ÇUKUROVA UNIVERSITY
INSTITUTE OF NATURAL SCIENCE AND
APPLED SCIENCES

MSc THESIS

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DYNAMIC MODELLING AND OPTIMIZATION OF CARBON MANAGEMENT STRATEGIES IN CIL: A CASE STUDY OF NEW LIBERTY PROCESSING PLANT (LIBERIA)

DEPARTMENT OF MINING ENGINEERING

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ABSTRACT

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This thesis presents the Dynamic Modelling and Optimization of Carbon Management Strategies in the gold mine of the company Avesoro Resources in its New Liberty Process Plant, Liberia, which is the subject of this specification aims to increase its gold production and improve the quality of operations in this sector. The primary aim of the model is to investigate different carbon management strategies of the Carbon in Leach (CIL) process. It was provided more control and monitoring means the operations in the said sector.

This study was conducted in two parts: Mineral Processing and Metallurgical tests and Optimization and Stimulation of Carbon in Leaching Process. The mineral processing and metallurgical testing were deal with test work samples and leach optimization test. This aims to know the optimization of the leach and grinding test work. The optimization and stimulation of carbon is an important step in gold processing that involves simultaneous adsorption and leaching. The focus of this part was to study the effect of operating parameters like cyanide concentration, pH, oxygen concentration and mean particle diameter on the overall efficiency of CIL circuit. A dynamic model based on first principles was developed for the entire CIL circuit.

Leach test at pH 10.50 and different cyanide dosages: it was observed that the recovery of leach at 0.70 kg/ton in comparison to 1.20 kg/ton yields 81.27% and 78.77%, respectively. Test conducted at 1.20 kg/ton had a cyanide consumption of 0.96 kg/ton when comparing it to 0.37 kg/ton of test at 0.70 kg/ton NaCN dosage. The Dissolved Oxygen before and after leach test were below 6.00 ppm. Under leach test on fresh ore at pH 9.50 and 10.00 and 1 kg/t NaCN: it was concluded that after the test the +106, and +75 microns there is a certain degree of locked gold. Above all the gravity recoverable gold when present on a sample will always have a slow reaction kinetics and therefore residence time is essential during leaching. The test residue is high due to locked gold inside coarse material. The cyanide consumption is lower as compared to the main plant trends due to the fact that gold that was available was not in contact with cyanide.

From SIMCIL data on comparing the first baseline modelled profile and average plant profile, gold on carbon (not full GIC of gold on carbon/solution/solids) but of course almost all the gold is on the carbon. The full GIC was calculated and shown in the model. The elution frequency column provides elutions/day and elutions/week. Locking all scenarios, with the respect to carbon distribution. We generally found that even carbon distribution (same mass of carbon mass in each tank) was given a solution loss similar to or better than uneven profiles and has the advantage of simplicity.

Key Words: CIL, Simulation, Leach, Mass Balance, Circuit modelling, Adsorption

ÖZ

YÜKSEK LİSANS TEZİ

CIL DE KARBON YÖNETİM STRATEJİLERİNİN DİNAMİK MODELLENMESİ VE OPTİMİZASYONU: NEW LIBERTY TESİS ÖRNEK ÇALIŞMASI (LIBERYA)

Cheick Ahmed Tidiane SOUMAH

ÇUKUROVA ÜNİVERSİTESİ
FEN BİLİMLERİ ENSTİTÜSÜ
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Bu tez, Avesoro Ressources firmasının New Liberty Proses Tesisinde (Liberya) altın madeninde karbon yönetimi stratejilerinin dinamik modellenmesini ve optimizasyonunu sunmakta olup, bu şartnamenin konusu altın üretimini artırmayı ve bu sektördeki operasyonların kalitesini artırma amacı taşımaktadır. Modelin temel amacı, (CIL) prosesinde karbon yönetimi stratejilerini incelemektir. Söz konusu sektördeki operasyonların daha fazla kontrol ve izleme anlamına gelmesini sağlayacaktır. Bu çalışma iki kısımda yürütülmüştür: Cevher hazırlama ve metalürjik testler ve liçte karbonun optimizasyonu ve simülasyonu. Cevher hazırlama ve metalürji testi, test çalışma örnekleri ve liç optimizasyon testi ile ilgilidir. Karbonun optimizasyonu ve simülasyonu, eşzamanlı adsorpsiyon ve liç içeren altın işlemede önemli bir adımdır. Bu bölümün odak noktası, siyanür konsantrasyonu, pH, oksijen konsantrasyonu ve ortalama partikül çapı gibi işletim parametrelerinin CIL devresinin genel verimliliği üzerindeki etkisini incelemektir. Tüm CIL devresi için ilk prensiplere dayalı dinamik bir model geliştirilmiştir.

