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Flowsheet development for the Sukari gold project in Egypt

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The Sukari project, which is owned by Centamin Egypt Ltd, is located in southern Egypt, 770 km south of Cairo but close to the Red Sea. The project is the first new gold development in Egypt and was commissioned in the middle of 2009.

In 2007, Centamin purchased the CIL plant from the Kori Kollo mine in Bolivia. This plant was originally designed to treat 6.5 Mtpa of oxide ore.

The Sukari deposit contains an oxide zone close to surface with a thick zone of mixed oxide and sulphides below this. The sulphides contain relatively coarse pyrite and arsenopyrite with fine <10 μm gold inclusions. Eventually, the sulphide ore will be mined from underground.

Metplant Engineering Pty Ltd, a subsidiary of Bateman Engineering Pty Ltd was contracted to design modifications and additions to the Kori Kollo plant to enable it to treat the Sukari ores.

The plant was designed to treat the three ore types in stages. Throughput was set to 4 Mtpa, due to Sukari ore being harder, at a head grade of 1.7 g/t to produce 200 000 oz of gold.

The oxide circuit was based on the use of the Kori Kollo plant and the difference with mixed ore was that the oxide circuit would treat flotation tailings rather than ore. This circuit has a leach capacity of 26 000 m^3 and residence time of 24 hrs

The main feature of the upgraded mixed ore plant was the addition of a flotation circuit to produce a concentrate which would be reground and treated in a separate leach and elution circuit.

Ore is crushed in a gyratory crusher and the comminution circuit uses a conventional SABC producing a product with a P_{80} of 150 microns. The throughput of 580 tph of oxide ore and 500 tph of mixed ore was set to match the mills from Kori Kollo, with the addition of the pebble crusher to accommodate the more competent Sukari ore.

The flotation circuit comprises a single rougher circuit to maximize recovery. The concentrate is thickened prior to regrinding which is carried out in a tower mill and two SMD mills operating in series. The final product, with a P_{80} of 12 microns, from the regrind circuit is leached in a CIL circuit with a total capacity of 2 550 m^3. Gold is recovered through a conventional pressure Zadra circuit.

Tailings from both circuits report to separate tailings thickeners and are pumped to a tailings storage facility. The circuit is designed to maximize the recovery of water.

The project lies in one of the driest deserts in the world and a feature of the operation is the use of sea water from the Red Sea. This water is pumped 25 km from the coast.

Background

The Sukari Hill gold mine is owned by Pharaoh gold mines (PGM), a 100% owned subsidiary of Centamin Egypt. The company has been exploring in Egypt since 1995. In May 2005, the company was granted a 160 km^2 exploration lease of the massive Sukari Hill project for 30 years.

The gold resource at Sukari had been exploited by the Pharaohs and the Romans, but the last previous workings had been by the British in the 1950's. The Sukari Project will be the first modern gold mine in Egypt.

In 2007, PGM purchased the Kori Kollo gold plant from Bolivia and moved it to Egypt.

In April 2007, Metplant Engineering Pty Ltd, a subsidiary of Bateman Engineering Pty Ltd, was contracted to design modifications and additions to the Kori Kollo plant to enable it to treat the Sukari ores. Metplant were responsible for the procurement of the new equipment and Centamin the construction and management.

Resource

In February 2007 Pharaoh Gold Mines completed a definitive feasibility study (DFS) which defined a mining reserve of 78.8 million tonnes at 1.47 g/t. The DFS study concluded that a plant operating at 4 Mtpa and producing 200 000 oz of gold per annum was economically robust. At the time of the DFS total construction costs were estimated at US$M216.

By February 2009, the measured and indicated resource had been increased to 191 Mt at 1.5 g/t.

Ore type

The Sukari ore body has an oxide zone at surface below which the ore is a mixture of oxides and sulphides. The ore then becomes predominantly sulphide. The host rock is a granitoid body of granodiotite of tonalite composition, referred to as Sukari Porphyry.

