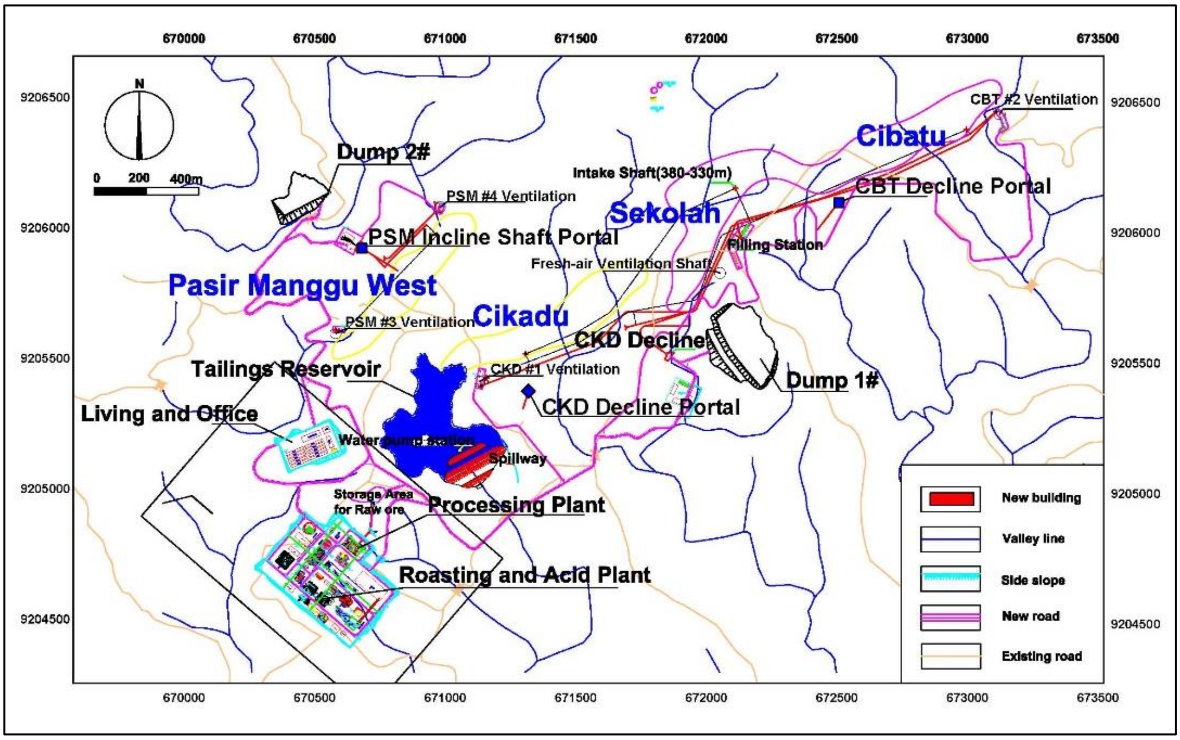


Figure 14-1: Simplified General Layout for the Project

15 Environmental and Social Assessment

15.1 Environmental and Social Review Objective

The objective of this environmental and social due diligence review is to identify and or verify the existing and potential environmental and social liabilities and risks, and assess any associated proposed remediation measures for the Ciemas Project. The site visit for this environmental review was undertaken in September 2017, on which time limited mining and processing activities took place, such as trenching and pitting of gold ores, and temporal cyanidation leach processing.

15.2 Environmental and Social Review Process, Scope and Standards

The process for the verification of the environmental compliance and conformance for the Ciemas Project comprised a review and inspection of the project's environmental management performance against:

- Indonesian national environmental regulatory requirements (see Appendix 3);
- World Bank/International Finance Corporation ("IFC") environmental and social standards and guidelines (see Appendix 3); and
- Internationally recognized environmental management practices (Appendix 4).

15.3 Status of Environmental Approvals

Indonesian national mining and environmental laws both require mining companies developing projects that are deemed to have significant potential environmental and/or social impacts to produce an environmental impact assessment and planning document (called an *Analisa Mengenai Dampak Lingkungan* or "AMDAL" in Indonesian). An AMDAL consists of an environmental impact assessment (called *Analisis Dampak Lingkungan* or "ANDAL"), an environmental management plan (a *Rencana Pengelolaan Lingkungan* or "RKL"), and an environmental monitoring plan (*Rencana Pemantauan Lingkungan* or "RPL").

SRK has sighted two original AMDALs (each including an ANDAL, RKL, and RPL) for the Ciemas Project, and the Company has provided English translations of these documents. The main AMDAL issued to the Company is dated August 2010 and its approval by the Regent of Sukabumi is dated 16 August 2010. SRK notes that this approval was based on a recommendation by the Chairman of the AMDAL Appraisers Commission of Sukabumi Regency dated 21 July 2010, and was copied to the State Minister of Environment. The other original ANDAL issued to Liek Tucha is dated August 2000. SRK has not sighted the RKL, RPL, and approval for this original ANDAL. The Company states that in 2000, the RKL and RPL were not required and that an approval for this original ANDAL was not issued by the government. However, the Company also states that the AMDAL dated August 2010 covers the environmental management of both mining license areas.

15.4 Environmental Compliance and Conformance

SRK notes that the AMDALs and its approval for the Company have been compiled in accordance with the relevant national Indonesian laws, regulations, and decrees.

SRK notes that the AMDALs for the Company contain the project's Environmental Management Plan (RKL) and Environmental Monitoring Plan (RPL). In addition, 2016 1st half year RKL-RPL implementation report, and 2016 2nd half year RKL-RPL implementation report were also provided to SRK. SRK has reviewed these documents against recognised international industry environmental management standards, guidelines, and practices. SRK provides comments in respect to the project's proposed environmental management measures.

15.5 Land Disturbance and Flora and Fauna

Based on SRK's site visit, existing terrain, soil, meteorological conditions are favourable for the growth of vegetation. The Company's RKLs, and RPLs contain proposed measures for controlling and monitoring soil erosion and minimising loss of flora and fauna habitat. However, SRK notes that the proposed measures for minimising and monitoring the project's overall land disturbance and subsidence are not clearly defined. SRK notes that the recognised international practice is to establish operational procedures for controlling/minimising land disturbance and subsidence that comprise the annual surveying and recording of areas of project land disturbance (including areas of disturbed land that have been rehabilitated and mined-out areas to be backfilled).

SRK also notes that the AMDALs do not specify whether there are rare, endangered, and/or significant flora and fauna within the project area.

15.6 Waste Rock/Overburden Management

The Company RKLs, and RPLs contain proposed measures for controlling and monitoring soil erosion and sedimentation. SRK notes that these proposed management measures can be applied to any storage of waste rock or overburden, however, the Company RKLs and RPLs do not provide any specific information with respect to the proposed management of the project's waste rock/overburden, in particular, the proposed measures/design for waste rock storage, geochemical/acid rock drainage ("ARD") assessment of the waste rock, and any potential for leaching/ARD risks/impacts (including drainage/flood and seepage management).

SRK notes that the recognised international practice is to complete a waste rock geochemical characterisation/ARD assessment and determine the potential for any significant leaching/ARD risks/impacts. The outcome of this assessment is then incorporated into a design for the proposed site waste rock dump.

15.7 Water Aspects

The project area is rich in water resources due to the high levels of precipitation; however, may not have enough water during dry season since no permanent water bodies nearby and poor groundwater productivity in this area. The Company states that the mine is supplied by groundwater for operating and domestic use.

The proposed water management measure provided within the Company RKLs and RPLs are:

- Stormwater/surface water drainage (including any mine dewatering): diversion channels, drainage systems, and sedimentation ponds are to be constructed around and within the mining and port areas;
- Surface water quality: water quality monitoring is to be conducted regularly;
- Processing water reuse and recycle system; and
- Oil separators and septic tanks to treat domestic wastewater.

SRK notes that the Company RKLs, and RPLs do not provide any details (i.e., with respect to design and management) for:

- A proposed site drainage system, including mine dewatering and stormwater drainage pathways, drainage pathways capacity analysis, and collection and discharge points/facilities; and
- Site hydrogeology and groundwater management, including limits for groundwater extraction and proposed extraction methods/facilities.

SRK notes that the recognised international practice is to complete the site hydrology/hydrogeology assessment and incorporate the results into designs for the proposed site surface water and groundwater management, especially into the detailed design of the processing water reuse and recycling system (once the gold processing method is determined).

15.8 Air Emissions

The proposed site dust and gas emission management measures provided within the Company RKLs, and RPLs are:

- Regular watering of roads and open areas with water trucks;
- Maintaining surface moisture on ore stockpiles with water sprays;
- Setting vehicle speed limits at designated areas and limiting the frequency of vehicle traffic;
- Utilizing special vehicles to carry gold ore which meet emissions requirements;
- Conducting regular preventative maintenance on vehicles and heavy equipment;
- Age restriction on vehicles that are used;
- Keeping gold ore in closed storage spaces;
- Maintaining regular ambient air quality monitoring (i.e., dust and gas monitoring) at the site boundary; and
- Recording and responding to any public complaints in relation to any site dust/gas emissions.

SRK notes that the above proposed site dust and gas emission management measures are in line with recognised international industry environmental management guidelines and practices. During the site visit, SRK did not observe any significant dust emissions mainly due to very limited mining and processing activities. However, SRK also did not observe any water trucks on site. SRK notes that the recognised international practice is to develop site operating procedures for these dust and gas emission management measures.

15.9 Noise Emissions

The proposed site noise emission management measures provided within the Company RKLs and RPLs are:

- Scheduling mobile equipment usage and materials transport during daylight hours;
- Setting vehicle speed limits at designated areas (e.g., at or near residential areas) and limits on the frequency of vehicles;
- Use of hearing protection for relevant personnel;
- Ensuring that vehicles are suitable for use and conducting regular preventative maintenance on vehicles and heavy equipment;
- Age restrictions on vehicles in use;
- Regular ambient noise quality monitoring at the site boundary and in residential areas; and
- Regularly liaising and consulting with the surrounding residents on any perceived issues with site noise emissions.

SRK notes that the above proposed site noise emission management measures are generally in line with recognised international industry environmental management guidelines and practices. SRK notes that the recognised international practice is to develop site operating procedures for these noise emission management measures.

15.10 Hazardous Materials Management

The main hazardous materials for the project's mining operations will mainly comprise the storage and handling of hydrocarbons (fuels and lubricants), and chemicals (cyanide and others once the processing plant is built). SRK notes that the Company's mining RKLs do not make any references to the management of hazardous materials.

SRK notes that the recognised international practice is to identify and quantify the hazardous materials for the project, and to design and construct all storage and handling facilities with secondary containment.

15.11 Waste Management

15.11.1 Waste Oil

Waste oil will be generated from the maintenance of the project's mobile equipment (i.e., within the proposed site workshops). SRK notes that the Company's RKLs do not discuss any proposed management measures for this waste oil. SRK notes that the recognised international practice is for all waste oil generated to be stored within site facilities that have secondary containment, and that the options for off-site recycling/disposal should be assessed.

15.11.2 Solid Wastes

Domestic and industrial solid wastes will be generated from the project operations. SRK notes that the Company's RKLs refer to the collection of solid wastes but do not provide any detail on the final disposal method and facilities (i.e., landfill, incineration, off-site collection, etc.). SRK notes that the recognised international practice is to identify and quantify the solid wastes for the project, and that the disposal method and facilities also are defined, designed and operated in line with relevant international guidelines.

15.11.3 Sewage and Oily Wastewater

Domestic sewage will be generated from the general project operations. Oily wastewater will be generated from the washdown and servicing of mining mobile equipment. SRK notes that the Company's RKLs refer generally to wastewater treatment but do not provide any detail on the final disposal method and facilities (i.e., sewage treatment plant, septic tanks, etc.).

The Company's mining RKLs do not refer to the management of oily wastewater. However, the Company's RKL does refer to an oil catcher for treating "*oil from workshop*".

SRK notes that the recognised international practice is to identify and quantify the domestic sewage and oily wastewater wastes for the project, and that the disposal methods and facilities also are defined, designed and operated in line with relevant international guidelines.

15.12 Contaminated Sites Assessment

The Ciemas Project has the potential to generate contaminated areas of land from spillages of fuels and oils. SRK has not sighted, as part of this review, a documented operational process to assess and remediate any areas of suspected contamination. SRK notes that the recognised international practice is to develop a contaminated sites assessment and management process.

15.13 Operational Environmental Management Plan

The RKLs and RPLs for the Ciemas Project provide the basis for the project's operational Environmental Management Plan ("EMP"). However, these have yet to be developed into an operational EMP. SRK notes that the recognised international practice is to develop an operational EMP, which provides detailed actions, schedules and responsibilities, and is reviewed and updated as the operational situation changes. EMPs are generally updated on an annual basis as part of the overall operational planning.

15.14 Emergency Response Plan

The recognised international industry practice for managing emergencies is for a project to develop and implement an Emergency Response Plan ("ERP"). The general elements of an operational ERP are:

- Administration: policy, purpose, distribution, and definitions of potential site emergencies and organisational resources (including setting of roles and responsibilities);
- Emergency response areas: command centres, medical stations, and muster and evacuation points;

- Communication systems: internal and external communications;
- Emergency response procedures: work-area specific procedures (including area-specific training);
- Checking and updating: prepare checklists (role and action lists and equipment checklists) and undertake regular reviews of the plan; and
- Business continuity and contingency: options and processes for business recovery from an emergency.

SRK notes that a documented operational ERP has yet to be developed for the Ciemas Project.

15.15 Site Closure Planning and Rehabilitation

The AMDALs contain general references to the proposed site rehabilitation, but the AMDALs do not contain any detailed rehabilitation scoping and planning information. However, SRK notes that the RKLs for the Ciemas Project includes a summary of site reclamation to be implemented under the "*Land Reclamation*" sections of the RKLs.

The recognised international industry practice for managing site closure and rehabilitation is to develop and implement an operational site closure and rehabilitation planning process and document this through an operational Closure and Rehabilitation Plan. This operational closure planning process generally includes the following components:

- Identify all site closure stakeholders (e.g., government, employees, community);
- Undertake stakeholder consultation to develop agreed site closure criteria and post operational land use;
- Maintain records of stakeholder consultation;
- Establish a site rehabilitation objective in line with the agreed post operational land use;
- Describe/define the site closure liabilities determined against agreed closure criteria;
- Establish site closure management strategies and cost estimates to address/reduce site closure liabilities;
- Establish a cost estimate and financial accrual process for site closure; and
- Describe the post site closure monitoring activities/program to demonstrate compliance with the rehabilitation objective/closure criteria.

SRK notes that a documented operational Closure and Rehabilitation Plan has yet to be developed for the Ciemas Project. For this moment, total amount of Rp 500,000,000 has been deposited in the bank as a reclamation guarantee to local authority. The unit reclamation cost for the Ciemas Project is estimated to be Rp 10,000 per square meter. The estimated cost on mine closure is USD 3.4 million approximately.

15.16 Social Aspects

The project area is administratively located in the District Ciemas and District Simpenan. The mining and processing activities are exactly located in the Village Mekarjaya, Village Ciemas and Village Cihaur. The Village Mekarjaya has the highest population density namely 3 inhabitants per ha, while the Village Cihaur District Simpenan has the lowest population density namely one person per ha. By such population rate, villages in the study area can be categorized has low population.

Based on interviews with local villagers and district officers, business activities/means of livelihood of most people are argrucutural industry and mining industry. The dominant means of livelihoods of the local people are farmers, employee/private company workers, and trade/services. Other job opportunities mostly offered are working for gold mining companies based on the results of interviews with the local people.

Currently the power is supplied via the local grid, and diesel generators are another alternative. A large-capacity power station and a port project are under construction in Pelabuhan Ratu, which is about 12 km away from the mine site in a straight line.

The Ciemas Project's RKLs and RPLs contain comprehensive summaries of the project's proposed social management measures. These measures comprise the following:

- Public perceptions and public attitudes: public consultation will be undertaken throughout all phases of the project, including establishing a process to record and respond to local public complaints;
- Improve local economic conditions: setting local employment/recruitment targets and giving priority to employing local residents, utilising and/or supporting local businesses and undertaking technical skills training programs for local employment candidates; increasing local revenues in the Sukabumi Region through payment of local royalties and taxes;
- Public health and amenity: manage/minimise air (dust) and noise impacts, monitor the quality of the local water supply; monitor local public health conditions; and provide information to the local community; and
- Site land reclamation planning: consult with local residents on site reclamation planning, employ local residents on site closure works, and provide training and redeployment support for local employees and businesses.

SRK notes that the above proposed social management measures are in line with relevant recognised industry international guidelines and practices. In addition, SRK has sighted, as part of this review, some community land access/compensation agreements for the development of the Ciemas Project.

The Company has stated that the Indonesian government is focusing on investment attraction and increasing employment opportunities. The Company has also stated that they intend to recruit a majority of the project employees from the local population, which will benefit the local economy. At the time of this report completion, the Ciemas Project allocated USD 100,000 for the prospective local villagers' relocation. Furthermore, 2% of gross profit will be spend on contribution to local responsibility.

15.17 Evaluation of Environmental and Social Risks

The sources of inherent environmental and social risks are project activities that may result in potential environmental and social impacts. These project activities have been previously described within this report. Based on the site visit observations and the review of the proposed management measures within the provided documents, SRK notes that the sites are generally being managed to meet minimum Indonesian National requirements listed in the related environmental approvals.

The significant inherent environmental and social risks for the Ciemas Project are:

- Land disturbance and subsidence;
- Poor water management (i.e., stormwater/surface water drainage including any mine dewatering);
- Waste rock stockpiling/waste rock dump management;
- Poor dust management; and
- Soil and groundwater contamination (i.e., poor hydrocarbon storage and handling).

It is SRK's opinion that the environmental and social risks for the Ciemas Project are categorised as between moderate/tolerable to high risks. More environmental management efforts are required to mitigate these risks into manageable levels.

16 Workforce

16.1 Enterprise Structure

The operating company of the Ciemas Gold Project will be supervised and led under the management of Wilton, while the direct operation will be led by the general manager. Within the mine, there will be some functional departments and various workshops (plants and factories). The workshops (working units) designed for the project include the mining areas, acid making factories, and ore processing and smelter plants, as well as associate and production supportive facilities.

Functional departments include human resources, accounting, HSE, and purchase and sales, in addition to the general and deputy general managers.

16.2 Working Scheme

It is proposed that the working days per year will be 330 days. The working schemes for the production and administrative personnel are as below:

- For mining, processing and smelter, and acid manufacturing workshops, there will be continuous 7 days per week, three shifts per days and 8 hours per shifts, and 330days/a, which will require 4 personnel crews.

- For the administration personnel, there will be one shift per day, and 8 hours per shift, and 250days/a, and will be adjusted when it is necessary.

16.3 Personnel Requirements

The overall personnel requirements for the project is 711 personnel, including 661 people in the production workshops, ands 50 people in the general administration departments. Table 16-1 gives the details.

Table 16-1: A List of Manpower Proposed for the Project

No.	Working / Production Unit	Quota (person)	Percentage of total
A	**Mining Total**	**434**	**65.66%**
1	Administration	3	0.45%
2	Mining technical personnel	24	3.63%
3	Geological technical personnel	3	0.45%
4	Mine operators	376	56.88%
5	Maintenance personnel	28	4.24%
B	**Roasting and acid-making**	**94**	**14.22%**
1	Administration and service	14	2.12%
2	Production workshop	35	5.30%
3	Supportive production workshop	45	6.81%
C	**Ore processing and smelter plants**	**117**	**17.70%**
1	Administration personnel	8	1.21%
2	Plant operators	24	3.63%
3	Carbon cyaniding workshop (tailings)	16	2.42%

4	Roasting residue re-milling workshop	4	0.61%
5	Carbon cyaniding workshop	8	1.21%
	(Roasting residue of concentrates)		0.00%
6	Gold extraction workshop	16	2.42%
7	Supportive production workshop	41	6.20%
D	**Associate Supportive workshop**	**16**	**2.42%**
1	Machinery repairing	6	0.91%
2	Purification station of living water	4	0.61%
3	Drivers	6	0.91%
E	**Total Production Workshops**	**661**	**100.00%**
F	**Management and Support Departments**		
1	General Manager	1	
2	Vice General Manager	2	
3	HR Department	10	
4	Finance Department	8	
5	HSE Department	6	
6	Purchase and Sales Department	15	
7	Driver	8	
	Total Administration	**50**	
G	**Grand total**	**711**	

17 Capital Expenditures and Operating Expenses

17.1 Capital Expenditures (CAPEX)

17.1.1 Summary

The production capacity of the Ciemas Gold Project is designed as 495,000tpa of mining and ore processing, as well as 65,000tpa flotation concentrate roasting and associated acid manufacture. According to the construction plan provided by Wilton, the initial construction commencing in 2018 will include the mining system and the ore processing system without including the further processing of flotation concentrates, and the roasting system for further processing of the flotation concentrates as well as associated facilities, such as acid manufacture plant will be constructed in 2023. Table 17-1 summarizes the initial capital expenditures for the project. Table 17-2 summarizes the sustaining CAPEX for the project after the production of the Project.