pH 10.50 ve farklı siyanür dozajlarında liç testi: 1,20 kg/ton'a kıyasla 0,70 kg/ton'da liç geri kazanımının sırasıyla %81,27 ve %78,77 verim sağladığı gözlenmiştir. 1,20 kg/ton'da gerçekleştirilen test, 0,70 kg/ton NaCN dozajındaki 0,37 kg/ton test ile karşılaştırıldığında 0,96 kg/ton siyanür tüketimine sahiptir. DO'lar liç testi öncesi ve sonrası 6,00 mg/L'nin altındadır. pH 9,50 ve 10,00 ve 1 kg/t NaCN'de taze cevher üzerinde liç testi altında: Testten sonra +106 ve +75 mikronlarda belirli bir derecede kilitli altın olduğu sonucuna varılmıştır. Her şeyden önce, bir numune üzerinde mevcut olduğunda gravite ile geri kazanılabilir altın her zaman yavaş bir reaksiyon kinetiğine sahip olacaktır ve bu nedenle liç sırasında kalma süresi esastır. Kaba malzeme içindeki kilitli altın nedeniyle test kalıntısı yüksektir. Mevcut altının siyanürle temas etmemesi nedeniyle siyanür tüketimi ana fabrika trendlerine göre daha düşüktür.

İlk temel modellenmiş profil ile ortalama bitki profilinin karşılaştırılmasıyla ilgili SIMCIL verilerinden, karbon üzerinde altın (karbon/çözelti/katılar üzerinde altının tam GIC'si değil) ama tabii ki altının neredeyse tamamı karbon üzerindedir. Tam GIC hesaplandı ve modelde gösterilmiştir. Elüsyon frekansı sütunu, elüsyon/gün ve elüsyon/hafta sağlamaktadır. Karbon dağılımı ile ilgili tüm senaryolara kilitlenmiştir. Genel olarak, karbon dağılımının bile (her tankta aynı karbon kütlesi kütlesi) eşit olmayan profillere benzer veya daha iyi bir çözelti kaybı verildiğini ve basitlik avantajına sahip olduğu tespit edilmiştir.

Anahtar kelimeler: CIL, Simülasyon, Liç, Kütle balansı, Devre modelleme, Adsorpsiyon

EXTENDED SUMMARY

This section highlights the grinding circuit balance, leach optimization and CIL simulation.

During this study, the operation was problematical for a variety of reasons. These issues can be summarised as availability of planned ore, high proportion of oxide material in the ore feed, oversize feed to the ball mill resulting in inefficient grinding and excessive stone discharge from the ball mill trommel, poor grinding ball quality, ball mill liner and grate material problems, under-utilisation of the Vertimill, excessive wear in the grinding circuit, gravity circuit feed screen capacity problems, lower than expected gravity circuit gold extraction due to insufficient gravity concentrator capacity, oxygen plant operational problems; oxygen sparging issues in pre-oxidation and CIL tanks, poor CIL leach extraction, poor carbon management in the CIL circuit, cyanide detoxification circuit performance issues and availability of reagents and maintenance spares.

Several plant modifications have been implemented to address these issues and further changes were planned for implementation. These changes should reduce plant downtime, enhance plant throughput, and improve plant performance.

For the CIL simulation the treatment circuits at the Bea Mine (Liberia) were treated new ores from the Ndablama and the New Liberty underground deposits. The treatment rates, amount of gravity gold, leach feed grades and extraction rates were varied over time. There will be two identical comminutions, leaching a CIL plants. This model was independently verified. One of the limiting factors of model verification was the availability of useable plant data for model building and verification. By using the daily data, all operating parameters was manipulated and controlled during this study.