Mineralogy

Samples of the ore used for metallurgical testwork were examined to determine the nature of the gold and sulphide minerals. The conclusions drawn were:

- Sulphide minerals are present with an average sulphur content of 1%
- Gold occurs as native and argentian gold as fine inclusions in pyrite and arsenopyrite with pyrite the most common sulphide mineral
- Gold grains were observed from 0.5 to 15 μm in pyrite ranging from 40 to 180 μm
- A single gold grain was observed in arsenopyrite.

Mining

The ore will be mined by conventional open pit methods, although studies are underway to assess the potential for underground mining of higher grade deeper zones.

Testwork

Between 2000 and 2006, PGM had leaching and flotation testwork carried out at different metallurgical laboratories. The samples tested from 2000 to 2003 were, however, not considered representative and, consequently, a new programme was initiated.

Testwork conducted in 2005 was performed on a composite sample assaying 1.7 g/t gold and 1.1% total sulphur. This sample was representative of the ore body and indentified that a process which adopted flotation, followed by concentrate regrind and CIL as the preferred process route. The composite samples and products from the testwork were used to establish parameters for the plant design.

Variability testwork was conducted to confirm that the circuit was suitable for different ore types and to establish a range of physical parameters in particular obtaining work indices for a range of materials.

The Sukari drill samples were not assayed for sulphur, so a study was conducted to compare geological logging and metallurgical response. The results showed that the degree of oxidation established by the geologists was a suitable method to predict the performance.

A subsequent series of tests was conducted at a different laboratory in 2006 in which composite samples based on mineralogical composition. The tests confirmed that metallurgical performance was related to degree of oxidation. The products from the tests were also used to further investigate regrinding of the concentrate and leaching of the flotation tailings.

Kori Kollo plant

PGM made a decision to purchase the Kori Kollo gold plant from Bolivia. The Kori Kollo plant was built in 1993 and was designed to process 6.5 Mtpa of ore through a CIL plant.

A heap leach operation continued at Kori Kollo so the purchase did not include the elution gold room, carbon regeneration or reagent sections. The key equipment list given in Table III specifies the items that were used from the Kori Kollo plant and those purchased.

Once the plant was dismantled, some items, in particular the screens, the flocculant system and some pumps, were found to be in poorer condition than originally assessed. These items were subsequently replaced.

Design

Design basis

Based on the ore type, the plant was designed to be operated in three stages; however, the total plant will be installed from Day 1.

Engineering

The main engineering challenges were incorporating the new equipment into the Kori Kollo plant.

The original Kori Kollo plant comprised a crushing and grinding circuit, a leach circuit and an elution and gold recovery circuit. The Kori Kollo plant was commissioned in 1993.

Crushing

The Kori Kollo gyratory crushers power was considered too small for the Sukari ore and an alternative second hand crusher with a 375 kW motor was purchased from Spain.

A new section of the crushing plant was built to incorporate the gyratory and existing apron feeder. The ore from the crusher feeds onto a 74 m cross conveyor then onto a 167 metre conveyor feeding a 15 000 tonne stockpile. The longer conveyor was from the Kori Kollo plant.