Table 17-1: Summary of Initial CAPEX Required for the Ciemas Gold Project

Cost centres	Cost (1000 USD)	%
Mining	34,005.8	34.2%
Ore processing	19,090.7	19.2%
TSF	6,922.3	7.0%
Infrastructure	17,808.2	17.9%
Others	10,492.2	10.6%
Subtotal	88,319.0	88.9%
Contingency	11,039.9	11.1%
Total	**99,358.9**	**100.0%**

Note: Totals may not sum due to rounding

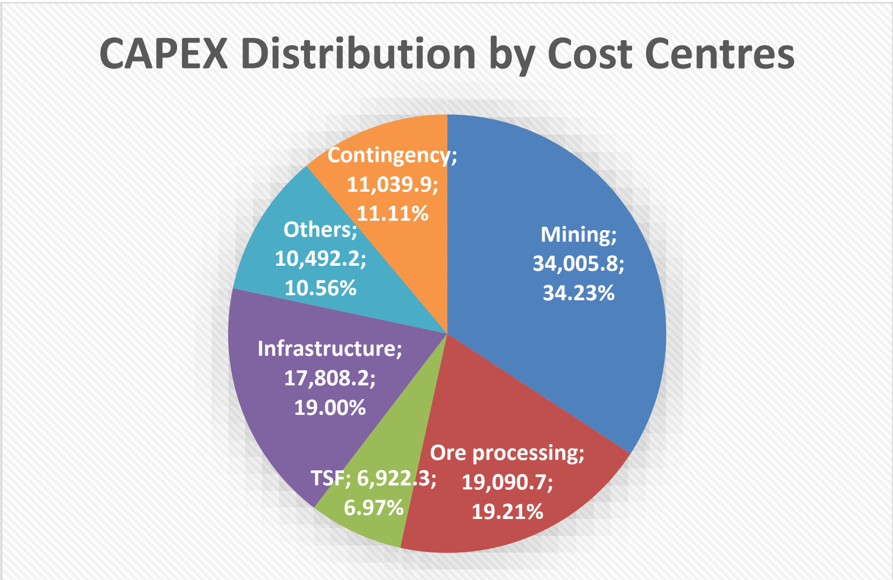


Figure 17-1: Distribution of Initial CAPEX by Cost Centres

Table 17-2: Summary of Sustaining CAPEX Required for the Ciemas Gold Project (1000 USD)

Cost Centres	Development	Construction	Procurement	Installation	Other	Subtotal
Mining	7,269.1					7,269.1
Ore processing		628.1	2,151.9	540.6		3,320.6
Roasting		11,838.3	20,927.4	9,062.7		41,828.4
Infrastructure		3,125.8	1,043.1	283.1		4,452.0
Total Engineering	**7,269.1**	**15,592.3**	**24,122.4**	**9,886.4**		**56,870.1**
Other costs					4,524.9	4,524.9
Engineering + Other	**7,269.1**	**15,592.3**	**24,122.4**	**9,886.4**	**4,524.9**	**61,395.0**
Contingency					7,674.4	7,674.4
Total Sustaining Capital	**7,269.1**	**15,592.3**	**24,122.4**	**9,886.4**	**12,199.3**	**69,069.4**

Note: Totals may not sum due to rounding

In addition to the CAPEX, there will be USD7 million as working capital for mining and processing which will be invested into the project in 2020, and USD 5 million as working capital for the roasting and acid manufacture in 2024, and the working capital will be recovered with closure of the project. It is estimated that the mine closure cost is USD 3.4 million.

The CAPEX, working capital, as well as mine closure cost will be invested into the project as scheduled in Table 17-3.

Table 17-3: Summary of CAPEX Investment Schedule for the Ciemas Gold Project (1000 USD)

Year	2018	2019	2020	2021	2022	2023	2024	2025	2026	Total
Mining	7,495.3	26,510.3	3,034.0	2,354.0	1,475.0	86.0				40,954.6
Ore processing	7,636.3	11,454.4				3,320.6				22,411.3
TSF	2,768.9	4,153.4								6,922.3
Infrastructure	7,123.3	10,684.9				4,452.0				22,260.2
Roasting and acid						41,828.4				41,828.4
Other cost	4,196.9	6,295.3				4,524.9				15,017.1
Contingency	3,652.6	7,387.3	379.3	294.3	184.4	6,776.5				18,674.2
Working Capital			7,000.0				5,000.0		(12,000.0)	-
Mine closure									3,400.0	3,400.0
Total	**32,873.2**	**66,485.6**	**10,413.3**	**2,648.3**	**1,659.4**	**60,988.4**	**5,000.0**		**(8,600.0)**	**171,468.1**

Note: Totals may not sum due to rounding

17.1.2 Basis for the Estimate

Based on the materials provided by the owner, and with reference to other similar projects in Indonesia, the basis of the estimate is as below:

- Manpower cost: for the construction of the project, the manpower fee for construction is USD 35/working day;
- Material costs: the prices and source of the majority of materials for the project have been provided in Table 17-4.

Table 17-4: Prices and Sources of Majority of Materials

No.	Material	Unit	Unit Price (USD)	Source
1	Local construction materials			
1.1	Cement	t	88	Indonesia

No.	Material	Unit	Unit Price (USD)	Source
1.2	Timber	m^3	250	Indonesia
1.3	Ren brick	Piece	0.6	Indonesia
1.4	Sand	m3	25	Indonesia
1.5	Gravel	m3	25	Indonesia
1.6	Fuel	kg	0.9	Indonesia
1.7	Diesel	kg	0.8	Indonesia
1.8	Explosive	kg	1.4	Indonesia
1.9	Detonator	Piece	4.4	Indonesia
2	Steel materials (Pipe, Steel-work and steel plate etc.)			China
3	Other materials, pipes for cable and accessary, grounding for lighting, and lights etc.			China
4	Materials for thermal insulation, water proofing, anticorrosion, and painting etc.			China
5	Plates of roof, walls; doors and windows, various floor tiles, fire-proof plates and anti-static plates etc.			China
6	Supportive construction materials(such as welding electrodes)			China

- Machinery costs: the prices for rental machinery per set x shift for the project have been provided in Table 17-5.

Table 17-5: Prices of Major Machinery per Set by Shift for Construction of the Project

No.	Name	Unit	Unit Price (USD)	Remark
1	Auto-dump truck (20t)	Set.shift	150	Local rental
2	Excavator (1m^3)	Set.shift	170	Local rental
3	Crane truck（50t）	Set.shift	950	Local rental
4	Bulldozer (320HP)	Set.shift	200	Local rental

- Equipment cost: tenderers' quotations were used for major equipment, and the transportation and incidental expenses for the equipment include the expenses within China (5% of the equipment costs) and overseas (10% of the equipment costs to cover the fees from a Chinese port to the project site, and the customs duty); and the expenses for the back-up parts for the equipment is costed at 5% of the equipment costs;
- Custom duty and value added tax: according to the data provided by the owner, the overall custom duty is 0% and the value added tax is 10%;
- The costs of the development engineering, construction and installation were estimated by referring to other similar projects, and making some adjustments based on the actual manpower costs, material costs and machinery costs for the project;
- Other basis included the connecting technical conditions among various disciplines for the study and design, as well as the data provided by the owner;
- Other costs and contingency are proposed.

17.1.3 Analysis of the Capital Expenditures

The distribution of the initial capital expenditures among various usages is provided in Table 17-6 and Figure 17-2.

Table 17-6: Distribution of Initial CAPEX in Various Usages

No.	Investment Usage	CAPEX (× USD1000)	Proportion of total (%)
1	Development engineering	8,269.5	8.3%
2	Construction engineering	28,795.2	29.0%
3	Equipment procurement	30,065.9	30.3%
4	Installation Engineering	10,696.3	10.8%
5	Other costs	10,492.2	10.6%
6	Contingency	11,039.9	11.1%
	Grand Total	**99,358.9**	**100.0%**

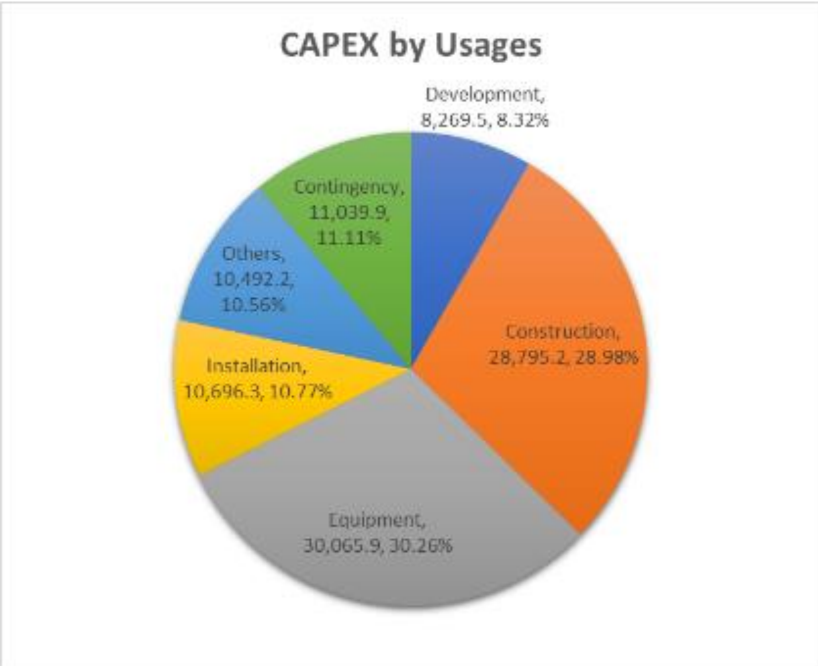


Figure 17-2: Distribution of Initial CAPEX in Cost Usages

17.1.4 Quantity of Major Engineering Work

The quantity of major engineering work has been summarized on Table 17-7.

Table 17-7: A List of Quantity of Major Engineering Work

No.	Item		Unit	Quantity
1	Development engineering		m^3	73,115
2	Preparation and cut engineering		m^3	12,121
3	Support concrete		m^3	8,953
4	Earthwork removal		m^3	947,100
5	Rock removal		m^3	405,900
6	Fill-back		m^3	466,200

No.	Item		Unit	Quantity
7	Road		m^2	103,200
8	Harden sites		m^2	30,000

17.1.5 Other Comments

- The estimate of the CAPEX does not include the fees for treatment of special foundations and side slopes;
- The estimate does not include the fluctuation of exchange rates of currencies;
- The estimate does not include inflation and price increasing factors;
- The estimate does not allow for the factors of changing laws and political policies of local governments;
- The owner provided the land-using fee, house expropriation fee and mining right fee, and the fees should be further verified.

17.1.6 Summary of Detailed CAPEX

17.1.6.1 Initial CAPEX

Table 17-8 gives the detailed budget for the initial CAPEX of the Ciemas Gold Project.

Table 17-8: Detailed Budgets for the Initial CAPEX of the Project (1000 USD)

I. Mining system

No.	Item	Development	Construction	Procurement	Installation	Other	Subtotal
A	**Mining engineering**	**8,269.5**	**2,317.0**	**13,315.0**	**3,245.0**		**27,146.5**
1	Cikadu, Sekolah and Cibatu mining sections	6,611.5	279.0	2,296.0	1,104.0		10,290.5
1.1	Ramps	1,969.4		597.0	142.0		2,708.4
1.2	Shaft for entering air	395.6			120.0		515.6
1.3	1# ventilation shaft	295.4			88.0		383.4
1.4	2# ventilation shaft	350.9			101.0		451.9
1.5	levels and stations etc.	3,115.4		145.0	198.0		3,458.4
1.6	Water drainage system	291.5		658.0	238.0		1,187.5
1.7	Mining and cutting engineering	193.3					193.3
1.8	Rooms for ventilation equipment		279.0	896.0	218.0		1,393.0
2	Pasir Manggu W mining section	1,658.0	412.0	1,821.0	712.0		4,603.0
2.1	Decline	255.2	321.0	920.0	237.0		1,733.2
2.2	3# ventilation shaft	57.4					57.4
2.3	4# ventilation shaft	48.0			60.0		108.0

No.	Item	Development	Construction	Procurement	Installation	Other	Subtotal
2.4	levels and stations etc.	543.9		226.0	256.0		1,025.9
2.5	Water drainage system	206.0		337.0	88.0		631.0
2.6	Mining and cutting engineering	547.5					547.5
2.7	Rooms for ventilation equipment		91.0	340.0	71.0		502.0
3	Mining equipment			5,376.0	205.0		5,581.0
4	Back-filling plant		1,442.0	2,877.0	725.0		5,044.0
5	Air compressor station		184.0	945.0	499.0		1,628.0
B	**Mining industrial sites**		**3,073.4**	**594.4**	**200.2**		**3,868.0**
1	Fuel station		34.2	171.1	14.4		219.8
2	Automobile repairing workshop		488.0	280.9	98.8		867.6
3	Workshop of maintenance and repairing for locomotives		512.4	142.4	78.0		732.7
4	General storage rooms		180.2		9.0		189.2
5	Waste dumping sites		1858.7				1,858.7
C	**Public engineering and buildings**			**706.4**	**2,284.9**		**2,991.3**
1	Overhead powerlines in mining sites				1,388.7		1,388.7
2	Communication and instrument control system			706.4	241.3		947.7
3	Pipe network				654.8		654.8
	Total Engineering Cost	**8,269.5**	**5,390.4**	**14,615.8**	**5,730.1**		**34,005.8**

Note: Totals may not sum due to rounding

II. Ore processing system

No.	Item	Development	Construction	Procurement	Installation	Other	Subtotal
A	**Ore processing Engineering**		**3,920.2**	**9,596.4**	**2,776.4**		**16,292.9**
1	Coarse crushing workshop		355.2	484.8	166.7		1,006.7
2	Intermediate storage of ore		375.8	135.0	84.7		595.4
3	Mining and flotation workshop		1,846.2	4,887.8	1,466.3		8,200.3
4	Concentrate de-watering work shop		138.5	934.0	140.1		1,212.6
5	Carbon-cyanidation workshop		315.2	1,534.5	460.3		2,310.0
6	Gold extraction workshop		840.4	1,154.3	346.3		2,340.9

7	Air compressor station		48.9	466.0	112.0		626.9
B	**Supportive production facilities**		**688.7**	**350.4**	**124.0**		**1,163.1**
1	Automobile repairing workshop		340.0	121.7	12.2		473.8
2	Front water returning system		132.6	82.1	35.1		249.8
3	Cooling of recycling water		13.9	134.6	46.7		195.2
4	General storage rooms		202.3	12.0	30.0		244.3
C	**Public engineering facilities**			**907.7**	**727.0**		**1,634.7**
1	20kV power cable and lines				545.4		545.4
2	Communication and instrument control system			907.7	181.5		1,089.2
	Total Engineering cost		**4,608.9**	**10,854.5**	**3,627.3**		**19,090.7**

Note: Totals may not sum due to rounding

III. Tailings storage facility

No.	Item	Development	Construction	Procurement	Installation	Other	Subtotal
A	**Tailings storage facility**		**6,292.7**	**423.2**	**206.4**		**6,922.3**
1	Tailings dam		1,586.5				1,586.5
2	Flood drainage system		492.3				492.3
3	Seepage-proofing system		2,787.8				2,787.8
4	Clearing of tailings pond area		364.8				364.8
5	Arris body for drainage		814.6				814.6
6	Drainage ditch on the dam abutment		17.5				17.5
7	Tailings transportation system			99.3	79.4		178.7
8	Water returning system for TSF		229.2	324.0	127.0		680.2

Note: Totals may not sum due to rounding

IV. Infrastructures

No.	Item	Development	Construction	Procurement	Installation	Other	Subtotal
A	**Infrastructures**		**15,629.0**	**5,215.5**	**1,415.7**		**22,260.2**
1	Main office building		190.0	12.9	29.6		232.6
2	Shower rooms		83.5		20.3		103.8
3	Dinning rooms		70.4	6.2	12.6		89.2
4	Dormitory		675.9	49.2	116.8		841.8
5	Back-up disel generators		131.8	1,753.3	406.2		2,291.2

6	Fresh production water supplying system		177.3	0.0	402.3		579.6
7	Living water supply and drainage systems		21.0	502.1	124.4		647.5
8	Communication and instrument control system		0.0	1,200.9	261.7		1,462.6
9	General latout and transportation engineering		14,030.6	1,648.9			15,679.6
9.1	Earth and rock engineering		8,908.3				8,908.3
9.2	Hardening of roads and sites		4,107.7				4,107.7
9.3	Gates and walls		260.9				260.9
9.4	Equipment for general layout and transportation			1,648.9			1,648.9
9.5	Narrow gauge railway for mining		21.8				21.8
9.6	Block walls for ore dressing and smelter		731.9				731.9
10	Explosive storage facility		248.4	42.2	41.6		332.2
	Initial infrastructure (80%)		**12,503.2**	**4,172.4**	**1,132.5**		**17,808.2**

Note: Totals may not sum due to rounding

V. Other costs

No.	Item	Development	Construction	Procurement	Installation	Other	Subtotal
A	**Other fees**						
1	Owner's costs (including items 1.1 to 1.9)					4,000.0	4,000.0
1.1	Management fee of owners						
1.1	Accounting fees						
1.2	Permitting fees						
1.3	Land-using fees (including re-location fee)						
1.4	Fees for mining right						
1.5	Fees for environmental protection and security						
1.6	Legal consulting fees						
1.7	Fees for connecting water, electricity and communication						
1.8	Fees for external roads						
1.9	Fees for temporary` facilities						

2	Safety protection and security fee on-site					324.0	324.0
3	Fees for feasibility study					294.2	294.2
4	Fees for tests and researches					500.0	500.0
5	Fees for geotechnical exploration					233.5	233.5
6	Fees for engineering designs					2723.9	2723.9
7	Maintenance and repairing fees for mine tunnels					778.3	778.3
8	Trial production fees					480.1	480.1
9	Training fee for workers and fees for entering the mine early					660.0	660.0
10	Fees of office and living furniture					198.0	198.0
11	Fees of production related tools, meters and furniture					300.1	300.1
	Total					**10492.2**	**10492.2**

Note: Totals may not sum due to rounding

VI. Contingency

No.	Item	Development	Construction	Procurement	Installation	Other	Subtotal
A	**Contingency**					**11,039.9**	**11,039.9**

17.1.6.2 Sustaining CAPEX

The capital investment after the mining and processing production of the project in 2020 will be considered as the sustaining CAPEX, and maining include the CAPEX for further development in mining, additional facility in processing and the roasting and associated acid manufacture facility and other supportive facilities. Table 17-9 gives the summarized budget for the sustaining CAPEX of the Ciemas Gold Project. Table 17-10 lists the details about the sustaining CAPEX.