Leach test at pH 10.50 and different cyanide dosages: it was observed that the recovery of leach at 0.70 kg/ton in comparison to 1.20 kg/ton yields 81.27 % and 78.77%, respectively. Test conducted at 1.20 kg/ton had a cyanide consumption of 0.96 kg/ton when comparing it to 0.37 kg/ton of test at 0.70 kg/ton NaCN dosage. The DO's before and after leach test were below 6.00 ppm. Under leach test on fresh ore at pH 9.50 and 10.00 and 1 kg/t NaCN: it was concluded that after the test the +106, and +75 microns there is a certain degree of locked gold. Above all the gravity recoverable gold when present on a sample will always have a slow reaction kinetics and therefore residence time is essential during leaching. The test residue is high due to locked gold inside coarse material. The cyanide consumption is lower as compared to the main plant trends due to the fact that gold that was available was not in contact with cyanide.

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The models and elution simulation were first calibrated (baseline simulation) using current operating data and then a suite of simulations were undertaken with varying inputs variables.

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LIST OF SYMBOLS

g/t	: gram per ton
km	: kilometre
WAD	: weak acidic dissolutions
ppm	: parts per million
SMBS	: Sodium Metabisulphite
%	: percentage
m	: meter
PLC	: Programmable Logic Controllers
HMI	: Human Machine Interface
kPa	: Kilopascal
mm	: millimetre
mg/L	: milligram per litter
ml	: millilitre
kg/ton	: Kilogram per ton
hrs	: hours
µm	: micron
t/s	: ton per hour
g/l	: gram per litter
µm	: micron
t/h	: ton per hour
CIL	: Carbon-In-Leach
CIP	: Carbon-In-Pulp
GIC	: Gold-In-Carbon
DO	: Dissolved Oxygen
GRG	: Gravity Recoverable Gold
QA	: Quality Assurance
QC	: Quality Control

1. INTRODUCTION

The carbon in leach (CIL) step in a gold processing is a continuous process that use activated carbon in cascade of large agitated the tanks, which have been widely used to recover or concentrate precious metal in gold extraction plants. In this process the activated carbon is moved from one tank to another in counter current with the ore pulp until the recovery of the loaded carbon in the first tank.

The use of carbon for the recovery of gold was first noted in the 19^{th} century, where the gold was recovered by combustion of the loaded carbon and smelting its ash. However, there were no means of reusing the carbon at the time, and when coupled with improvements in an alternative gold recovery process. The Merrill-Crowe process activated carbon remained an uneconomic alternative until the 1970s.

This thesis presents the dynamic modelling and optimization of carbon management strategies in the gold mine of the company Avesoro Resources in its New Liberty Process Plant, which is the subject of this specification aims to increase its gold production and improve the quality of operations in this sector. The primary aim of the model is to investigate different carbon management strategies of the Carbon in Leach (CIL) process. It will provide more control and monitoring mean the operations in the said sector.

This study was carried out in two parts: Mineral Processing and Metallurgical tests and Optimization and Stimulation of Carbon in Leach. The Mineral processing and metallurgical testing were deal with test work samples and leach optimizations tests. This aims to know the optimisation of the leach and grinding test work. The optimisation and stimulation of carbon is an important step in gold processing that involves simultaneous adsorption and leaching.

The focus of this part is to study the effect of operating parameters like cyanide concentration, oxygen concentration and mean particle diameter on the overall efficiency of CIL circuit. A dynamic model based on first principles is developed for the entire CIL circuit.

1.1. History

The first exploration work at the property was carried out by Golden Limbo and comprised desktop studies, a review of satellite imagery, target selection and acquisition of a portfolio of possibilities. In 1997, Mano River Resources (Mano) collected preliminary channel samples across the artisanal workings, where primary rock was exposed. During reconnaissance work numerous targets for gold mineralization were identified through geological mapping, supported by soil and stream geochemical sampling programmes.

Exploration by BMMC at the Bea-MDA property since 2011 has followed a systematic process of reconnaissance work, grab-sampling followed by soil geochemistry, mapping, trench sampling and eventually drilling. BMMC completed a feasibility study in October 2012 and subsequent to this carried out additional work with a view to optimising the project. This optimisation work was reported in the report titled New Liberty Gold Project, West Africa, and Updated Technical Report, dated 3 July 2013 (Technical Report, 2013).