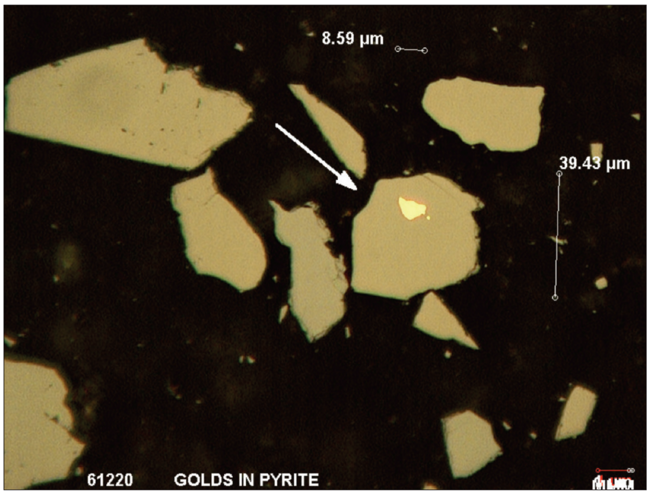

Figure 1. Roger Townend: occurrence of gold in Pyrite

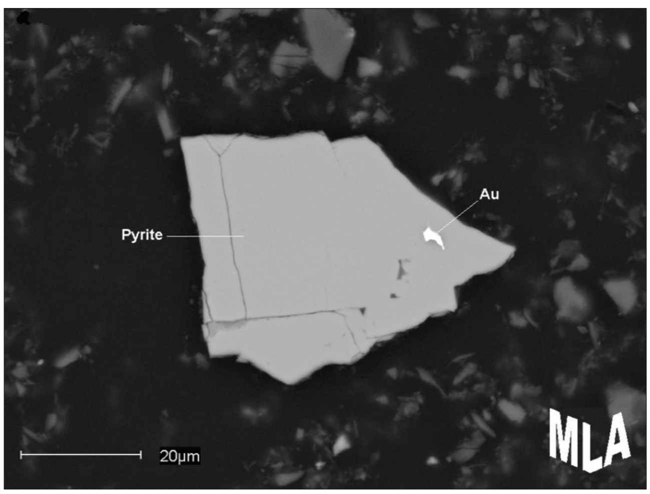

Figure 2. JK Tech: `mineral liberation analyser gold deportment in Pyrite

Table I
Key process design criteria

Criteria	Units	Stage 1	Stage 2	Stage 3
Throughput	Mtpa	4.6	4.0	4.0
Throughput	tph	580	500	500
Plant availability	%	91.3	91.3	91.3
Head grade	g/t Au	1.55	1.76	1.83
Comminution				
Crushing work index - range	kWh/t	4.3 - 40	4.3 - 40	4.3 - 40
Rod mill work index - design	kWh/t	23.2	23.2	23.2
Ball mill work index - design	kWh/t	16.2	19.1	19.1
Cyclone overflow P_{80}	µm	150	150	150
Cyclone overflow solids density	%	43	42	42
Concentrate regrind P_{80}	µm	N/A	12	12
Flotation				
Feed solids density	%	N/A	35	35
Residence time	min	N/A	32	32
Concentrate production - average	t/h	N/A	25	25
Concentrate production - design	t/h	N/A	40	40
Gold recovery	%	N/A	71	96
Collector - PAX	g/t	N/A	75	75
Promoter – A404	g/t	N/A	50	50
Frother	g/t	N/A	20	20
Leach				
Oxide / flotation tails leach / CIL residence time	hrs	24	34	N/A
Flotation concentrate leach / CIL residence time	hrs	N/A	43	43
NaCN – oxide / tails	kg/t	1.74	0.6	
NaCN – concentrate	kg/t		3.35	3.35
Dewatering				
Oxide / tails specific settling rate	$t/m^2/h$	1.4	1.4	1.4
Concentrate specific settling rate	$t/m^2/h$	N/A	0.25	0.25
Flocculant addition - total	g/t	20	20	20

Table II
Predicted metallurgy

Parameter	Units	Stage 1	Stage 2	Stage 3
Throughput	tph	580	500	500
Head grade	g/t	1.55	1.76	1.83
Overall recovery				
Oxide / tails	%	90.2	18.1	
Flotation concentrate	%		69.3	89.7
Gold production				
Gold production - Oxide / tails	000's oz/ a	209	44	
Gold production - concentrate	000's oz/ a		154	212

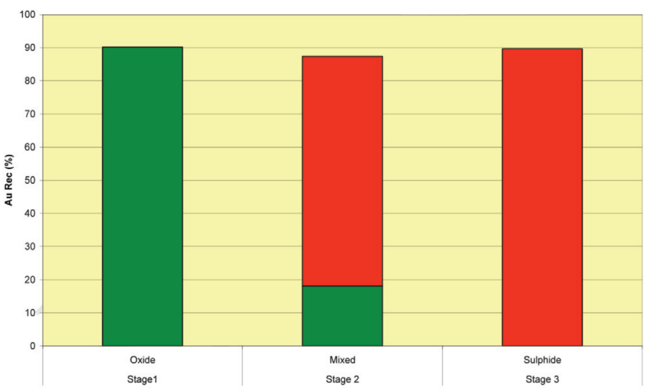

Figure 3. Overall gold recovery by ore type

Grinding

The Sukari ore is harder than the Kori Kollo ore. Consequently a number of changes resulted. The grinding circuit was modeled by SMCC Ltd.