Table 17-9: Summarized Budgets for the Sustaining CAPEX of the Project (1000USD)

Cost Centres	Development	Construction	Procurement	Installation	Other	Subtotal
Mining	7,269.1					7,269.1
Ore processing		628.1	2,151.9	540.6		3,320.6
Roasting		11,838.3	20,927.4	9,062.7		41,828.4
Infrastructure		3,125.8	1,043.1	283.1		4,452.0
Total Engineering	**7,269.1**	**15,592.3**	**24,122.4**	**9,886.4**		**56,870.1**
Other costs					4,524.9	4,524.9
Engineering+Other	**7,269.1**	**15,592.3**	**24,122.4**	**9,886.4**	**4,524.9**	**61,395.0**
Contingency					7,674.4	7,674.4
Total Sustaining Capital	**7,269.1**	**15,592.3**	**24,122.4**	**9,886.4**	**12,199.3**	**69,069.4**

Note: Totals may not sum due to rounding

Table 17-10: Detailed Budgets for the Sustaining CAPEX of the Project

I. Mining (1000 USD)

Engineering	Total	2020	2021	2022	2023	2024
CKD_Decline	1,042.4	118.9	855.7	67.8	-	
SEK_Decline	538.2	538.2	-	-	-	
SEK_Incline	200.6	200.6	-	-	-	
CBT_Access_420	124.5	124.5	-	-	-	
Level Drive	4,697.8	1,103.5	1,198.4	1,198.4	1,164.4	33.1
Temporary Incline_490	128.4	128.4	-	-	-	
#4 Vent. Shaft	24.7	24.7	-	-	-	
Level Drift	512.5	512.5	-	-	-	
Total	7,269.1	2,751.3	2,054.1	1,266.2	1,164.4	33.1

Note: Totals may not sum due to rounding

II. Ore processing plant

No.	Item	Development	Construction	Procurement	Installation	Other	Subtotal
A	Engineering		**628.1**	**2,151.9**	**540.6**		**3,320.6**
1	Lime curing workshop		153.1	359.4	76.0		588.5
2	Re-milling of baking residue workshop		285.9	871.9	188.3		1,346.0
3	Workshop of carbon cyanidization of baking residues		189.1	920.7	276.2		1,386.0

Note: Totals may not sum due to rounding

III. Roasting and acid manufacture

No.	Item	Development	Construction	Procurement	Installation	Other	Subtotal
A	**Engineering**		**9,654.6**	**18,120.3**	**6,976.7**		**34,751.6**
1	Prime material supply system		1,343.9	381.2	111.1		1,836.2
2	Baking process		2,814.7	9,172.0	2,328.5		14,315.2
3	Boiling for residual heat		233.3	1,067.6	323.8		1,624.7
4	Acid manufacture system		3,376.5	5,504.1	3,276.0		12,156.6
4.1	*Purification process*		622.2	1,475.2	732.3		2,829.7
4.2	*Dry absorption process*		1,112.7	1,102.8	623.0		2,838.6
4.3	*Transforming process*		368.0	1,661.3	1,258.9		3,288.2
4.4	*Storage and load facilities for sulphate*		1,112.0	1,144.9	611.8		2,868.6
4.5	*De-sulphuration from waste gas of sulphate*		161.6	119.9	50.0		331.5
5	Water recycling system		794.6	540.9	185.8		1,521.3
6	Treatment systems for waste acid and water		1,055.2	1,153.5	601.9		2,810.6

6.1	*Waste water treatment system*		514.1	670.1	308.1		1,492.3
6.2	*Treatment process of waste acid*		541.1	483.4	293.8		1,318.3
7	Air compressor station		36.4	301.0	149.5		486.9
B	**Supportive Production facility**		**1,210.5**	**1,793.0**	**624.6**		**3,628.2**
1	Main building for sulphate manufacture		398.6	756.1	410.7	-	1,565.4
2	Electricity generator by using residual heat		113.3	1,036.9	214.0	-	1,364.2
3	Storage of coarse arsenic		446.0	-	-	-	446.0
4	Storage of baking residues		252.5	-	-	-	252.5
C	**Public facility**		**973.2**	**1,014.1**	**1,461.3**	**-**	**3,448.6**
1	Communication and instrument control system		-	1,014.1	317.2	-	1,331.3
2	Pipe network		973.2	-	1,144.1	-	2,117.3
	Total Engineering cost		**11,838.3**	**20,927.4**	**9,062.7**	**-**	**41,828.4**

Note: Totals may not sum due to rounding

IV. Infrastructure

No.	Item	Development	Construction	Procurement	Installation	Other	Subtotal
	Sustaining infrastructure		3,125.8	1,043.1	283.1		4,452.0

V. Other cost

No.	Item	Development	Construction	Procurement	Installation	Other	Subtotal
A	**Other fees**						
1	Owner's costs (including items 1.1 to 1.9)					1,000.0	1,000.0
1.1	Management fee of owners						
1.1	Accounting fees						
1.2	Permitting fees						
1.3	Land uusing fees (including re-location fee)						
1.4	Fees for mining right						
1.5	Fees for environmental protection and security						
1.6	Legal consulting fees						
1.7	Fees for connecting water, electricity and communication						

1.8	Fees for external roads						
1.9	Fees for temperary facilities						
2	Safety protection and security fee on-site						
3	Fees for feasibility study						
4	Fees for tests and researches						
5	Fees for geotechnical exploration					284.4	284.4
6	Fees for engineering designs					1,990.5	1,990.5
7	Maintenance and repairing fees for mine tunnels					568.7	568.7
8	Trial production fees					419.3	419.3
9	Training fee for workers and fees for entering the mine early						
10	Fees of office and living furniture						
11	Fees of production related tools, meters and furniture					262.1	262.1
	Total					**4,524.9**	**4,524.9**

Note: Totals may not sum due to rounding

VI. Contingency

No.	Item	Development	Construction	Procurement	Installation	Other	Subtotal
A	**Contingency**					**7,674.4**	**7,674.4**

17.1.7 Cost for Mine Closure

SRK estimated that the CAPEX for mine closures will be USD3.4 million, which is considered in the owner's cost. A total of 175 ha disturbed area will be rehabilitated with a unit rehabilitation cost of USD 8,000 per ha, and the unit cost is based on the Indonesian industrial practice. Therefore, USD 1.4million is estimated for the rehabilitation of disturbed area only. In addition, USD 2.0 million is estimated for the buildings and facilities demolition cost. SRK considers that USD 3.4 million is a reasonable estimate of cost for mine closure.

17.1.8 Residue Value

Since the project will have a LOM of 7 years in the FS study, and it is proposed that the depreciation of the fixed assets will be at 10%, and amortization of the intangible assets at 10% for 10 years, the residual value of the CAPEX for the mining and ore processing will be determined by the depreciation and amortization schedules of the CAPEX assets for mining and processing. The Roasting and Acid Manufacture facilities will be put into production in years 5 of mining production, their residue value will be determined accordingly. Some funds have been allocated for further exploration as the intangible CAPEX for the project. It is SRK's opinion that upon the success of the further exploration programs, the LOM should be extended.

17.1.9 Investment Plan

Considering that the construction of the mining and processing facilities, and associated supportive facilities and infrastructure will start on July 1, 2018, and the construction of the Roasting and Acid Manufacture facilities will start on January 1, 2023, SRK has summarized the investment schedule as in Table 17-11.

Table 17-11: Investment Plan for Ciemas Gold Project (1000 USD)

Year	2018	2019	2020	2021	2022	2023	2024	2025	2026	Total
Mining	7,495.3	26,510.3	3,034.0	2,354.0	1,475.0	86.0				40,954.6
Ore processing	7,636.3	11,454.4				3,320.6				22,411.3
TSF	2,768.9	4,153.4								6,922.3
Infrastructure	7,123.3	10,684.9				4,452.0				22,260.2
Roasting and acid						41,828.4				41,828.4
Other cost	4,196.9	6,295.3				4,524.9				15,017.1
Contingency	3,652.6	7,387.3	379.3	294.3	184.4	6,776.5				18,674.2
Working Capital			7,000.0				5,000.0		(12,000.0)	-
Mine closure									3,400.0	3,400.0
Total	**32,873.2**	**66,485.6**	**10,413.3**	**2,648.3**	**1,659.4**	**60,988.4**	**5,000.0**		**(8,600.0)**	**171,468.1**

Note: Totals may not sum due to rounding

17.1.10 Estimate Accuracy

For the budgeted capital expenditures of the Ciemas Gold project, SRK/NERIN consulted with various manufacturers and construction companies, as well as Wilton, the owner for the prices of equipment and rates for development, and used some database costs, and estimated costs, such as benchmarks and comparable costs from other similar projects. The following tables show the accuracy of the CAPEX estimates. In general, the CAPEX for development and equipment is deemed as the Budget Quotation accuracy, the CAPEX for construction and installation for the major engineering, as well as the Others cost is the Database Cost, and the CAPEX for construction and installation for the supportive and public engineering, and infrastructure is deemed as the Estimate Cost. The sustaining CAPEX for construction and installation is deemed as Estimated Cost, except of those for ore processing plant. From the tables, SRK considers that the accuracies for both initial and sustaining CAPEX are greater than 85%.

Table 17-12: Overall Estimate Accuracy for the Initial CAPEX of the Project

Item	Amount (1000 USD)	Accuracy	% of Total
Budget Quotes	38,335.4	95%	43.4%
Database costs	29,249.7	85%	33.1%
Estimated costs	20,733.9	75%	23.5%
Total/Average	**88,319.0**	**86.99%**	**100.0%**

- Contingency was not included

Table 17-13: Detailed Estimate Accuracy for the Initial CAPEX of the Project

Item	Amount (1000 USD)	Accuracy	% of Total
Mining engineering	27,146.5	92.95%	30.74%
Mining industrial sites	3,868.0	78.07%	4.38%
Mining Public engineering and buildings	2,991.3	79.72%	3.39%

Overall Mining	**34,005.8**	**90.10%**	**38.50%**
Ore processing Engineering	16,292.9	90.89%	18.45%
Processing Supportive production facilities	1,163.1	81.03%	1.32%
Processing Public engineering facilities	1,634.7	86.11%	1.85%
Overall Ore processing	**19,090.7**	**89.88%**	**21.62%**
Tailings storage facility	**6,922.3**	**85.61%**	**7.84%**
Initial infrastructure	**17,808.2**	**79.69%**	**20.16%**
Others	**10,492.2**	**85.00%**	**11.88%**
Subtotal	**88,319.0**	**86.99%**	**100.00%**

- Contingency was not included

Table 17-14: Overall Estimate Accuracy for the Sustaining CAPEX of the Project

Item	Amount (1000 USD)	Accuracy	% of Total
Budget Quotation	31,391.5	95%	51.13%
Database costs	5,693.6	85%	9.27%
Estimate costs	24,309.8	75%	39.60%
Total/Average	**61,395.0**	**86.15%**	**100.00%**

- Contingency was not included

Table 17-15: Detailed Estimate Accuracy for the Sustaining CAPEX of the Project (1000 USD)

Item	Amount (1000 USD)	Accuracy	% of Total
Mining	7,269.1	95.00%	11.84%
Ore processing	3,320.6	91.48%	5.41%
Roasting overall	41,828.3	85.01%	68.13%
Roasting major	34,751.6	85.43%	56.60%
Roasting supportive	3,628.2	84.88%	5.91%
Roasting Public	3,448.6	80.88%	5.62%
Infrastructures	4,452.0	79.69%	7.25%
Others	4,524.9	85.00%	7.37%
Total/Average	**61,395.0**	**86.15%**	**100.00%**

- Contingency was not included

17.2 Operating Expenses (OPEX)

17.2.1 Summary

It is estimated that the annual direct operating expenses in the full production years will be USD 14.45 million for mining, USD 16.84 million for ore processing and smelter, USD 2.61 million for roasting and acid manufacturing, USD 0.392 million for public supportive workshops, USD 8.2 million for G & A. Table 17-16 gives details.

Table 17-16: Average Direct Operating Expenses in Full Production Years

Cost centre	Total Cost (1000USD)	Unit cost (USD/t ore)
Mining	14,449	29.19
Processing and smelter	16,845	34.03
Roasting and acid manufacturing	2,611	5.27
Supportive infrastructure	392	0.79

General and administration	8,206	16.58
Total	**42,503**	**85.86**

It is proposed that the roasting and acid manufacture facility will be constructed later in 2023. From 2020 to 2023, the products of the project will be gold concentrates from flotation and gold dore from the leaching of flotation tailings. In the period, the operating expenses have been given in Table 17-17. The average operating cost is US$69.26/t ore.

Table 17-17: Average Direct Operating Cost in 2020-2023 (assuming full production)

Cost centre	Total Cost (1000 USD)	Unit cost (USD/t)
Mining	14,449	29.19
Processing and smelter	12,509	25.27
Supportive infrastructure	392	0.79
General and administration	6,935	14.01
Total	**34,285**	**69.26**

17.2.2 OPEX for Mining

Table 17-18 lists the direct operating costs of mining during full production years, and Table 17-19 and Table 17-20 list the details of pure mining and development direct costs.

Table 17-18: Average Direct Operating Expenses of Mining in Full Production Years

No.	Cost centre	Unit	CSC Complex	PSM Section	LOM Total (M USD)	LOM AVG (USD)
1	Mining Total	USD/t ROM	18.25	18.25	41.82	18.25
	Supportive material	USD/t ROM	6.89	6.89		
	Fuel	USD/t ROM	2.75	2.75		
	Power	USD/t ROM	2.50	2.50		
	Payroll	USD/t ROM	5.67	5.67		
	Water	USD/t ROM	0.43	0.43		
2	Stope development total	USD/t ROM	12.72	1.29	29.69	10.94
	Supportive material	USD/t ROM	4.30	0.44		
	Fuel	USD/t ROM	0.94	0.10		
	Power	USD/t ROM	2.43	0.25		
	Payroll	USD/t ROM	5.05	0.51		
3	**Total**	**USD/t ROM**	30.97	19.54	79.20	29.19

Table 17-19: Average Direct Operating Expenses for Pure Mining in ROM

No.	Item	Unit	Unit Price (USD)	Unit consumption (/t. ROM)	Unit Cost(USD/t. ROM)	Total consumption	Total Cost (k USD)
	ROM	kt				495	
1	Supportive materials				6.89		3,413
	Explosives	kg	1.30	0.62	0.81	306,900	399
	Detonating tube	m	0.20	1.31	0.26	648,450	130
	Non-electric detonator	Piece	0.30	0.44	0.13	217,800	65
	Bore bit	head	4.30	0.02	0.08	9,009	39

	Drill rod	piece	11.00	0.01	0.07	3,020	33
	Rebar	kg	1.00	0.65	0.65	321,750	322
	Anchor bolt	piece	3.00	0.01	0.03	4,950	15
	Cement	kg	0.05	65.00	3.25	32,175,000	1,609
	Tire	piece	600.00	0.00	0.24	198	119
	Others				1.38		683
2	Fuel				2.75		1,266
	Diesel	l	1.00	2.75	2.75	13,611	1,266
3	Power				2.93		1,451
	Electricity	kwh	0.1		2.50	12,393	1,239
	Water	m^3	1.00	0.43	0.43	211,930	212
4	Payroll	1000UDS/person/a	8.00		5.67	351	2,808
5	**Total**				**18.25**		**8,937**

Table 17-20: Average Direct Operating Expenses for Development in Full Production Years (per volume)

Item	Unit	Unit Price (USD)	Unit consumption (m^3)	Unit Cost (USD/m^3)
Stope development	m^3			
Supportive materials				21.45
Explosives	kg	1.30	4.43	5.75
Detonating tube	m	0.20	17.59	3.52
Non-electric detonator	piece	0.30	5.86	1.76
Bore bit	head	4.30	0.24	1.05
Drill rod	piece	11.00	0.08	0.90
Anchor bolt	piece	3.00	0.15	0.44
Cement	kg	0.05	5.28	0.26
Tire	piece	600.00	0.003	1.76
Others		1.00	6.01	6.01
Diesel	kg	1.00	4.69	4.69
Electricity	kWh	0.10	121.55	12.15
Payroll	1000UDS/person/a	8.00	3.15	25.21
Total			-	63.50

17.2.3 OPEX for Roasting and Acid Manufacturing

Table 17-21 gives detailed operating costs for 65 kta sulfuric acid/a. The unit cost is also per tonne acid, so a conversion should be done to convert the unit cost to per tonne ore.

Table 17-21: Average Direct Operating Expenses for Roasting and Acid Manufacturing in Full Production Years

No.	**Item**	**Unit**	**Unit Price (USD)**	**Unit consumption (/t. acid)**	**Unit Cost (USD/t.acide)**	**Total consumption**	**Total Cost (x1000USD)**
	Sulphur acid production	t				65,000	
1	Supportive materials				3.27		212.55
	Vanadium catalyst	m^3	5091.35		0.76	9.7	49.4
	Sodium sulphide (60%)	t	398.1		1.13	184.5	73.45

	Sodium hydroxide (100%)	t	500		0.32	41.6	20.8
	Hydrochloric acid (100%)	t	425.8		0.15	22.9	9.75
	Limestone	t	39.22		0.54	895.0	35.1
	Cloth bag				0.02		1.3
	Others				0.33		21.45
2	Fuel				0.27		17.55
	Light diesel	t	1073.62		0.27	16.3	17.55
3	Power				25.06		1628.9
	Electricity	k.kWh	100		31.83	20689.5	2068.95
	Minus electricity generated from residual heat	k.kwh	100		-12	-7800.0	-780
	Fresh water	m^3	1		5.23	339950.0	339.95
4	Employees' salaries	1000UDS/ person/a	8		11.57	94.0	752.0
5	Total (acid)				40.17		2611.0

17.2.4 OPEX for Ore Processing and Smelter

Table 17-22 lists the direct operating costs related to ore processing and smelter in the full production years. Please note that the tonnage for crushing and carbon cyaniding of roasting residues is 132 kt/a rather than the 495kt/a of ROM.

Table 17-22: Average Direct Operating Expenses for Ore Processing and Smelter in Full Production Years

No.	Item	Unit	Unit Price (USD)	Unit consumption (/t. ore)	Unit Cost (USD/t. ore)	Total consumption	Total Cost (x1000USD)
	Ore processed	kt				495.00	
1	Supportive materials				24.32		12034.79
1.1	Ore processing, carbon cyaniding of tailings etc.						
	Steel ball	kg	1.00	1.00	1.00	495.00	495.00
	Liner plate	kg	1	0.35	0.35	173.25	173.25
	Tubber belt	m^2	40	0.0015	0.06	742.50	29.70
	Screen cloth	kg	1	0.01	0.01	4.95	4.95
	Engine oil	kg	2	0.03	0.06	14.85	29.70
	Yellow oil	kg	2	0.07	0.14	34.65	69.30
	Impeller cover plate	kg	1.1	0.3	0.33	148.50	163.35
	Filter cloth	m^2	20	0.005	0.10	2475.00	49.50
	PAX	kg	2.15	0.25	0.54	123.75	265.72
	MIBC	kg	4.00	0.02	0.08	9.90	39.60
	Copper sulfate	kg	1.90	0.50	0.95	247.50	470.25
	Lime	kg	0.04	2	0.08	990.00	38.82
	NaCN	kg	4.50	2	9.00	990.00	4455.00
	NaOH	kg	0.50	0.05	0.03	24.75	12.38
	Active carbon	kg	1.95	0.2	0.39	99.00	193.05
	Hydrochloric acid	kg	0.43	0.02	0.01	9.90	4.22
	Cloth bag	m^2	50.67	0.003	0.14	1400.00	70.94
1.2	Water supplying and drainage system						
	Copper sulfate	kg	1.90	0.12	0.22	57.09	108.47
	Lime	kg	0.04	4.00	0.16	1980.00	77.65
	SMBS	kg	2	1.85	3.71	917.40	1834.80
	NaHS	kg	0.50	0.02	0.01	12.05	6.02
	FeSO4	kg	0.2	4.00	0.80	1980.00	396.00
1.3	Crushing and carbon cyaniding of baking residues etc.						
	Steel ball	kg	1.00	1.00	0.267	132.00	132.00
	Liner plate	kg	1.00	0.25	0.067	33.00	33.00
	Tubber belt	m^2	40.00	0.00	0.016	198.00	7.92
	Engine oil	kg	2.00	0.02	0.011	2.64	5.28
	Yellow oil	kg	2.00	0.05	0.027	6.60	13.20
	Impeller cover plate	kg	1.10	0.20	0.059	26.40	29.04
	Lime	kg	0.04	2.00	0.021	264.00	10.35
	Sodium cyanide	kg	4.50	2.00	2.400	264.00	1188.00
	NaOH	kg	0.50	0.20	0.027	26.40	13.20

	Active carbon	kg	1.95	0.05	0.026	6.60	12.87
	Hydrochloric acid	kg	0.43	0.02	0.002	2.64	1.12
	Others				3.23		1601.13
2	Power				7.83		3874.20
	Electricity	k.kWh	100.00	7.30	7.30	36148.38	3614.84
	Water	m^3	1.00	0.37	0.52	259366.14	259.37
3	Employee's salaries	1000UDS/ person/a	8.00		1.89	117	936.00
4	Total				34.03		16844.99

17.2.5 OPEX for Supportive Production Workshops

Table 17-23 gives the details about OPEX for the supportive production workshop.