Since this time, the Company has continued to conduct further evaluation work at New Liberty, including grade control drilling to produce a better geological understanding of the deposit at a mining scale.

1.1.1. The New Liberty Gold Mine

The key asset in Avesoro Resources' portfolio is the New Liberty Gold Mine, which is Liberia's First and largest commercial gold mine. Construction at New Liberty commenced in April 2014, with first gold poured from the process plant on 31 May 2015 and gold sales commencing in August 2015. Commercial Production was declared at New Liberty on 1 March 2016. New Liberty is an open-pit mining operation, and the process plant has an industry standard flowsheet consisting of a two-stage crushing, ore stockpiling, milling and gravity and carbon in leach plant with an annual throughput capacity of 1.1 million tons of ore.

The New Liberty Gold Mine has a NI 43-101 – Standards of Disclosure for Mineral Projects (“NI 43-101”) compliant Proven and Probable Mineral Reserve estimate of 7.4 million tons grading 3.03 g/t for 717,000 contained ounces of gold and a NI 43-101 compliant Mineral Resource estimate comprised of 9.6 million tons grading 3.2 g/t for 985,000 ounces in the Measured and Indicated category and 6.4 million tons grading 3.0 g/t for 620,000 ounces in the Inferred category.

The New Liberty Gold Mine is located within the Bea-MDA property, in Grand Cape Mount County in the north-western portion of the Republic of Liberia, approximately 100 km north-west of the capital, Monrovia.

From the capital there is approximately 80 km of excellent paved road to the town of Daniels Town and then a small 20 km section of laterite road to the project site. Avesoro Resources has recently upgraded the laterite section of road and installed three new culvert-type bridges to facilitate the transportation of equipment to site. Road access is all year round.

1.2. Overview of Gold Processing

To illustrate the operation of a typical gold plant, a process flow chart of New Liberty Gold Plant is shown in Figures 1.1-1.4.