The work index of 19.1 kWh/t for mixed and sulphide ores was established at the 85th percentile of the results from a range of different rock types.

One of the samples tested was a sample designated weathered ore. The work index of this material, at 16.2 kWh/t, was used as the figure for oxide ore.

The feed to the grinding circuit was established at an 80% passing 113 mm based on the size of the product from the gyratory and the product P_{80} 150 microns derived from the leach and flotation testwork.

- A decision was made to use the grinding circuit without any additional milling capacity. This restricted the throughput to 4 Mtpa of mixed and sulphide ore. This throughput served as the basis for the DFS study in 2007
- A pebble crusher was required in the circuit to break the more competent scats from the SAG mill
- As the oxide ore is softer, the throughput will be increased to 4.6 Mtpa in stage 1.

To achieve the required throughput the SAG mill ball load was set at 15% and the pebble port size was increased to 80 mm. The mill manufacturer FFE (now FL Smidth) confirmed that the mill could be operated with a 15% charge.

Table III
Key equipment

Equipment	Size and make	Source
Primary crusher	Allis Chalmers 54” x 74” gyratory - 375 kW	Spain S/H
SAG mill	FFE - 8.53 m dia x 4.27m high aspect – 5600 kW	KK
Ball mills	2 off - FFE 5.03 m x 9.29m overflow – 4100 kW	KK
Pebble crusher	Metso HP 500 – 375 kW short head cone crusher	New
Cyclones	24 off - 380 mm Krebs	KK
Oxide tails leach and CIL tanks	8 off 15.85 m x 16.46 m 3 050 m^3 tanks fitted with Kemix intertank screens	KK
Tails thickener	22.9 m dia Enviroclear	KK
Final tails thickener	23.0 m dia Outotec	New
Flotation	6 off 100 m^3 Outotec tank cells	New
Concentrate thickener	14.0 m dia Outotec thickener	New
Regrind mills	1 off 1 250 VTM Metso Vertimill – 932 kW	New
	2 off Stirred Mill Detritor – 355 kW	New
Concentrate leach and CIL tanks	9 off 6.9 m dia x 7.6 m 284 m^3 tanks	New
Oxide/ tails leach elution	8 ton column	New
Concentrate leach elution	2 ton column	New
Electrowinning oxide / tails	4 off 800 x 800 mm – 12 cathode cells	New
Electrowinning concentrate	3 off 800 x 800 mm – 12 cathode cells	New
Carbon regeneration kiln	Anzac 600 kg/h	New