Table 17-23: Average Direct Operating Expenses for Supportive Production Workshop in Full Production Years

No.	Item	Unit	Unit Price (USD)	Unit consumption (/t. ore)	Unit Cost (USD/t.ore)	Total consumption	Total Cost (x1000USD)
	Processing ore	kt				495	
1	Fuel				0.40		198.00
	Diesel	l	1.00			198000.00	198.00
2	Power				0.13		66.00
	Water	m^3	1.00		0.13	66000.00	66.00
					0.00		
3	Employee's salaries	1000UDS/person/a	8.00		0.26	16	128.00
4	Total				0.79		392.00

17.2.6 OPEX for G&A

Table 17-24 gives the details about OPEX for the general and administration.

Table 17-24: Average Direct Operating Expenses for G&A in Full Production Years

Items	Total Cost (USD)	USD/t ore	Note
General Management	200,000	0.40	International and domestic traveling cost to the mine site
Training during operation	334,400	0.68	5% of the salary for the all full time staff
Business Sustainability	800,000	1.62	Community investment
The environmental and permitting	150,000	0.30	
Human resources	200,000	0.40	
Health & Safety	200,000	0.40	
Security	1,000,000	2.02	
Technology	150,000	0.30	
Supply Chain and Sales	2,000,000	4.04	
Site Administration (catering and salary)	3,171,300	6.41	catering of USD 10/person, salary of general management team, maintenance, and PPE
Total	**8,205,700**	**16.58**	

G&A costs for the operations phase of the project were established from current knowledge of the site costs and the proposed operations structure. Costs were estimated by area for business sustainability; finance; environmental & permitting; human resources; training; health & safety; security; technology; supply chain; site administration; and general management and cover the Jakarta office, Pelabuhan Ratu office, and site office.

General Management costs include the overall travel budgets for all teams, and office administration for Jakarta office, Pelabuhan Ratu office, and site office which are shared by the teams. Land easement costs and costs allocated to expatriates are also included in G&A under general management.

International and domestic travel is built up starting with on-site personnel and incorporates the rotation schedules. Quotations for international and domestic flights will be acquired through the travel agencies. Ground transportation costs are built in the same way using local vendors for pricing quotations.

Training during operation
The training budget covers training allocated for staff in addition to internships for the G&A, mining, process and surface infrastructure departments. Training was estimated as a percentage of total annual salary. The training and internship costs are captured in the HR area.

Business Sustainability
Business sustainability focuses on developing and maintaining relationships with the local community and government via community investment, donations, hosted events and publications.

The community investment will be to benefit stakeholders in the various government bodies and local residents. The business sustainability group is primarily located in Pelabuhan Ratu, Jakarta, or project site.

The environmental and permitting group will manage all environmental requirements including monitoring, reporting, permits, government relations related to environmental, and protection and rehabilitation of the flora and fauna on the project site. The budget includes the following areas:
- Hydrological and meteorological stations maintenance
- Environmental management plan implementation, monitoring and reporting
- Water monitoring
- Rehabilitation maintenance
- Waste management
- Environmental permitting

Staff for environmental and permitting will be located in Pelabuhan Ratu, Jakarta, or project site.

Human resources focus on the management of the labour force including payroll, recruitment, termination, company events, labour law implementation and expatriate integration. The staff for human resources are located in Pelabuhan Ratu, Jakarta, or project site.

Health & Safety
This area primarily focuses on the site-implemented program for health and safety. Operating costs for this area include safety audits, seminars, safety training, and safety and medical supplies. Specific safety items such as mine rescue equipment and supplies are captured in the mining operating costs.

Security
Operating costs for this area include the security contractor and the security superintendent employee costs who will manage the security contractor.

Technology

The G&A technology budget includes the servers, software and networking infrastructure for Pelabuhan Ratu, Jakarta, or project site. IT costs cover only what is in country and related to the projects operations. Exploration and geology servers are not included in the project operating costs for Technology.

The costs are estimated for providing services for personal devices such as laptop computers, cell phones and software. Each general area: G&A, mining, process, and surface operation will carry the costs for the devices for their staff. IT services will be managed from Jakarta with remote support, in addition to site support

Supply and sales Chain

The supply chain team will plan and manage the procurement, logistics and importation of goods. The supply chain team will be based in Pelabuhan Ratu, Jakata, or project site. The main warehouse will be located on the project site.

Site Administration

Site administration will provide support to the camp organization, maintenance and catering. The site administration staff will be based on site. The cost of catering for the staff is based on the on-site man-days calculated from the estimated operations labour force. PPE and uniforms must be renewed annually under the law, the cost for this (estimated per employee) is included in this area because the supply will be managed by site administration.

18 Economic Analysis

An economic analysis for Ciemas Gold Project is carried out using a discount cash flow approach on an after-tax basis. The internal rate of return (IRR) on total investment was calculated based on 100% equity financing. The Net Present Value (NPV) was calculated from the cash flow generated by the project based on a discount rate of 9.0%. The payback period based on the undiscounted annual cash flow of the project was also indicated as a financial measure. A sensitivity analysis was also performed for the after-tax base case to assess the impact of variations of the project capital costs, operating costs and price of gold.

The economic analysis presented in this section contains forward-looking information with regards to the ore reserve estimates, commodity prices, exchange rates, proposed mine production plan, projected recovery rates and processing costs, infrastructure construction costs and schedule. The results of the economic analysis and are subject to numerous known and unknown risks, uncertainties and other factors that may cause actual results to differ materially from those presented here.

The economic analysis is based on the estimates for ore reserves only.

18.1 Assumptions and Basis

The Economic Analysis was performed using the following assumptions and basis:

- The Project Executive Schedule developed in this Study, considering key project milestones;
- The Financial Analysis is performed for the entire Ore Reserves estimated in this Study;
- Commercial pilot production is scheduled to begin in the fourth quarter (Q4) of 2019. The first full year of production is therefore 2020. One and a half years are for construction that is, from Q3 2018 to the end of 2019. Operations are estimated to span a period of approximately 7 years, which includes a last 2 years capacity ramp-down;
- Real term United States Dollar (USD) values of Q2 2018 are used in the model and SRK considers the relative value as a constant over the LOM, i.e., no inflation or escalation factors are taken into account;
- Capital investments will be depreciated and/or amortized evenly over a 10 year period. The residual value will be recovered in the last year of the mine. The working capital will be fully reclaimed in the last year;
- All gold and/or concentrate is sold in the same year it is produced; and
- The transportation cost is assumed to be covered by the buyer and the cost is estimated ending as the mine gate.

The general assumptions and parameters used for this Study are summarized in Table 18-1 below.

Table 18-1: DCF Model Criteria

Description	Unit	Value
Volume		
Ore Reserve	kt	3,260
Au Grade	g/t	7.7
Processing Recovery Rate for Cons.	%	81.5
Processing Recovery Rate for Tailing	%	18.5
Conc. Roasting-CIL Recovery Rate (Incl. smelter)	%	90.8
Tailings CIL Recovery Rate（Incl. smelter）	%	68.7
Recovery Rate for Cons. Roast-CIL	%	74.0
Recovery Rate for Tailing CIL	%	12.7

Final Recovery Rate for ROM to Gold	%	86.7
Capacity and Schedule		
Mine Capacity	ktpa	from 400 to 511
Mill Capacity	ktpa	495
Mine Life	year	7
Construction	year	1.5
Mine Economy		
Initial Capital	USD M	99.4
Sustaining Capital	USD M	68.7
Mine Closure	USD M	3.4
Working Capital	USD M	12.0
U/G Mining Cost	USD/t ROM	29.2
O/C Mining Cost	USD/t ROM	4.8
O/C Stripping Cost	USD/t rock	1.5
Processing and smelter Cost (Excluding Roasting)	USD/t ROM	25.3
Processing and smelter Cost (Including Roasting)	USD/t ROM	34.0
Roasting Cost	USD/t ROM	5.3
Supportive infrastructure Cost	USD/t ROM	0.8
G & A Cost Excl Roasting	USD/t ROM	14.0
G & A Cost Incl Roasting	USD/t ROM	16.6
Payable Rate for Cons. (before Roasting)	%	95
Refine Charge for Cons. (before Roasting)	USD/t Cons.	190
Royalty to Gold Revenue	%	3.75
Enterprise Income Tax	%	25

18.2 Gold Prices and Revenue

The consensus market forecast ("CMF") as presented herein, is derived from the median of analysts' forecasts and is presented in real terms. The base information used to derive the CMF is sourced from the Energy and Metals Consensus Forecast, published by Consensus Economics Inc. The CMF gold prices release in Q2 2018 (20-Apr-2018) are presented in Table 18-2.

Table 18-2: Consensus Gold Price Forecast

Commodity	Unit	SPOT price (14-Apr-2018)	2018	2019	2020	2021	2022	2023	2024	LTP
Gold	USD/oz	1,349	1,300	1,300	1,270	1,250	1,240	1,220	1,220	1,220
	USD/g	43.37	41.80	41.80	40.83	40.19	39.87	39.22	39.22	39.22

From the consensus forecasts it can be seen that gold prices are expected to decrease over the coming years from a forecast of USD1,300 in 2019 to a Long-Term Price of USD1,220 in 2023, a decrease of 2.15%.

The products of Ciemas Project are gold dore and gold concentrate from year 2019 to 2021. The payable rate is assumed to be 95% and the refine charge is USD 190 per tonne concentrate. The sulphuric acid (98%) is assumed a price of USD 60 per tonne. The product of As_2O_3 is assumed could not be sold and a cost of USD 5 per tonne for storage. The annual revenue over LOM is presented in Figure 18-1 below. The LOM revenue forecast is as shown in Table 18-3 below.

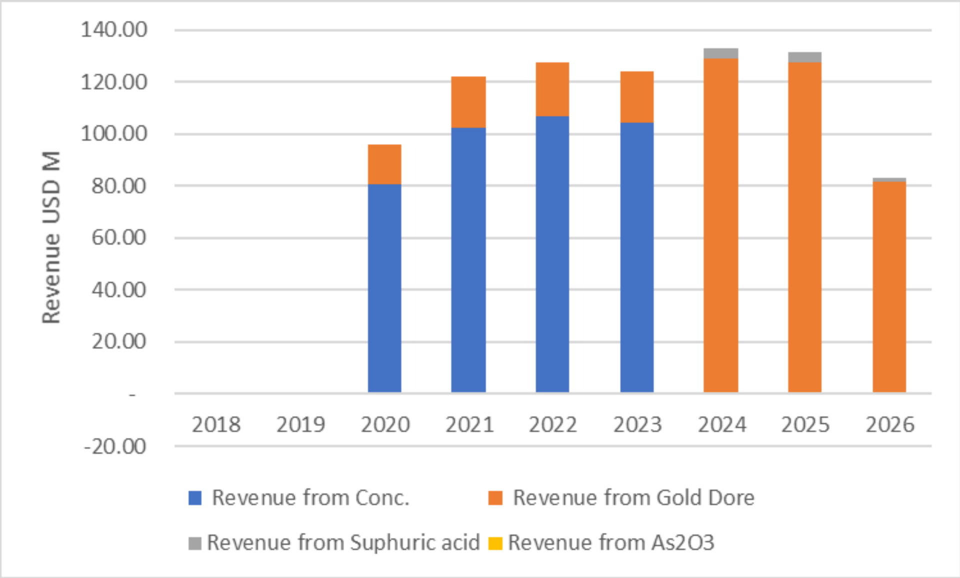


Figure 18-1: Annual Revenue by Products

Table 18-3: LOM Revenue Forecast

Revenue	Unit	LOM Total
Revenue from Conc.	USD M	393.7
Revenue from Gold Dore	USD M	413.8
Revenue from Sulphuric acid	USD M	9.0
Revenue from As2O3	USD M	-0.04
Total Revenue	USD M	816.5

18.3 Operating Costs

The operating costs are estimated in two (2) phases, which are excluding roasting from the beginning to the year 2023and including roasting from 2024 to the end of LOM. The summary of operating costs is shown in Table 18-4 below. The distribution of operating costs is presented in Figure 18-2 and Figure 18-3.

Table 18-4: Unit Operating Costs

Cost centre	Unit	Cost Excl. Roasting	Cost Incl. Roasting
U/G Mining	USD/ t ROM	29.2	29.2
O/C Mining	USD/ t ROM		4.8
O/C Stripping	USD/ t		1.5
Processing and smelter	USD/ t Milled	25.3	34.0
Roasting and acid manufacturing	USD/ t Milled	-	5.3
Supportive infrastructure	USD/ t Milled	0.8	0.8
General and administration	USD/ t Milled	14.0	16.6
Total		69.3	85.9

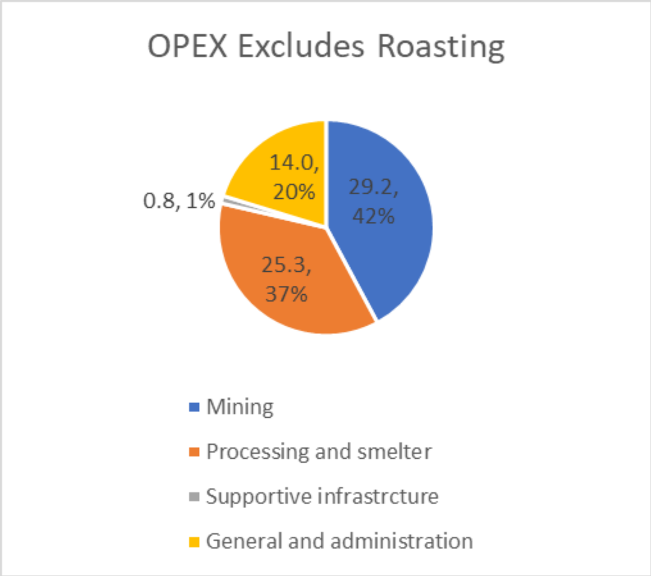


Figure 18-2: Costs Excludes Roasting Components (2019-2023)

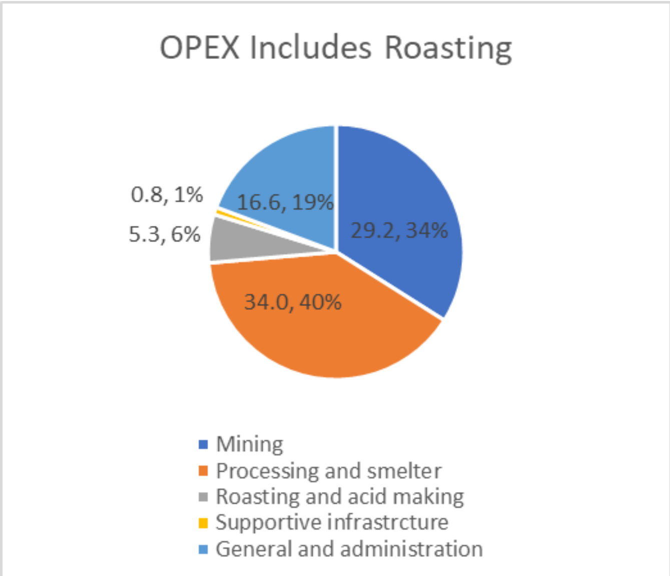


Figure 18-3: Costs Includes Roasting Components (2024-2026)

The annual operating cost and LOM forecast operating cost are presented in Figure 18-4 and Table 18-5 below.

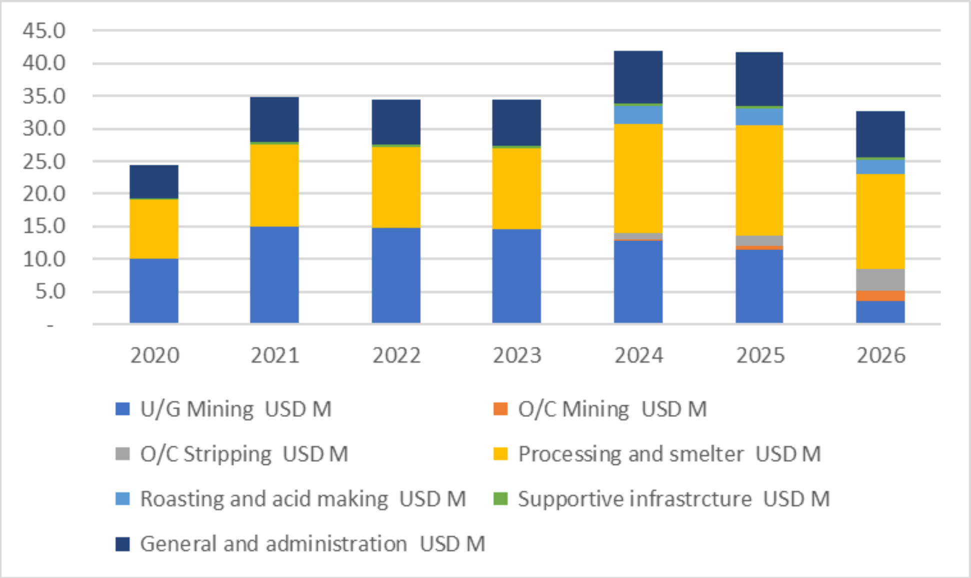


Figure 18-4: Annual Operating Costs by Cost Centre

Table 18-5: LOM Operating Costs Forecast

Cost centre	Unit	LOM Total
U/G Mining	USD M	82.2
O/C Mining	USD M	2.0
O/C Stripping	USD M	6.1
Processing and smelter	USD M	94.8
Roasting and acid manufacturing	USD M	7.5
Supportive infrastructure	USD M	2.6
General and administration	USD M	49.3
Total	USD M	244.4

18.4 Capital Expenditures

The capital expenditure is contained in three parts, which are initial (pre-production) capital, sustaining capital and working capital. It should be noted that the sustaining capital includes a large part for roasting and acid plant in year 2023. The annual distribution of Capital expenditure over the LOM is shown in Figure 18-5 below. The forecast LOM total capital expenditure is presented in Table 18-6. The working capital and residual value of assets will be fully reclaimed at the end of LOM.

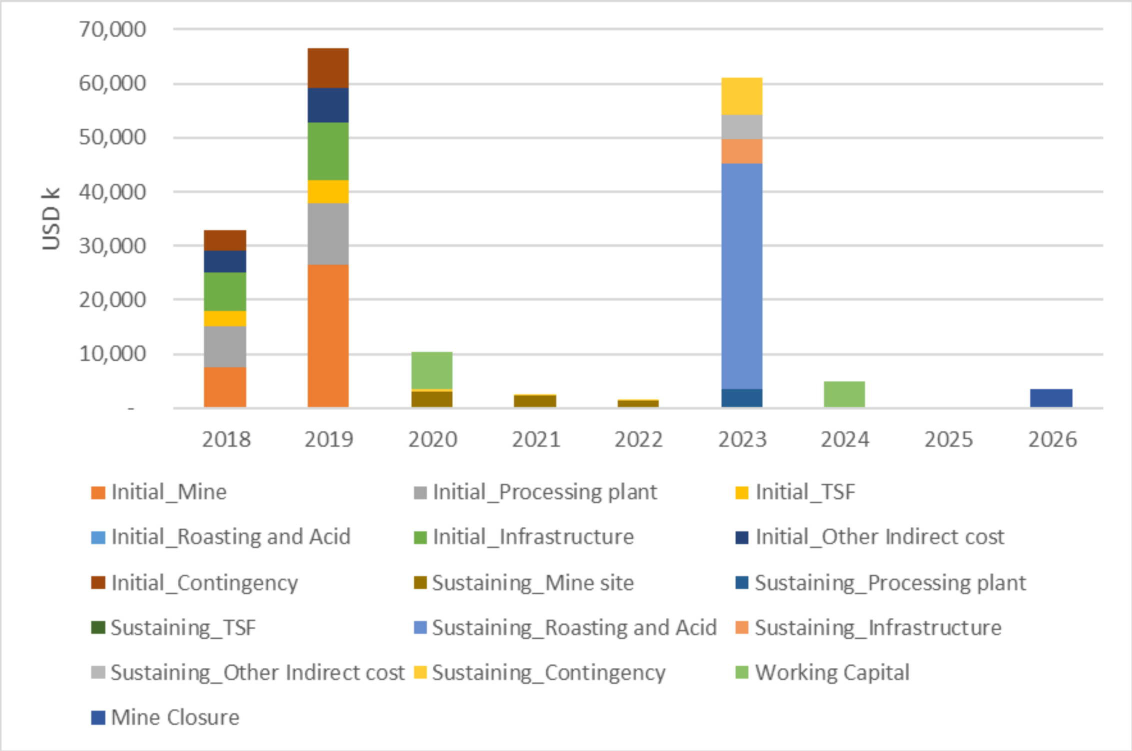


Figure 18-5: Annual Capital Expenditure

Table 18-6: Forecast LOM Capital Expenditure

Cost Centre	Unit	LOM Total
Initial_Mine	USD k	34,006
Initial_Processing plant	USD k	19,091
Initial_TSF	USD k	6,922
Initial_Roasting and Acid	USD k	-
Initial_Infrastructure	USD k	17,808
Initial_Other Indirect cost	USD k	10,492
Initial_Contingency	USD k	11,040
Initial_Total	USD k	**99,359**
Sustaining_Mine site	USD k	6,949
Sustaining_Processing plant	USD k	3,321
Sustaining_TSF	USD k	-
Sustaining_Roasting and Acid	USD k	41,828
Sustaining_Infrastructure	USD k	4,452
Sustaining_Other Indirect cost	USD k	4,525
Sustaining_Contingency	USD k	7,634
Sustaining_Total	USD k	68,709
Mine Closure	USD k	3,400
Working Capital	USD k	**12,000**

18.5 Depreciation and Tax

SRK used a 10% straight line depreciation method in its tax calculations. The initial and sustaining capital including indirect and contingency are considered depreciated and/or amortized. The annual depreciation over LOM is presented in Table 18-7 below.