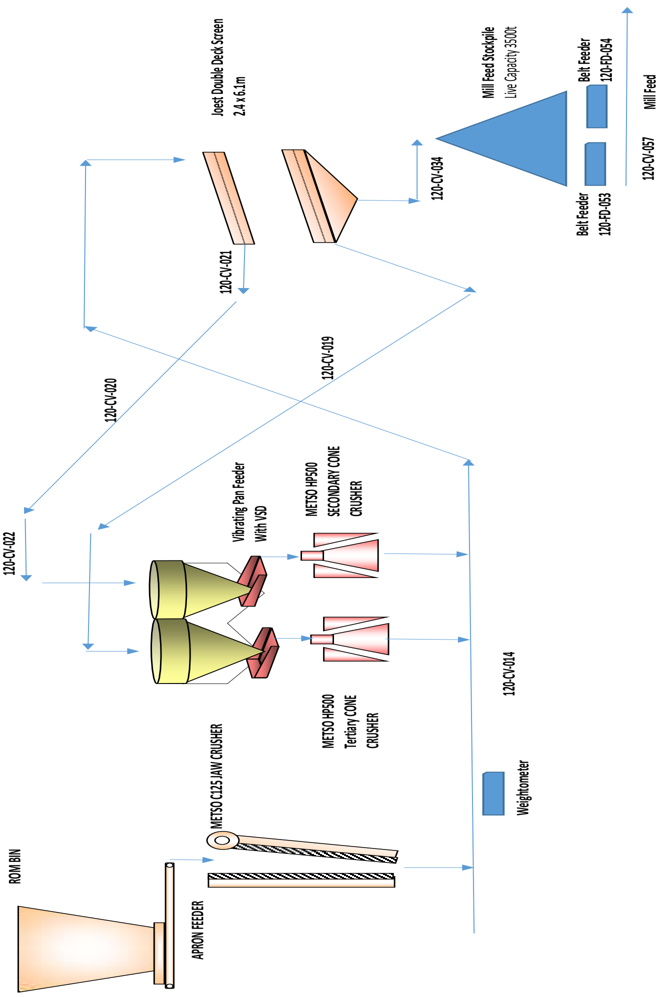


Figure 1.1. Crushing circuit

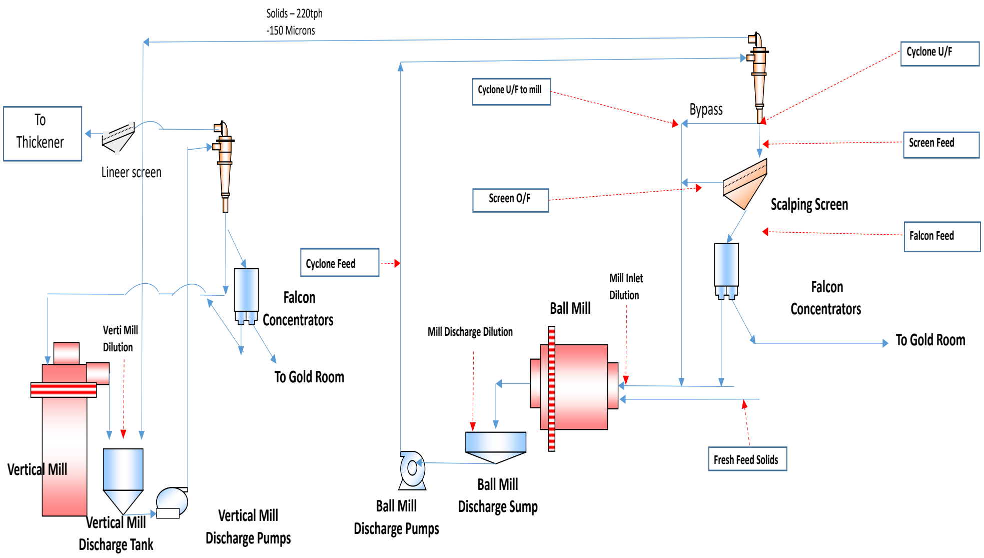


Figure 1.2. Grinding and gravity circuit

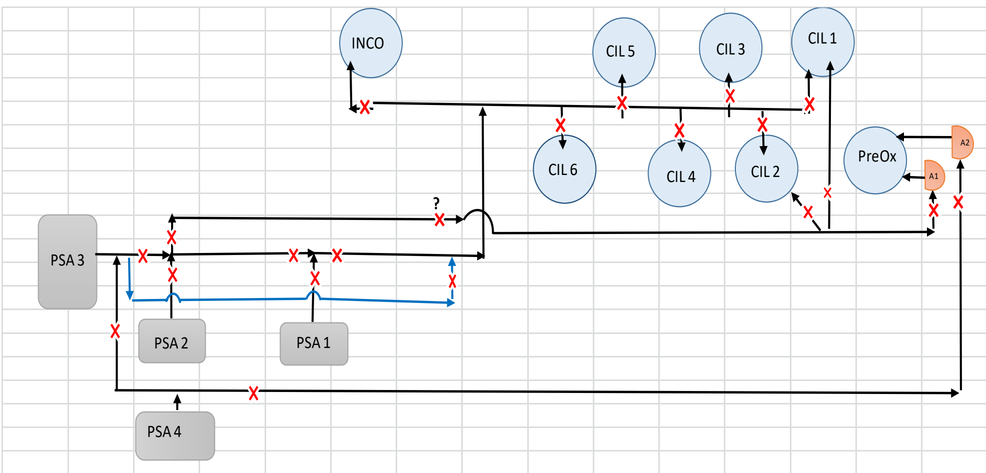


Figure 1.3. CIL circuit

- **Inco Detox**

The Acidic Pre-Leach Tank serves a dual purpose by destructing the cyanide in the slurry to below the required 20 ppm WAD, using the INCO process, and to leach the arsenic present in the ore body before precipitating in tanks downstream.

The INCO process is based upon conversion of WAD (weak acidic dissolutions) cyanides to cyanate using a mixture of SO_2 and air, in the presence of a soluble copper catalyst at a controlled pH.

With the ore being Ni rich, detox is carried out at pH levels of between 5.0 – 6.0 as opposed to the conventional pH of 8.0 – 9.0.

The addition of SMBS in this process has a dual purpose, firstly taking part in the detoxification reaction, and secondly lowering the pH (due to the formation of sulphuric acid) to such an extent favouring the leaching process.