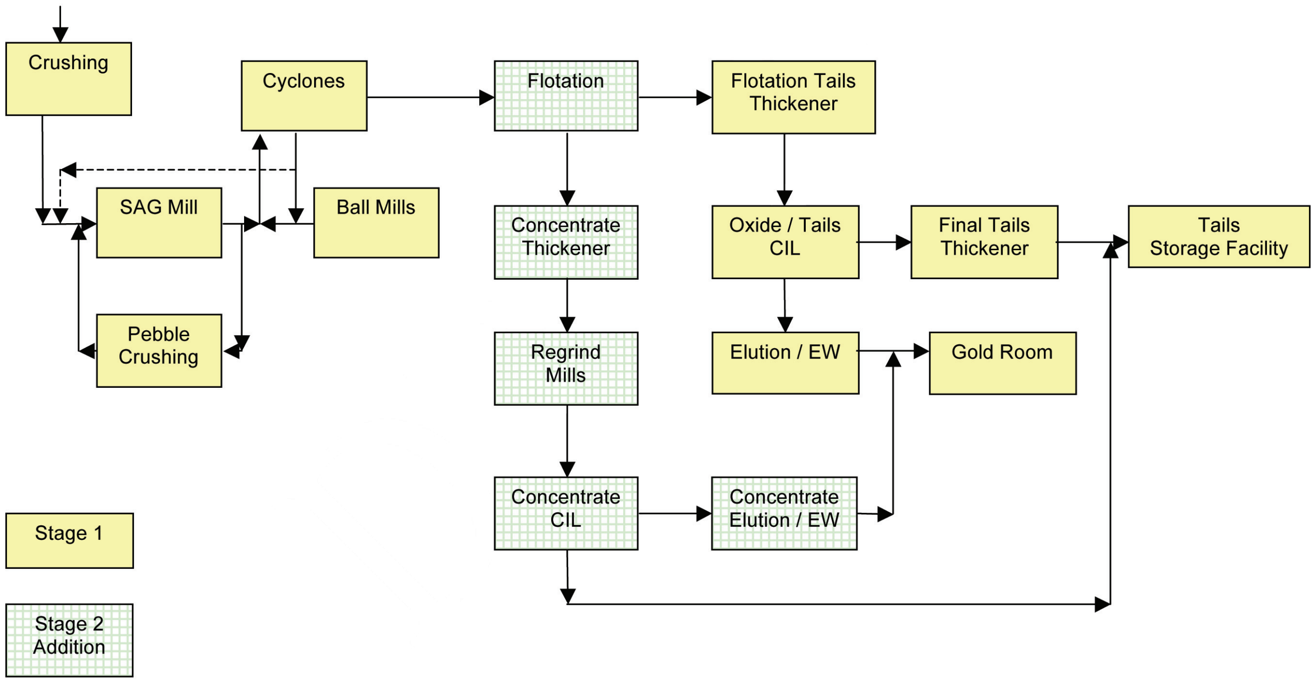

Figure 4. Stage 2 mixed ore schematic flowsheet

The pebble crusher feed rate was estimated to be 120 to 140 tph, but with the potential for feed rates to be 50% above this. An HP 500 crusher capable of treating up to 240 tph was, therefore, selected for the duty.

The 24 x 380 mm cyclones ex Kori Kollo, were deemed to be suitable for the circuit.

Oxide/tails leaching

In stages 1 and 2, first the oxide ore and then the flotation tails will be leached. The configuration of the plant will be one leach tank followed by 7 CIL tanks with a total capacity of 26 000 m^3. The leach time will be 24 hours for the oxide ore and 34 hours for the tailings. The 7 stages of adsorption result in minimum solution losses with a predicted gold tenor of 0.02 g/l.

Air lifts will be used for carbon advance and carbon loadings are designed at 2500 g/t.

The Kemix intertank screens that were used on the Kori Kollo plant have been re used.

Flotation

The sulphide ore responds readily to standard flotation techniques. The simple, rougher circuit operates at a natural pH of approximately 8. Potassium amyl xanthate (PAX) was selected as the collector and Aero 404 as a promoter which enhanced gold recovery in testwork. A series of tests showed that copper sulphate increased the weight recovery but did not improve the gold recovery.

On the sulphide ore, the gold recovery was 96% and the sulphide sulphur recovery 97%.

Weight recovery

The average mass pull at a head grade of 1% sulphur was 5% which equates to 25 tph. The maximum design was set at an 8% pull which was based on a sulphur grade of 1.8% which was at the 90th percentile of core assays. This throughput was most significant in sizing the regrind mill circuit.

The gold grade of the concentrate was 25 and 35 g/t for stages 2 and 3, respectively. The copper of the concentrate is below 200 ppm so the cyanide consumption will not be affected.

Frother

The original frother selected for the process was methyl isobutyl carbinol (MIBC). As MIBC has a flash point of 43°C it requires additional safety measures including explosion proof motors and switches to be employed when designing the reagent circuit. Tests conducted with alternate frothers showed that Huntsman Polyfroth W22 gave an equivalent result to MIBC. W22 has a flash point of 71°C and, therefore, was selected as the installation cost was reduced and safety enhanced.

Leaching of flotation tailings

The residue grade from the flotation averaged 0.5 g/t Au. The tailings were subject to cyanidation. The testwork showed that an economical recovery of 71% could be achieved in 24 hours.

Regrinding

The gold is finely encapsulated, but not in solid solution, therefore, regrinding was investigated as the method for improving the recovery from the sulphides.

A selection of the key regrind tests from three sets of metallurgical testwork is given in the Table V.

On the basis of the initial testwork, 10 microns was selected as the P_{80} for the regrind.

Testwork by manufacturer, however, showed that regrinding in an Isa Mill would be limited to 12 μm. This was coarser than the targeted grind of 10 μm. A set of comparative tests was therefore conducted to determine whether there was any difference in grinding to 12 μm. These results are included in Table V for comparison.