Table 18-7: Depreciation and Residual

Item	Unit	2018 & 2019	2020	2021	2022	2023	2024	2025	2026
Annual Capex	USD M	99.4	3.4	2.6	1.7	61.0	-	-	3.4
Annual D&A	USD M		9.9	10.3	10.5	10.7	16.8	16.8	16.8
Accumulated Capex	USD M	99.4	102.8	105.4	107.1	168.1	168.1	168.1	171.5
Accumulated D&A	USD M	-	9.9	20.2	30.8	41.5	58.3	75.1	91.9
Residual	USD M	99.4	92.8	85.2	76.3	126.6	109.8	93.0	79.6

SRK has applied a royalty rate of 3.75% to gold revenue and an enterprise income tax of 25% to the earnings before tax (EBT). As the project will be fully invested by equity financing, no interest payments are calculated. Table 18-8 below presents the LOM profit and loss statement.

Table 18-8: LOM Profit and Loss Statement

Profit and loss	Unit	LOM Total
Revenue	USD M	816.5
OPEX	USD M	244.4
Royalty 3.75%@gold revenue	USD M	30.6
EBITDA	USD M	541.4
EBITDA Margin	%	66.3
D&A	USD M	91.9
EBIT	USD M	449.5
Interest balance	USD M	-
EBT	USD M	449.5
Tax Payable 25%@EBT	USD M	112.4
NPAT	USD M	337.2
NPAT Margin	%	41.3

18.6 Discount Rate

SRK has used a real discount rate of 9% in the economic analysis of the base case. The derivation of this rate is presented in Table 18-9 below.

Table 18-9: Discount Rate Calculation

Description	Value
Risk Free Rate- 10yr Bond	7.2%
Market Risk Premium	4.5%
Beta	1.2
Cost of Equity	12.7%
Debt Margin	3.0%
Cost of Debt	4.5%
Tax Rate	25.0%
Post-tax cost of debt	3.4%
Target Debt Equity Ratio [D/(D+E)]	0.0%
Weighted Average Cost of Capital- WACC	12.7%
Total Nominal WACC	12.7%
WACC is Nominal: Need to convert to Real Terms as model is in real terms	
Indonesia Inflation Rate	3.4%
WACC in Real terms	9.0%

18.7 Cashflow Model

The summary of Ciemas Gold Project after tax cashflow (ATCF) forecast over LOM is shown in Table 18-10 and annual distribution is presented in Figure 18-6.

Table 18-10: LOM ATCF Summary

Cashflow	Unit	LOM Total
Total Revenue	USD M	816.5
Total OPEX	USD M	244.4
Total CAPEX	USD M	171.5
Working Capital	USD M	12.0
Residual	USD M	79.6
Reclaim of working capital	USD M	12.0
Royalty 3.75%@gold revenue	USD M	30.6
Tax Payable 25%@EBT	USD M	112.4
Net Cashflow	USD M	337.2

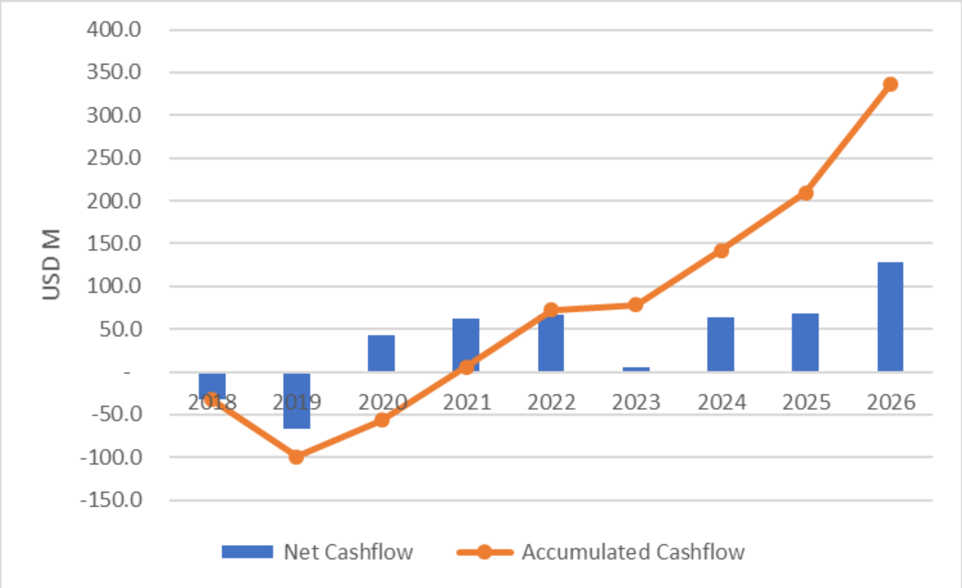


Figure 18-6: Annual ATCF Forecast

Table 18-11 below presented the simple DCF model.

Table 18-11: DCF Model Summary

Item	Unit	LOM Total	2018	2019	2020	2021	2022	2023	2024	2025	2026
Cash-in	**USD M**	**908.1**	**-**	**-**	**96.1**	**121.8**	**127.3**	**124.2**	**132.8**	**131.4**	**174.5**
Total Revenue	USD M	816.5	-	-	96.1	121.8	127.3	124.2	132.8	131.4	82.9
Residual	USD M	79.6	-	-	-	-	-	-	-	-	79.6
Reclaim of working capital	USD M	12.0	-	-	-	-	-	-	-	-	12.0
Cash-out	USD M	**570.9**	**32.9**	**66.5**	**53.0**	**60.1**	**60.3**	**118.6**	**69.2**	**63.6**	**46.7**

Total OPEX	USD M	244.4	-	-	24.4	34.8	34.5	34.4	42.0	41.7	32.7
Total CAPEX	USD M	171.5	32.9	66.5	3.4	2.6	1.7	61.0	-	-	3.4
Working Capital	USD M	12.0	-	-	7.0	-	-	-	5.0	-	-
Royalty 3.75%@gold revenue	USD M	30.6	-	-	3.6	4.6	4.8	4.7	5.0	4.9	3.1
Tax Payable 25%@EBT	USD M	112.4	-	-	14.5	18.0	19.4	18.6	17.3	17.0	7.6
Net Cashflow	**USD M**	**337.2**	**-32.9**	**-66.5**	**43.1**	**61.7**	**67.0**	**5.6**	**63.6**	**67.8**	**127.7**

18.8 Cashflow Projection

Table 18-12 below presented the projection of the DCF model of Ciemas Gold Project. As the NPV of this project in the base case is positive, it is reasonable to convert Mineral Resource to Ore Reserve.

Table 18-12: Economic Analysis Summary

Item	Unit	Value
Base Case NPV @ 9%	USD M	180.3
Base Case IRR	%	43.0
Base Case Payback Years	Years from July 2018	3.4
NPV @ 4%	USD M	254.8
NPV @ 6%	USD M	221.8
NPV @ 8%	USD M	193.1
NPV @ 10%	USD M	168.1
NPV @ 12%	USD M	146.2
NPV @ 14%	USD M	126.9
NPV @ 16%	USD M	110.0

The NPV at different discount rate curve is presented in Figure 18-7 below.

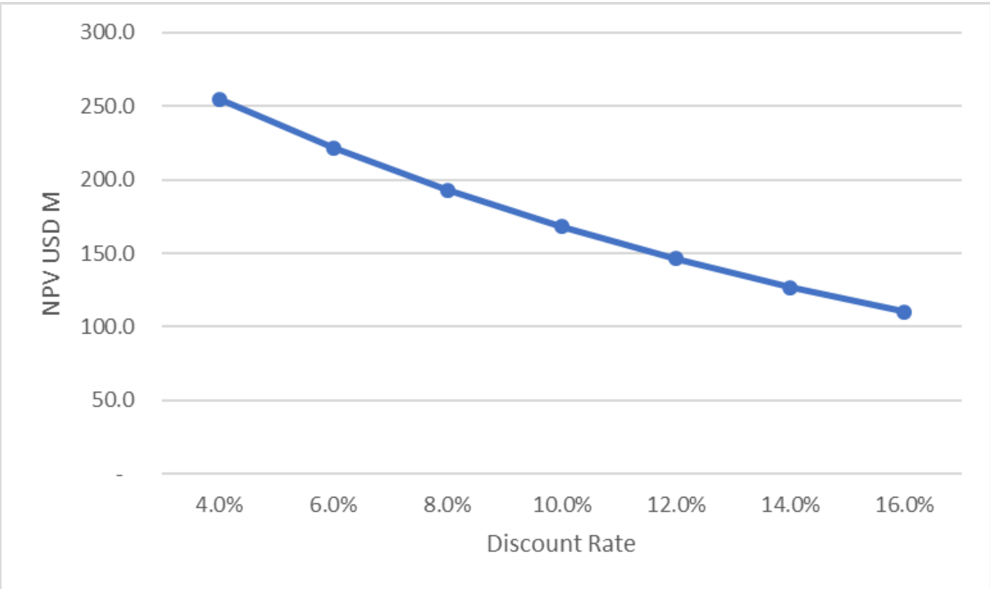


Figure 18-7: NPV at Different Discount Rates

18.9 Sensitivity Analysis

Sensitivity analysis has been conducted based on the base scenario against the changes of capital cost estimates ("CAPEX"), operating cost estimates ("OPEX"), and production revenue. Many parameters affect the production revenue, such as ore grade, recovery rates, and product price. To simplify the calculations, SRK used the price of gold as a representative for the analysis. Table 18-13 below and Figure 18-8 gives the results of NPVs compared to changes in CAPEX, OPEX, and forecast gold price.

Table 18-13: NPV (@9% discount rate) vs. CAPEX, OPEX and Gold Price (in USD Million)

Sensitivity	Revenue	OPEX	CAPEX
-25%	83.1	210.0	202.0
-20%	102.6	204.1	197.7
-15%	122.0	198.1	193.3
-10%	141.4	192.2	189.0
-5%	160.9	186.3	184.7
0%	180.3	180.3	180.3
5%	199.8	174.4	176.0
10%	219.2	168.4	171.6
15%	238.6	162.5	167.3
20%	258.1	156.5	163.0
25%	277.5	150.6	158.6

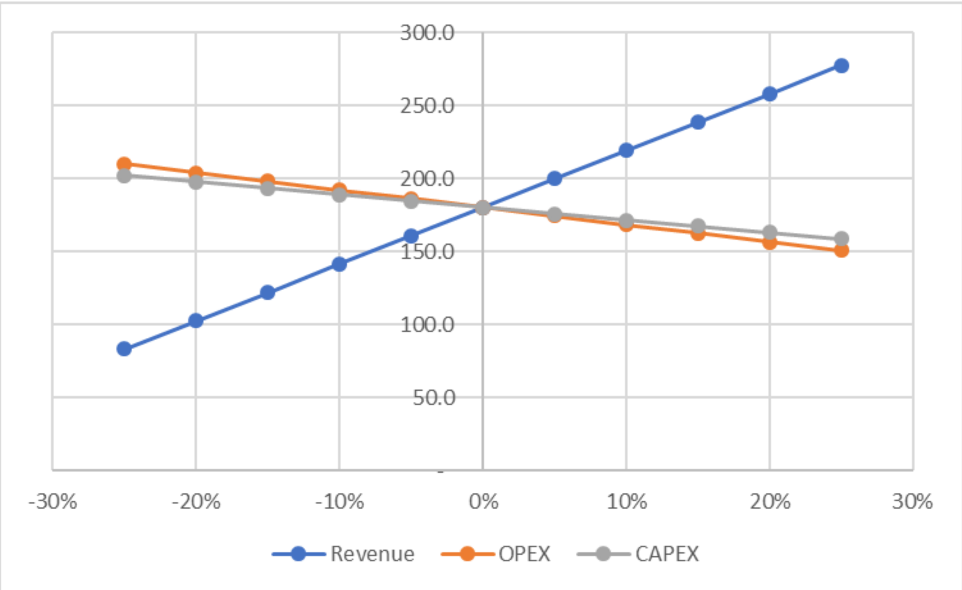


Figure 18-8: Sensitivity Analysis on NPV

It can be seen that the changes in CAPEX have a little more effect on the Project's NPV than changes in OPEX do, while changes in price of gold have the most significant effect on NPV.

To show the effect of gold price on the Project's NPV, SRK conducted a further sensitivity analysis on gold price to the Project and the results are presented in Table 18-14 below. The break-even price (NPV=0 at 9% discount) is around a change of -46.2% from the base scenario prices. On this basis SRK considers that the Project is economically feasible, and the conversion of the ore reserves is reasonable.

Table 18-14: Summary of Sensitivity on Gold Price to Project

Scenario	Gold Price	NPV @9%	IRR	Payback Period
Unit	USD/oz	USD M	%	Years
Base Case	1300-1220	180.3	43.0	3.4
Case -20%	1040-976	102.6	29.3	4.0
Case -10%	1170-1098	141.4	36.3	3.7
Case +10%	1430-1342	219.2	49.4	3.2
Case +20%	1560-1464	258.1	55.5	3.0
Case Break-even (-46.2%)	700-657	-	9.0	

19 Project Implementation

19.1 Extent and Principles of the Design

Wilton may decide the methodology of selecting construction companies and teams for building the project. Although there was some engine engineering done in the project for different purposes, it is SRK's opinion that the existing facilities are basically temporary. All engineering, development and construction for the project will be newly conducted. Wilton may either retain a professional design-equipment-procurement-construction (DEPC) company or manage the construction of the project by itself.

The detailed designs and preparation of the construction would start on April 1, 2018, and take 3 months. The formal construction is designed to start on July 1, 2018. There could be possible delay on the design of project implementation, but generally this design can be referred as an overall implementation schedule.

In the feasibility study, SRK proposed to having three development teams for the mine construction, and others teams for ore processing and smelter plants, tailings storage facility, baking and acid manufacture workshop, and surface infrastructures etc.

19.2 Milestones

Milestone 1: On July 1, 2018, the construction of the project will formally start after three months of modification of design, detailed designs of engineering and constrution preparations. For the mine development schedule, please refer to the mining section. The construction of other facilities will also start in July 2018. The construction will start in the 3 sections of Cikado, Sekolah and Cibatu, and the construction at Pasir Manggu West will start in January 2019;

Milestone 2: in October 2019, mining of ore may start in Pasir Manggu West;

Milestone 3: At the end of 2019, the main construction of mines, plants and other facilities will be completed, and from January 2020 the plants may be put in a trial production phase for three months;

Milestone 4: In April 2020, after tuning-up of flowsheets during the trial production period, the project will be formally in production;

Milestone 5: in 2021, the production can reach its full capacity of 1500tpd;

Milestone 6: in 2023, the construction of the roasting and associated facilities starts, and will be completed in one year;

Milestone 7: in 2024, the gold concentrates will be treated further by roasting to produce gold dore;

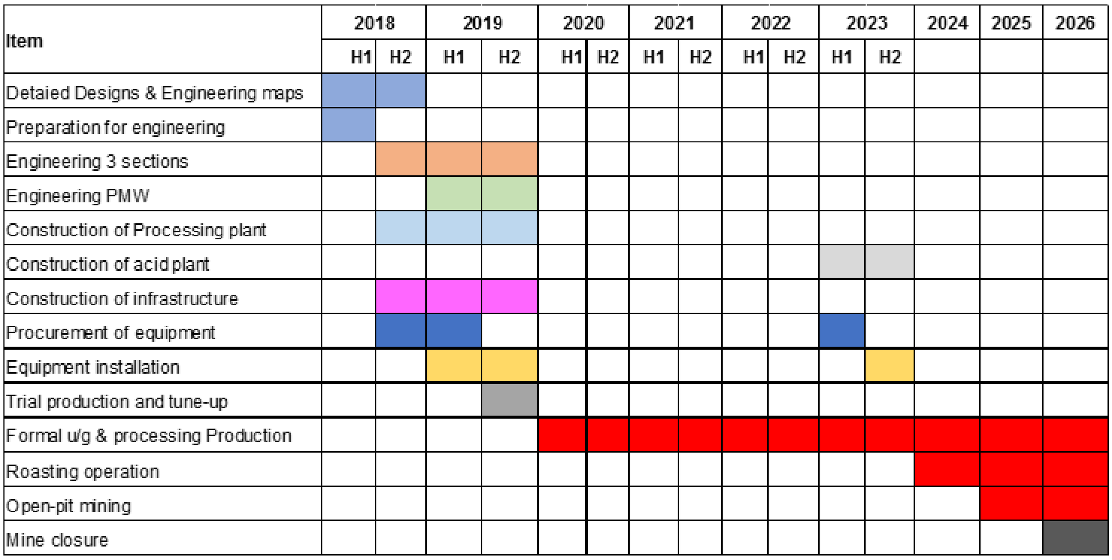
Milestone 8: in 2025 and 2026, open-pit mining will be conducted;

Milestone 9: in 2026, the mile will be closed, unless further exploration defines additional mineral resources to extend the life of the mine.

19.3 Flowchart

Table 19-1 shows the project schedule.

Table 19-1: Project Schedule



19.4 Implementation Strategy

The implementation strategy for the project is dictated by the duration of the construction of the declines at Pasir Manggu West section and other three sections, which provide accesses to the deposits, the estimated duration for this construction is 18 months. It is possible to build all the surface facilities including the process plant and associated infrastructure during this period. Therefore, the construction of the mine accesses is the critical path.

Project implementation will be led by a team from Wilton who will be responsible for the arrangements of detailed engineering, procurement, contract administration and construction management.

The project implementation strategy will include several types of contracting; Construction Contracts, Services Contracts, Build, Operate and Transfer Contracts and Engineering, Procurement and Construction Contracts (EPC), which will be administered by the team.

Wilton may also contract a company, as a Prime Contractor to manage the entire project, including engineering, procurement, contracts, construction management and commissioning (EPCM). This Prime Contractor will prepare the engineering for the entire project, integrating third party designs and providing engineering for the other areas.

Design contracts will be awarded to specialist companies for the TSF, back-filling plant, and ore processing plant. These engineering contractors will be managed by the team or the Prime Contractor, if any. Technical responsibility will remain with the engineering contractor and the Prime Contractor (if any) will manage the work.

20 Conclusions and Recommendations

20.1 Conclusions

This IQPR summarise key technical aspects of the Ciemas Project. This feasibility study work completed in 2018 was to test the viability of the current Mineral Resources of the Ciemas Gold Project. The work was done to a feasibility level of detail. The cost estimates calculated at above 86% accuracy are within the ±15% accuracy of a feasibility study.

The IRR of the project based on the described techno-economic solution is positive at 43.0%, and the NPV is USD180.3 M. Therefore, the project is considered to being economically viable, and the ore reserve statements declared in the report are valid.

20.2 Recommendations on Further Work

Geological Exploration and Mineral Resources:

Since the life of the mine of the project is only 7 years in the study, and the ore bodies are still open to depth and discovered in other prospects, SRK recommends to further conduct exploration work for

- The extensions to depth of the ore bodies defined in the four mining sections, in order to define more high-category mineral resources and extend the LOM of the project;

- In other prospects to discover more gold mineralization zones, as well as other mineralizations, in order to define more mineral resources of the project.

Geotechnical Aspects:

The design analyses represent the core of rock mechanics practice. It is understood that knowledge of the strength, structure and stability of the rock mass for underground mining is critical for technical design purposes and for accurate cost estimation. Well-organized ongoing geotechnical studies, including data collection and processing, kinematic stereographic analyses, stability analyses and numerical stimulation, is strongly recommended to provide sound configuration-specific data for the modification of major design parameters of diverse mining components.