Copper Sulphate ($CuSO_4$) is primarily used as a catalyst in the reaction.

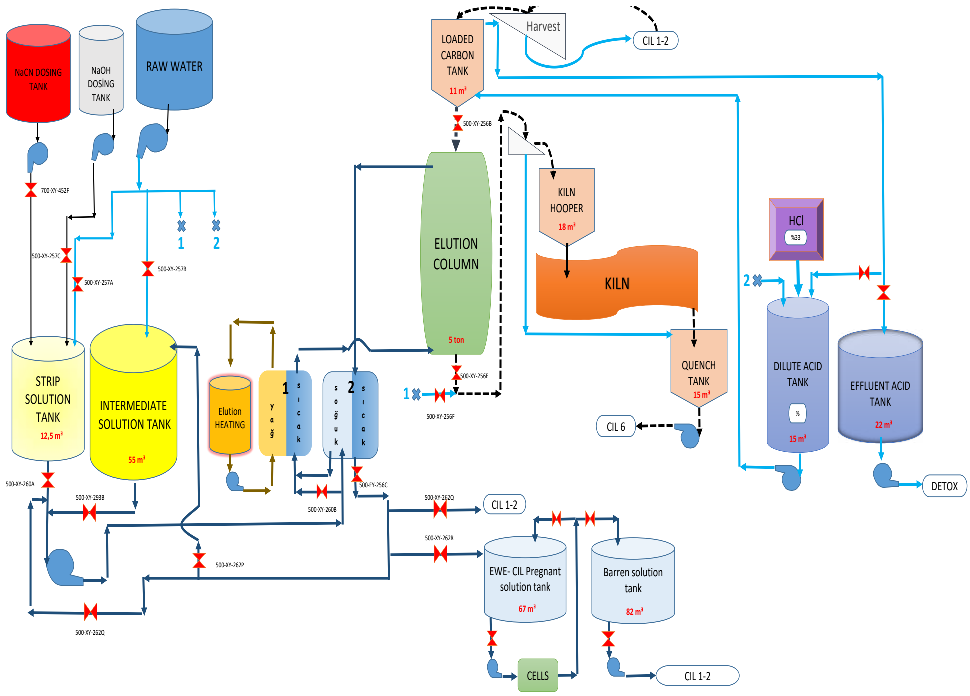


Figure 1.4. Elution circuit

Firstly, run of mine ore is crushed and then milled. The ore composition and characteristics will dictate the type of crushing circuit that is used. The ore is milled to approx. 74 micron and then leached in cyanide solution. During leaching the gold in the ore complexes with cyanide to form a gold-cyanide complex. Leaching typically takes between 10 to 30 hours. The rate of leaching is affected by a number of factors including the cyanide and oxygen concentration, pH, temperature, surface area of gold exposed, agitation and the presence of other ions.

Carbon in leach process, leaching and adsorption occur simultaneously. This circuit is also known as Carbon in Leach (CIL) circuit. It is this circuit that was the focus of this theses.

The Carbon in Leach (CIL) circuit is used to leach out gold from the easily recoverable oxide ore or act as a scavenger for the remaining gold from the circuit. Oxide material from the thickener is pumped to the CIL pre-oxidation tank. The feed enters the pre-oxidation tank through a mixing box and can be bypassed to CIL tank no. 1 if the pre-oxidation tank is off-line.

An Aachen reactor pump circulates the slurry in the tank through Aachen reactors which contacts the slurry with oxygen. The Aachen reactor pump is supplied with high pressure gland seal pump. Oxygen is also injected at the bottom of the tank through Oxygen spargers. Introduction of Oxygen with the use of the reactors and spargers is aimed at increasing the dissolved Oxygen level which enhances leach kinetics and Carbon loading.

Lime is added to the pre-oxidation tank to increase the slurry pH to the required leach pH level before Cyanide is added. The leach pH level must be maintained at 10.5 or higher. Lead Nitrate is also added to enhance leaching kinetics. Hydrogen peroxide can be dosed as an Oxygen replacement if Oxygen is unavailable and can also be used as a top-up oxidant if required.