The conclusions reached were that the gold extraction at 12 μm was not less than at 10 μm and that minor differences of less than 1% were due to assay variations. The tests showed similar reagent consumptions were required at both grind sizes. On this basis a regrind size of 80% passing 12 μm was adopted for the plant design.

Discussions were held with users of both the horizontal and vertical mills. The opportunity to conduct testwork on the regrind options was restricted however due to the relatively large amounts of concentrate required not being available. The final decisions were therefore based on suppliers' assessments and discussions with other operations using the mills.

Horizontal mill

Net energy consumption

The manufacturer determined that the power required to regrind the concentrate was as per Table V. The two stages of feed size are due to different estimates of the rougher concentrate sizing from different sources. A sample submitted for testing was much finer than expected so the results were not directly used in the evaluation. The net energy consumption (NEC) is based on using sand as a media. The use of ceramic media will reduce the power requirement. A single 3 000 kW mill was selected for the duty.

The manufacturer's calculations also highlighted that increasing the regrinding P_{80} size from 10 to 12 μm resulted in a reduction 23 kWh/t NEC. This was a significant cost saving.

Due to the site conditions and high ambient temperatures the manufacturer stated that the product size would be limited by the expected heat generation in the mill with a temperature limit of 65°C. Higher temperatures could cause premature failure of the rubber lining. Metplant also considered that high pulp temperatures would result in increased cyanide usage in the concentrate leach.

In conjunction with manufacturer and PGM, external cooling options and the use of a closed circuit were investigated. The use of refrigeration cooling in the circuit increased the costs substantially, so a decision was made to investigate an alternative to a horizontal mill.

Vertical mills

The alternative regrind mill option considered was to use vertically stirred mills. The manufacturer predicted that temperatures using the vertical mills would be 10°C lower than the horizontal mill.

The vertical mills are arranged differently from the horizontal mill with the three mills in series.

The vertical mills were also considered more suitable to the remote location in Africa as the mills require less skilled personnel to operate and maintain and lower spares cost. Ceramic media was selected for the vertical mills.

The Vertimill operates in closed circuit with cyclones. The flowsheet is shown in Figure 5.

Table IV
Laboratory recovery v regrind sizing

Grind Size (μm)	Au Recovery %
70	68
40	82
30	83
20	86
10	92
5	92
10	91 - 92
12	92

Table V
Horizontal mill–net energy consumption

Concentrate feed size (μm)	Product size (μm)	Net energy consumption kWh/t
	106	69 10
	69	12 49
Total		**59**
Recommended limit		50

Table VI
Vertical Regrind Mills

Regrind stage	Mill type	Power (kW)	Media
1	Vertimill	932	12 mm SS Balls
2	SMD	355	3.5 mm Ceramic
3	SMD	355	3.5 mm Ceramic

Thickening

The thickening was straightforward with the settling parameters of 0.25 t/m²/h for flotation concentrate and 1.4 t/m²/h for tailings. The thickener from Kori Kollo was in poor condition and, therefore, a new unit was purchased for the final tails duty. Both the flotation tails and final tails units were sized at 23 m and the concentrate thickener at 14 m. A non ionic flocculant, Magnafloc 342, was selected for all thickening duties at the operation.

The concentrate is thickened to 40% solids prior to regrinding. The material is not thickened after leaching and is pumped directly to the final tailings hopper and combined with the thickened flotation tailings.

Elution and electrowinning

The Oxide/tails and Concentrate CIL circuits have been designed with separate acid wash, elution and electrowinning systems. This was done due to the disparity in the size of the requirements in the different stages of the operation. As a consequence, the elution column for the Oxide/tails circuit is a 9.7 ton unit whereas the Concentrate column is a 2.4 ton unit.

Four new electrowinning cells were purchased for the Oxide tails system and three additional units for the Concentrate system. Both sets of units will use a stainless steel wire and sludge operation.

The two gold rooms are incorporated into one housing both electrowinning circuits and using one set of gold room ancillary equipment.