The objective of geotechnical studies is to define the mechanical properties and state of the medium in which mining is to take place. It involves determination of the strength and deformation properties of the various lithological units represented in and around the orebody, definition of the geometric and mechanical properties of pervasive jointing, and location and description of the properties of discrete structural features. An estimate of the in situ strength of the medium may then be made from the properties of the constituent elements of the mass. The geotechnical studies should also include determination of the in situ state of stress in the mine area, and investigation of the hydrogeology of the orebody and environs. Having defined the prevailing conditions in the rock mass in an analytically tractable way, the mechanical performance of selected mining configurations and excavation geometries can be predicted using appropriate mathematical or numerical techniques.

Moreover, monitoring rock mass performance should be carried out to ensure safety during construction and operation by giving warning of the development of excess ground deformations, groundwater pressures and loads in support and reinforcement elements, and to check the validity of the assumptions, conceptual models and values of rock mass properties used in design calculations in characterising the operational response of the rock mass to mining activity. The data required to generate understanding of rock performance are obtained by displacement and stress measurements made at key locations in the mine structure. In addition, visual inspections must be undertaken regularly to locate any structurally controlled failures and areas of anomalous response, and these should be mapped routinely. Finally, data should be collected on the production performance of each stope, and the final configuration of each stope should be surveyed and mapped. The aim in this case is to seek any correlation between rock mass local performance and stope productivity.

Hydrogeology:

In relation to the site hydrology, SRK recommends that the most recent climate data from nearby weather stations be obtained, as well as monthly monitoring data of flow rates and water quality from these creeks in the project area. Based on these data, flood lines of 1:100 year events should be delineated for this project area to avoid flooding of mining area, processing plant, TSF area, and ancillary facility areas. In addition, a storm water management system should be developed to separate the clean water from the dirty water, and to minimize the soil erosion.

SRK noted that a preliminary hydrogeological survey was in place, but that it lacks a complete hydrogeological model. Therefore, SRK recommends that a further comprehensive hydrogeological investigation be undertaken within the underground mining area to build a numerical groundwater model, based on the fact that most ore bodies are located below the water table.

Mining:

Backfill material should be test when there are tailings produced from the processing plant to optimize the proportion of cement, tailings, and aggregate if needed. The backfill material strength and quantity of cement added are based on some assumptions by the reason that there is no tailing produce by the planned processing plant. Therefore further test on backfill material and optimization are need.

Local support method special study when the development opened if needed. The FS proposed support method is the general understanding based on the geo-tech data which is existing as day off. So the further studies may need if some bad ground situation be met.

Ore Processing:

There were abundant experiments and tests done in the past with various types/locations of samples. A scale-up (expanding) test of composite samples from these deposits is recommended to be conducted to optimize the flowsheet in order to achieve higher recovery rate. The expanding test can be also aimed to verify the feasibility of "fine grinding – chemical peroxidation – leaching" flow.

The smelting plant and its auxiliary facilities were comprehensively designed in accordance with the process determined/suggested by the metallurgical test and the actual conditions of the mine. The designed metallurgical production scale is 495 ktpa, which deals with the primary ore of underground mining in the early stage, and the mixed ore of the primary ore of underground mining and the oxidized ore of open mining in the later stage. Further monitoring and tests are recommended to be followed up during the production.

Cyanide leaching test on Roasting Calcine mixed with flotation tailings material is recommended to determine the Calcine, and the feasibility of tailings cyaniding carbon slurry mixed material, namely the feasibility of the merger the tailing slurry and Roasting Calcine slurries to save the costs.

Detailed design is recommended on the oxidation roasting plant of flotation concentrate and its affiliated facilities for purifying arsenic collection and sulfuric acid production and other auxiliary facilities.

The construction investment of TSF could be lower as currently the TSF has been designed completely coated and impermeable. Toxic test of tailings wastewater is suggested to determine the feasibility of no coating film

Environmental Aspect:

The environmental impact assessment (AMDAL) for the Ciemas Project is recommended to be updated to involve the roasting and acid making system and the management measures of As_2O_3 should be particularly studied in the updated AMDAL as well.

The reclamation plan and post-mining plan should be developed to establish reclamation/closure management strategies and cost estimates to address/reduce reclamation/closure liabilities.

The development of the corporate social responsibility (CSR) and community development (CD), which identify the project's social responsibility and ensure the community engagement, is underway. Six themes (infrastructure, economy, education, health, environment and donation) are suggested for the CSR/CD development.

21 Project Qualitative Risk Analysis

Mining is a relatively high risk industry. In general, the risk may decrease from exploration, development, through to production stage. The Ciemas Gold Project is an advanced exploration/development project with some previous production. Risks exist in different areas. SRK considers various technical aspects which may affect the feasibility and future cash flow under the proposed production schedule of the project, and conducts a qualitative risk analysis which has been summarised in Table 21-1. In this risk analysis, various risk sources/issues have been assessed for Likelihood and Consequence, and then a Risk Rating has been assigned. The qualitative risk analysis uses the following definitions for likelihood and consequence:

The Likelihood of a risk is considered within a certain time frame, e.g., five years, as:

- **Likely:** will probably occur;
- **Possible:** may occur; or
- **Unlikely:** unlikely to occur.

The Consequence of a risk is classified as:

- **Major Consequence:** the factor poses an immediate danger to the Project that, if uncorrected, will have a material effect on the Project cash flow and performance and could lead a project failure;
- **Moderate Consequence:** the factor, if uncorrected, will have a significant effect on the Project cash flow and performance; or
- **Minor Consequence:** the factor, if uncorrected, will have little or no effect on the Project cash flow and performance.

The overall risk assessment combines the Likelihood and Consequence of a risk, and be classified as Low (unlikely and possible minor risks, and unlikely moderate risk), Medium (likely minor, possible moderate, and unlikely major risks), and High (likely moderate and major, and possible major risks).

Table 21-1: Project Risk Assessment of the Ciemas Gold Project

Risk Source/Issue	Likelihood	Consequence	Risk Rating
Geology and Resource			
Lack of Significant Resource	Unlikely	Moderate	Low
Lack of Significant Reserve	Unlikely	Major	Medium
Unexpected Groundwater Ingress	Possible	Moderate	Medium
Mining			
Significant Production Shortfalls	Unlikely	Major	Medium
Low Production Pumping System Adequacy	Unlikely	Moderate	Low
Significant Geological Structure or Geotechnical Issues	Possible	Moderate	Medium
Excessive Surface Subsidence	Unlikely	Minor	Low
Poor Ground Conditions	Possible	Moderate	Medium
Poor Mine Plan	Possible	Moderate	Medium
Poor Stability of Backfilling System	Possible	Moderate	Medium
Ore Processing			
Lower Yields (output / raw ore)	Possible	Moderate	Medium
Lower Recovery	Possible	Moderate	Medium
High Production Cost	Likely	Moderate	Medium
Poor Plant Reliability	Unlikely	Moderate	Medium
Environmental and Social			
Land disturbance, rehabilitation and site closure	Possible	Moderate	Medium
Poor Water management (i.e. stormwater/surface water drainage – including any mine dewatering).	Possible	Moderate	Medium
Poor Waste rock stockpiling/ dumping management	Possible	Moderate	Medium
Land contamination (i.e. hydrocarbon storage and handling).	Possible	Moderate	Medium
Social aspects (i.e. local community interactions)	Possible	Moderate	Medium
Capital and Operating Costs			
Project Timing Delays	Possible	Moderate	Medium
Capital Cost Increases	Possible	Moderate	Medium
Operating Cost Underestimated	Possible	Moderate	Medium
Project Implementation			
Construction or Production Delay	Possible	Moderate	Medium

SRK notes that variations in the market price of gold may affect the project's economic analysis as shown in this report; however it is considered as a low risk with improbable likelihood of gold prices falling below the value that would cause a negative NPV and subsequently hinder appropriate Ore Reserves estimation for the project.

West Java is known as a tectonically active area subject to relatively frequent earthquakes, according to historical records. Seismic events are considered as a possibly medium rating factor; however, such risk is not possible to evaluate or control. It is recommended that geological and engineering analysis and procedures should be performed to protect staff and infrastructure from earthquakes.

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Appendices

Appendix 1: JORC Code Table 1

Section 1: Sampling Techniques and Data

Criteria	JORC Code explanation	Commentary – Assessment of the Ciemas Project
Sampling techniques	• Nature and quality of sampling (eg cut channels, random chips, or specific specialised industry standard measurement tools appropriate to the minerals under investigation, such as down hole gamma sondes, or handheld XRF instruments, etc). These examples should not be taken as limiting the broad meaning of sampling. • Include reference to measures taken to ensure sample representivity and the appropriate calibration of any measurement tools or systems used. • Aspects of the determination of mineralisation that are Material to the Public Report. • In cases where 'industry standard' work has been done this would be relatively simple (eg reverse circulation drilling was used to obtain 1 m samples from which 3 kg was pulverised to produce a 30 g charge for fire assay'). In other cases more explanation may be required, such as where there is coarse gold that has inherent sampling problems. Unusual commodities or mineralisation types (eg submarine nodules) may warrant disclosure of detailed information.	• Sampling of split drill cores and reverse circulation ("RC") drill chips has been applied to the Ciemas Project. Specific gravity samples were taken from oxidised outcrops and fresh zones (drill cores). No other specialized sampling techniques were included in the exploration database used for Mineral Resource estimation. • The data used for Mineral Resource estimation was solely derived from drill holes (including RC drillholes and diamond drillholes). • There were sufficient surface works (trenching, geochemistry, pitting, percussion drilling) done in the four properties (Pasir Manggu, Cikadu, Sekolah and Cibatu) previously (some 20 years ago). These were only used to guide the interpretation of mineralisation near the surface. • Diamond drill cores with mineralisation indication (predominately by observing sulphide in tectonic breccia or quartz veins) were sampled by split cuts at 1 m intervals generally and RC chips were collected and split at intervals of about 1 m. • Sample representativity was guaranteed by systematic drilling conducted on a basic exploration grid of 40 m by 40 m, with in-fill grid of 20 m by 20 m and inferred grid of 80 m by 80 m. The exploration grids were designed and deployed in a way similar to what is done in other rock gold deposits.
Drilling techniques	• Drill type (eg core, reverse circulation, open-hole hammer, rotary air blast, auger, Bangka, sonic, etc) and details (eg core diameter, triple or standard tube, depth of diamond tails, face-sampling bit or other	• Core drilling of exploration programme in 2012 and 2013 was completed by standard triple tube rigs in the Ciemas Project. Drill cores were HQ3 - (61.1 mm) size. Every 21m orientation spear

Criteria	JORC Code explanation	Commentary – Assessment of the Ciemas Project
	type, whether core is oriented and if so, by what method, etc).	marks were recorded for core orientation purposes. ● RC drilling was conducted using standard RC rigs and samples were taken every 1 m. ● SRK did not assess the techniques of drilling completed by former concession owners Parry, Terrex and Meekatharra who did exploration in 1980s and 1990s because there was not enough data on them, however SRK performed verification of the drilling results through additional diamond drilling in 2012, which has proved the data derived from previous drilling was managed properly. It was reported that most of the historical drill cores were HQ-sized with some minor NQ-sized drilling.
Drill sample recovery	● Method of recording and assessing core and chip sample recoveries and results assessed. ● Measures taken to maximise sample recovery and ensure representative nature of the samples. ● Whether a relationship exists between sample recovery and grade and whether sample bias may have occurred due to preferential loss/gain of fine/coarse material.	● Core recovery rates of historical drilling conducted by Parry, Terrex and Meekatharra are unknown as not enough data relating to core recovery is available. The re-printed historical DDH and RCH log books recorded the lithology and sample intervals as well as coordinates however there was no information about recovery. Original drillhole logging sheets were found for only a few historical DDHs and SRK noticed the manuscripts recorded core recoveries generally above 85%. Except some core residuals, there are no cores available for recalculating the historical drill sample recoveries. ● For new drilling programme conducted since 2012, the measurements of cores and footage (length) drilled in each run were recorded in the drilling logs and were reviewed by both P.T. ASI and SRK site geologists. In general the core recovery of the drilling programme conducted by Wilton is high, averaging about 95%. The average recovery of mineralised intervals is even higher .

Criteria	JORC Code explanation	Commentary – Assessment of the Ciemas Project
		● For low recovery of cores, the related samples were considered very carefully when using them for grade interpolation, usually with the measures of additional constraints and weight, or even delete the low recovery sample from the grade interpolation. The additional assessment / check was performed to adjust the grade interpolation done by the computer with an uniform inputs of estimation parameters. ● The gold mineralisation is related to breccia and fractured zones, as well as to structurally-controlled alteration rocks. The mineralised intervals are sometimes fractured but the recovery is high as the drilling programme implemented a strict protocol for high core recovery – the drilling team was instructed by field geologists to slow down to avoid getting a low recovery in particular when encountering a breccia zone. Core recovery and assay grades are not correlated, as SRK observed.
Logging	● Whether core and chip samples have been geologically and geotechnically logged to a level of detail to support appropriate Mineral Resource estimation, mining studies and metallurgical studies. ● Whether logging is qualitative or quantitative in nature. Core (or costean, channel, etc) photography. ● The total length and percentage of the relevant intersections logged.	● Logging was done for the historical drilling completed by Parry, Terrex and Meekatharra. Basic information including sample location, lithology, drillhole dip and azimuth and assays is available. ● In the drilling programmes in 2012 and 2013, the core samples have been geologically (lithology, structure, alteration, mineralisation, geotechnical features) logged to a level of detail supporting geological interpretation and Mineral Resource estimation. ● In 2012 and 2013 all cores have been logged and the logs were recorded in a standard logging sheet format and then stored electronically. ● In 2012 SRK geologist performed QA/QC on site, drill cores were photographed during logging.
	● If core, whether cut or sawn and whether quarter,	● The sub-sampling techniques and sample

Criteria	JORC Code explanation	Commentary – Assessment of the Ciemas Project
Sub-sampling techniques and sample preparation	half or all core taken. ● If non-core, whether riffled, tube sampled, rotary split, etc and whether sampled wet or dry. ● For all sample types, the nature, quality and appropriateness of the sample preparation technique. ● Quality control procedures adopted for all sub-sampling stages to maximise representivity of samples. ● Measures taken to ensure that the sampling is representative of the in situ material collected, including for instance results for field duplicate/second-half sampling. ● Whether sample sizes are appropriate to the grain size of the material being sampled.	preparation conducted at the time Parry, Terrex and Meekatharra were in charge of the Ciemas Project are not known. ● In 2012 and 2013, drill cores were split and half core samples were taken . RC drilling chips were collected every 1 m. Both core and RC chips were sampled in the logging and sampling yard by the geologists from Wilton, after logging and photographing. ● In the 2012 exploration programme, SRK field geologists managed the on-site QA/QC. Sampling was directly supervised by SRK. ● SRK geologist inserted several coarse blanks and field duplicates (quarter core and/or chip rejects) in 2012 for random checks in 2012. Since 2012, all samples related to the Mineral Resource estimation of Ciemas Project were prepared by Intertek, a Jakarta-based laboratory belonging to recognised international organization. ● Sample preparation in Intertek Jakarta followed a standard procedure for gold sample preparation, consisting in coding, weighing, crushing, splitting, and pulverising, in agreement with and internationally recognised practice. ● Intertek performed its own QC procedures including the insertion of blank, duplicate and standard samples at a frequency higher than 1:20. ● SRK geologist visited Intertek Jakarta in April and September 2012 and is satisfied with its workflow of sample preparation and QC protocol.
Quality of assay data and laboratory tests	● The nature, quality and appropriateness of the assaying and laboratory procedures used and whether the technique is considered partial or total. ● For geophysical tools, spectrometers, handheld XRF instruments, etc, the parameters used in determining the analysis including instrument make and model, reading times, calibrations factors applied and their	● Prior to Wilton's exploration beginning in 2009, samples were assayed by laboratories Kep Seksi Kimia Mineral, Inchcape Testing Service and PT. Inchcape Utama Service. The Inchcape Laboratory was subsequently re-named "Intertek". ● Samples taken after 2009 and prior to 2012 were

Criteria	JORC Code explanation	Commentary – Assessment of the Ciemas Project
	derivation, etc. ● Nature of quality control procedures adopted (eg standards, blanks, duplicates, external laboratory checks) and whether acceptable levels of accuracy (ie lack of bias) and precision have been established.	only used for mineral identification and Wilton's verification purpose and were not used in the Mineral Resource estimation. ● All samples taken since 2012 (and included in this Mineral Resource estimate) were analysed by Intertek Jakarta with its internal QC procedures, including the insertion of standards, duplicates and blanks. ● Assaying for gold was done by fire assay with atomic absorption spectrometry ("AAS") and other elements including Ag, As, Cu, Pb and Zn were also determined by AAS. ● PT ASI inserted standard samples (four types of CRMs) into the last batch of samples in 2012 at a ratio of 1:20. The results of external standard samples did not reveal any considerable issues that needed to be revisited for the sample analysis. ● No external checks have been performed for the samples assayed since 2012. SRK recommends selecting about 5% - 10% of total assayed samples for external checks.
Verification of sampling and assaying	● The verification of significant intersections by either independent or alternative company personnel. ● The use of twinned holes. ● Documentation of primary data, data entry procedures, data verification, data storage (physical and electronic) protocols. ● Discuss any adjustment to assay data.	● SRK project team initially visited the project site in March 2012. In March and April 2012, Wilton drilled 9 diamond drill holes for data verification purpose. These drillholes were planned by SRK and the drilling and sampling processes were closely supervised by SRK geologists. ● From October 2012 to January 2013, Wilton drilled additional 15 drillholes for verification and in-fill purpose. The drilling was following the protocols prepared by SRK and the drilling and sampling of the first 8 holes were supervised by SRK;the other 7 holes were inspected by SRK in June 2014. ● During each site visit SRK geologists inspected the exploration ground, mineralisation, drill cores and sealed borehole collar.

Criteria	JORC Code explanation	Commentary – Assessment of the Ciemas Project
		● An SRK mining engineer, a processing engineer and an environmental scientist visited the Ciemas tenement in April 2012 and March 2013, respectively. ● SRK geologists visited the primary laboratory – Intertek Jakarta and assessed that the laboratory was certified and capable to perform the sample preparation and assaying. ● There were no twin holes drilled to the Ciemas Project however 5 holes were located near (less than 10 m apart) previous DDH or RCH. Additional in-fill drill holes revealed geological continuity of gold mineralisation as interpreted. ● Overall, SRK was satisfied of the verification results and was of the opinion that the integrated database used for Mineral Resource estimation for the four properties was reliable and reasonable.
Location of data points	● Accuracy and quality of surveys used to locate drill holes (collar and down-hole surveys), trenches, mine workings and other locations used in Mineral Resource estimation. ● Specification of the grid system used. ● Quality and adequacy of topographic control.	● Each drillhole coordinate was surveyed using Universal Transverse Mercator ("UTM") coordinate system. Prior to Wilton the survey employed a localised coordinate system and these local coordinates were all reconciled to the UTM system and checked by recent survey. The coordinates were consistent with or transferable to the coordinates specified in the mining licence (IUP-OP). ● A network of 22 benchmarks have been installed across the project area using survey grade differential Global Positioning System ("GPS") methods and all surveys of drill collars are done sing total station equipment referenced to these benchmarks. ● Since 2012 downhole survey has been generally performed every 50 m downhole by the drilling team using micro-camera "Proshot". ● The surveys ensured the locations of data points used for Mineral Resource estimation were

Criteria	JORC Code explanation	Commentary – Assessment of the Ciemas Project
		accurate.
Data spacing and distribution	● Data spacing for reporting of Exploration Results. ● Whether the data spacing and distribution is sufficient to establish the degree of geological and grade continuity appropriate for the Mineral Resource and Ore Reserve estimation procedure(s) and classifications applied. ● Whether sample compositing has been applied.	● The drilling grids were generally 40 m by 40 m. The spacing between exploration lines was generally 40 m. The in-fill drilling grids were about 20 m by 20 m, and grids of approximately 80 m by 80 m were used to explore the resource boundaries. ● Sample length was generally 1 m, Samples were continuously taken over all mineralised zones and their direct host walls. ● All samples were composite to 1m within the geological model.
Orientation of data in relation to geological structure	● Whether the orientation of sampling achieves unbiased sampling of possible structures and the extent to which this is known, considering the deposit type. ● If the relationship between the drilling orientation and the orientation of key mineralised structures is considered to have introduced a sampling bias, this should be assessed and reported if material.	● Where possible the drill holes were planned and executed in exploration lines perpendicular to the overall strike of the gold veins. ● Holes have been drilled at dip angles varying from -55° to – -90° depending on terrain and to intercept the mineralisation perpendicularily,. ● There was no sample bias due to the angle of drilling. ● The dip angle and azimuth were used in a 3D modeling to reflect actual sampling location and orientation.
Sample security	● The measures taken to ensure sample security.	● The sample security prior to Wilton's management of this project is unknown. Previous sample rejects and duplicates are not available. ● Wilton's samples were taken and secured by the Company. Samples were transported to the Intertek laboratory with the Company's own vehicles. ● Coarse rejects and pulps were returned from Intertek and were transported to each project site by the Company. ● All remaining drill cores, coarse rejects and pulps were secured at the core shack of the project site by the Company personnel.
Audits or reviews	● The results of any audits or reviews of sampling techniques and data.	● Site visits and on-site supervision had been performed by SRK geologists. Exploration,

Criteria	JORC Code explanation	Commentary – Assessment of the Ciemas Project
		sampling techniques, QA/QC protocols and data collection had been reviewed to ensure correct procedures and protocols were followed and that the data collected was reliable and accurate for Mineral Resource estimation and reporting to the JORC Code 2012 edition standards(best practice). ● PT. ASI, an Indonesia based profession provided technical support to the Ciemas Project. Professor Zhengwei Zhang from Chinese Academy of Sciences led a team doing scientific research in the project area and published scientific papers.