Process control

The philosophy adopted for Sukari allows for all major components to be monitored and, in most instances controlled by the control room operator, whilst still maintaining a high level of safe plant operability.

A SCADA system that allows for drive and process parameter status, trending of process variables and alarming forms the basis for plant control.

This relatively basic control system was considered the most suitable for the remote location.

Future provisions

The design has incorporated a number of provisions for additional equipment should results or circumstances require them.

Secondary crushing

The crusher layout has been prepared so that a secondary crushing circuit can be installed. Instead of the primary crusher discharge being conveyed to the stockpile, it would be diverted to a secondary crusher operating in closed circuit with a screen.

Gravity circuit

Testwork has been conducted which shows that a gravity circuit may be required. The circuit has been designed so that the cyclone underflow recycle pump can be diverted to a future gravity section.

Services

Water

Sukari is in one of the driest deserts in the world. The average rainfall is less than 1mm per year, whilst the evaporation rate is over 4 metres per year. The plant has been designed to use sea water which will be pumped from the Red Sea 22 km from the minesite. The delivery system will require comprise twin 600 mm pipelines with six booster stations. Each booster station will consist of a tank and three 132 kW pumps, (two operating and one standby). Power will be delivered by an overhead HV line and fibre optics will be used for control.

The water was tested and as expected had a buffering effect above pH 10. The operating pH in the plant will therefore be kept between 9.5 and 10.

The water will be used in the process plant grinding, leach and flotation circuits. A proportion of the water will be treated in a reverse osmosis plant to produce fresh water for the gold circuit, mine services and power plant. A portion of this water will be further treated to produce potable water for safety showers, personnel use and the camp.

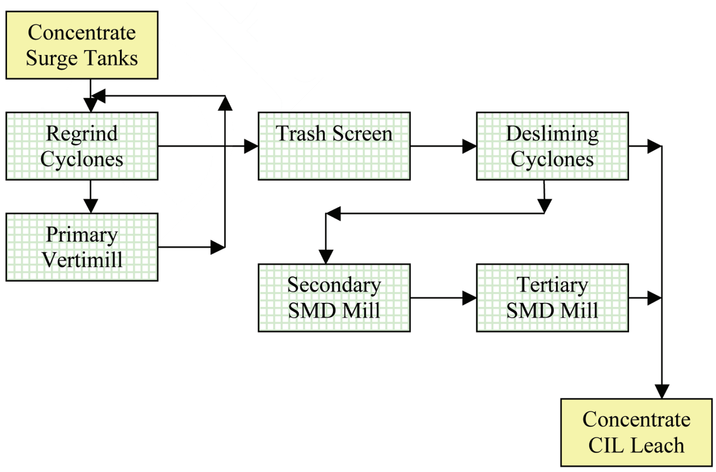

Figure 5. Regrind mill circuit

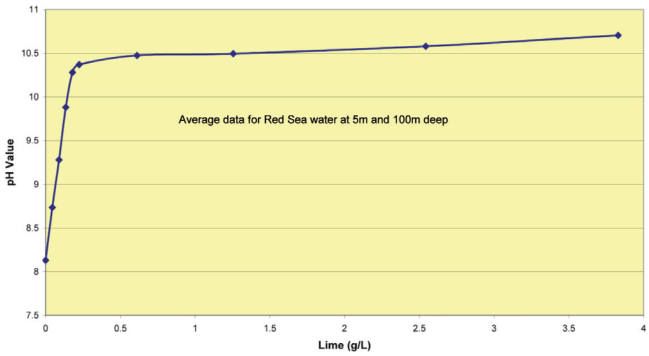

Figure 6. Lime buffer curve: Red Sea water

Power

Power for the operation will be supplied by diesel fired generators. PGM purchased a second hand plant from Isparta in Turkey comprising 4 x 8,6 MVA generators. A further 6 2 MVA generators were subsequently added.

The facility will supply 6.3 kV and 4.16 kV for distribution within the plant area for the high voltage duties. The two different voltages are required for the new and second hand equipment respectively. A 34.5 kV power line has been installed for the booster water pumping stations from the Red Sea to the plant. The operating voltage for the rest of the equipment is 380V 50 Hz. A centralised PLC with indication of equipment status and load controls the entire plan.