Section 2: Reporting of Exploration Results

Criteria	JORC Code explanation	Commentary – Assessment of the Ciemas Project
Mineral tenement and land tenure status	● Type, reference name/number, location and ownership including agreements or material issues with third parties such as joint ventures, partnerships, overriding royalties, native title interests, historical sites, wilderness or national park and environmental settings. ● The security of the tenure held at the time of reporting along with any known impediments to obtaining a licence to operate in the area.	● Tenure information and project location are detailed in Section 2.2 of this report.
Exploration done by other parties	● Acknowledgment and appraisal of exploration by other parties.	● Where possible SRK reviewed the data derived from the previous exploration done by Parry, Terrex and Meekatharra and planned additional verification drillholes. ● Wilton's exploration completed in 2012 was under the guidance of an exploration protocol established by SRK, which supervised the exploration.
Geology	● Deposit type, geological setting and style of mineralisation.	● Detailed in Section 4 of this report.
Drill hole Information	● A summary of all information material to the understanding of the exploration results including a tabulation of the following information for all Material drill holes: - easting and northing of the drill hole collar - elevation or RL (Reduced Level – elevation above sea level in metres) of the drill hole collar - dip and azimuth of the hole - down hole length and interception depth - hole length. ● If the exclusion of this information is justified on the basis that the information is not Material and this exclusion does not detract from the understanding of the report, the Competent Person should clearly explain why this is the case.	● Drillholes are used for Mineral Resource estimation ● More detailed individual exploration drillhole sample results and downhole intercepts are available on request.
	● ·In reporting Exploration Results, weighting	● The sample data derived from drilling was

Criteria	JORC Code explanation	Commentary – Assessment of the Ciemas Project
Data aggregation methods	averaging techniques, maximum and/or minimum grade truncations (eg cutting of high grades) and cut-off grades are usually Material and should be stated. ● Where aggregate intercepts incorporate short lengths of high grade results and longer lengths of low grade results, the procedure used for such aggregation should be stated and some typical examples of such aggregations should be shown in detail. ● The assumptions used for any reporting of metal equivalent values should be clearly stated.	compiled into an integrated database with information of collars, downhole surveys and sample assays. ● Evaluation of outliers was performed according to the basic analysis of composite samples. Grade capping was applied for controlling extreme high grade outliers at each mineralised vein, as detailed **in** Section 5.7 . ● No metal-equivalence approaches were applied.
Relationship between mineralisation widths and intercept lengths	● These relationships are particularly important in the reporting of Exploration Results. ● If the geometry of the mineralisation with respect to the drill hole angle is known, its nature should be reported. ● If it is not known and only the down hole lengths are reported, there should be a clear statement to this effect (eg 'down hole length, true width not known').	● The mineralised bodies were modeled according to the sample interceptions and mineralisation widths were reported as "true thickness" according to the modeled bodies. ● The geometry of the mineralisation with respect to the drillhole angle is known.
Diagrams	● Appropriate maps and sections (with scales) and tabulations of intercepts should be included for any significant discovery being reported These should include, but not be limited to a plan view of drill hole collar locations and appropriate sectional views.	● Geological map and sections with drillholes are shown in Section 4.2 .
Balanced reporting	● Where comprehensive reporting of all Exploration Results is not practicable, representative reporting of both low and high grades and/or widths should be practiced to avoid misleading reporting of Exploration Results.	● For the four properties of Pasir Manggu, Cikadu, Sekolah and Cibatu Mineral Resource estimates are reported ; other Exploration Results are not presented in this report.
Other substantive exploration data	● Other exploration data, if meaningful and material, should be reported including (but not limited to): geological observations; geophysical survey results; geochemical survey results; bulk samples – size and method of treatment; metallurgical test results; bulk density, groundwater, geotechnical	● SRK is not aware of any other material or substantive exploration data that has not been reported.

Criteria	JORC Code explanation	Commentary – Assessment of the Ciemas Project
	and rock characteristics; potential deleterious or contaminating substances.	
Further work	● The nature and scale of planned further work (eg tests for lateral extensions or depth extensions or large-scale step-out drilling). ● Diagrams clearly highlighting the areas of possible extensions, including the main geological interpretations and future drilling areas, provided this information is not commercially sensitive.	● Exploration potential has been discussed in this report. ● SRK is aware the Company is making a detailed exploration plan for further work which will be disclosed at a later.date

Section 3: Estimation and Reporting of Mineral Resources

Criteria	JORC Code explanation	Commentary – Assessment of the Ciemas Project
Database integrity	• Measures taken to ensure that data has not been corrupted by, for example, transcription or keying errors, between its initial collection and its use for Mineral Resource estimation purposes. • Data validation procedures used.	• The original exploration data was provided by Wilton. Diamond drilling and reverse circulation drilling data were combined for Mineral Resource estimation. • Prior to using the drilling data for Mineral Resource estimation, SRK performed a data verification programme by drilling 6 holes at each property area. Historical data was partly verified by the new drilling.
Site visits	• Comment on any site visits undertaken by the Competent Person and the outcome of those visits. • If no site visits have been undertaken indicate why this is the case.	• SRK Competent Persons visited theprojects in 2012 and 2013. Data verification and QA/QC programme to the projects were performed by SRK field geologists and approved and closely supervised by SRK team leader (Competent Person) Dr Anshun Xu FAusIMM (Director, Principal Geologist).
Geological interpretation	• Confidence in (or conversely, the uncertainty of) the geological interpretation of the mineral deposit. • Nature of the data used and of any assumptions made. • The effect, if any, of alternative interpretations on Mineral Resource estimation. • The use of geology in guiding and controlling Mineral Resource estimation. • The factors affecting continuity both of grade and geology.	• The geological interpretation was based on lithologyl, assays, structure and geotechnical information. • Drilling interceptions provided general confidence in the interpretation of gold mineralised veins. Ground geochemistry and channel (trenching) sample assays enhanced the level of confidence of the mineralisation and the interpretation of mineralised veins at the surface. • Geological continuity has been assessed in each cross section. • SRK's interpretation of mineralised veins/bodies was produced as a 3D wireframe model, which was supported by a similar interpretation from the Company and its consultants using 2D cross-sections; it was also supported by a similar interpretation of a 3D model made by PT. ASI.
Dimensions	• The extent and variability of the Mineral Resource expressed as length (along strike or otherwise), plan width, and depth below surface to the upper and	• A total of 24 mineralised vein zones are defined in the Pasir Manggu. Cikadu, Sekolah and Cibatu areas of the Ciemas Project. The

Criteria	JORC Code explanation	Commentary – Assessment of the Ciemas Project
	lower limits of the Mineral Resource.	geometric characteristics of the defined mineralised zones (veins) are detailed in Table 4-1 in the report.
Estimation and modelling techniques	● The nature and appropriateness of the estimation technique(s) applied and key assumptions, including treatment of extreme grade values, domaining, interpolation parameters and maximum distance of extrapolation from data points. If a computer assisted estimation method was chosen include a description of computer software and parameters used. ● The availability of check estimates, previous estimates and/or mine production records and whether the Mineral Resource estimate takes appropriate account of such data. ● The assumptions made regarding recovery of by-products. ● Estimation of deleterious elements or other non-grade variables of economic significance (eg sulphur for acid mine drainage characterisation). ● In the case of block model interpolation, the block size in relation to the average sample spacing and the search employed. ● Any assumptions behind modelling of selective mining units. ● Any assumptions about correlation between variables. ● Description of how the geological interpretation was used to control the resource estimates. ● Discussion of basis for using or not using grade cutting or capping. ● The process of validation, the checking process used, the comparison of model data to drill hole data, and use of reconciliation data if available.	● The Mineral Resource estimation presented in SRK's report was completed using Surpac software (version 6.3). ● Prior to modeling and estimation, the database was verified by SRK. The geological interpretations made previously in 2013 by SRK and the update made by other parties (Wilton and PT. ASI) have been reviewed. ● Wireframes of mineralised zones were modeled according to the interpretation made based on lithology, assays,structure and geotechnical information. Resource domains (wireframe of mineralisation) in the Ciemas Project were modeled using a cut-off grade of 0.8 g/t Au. ● Sample assays were composited to uniform 1 m length in the mineralised domains (wireframe of mineralised zones). ● Extreme high grades were assessed according to basic statistics of the composite assays and grade capping has been applied to the assays in each mineralised zone/vein. ● Geostatistical analysis has been performed and variograms were modeled with nugget effect and spherical structure(s). ● Ordinary kriging has been employed for the grade estimation where possible, and an inverse distance weighted ("IDW") method was applied to the mineralised zones ordinary kriging was not applicable. ● Detailed parameters for grade estimation are described in this report. ● Visual validation of block grades against drillhole grades; and global statistical validation of the mean composite grades versus block estimates have been applied. SRK is satisfied that the

Criteria	JORC Code explanation	Commentary – Assessment of the Ciemas Project
		estimation generally honored to the drilling data.
Moisture	● Whether the tonnages are estimated on a dry basis or with natural moisture, and the method of determination of the moisture content.	● Specific gravity analysis and grade assaying were conducted on a dry basis, therefore the tonnages were estimated on a dry basis. ● Moisture factor has not been considered for this Mineral Resource estimation.
Cut-off parameters	● The basis of the adopted cut-off grade(s) or quality parameters applied.	● A cut-off grade of 1.0 g/t Au has been applied for the resource estimation with assumptions summarized in Table 5-9 in the report, as below. - Gold price: 1,350 United States dollars (US$) per ounce; - Direct operational cost (C1 Cost) for combined open pit and underground mining: US$ 68 per tonne ore feed; - Mining dilution: 15%; - Mining recovery: 90% - Overall processing and metallurgical recovery: 90%. ● The parameters assumed by SRK are used to test for "reasonable prospects for eventual economic extraction". In SRK's opinion a cut-off grade of 1.0 g/t Au is suitable for the Mineral Resource reporting for the Ciemas Gold Project.
Mining factors or assumptions	● Assumptions made regarding possible mining methods, minimum mining dimensions and internal (or, if applicable, external) mining dilution. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential mining methods, but the assumptions made regarding mining methods and parameters when estimating Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the mining assumptions made.	● A combination of Open Pit and Underground mining was considered for the Ciemas Project, updated from previous scoping level studies where only underground mining was considered. ● The direct mining cost is assumed at US$ 30 per tonne ore mined including the costs of stripping and wastes to be mined or moved.

Criteria	JORC Code explanation	Commentary – Assessment of the Ciemas Project
Metallurgical factors or assumptions	• The basis for assumptions or predictions regarding metallurgical amenability. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider potential metallurgical methods, but the assumptions regarding metallurgical treatment processes and parameters made when reporting Mineral Resources may not always be rigorous. Where this is the case, this should be reported with an explanation of the basis of the metallurgical assumptions made.	• Initial metallurgical testwork has been conducted on the mineralisation of primary ores. Further metallurgical testwork for oxidised ores is in process. • Gravity separation with flotation process was considered previously to extract gold from the ores and gravity separation combined with cyanide in leaching ("CIL") is considered primarily for the project. • A combined processing and metallurgical cost at US$ 20 per tonne of ore feed was assumed when considering the mine economics for determining the resource cut-off grade.
Environmental factors or assumptions	• Assumptions made regarding possible waste and process residue disposal options. It is always necessary as part of the process of determining reasonable prospects for eventual economic extraction to consider the potential environmental impacts of the mining and processing operation. While at this stage the determination of potential environmental impacts, particularly for a greenfields project, may not always be well advanced, the status of early consideration of these potential environmental impacts should be reported. Where these aspects have not been considered this should be reported with an explanation of the environmental assumptions made.	• Environment costs were included in the costs of mining, processing and general & administrative costs. • No other substantial environmental risks were identified or assumed during the Mineral Resource estimation.
Bulk density	• Whether assumed or determined. If assumed, the basis for the assumptions. If determined, the method used, whether wet or dry, the frequency of the measurements, the nature, size and representativeness of the samples. • The bulk density for bulk material must have been measured by methods that adequately account for void spaces (vugs, porosity, etc), moisture and differences between rock and alteration zones within the deposit. • Discuss assumptions for bulk density estimates	• The density of ore was determined according to sample data collected at the project area. • Instead of bulk samples, small volumetric samples weighing about 5 – 10 kg were collected representing both fresh ores and oxidised ores. • As the Company's project development plan has been altered to open pit mining first and followed by underground mining, instead of the sole underground mining option considered previously in 2013, SRK recommends additional bulk density measurements to be conducted for

Criteria	JORC Code explanation	Commentary – Assessment of the Ciemas Project
	used in the evaluation process of the different materials.	the oxidised and supergene zones. ● Average ore density calculated from the known sample results is about 2.7 g/cm^3.
Classification	● The basis for the classification of the Mineral Resources into varying confidence categories. ● Whether appropriate account has been taken of all relevant factors (ie relative confidence in tonnage/grade estimations, reliability of input data, confidence in continuity of geology and metal values, quality, quantity and distribution of the data). ● Whether the result appropriately reflects the Competent Person's view of the deposit.	● The classification of Mineral Resource reflects confidence in the estimation based on both geological continuity and geostatistical analysis. SRK considered both the nature of drilling controls (interceptions) and distance and numbers of informing samples (drillholes). ● Measured Resource was limited to blocks at Pasir Manggu where an approximate grid of 20 m by 20 m was drilled. ● Indicated Resource was defined in those zones intersected generally by drillholes spaced no more than 50 m apart, and for blocks informed by at least 3 holes within a search radius of 100m . ● Inferred Resource was limited to within the area defined by the estimated blocks within the hard boundary of the solid model.
Audits or reviews	● The results of any audits or reviews of Mineral Resource estimates.	● This Mineral Resource estimate is an update of the Mineral Resource estimate in the IQPR prepared by SRK in 2013. Both internal and external peer reviews have been applied to that IQPR. ● 16 Additional drillholes have been integrated to this update. Some more geological information about new-surface mineralisation has been identified and taken into account. Modified assumptions for mining and processing sections have been applied. These are the basis for this update in resource domaining, grade estimation and resource classification. Other than for these changes, the Mineral Resource estimate presented in the IQPR dated July 2013 was the basis of this updated resource report. Changes with respect to the Mineral Resource estimate in the previous IQPR have been assessed by SRK. ● Peer reviews of this resource update have been

Criteria	JORC Code explanation	Commentary – Assessment of the Ciemas Project
		performed within SRK internally. ● SRK's Mineral Resource estimation was compared to the work done by PT. ASI in February 2014, and no significant discrepancies have been identified. ● SRK is not aware of any other audits or reviews undertaken for the Mineral Resource estimation.
Discussion of relative accuracy/ confidence	● Where appropriate a statement of the relative accuracy and confidence level in the Mineral Resource estimate using an approach or procedure deemed appropriate by the Competent Person. For example, the application of statistical or geostatistical procedures to quantify the relative accuracy of the resource within stated confidence limits, or, if such an approach is not deemed appropriate, a qualitative discussion of the factors that could affect the relative accuracy and confidence of the estimate. ● The statement should specify whether it relates to global or local estimates, and, if local, state the relevant tonnages, which should be relevant to technical and economic evaluation. Documentation should include assumptions made and the procedures used. ● These statements of relative accuracy and confidence of the estimate should be compared with production data, where available.	● The geometry of the interpreted orebody did not change significantly with the addition of drillholes for this resource update, although the resource categories may have been upgraded. ● Within certain parts of the deposit, the drill spacing and grade continuity are good enough to allow for a local estimation of the gold grades. ● Some local of antisanal mining (small scale) as observed during SRK's recent site visits in June and July 2014 proved the existence of the mineralisation at places which were intercepted by drilling.

Section 4: Estimation and Reporting of Ore Reserves

Criteria	Commentary
Resource estimate for conversion to Ore Reserves	• The gold Mineral Resource estimated by SRK team in June 2018. • The gold Ore Reserves are reported inclusive of Mineral Resources.
Site visits	• SRK undertook several visits to the Project and the recent visit was done by SRK team consisting geologists, mining engineers and environmental scientists in September 2018.
Study Status	• A report entitled "Feasibility Study Report for Ciemas Gold Project" has been compiled by engineers of SRK, dated July 2018. • The FS report associated by drawings indicates the project could be planned feasibility and viability, by technical and economic parameters/factors designed. • The Ore Reserve conversion has been undertaken from Mineral Resource considering the mine plan that is technically achievable and economically viable, as well as material Modifying Factors.
Cut-off parameters	• The cut-off grade of Au element is estimated based on the costs, forecasted gold price, processing and smelting recovery rates. • The break-even cut-off grade is 3.0g/t for run of mine (underground). • The marginal cut-off is 1.5 g/t for ROM (open pit)
Mining factors or assumptions	• Mechanised cut and fill mining method is proposed for CKD, SEK and CBT Sections. Dual declines access method is planned. the level height is 20m. Mine recovery factor is estimated 88.8%, and dilution factor is 8.6% • Traditional cut and fill mining method is designed for PSM section. Single incline shaft access method is proposed. The level height is 40m. Mine recovery factor is estimated 92.5% and dilution factor is 7.6%. • The overall production capacity of underground mine is 1,500tpd, or 495ktpa. • The selective mining unit ("SMU") is the block size of 4 m×4 m×2 m (X×Y×Z). • The mine schedule is that, 1.5-year construction, followed by a year production ramp-up, then 4 years full capacity production and 2 years ramp-down.
Metallurgical factors or assumptions	• The designed metallurgical flowsheet is "flotation + roasting + CIL", it is suitable for the ore property. Detailed in Sections 11 and 12; • The metallurgical flowsheet includes the operations of primary crushing, semi-autogenous milling, ball milling, flotation, concentrate roasting and acid making, flotation tailing CIL and roasting calcine CIL. Every operation is mature. The whole flowsheet is complex to suit for the recovery of refractory gold. • The metallurgical tests were conducted on different oxide degree of ore samples spreading in the deposit. The results indicate that the gold recovery is widely changed to different sample. In order to suit for all types of ore, the metallurgical plant adopts a combined method which can process various types of ore and achieve a satisfactory gold recovery. • The deleterious (harmful) elements for metallurgy are arsenic and sulphur which occurs as the minerals of arsenopyrite and pyrite. They enclose gold grains resulting in refractory, so flotation and roasting methods are applied to concentrate and decompose the arsenopyrite and pyrite for gold liberation. The liberated gold is easily cyanide leached. • There is no pilot plant test on bulk sample for the full flowsheet; but various laboratory metallurgical tests on different samples can determine the flowsheet is proper to the deposit. Wilton also conducted pilot production with pool leaching in 2017 and gold bullion had been produced. • The overall gold recovery and metallurgical costs have been considered during the conversion of ore reserves. The gold recovery is determined according to a lot of metallurgical tests and the flowsheet developed in the feasibility study.
Environmental	• There are no apparent and inherent environmental issues observed. • Environmental factors which may result a project cost variation have been considered in detailed cost estimates in the FS and this IQPR

Criteria	Commentary
	• Mine closure cost and relevant reclamation expenditures have been considered in cost estimates.
Infrastructure	• Facts which may result a project cost variation have been considered in detailed cost estimates in the FS and this IQPR. • Infrastructure costs have been considered in the cost estimates.
Costs	• The detailed cost estimates have been provided basis of economic considerations for Ore Reserves conversion.
Revenue factors	• The revenue factors are based on the tow products, which are gold concentrate and gold dore, according to the process flows and construction plan. • The assumed payable factor for gold concentrate is 95% for gold contained in that, and the refined charge is USD 190 per tonne concentrate. • The gold dore is assumed recovering 100% gold revenue. • The gold price forecasts are sourced from CONSENSUS MARKET FORECAST released report dated in 20-Apr-2018. The forecasted gold price is from the spot price of USD 1,349 per troy ounce down to USD 1,240 / oz. in 2022 and flat in USD 1,220/oz. since 2023.
Market assessment	• The market assessment has been performed. • Gold dore is the final product for the project; and the saleable concentrate will be part of the product in the first stage prior to the construction of roasting plant.
Economic	• The capital costs/operating costs forecasts are based on a baseline estimated by SRK in the feasibility study report. • The analysis shows that the after-tax (25% corporate income tax) NPV, at a discount rate of 9%, is estimated USD 180.1 million. The IRR is 43.0% and static payback period is 3.4 years. • The positive NPV suggests Ore Reserve defined is economically viable.
Social	• Section 15, no major social issues/ impact • The potential cost and/or tax have been considered in the cost estimates
Other	• Others detailed in recent FS • Other factors which may result a project cost variation have been considered in detailed cost estimates in the FS and this IQPR
Classification	• The classification is based on the resource category and the level of design by considering modifying factors
Audits or reviews	• Update of Previous report in 2013 • No other reviews
Discussion of relative accuracy/ confidence	• The accuracy of the Capex estimates is above 85% and is discussed in Section 17.1. • The operating cost estimates have been based on assumptions where SRK considered appropriate. The result is considered comparable with benchmark of similar projects.

Appendix 2: Resource and Reserve Summary Table

1. Pasir Manggu- JORC (2012) compliant resources as of 30 June 2018

Category	Mineral Type	Gross Attributable to Licence		Net Attributable to Issuer			Remarks
		Tonnes '000t	Grade (g/t Au)	Tonnes '000t	Grade (g/t Au)	Change [1] (%)	
Ore Reserves							
Proved							No Ore Reserves Reported in 2014
Probable	Quartz Vein Gold	587	6.6	587	6.6		
Total		587	6.6	587	6.6		
Mineral Resources [2]							
Measured	Quartz Vein Gold	100	7.3	100	7.3	-16.0	at cut-off grade 1.0 g/t Au
Indicated	Quartz Vein Gold	489	7.3	489	7.3	5.0	
Inferred	Quartz Vein Gold	242	4.9	242	4.9	14.0	
Total		831	6.6	831	6.6	3.3	

[1] Change from previous update as of 30 June 2014, changes are relative to contained metal as estimated; positive number denotes increase and negative number denotes decrease.

[2] Mineral Resources are inclusive of Ore Reserves.

2. Cikadu - JORC (2012) compliant resources as of 30 June 2018

Category	Mineral Type	Gross Attributable to Licence		Net Attributable to Issuer			Remarks
		Tonnes '000t	Grade (g/t Au)	Tonnes '000t	Grade (g/t Au)	Change [1] (%)	
Ore Reserves							
Proved							No Ore Reserves Reported in 2014
Probable	Structurally Altered Gold	986	8.0	986	8.0		
Total		986	8.0	986	8.0		
Mineral Resources [2]							
Measured							at cut-off grade 1.0 g/t Au
Indicated	Structurally Altered Gold	1,089	8.8	1,089	8.8	-3.5	
Inferred	Structurally Altered Gold	299	9.5	299	9.5	-6.2	
Total		1,388	9.0	1,388	9.0	-4.1	

[1] Change from previous update as of 30 June 2014, changes are relative to contained metal as estimated; positive number denotes increase and negative number denotes decrease.

[2] Mineral Resources are inclusive of Ore Reserves.

3. Sekolah - JORC (2012) compliant resources as of 30 June 2018

Category	Mineral Type	Gross Attributable to Licence		Net Attributable to Issuer			Remarks
		Tonnes '000t	Grade (g/t Au)	Tonnes '000t	Grade (g/t Au)	Change [1] (%)	
Ore Reserves							
Proved							No Ore Reserves Reported in 2014
Probable	Structurally Altered Gold	679	8.1	679	8.1		
Total							
Mineral Resources [2]							
Measured							at cut-off grade 1.0 g/t Au
Indicated	Structurally Altered Gold	700	9.1	700	9.1	-2.2	
Inferred	Structurally Altered Gold	453	7.3	453	7.3	28.3	
Total		1,154	8.4	1,154	8.4	6.5	

[1] Change from previous update as of 30 June 2014, changes are relative to contained metal as estimated; positive number denotes increase and negative number denotes decrease.

[2] Mineral Resources are inclusive of Ore Reserves.

4. Cibatu - JORC (2012) compliant resources as of 30 June 2018

Category	Mineral Type	Gross Attributable to Licence		Net Attributable to Issuer			Remarks
		Tonnes '000t	Grade (g/t Au)	Tonnes '000t	Grade (g/t Au)	Change [1] (%)	
Ore Reserves							
Proved							No Ore Reserves Reported in 2014
Probable	Structurally Altered Gold	1,008	7.9	1,008	7.9		
Total							
Mineral Resources [2]							
Measured							at cut-off grade 1.0 g/t Au
Indicated	Structurally Altered Gold	1,036	8.7	1,036	8.7	50.4	
Inferred	Structurally Altered Gold	455	7.0	455	7.0	-42.9	
Total		1,491	8.2	1,491	8.2	5.4	

[1] Change from previous update as of 30 June 2014, changes are relative to contained metal as estimated; positive number denotes increase and negative number denotes decrease.

[2] Mineral Resources are inclusive of Ore Reserves.

5. Cibak and Cipancar - JORC (2012) compliant resources as of 30 June 2018

Category	Mineral Type	Gross Attributable to Licence		Net Attributable to Issuer			Remarks
		Tonnes '000t	Grade (g/t Au)	Tonnes '000t	Grade (g/t Au)	Change [1] (%)	
Ore Reserves							
Proved							No Ore Reserves Reported in 2016
Probable							
Total							
Mineral Resources							
Measured							at cut-off grade 2.5 g/t Au
Indicated							
Inferred	Structurally Altered Gold and Quartz Vein	1,110	5.6	1,110	5.6	0.1	
Total		1,110	5.6	1,110	5.6	0.1	

[1] Change from previous update as of 31 August 2016, changes are relative to contained metal as estimated; positive number denotes increase and negative number denotes decrease. The change of 0.1% reported in metal quantity is due to the figure rounding discrepancies.

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Appendix 3: Indonesian Environmental Legislative Background

The Indonesian National Law on Mineral and Coal Mining (No.4 of 2009) ('Mining Law'), the Regulation for the Implementation of Mining Areas (No.22 of 2010), ('Mining Area Regulations') and the Regulation for the Implementation of Mineral and Coal Mining Business Activities (No.23 of 2010) ('Mining Regulations'), provide the main legislative framework for the administration and regulation of mining projects within Indonesia. The Law on Environmental Protection and Management (No.32 of 2009) ('Environmental Law') provides the main legislative framework for the regulation and administration of mining projects environmental impacts.

Mining Areas are those areas designated by the Central Government as 'open for mining'. These 'designated mining areas' are referred to as *Wilayah Pertambangan* (WP) and occur in the following three categories:

- Commercial mining business areas – *Wilayah Usaha Pertambangan* (WUP), are mining areas for larger scale mining.
- State reserve areas – *Wilayah Pencadangan Negara* (WPN), are mining areas reserved for the national strategic interest.
- People's mining areas – *Wilayah Pertambangan Rakyat* (WPR), are mining areas for small scale local mining.

Within these designated mining areas, mining licences may be issued under the following three categories:

- Mining Business Licence – *Izin Usaha Pertambangan* (IUP) is a general mining licence for conducting mining business activities within a WUP mining area.
- Special Mining Business Licence – *Izin Usaha Pertambangan Khusus* (IUPK) is a licence for conducting mining business activities within a specific WPN mining area.
- People's Mining Licence – *Izin Pertambangan Rakyat* (IPR) is a licence granted to Indonesian citizens/invertors only for conducting mining business of a limited size and investment, within a WPR mining area.

Both the Mining Law and the Environmental Law require mining companies that are developing projects that are deemed to have significant potential environmental and/or social impacts, to produce an environmental impact assessment and planning document *Analisa Mengenai Dampak Lingkungan* (AMDAL). An AMDAL consists of an environmental impact assessment, an environmental management plan and an environmental monitoring plan. An 'environmental management effort document', Upaya Pengelolaan Lingkungan (UPL) and Upaya Pengawasan Lingkungan (UKL) generally need to be prepared in any situation where it is deemed that an AMDAL is not required.

The following are further Indonesian laws, regulations, presidential decrees and statutes that provide environmental legislative support to the Mining Law/Regulations and the Environmental Law:

- The Law on Forestry (No.41 1999)
- Government Regulation (No. 24 2010) – regarding utilisation of forest areas
- Government Regulation (No. 78 2010) – concerning reclamation and post-mining
- Regulation of the Minister of Forestry (No.18 2011) Guidelines for Use of Forest Areas (Lend Use Permitting in Production Forest Areas and Protected Forest Area)
- Government Regulation (Presidential decree) (No.28 2011) – on the use of protected forest areas for underground mining
- Environmental Impact Assessment, Types of Businesses or Activities Required to Prepare (MOE Decree No.11, 1994)
- Environmental Management and Monitoring Procedures, Guidelines for (MOE Decree No.12, 1994)
- Indonesia: Environmental Regulations of Indonesia (Circular No.3 of 1987)

- Water Pollution, Control of (Gov’t Reg. No.20, 1990)
- Hazardous and Toxic Waste Management, Regulation Regarding (Gov’t Reg. No.19 1994)
- Hazardous and Toxic Wastes, Amendment of Regulation Regarding Handling (Gov’t Reg. No.12 1995)
- Environmental Impact Assessment, Regulation Regarding (Gov’t Reg. No.51 1993)
- Environmental Management and Monitoring Procedures, Guidelines for (MOE Decree No.12 1994)
- Hazardous and Toxic Waste Management, Regulation Regarding (Gov’t Reg. No.19 1994)
- Hazardous and Toxic Wastes, Amendment of Regulation Regarding Handling (Gov’t Reg. No.12 1995)

Appendix 4: Equator Principles and Internationally Recognised Environmental Management Practices

In seeking to obtain project financing or to list on a stock exchange, these institutions require the proponent to comply with such documents as the Equator Principles and the International Finance Corporation (IFC) Performance Standards and Guidelines. This is exemplified by the following preamble from the Equator Principles (July 2006):

Project financing, a method of funding in which the lender looks primarily to the revenues generated by a single project both as the source of repayment and as security for the exposure, plays an important role in financing development throughout the world. Project financiers may encounter social and environmental issues that are both complex and challenging, particularly with respect to projects in emerging markets.

The Equator Principles Financial Institutions (EPFIs) have consequently adopted these Principles in order to ensure that the projects we finance are developed in a manner that is socially responsible and reflect sound environmental management practices. By doing so, negative impacts on project-effected ecosystems and communities should be avoided where possible, and if these impacts are unavoidable, they should be reduced, mitigated and/or compensated for appropriately. We believe that adoption of and adherence to these Principles offers significant benefits to ourselves, our borrowers and local stakeholders through our borrowers' engagement with locally affected communities. We therefore recognise that our role as financiers affords us opportunities to promote responsible environmental stewardship and socially responsible development. As such, EPFIs will consider reviewing these Principles from time-to-time based on implementation experience, and in order to reflect ongoing learning and emerging good practice.

These Principles are intended to serve as a common baseline and framework for the implementation by each EPFI of its own internal social and environmental policies, procedures and standards related to its project financing activities. We will not provide loans to projects where the borrower will not or is unable to comply with our respective social and environmental policies and procedures that implement the Equator Principles.

The following Tables provide a brief summary of the Equator Principles and the IFC Performance Standards respectively. These documents are used by the EPFI's and stock exchanges in their review of the social and environmental performance of proponent companies.

Table A4-1: Equator Principles

Equator Principles	Title	Key Aspects (Summary)
1	Review and Categorisation	Categorise such project based on the magnitude of its potential impacts and risks
2	Social and Environmental Assessment	Conduct a Social and Environmental Assessment ("Assessment"). The Assessment should also propose mitigation and management measures appropriate to the nature and scale of the proposed project.
3	Applicable Social and Environmental Standards	The Assessment will refer to the applicable IFC Performance Standards, and applicable Industry Specific EHS Guidelines ("EHS Guidelines") and overall compliance with same.
4	Action Plan and Management System	Prepare an Action Plan (AP) which addresses the relevant findings of the Assessment. The AP will describe and prioritise the actions, mitigation measures, corrective actions and monitoring to manage the impacts and risks identified in the Assessment. Maintain a Social and Environmental Management System that addresses the management of these impacts, risks, and corrective actions required to comply with host country laws and regulations, and requirements of the applicable Standards and Guidelines, as defined in the AP.
5	Consultation and Disclosure	Consult with project affected communities. Adequately incorporate affected communities' concerns.
6	Grievance Mechanism	Establish a grievance mechanism as part of the management system. to receive and resolve concerns about the project by individuals or groups from among project-affected communities. Inform the affected communities about the grievance mechanism in the course of the community engagement process and ensure that the mechanism addresses concerns promptly and transparently, and is readily accessible to all segments of the affected communities.
7	Independent Review	Independent social or environmental expert will review the Assessment, AP and consultation process to assess Equator Principles compliance.
8	Covenants	Covenant in financing documentation: a) to comply with all relevant host country social and environmental laws, regulations and permits; b) to comply with the AP during the construction and operation of the project; c) to provide periodic reports not less than annually, prepared by in-house staff or third party experts, that (i) document compliance with the AP, and (ii) provide compliance with relevant local, state and host country social and environmental laws, regulations and permits; and d) to decommission the facilities, where applicable and appropriate, in accordance with an agreed decommissioning plan.
9	Independent Monitoring and Reporting	Appoint an independent environmental and/or social expert, or require that the borrower retain qualified and experienced external experts to verify its monitoring information.
10	EPFI Reporting	Each EPFI adopting the Equator Principles commits to report publicly at least annually about its Equator Principles implementation processes and experience, taking into account appropriate confidentiality considerations.

Table A4-2: IFC Performance Standards

IFC Performance Standard	Title	Objective (Summary)	Key Aspects (Summary)
1	Social and Environmental Assessment and Management Systems	Social and EIA and improved performance through use of management systems.	Social & Environmental Management System (S&EMS). Social & Environmental Impact Assessment (S&EIA). Risks and impacts. Management Plans. Monitoring. Reporting. Training. Community Consultation
2	Labour and Working Conditions	EEO. Safety and Health	Implement through the S&EMS. HR policy. Working condition. EEO. Forced & child labour. OH&S.
3	Pollution Prevention and Abatement	Avoid pollution. Reduce Emissions.	Prevent pollution. Conserve resources. Energy efficiency. Reduce waste. Hazardous materials. EPR. Greenhouse Gases
4	Community Health, Safety and Security	Avoid or minimise risks to community.	Implement through the S&EMS. Do risk assessment. Hazardous materials safety. Community exposure. ERP
5	Land Acquisition and Involuntary Resettlement	Avoid or minimise resettlement. Mitigate adverse social impacts	Implement through the S&EMS. Consultation. Compensation. Resettlement planning. Economic displacement
6	Biodiversity Conservation and Sustainable Natural Resource Management	Protect and conserve biodiversity	Implement through the S&EMS. Assessment. Habitat. Protected areas. Invasive species.
7	Indigenous Peoples	Respect. Avoid and minimise impacts. Foster good faith	Avoid adverse impacts. Consultation. Development benefits. Impacts to traditional land use. Relocation.
8	Cultural Heritage	Protect cultural heritage	Heritage Survey. Site avoidances. Consultation.

Summary Background Information on Some Key Internationally Recognised Environmental Management Practices.

The following provides background information on some key internationally recognised environmental management practices:

- **Land disturbance** – The main impact on the surrounding ecological environment is due to disturbance and contamination caused by surface stripping, waste rock and tailings storage, processing plant drainage, processing waste water, explosions, transportation and associated buildings that are erected. If effective measures are not taken to manage and rehabilitate the disturbed areas, the surrounding land can become polluted and the land utilization function will be changed, causing an increase in land degradation, water loss and soil erosion.
- **Flora and fauna** – Land disturbance from the development of mining and mineral processing projects may also result in impacts to or loss of flora and fauna habitat. The project development EIA should determine the extent and significance of any potential impacts to flora and fauna habitat. Where these potential impacts to flora and fauna habitat are determined to be significant, the EIA should also propose effective measures to reduce and manage these potential impacts.
- **Contaminated Sites Assessment** – The assessment, recording and management of contaminated sites within mining or mineral processing operations, is a recognised international industry practice (i.e. forms part of the IFC Guidelines) and in some cases a National regulatory requirement (e.g. an Australian environmental regulatory requirement). The purpose of this process is to minimise the level of site contamination that may be

generated throughout a project's operation while also minimising the level and extent of site contamination that will need to be addressed at site closure.

- A contaminated site or area can be defined as; '*An area that has substances present at above background concentrations that presents or has the potential to present a risk of harm to human health, the environment or any environmental value*'.
- Contamination may be present in soil, surface water or groundwater and also may affect air quality through releases of vapours or dust. Examples of typical contaminated areas within a mining/mineral processing project are spillages to soil/water of hydrocarbons and chemicals, and uncontained storage and spillages to soil/water of ores and concentrates. The process to assess and record the level of contamination basically involves a combination of visual (i.e. suspected contamination observed from spillages/releases) and soil/water/air sampling and testing (i.e. to confirm contaminant levels). Once the level of contamination is defined, the area's location and contamination details are then recorded within a site register.
- Remediation/clean up of contamination areas involves the collection and removal of the contaminated materials for treatment and appropriate disposal, or in some cases the in-situ treatment of the contaminated (e.g. use of bioremediation absorbents on hydrocarbon spillage). The other key component to the management of contaminated areas is to also remove or remedy the source of the contamination (e.g. place hydrocarbon storage and handling within secondary containment).

- **Environmental Protection and Management Plan** – The purpose of an operational Environmental Protection and Management Plan (EPMP) is to direct and coordinate the management of the project's environmental risks. The EPMP documents the establishment, resourcing and implementation of the project's environmental management programs. The site environmental performance is monitored and feedback from this monitoring is then utilised to revise and streamline the implementation of the EPMP.
- **Emergency Response Plan** – The IFC describes an emergency as 'an unplanned event when a project operation loses control, or could lose control, of a situation that may result in risks to human health, property, or the environment, either within the facility or in the local community'. Emergencies are of a scale that have operational wide impacts, and do not include small scale localised incidents that are covered under operational area specific management measures. Examples of an emergency for a mining/mineral processing project are events such as pit wall collapse, underground mine explosion, the failure of a TSF or a large scale spillage/discharge of hydrocarbons or chemicals. The recognised international industry practice for managing emergencies is for a project to develop and implement an Emergency Response Plan (ERP). The general elements of an ERP are:
-
- **Administration** – policy, purpose, distribution, definitions of potential site emergencies and organisational resources (including setting of roles and responsibilities).
- **Emergency response areas** – command centres, medical stations, muster and evacuation points.
- **Communication systems** – both internal and external communications.
- **Emergency response procedures** – work area specific procedures (including area specific training).
- **Checking and updating** – prepare checklists (role and action list and equipment checklist) and undertake regular reviews of the plan.
- **Business continuity and contingency** – options and processes for business recovery from an emergency.
- **Site Closure Planning and Rehabilitation** – The recognised international industry practice for managing site closure is to develop and implement an operational site closure planning process and document this through an operational Closure Plan. This operational closure planning process should include the following components:
- Identify all site closure stakeholders (e.g. government, employees, community etc.).

- Undertake stakeholder consultation to develop agreed site closure criteria and post operational land use.
- Maintain records of stakeholder consultation.
- Establish a site rehabilitation objective in line with the agreed post operational land use.
- Describe/define the site closure liabilities (i.e. determined against agreed closure criteria).
- Establish site closure management strategies and cost estimates (i.e. to address/reduce site closure liabilities).
- Establish a cost estimate and financial accrual process for site closure.
- Describe the post site closure monitoring activities/program (i.e. to demonstrate compliance with the rehabilitation objective/closure criteria).

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