The slurry overflows from the pre-oxidation tank into a launder and flows by gravity to CIL tank no.1. CIL tank no. 1 can be bypassed and the slurry gone to CIL tank no. 2 if the tank is off-line for maintenance.

Slurry and activated Carbon flows counter current to each other through CIL tank no. 1 to 6. Slurry from the pre-oxidation tank flows by gravity to CIL tank no. 1 and 2, and thereafter each subsequent tank, using interstate screens. These screens run continuously and “pump” the slurry from its tank into an overflow launder from where it flows to the next tank. The screens only transfer the slurry while retaining the Carbon in the tank.

Fresh / Regen carbon is added to tanks and is pumped from each tank to the preceding tank in the sequence using Carbon transfer pumps. Carbon is transferred daily from each tank and the operator must base the transfer time (time the pumps will be running) on the carbon concentration in each tank.

The Carbon concentration is determined by manual sampling carbon and slurry is transferred by a carbon transfer pump from tank no. 1 to the loaded carbon screen at the top of the elution columns where the carbon is screened out for elution. The loaded carbon screen is used for carbon form CIL, the screened-out slurry is returned to CIL tank no. 1, but can also be bypassed to tank no. 2.

The following reagents are dosed in the CIL tanks:

- Oxygen is added to the CIL tanks via three spargers in the bottom of each tank to enhance leach kinetics and Carbon loading.
- Cyanide is dosed in tank no 1 to affect the gold leaching. Cyanide is usually dosed in tank no. 1 but can also be dosed in tank no. 2 if tank no. 1 is off line for the maintenance.
- Lime can be dosed in CIL tank no. 1 – 4.
- Lead Nitrate can be dosed in tank no. 1 to enhance leach kinetics.
- Hydrogen Peroxide can be dosed in each CIL tank to serve as replacement oxidizer when oxygen is not available.

Three Detox tanks are used for cyanide destruction. Cyanide must be destroyed to below 20 ppm WAD before CIL tailings can be disposed of in the tailings dam according to Liberia legislation. The Detox facility achieves this by creating highly oxygenated slurry with the Aachen reactors and oxidizing the cyanide to cyanide in the presence of activated carbon catalyst.

1.3. Mineral Processing and Metallurgical

1.3.1. Background

The process plant was commissioned during 2015 and the initial operation was problematical for a variety of reasons. These issues can be summarised as availability of planned ore, high proportion of oxide material in the ore feed, oversize feed to the ball mill resulting in inefficient grinding and excessive stone discharge from the ball mill trommel, poor grinding ball quality, ball mill liner and grate material problems, under-utilisation of the Vertimill, excessive wear in the grinding circuit, gravity circuit feed screen capacity problems, lower than expected gravity circuit gold extraction due to insufficient gravity concentrator capacity, oxygen plant operational problems; oxygen sparging issues in pre-oxidation and CIL tanks, poor CIL leach extraction, poor carbon management in the CIL circuit, cyanide detoxification circuit performance issues, and availability of reagents and maintenance spares.

A number of plant modifications have been implemented to address these issues and further changes are planned for implementation. These changes should reduce plant downtime, enhance plant throughput and improve plant performance to beyond the levels assumed in the Feasibility Study.

1.3.2. Grinding Media and Components

Primary Mill: Primary milling essentially comprises a load-cell mounted 5.3 m (17.5’) diameter (inside shell) x 6.7 m (22’) long (flange-to-flange) ball mill fed from a crushed ore conical stockpile at a nominal rate of 210 dry tonnes per hour.

The ball mill (Figure 1.5) operated in closed circuit to produce a target grind of 80% passing 212 µm. The mill feed consists of fresh crushed ore, the primary classification cyclone cluster underflow, gravity concentrator scalping screen oversize and gravity concentrator tailings. The mill discharge density is controlled between 70-75% solids by mass by the addition of process water to the mill inlet.

Two scenarios exist when considering the amount of process water to be added yielding the required in mill density:

- Gravity Concentrator running
- Gravity Concentrator not running.

Gravity Concentrator Running: The cyclone underflow is diverted to the gravity concentrator, and no circulating load as such is circulated back to the mill.

*Gravity Concentrator **NOT** Running:* The dart valve feeding the gravity circuit is closed, effectively circulating the cyclone underflow back to the mill feed.

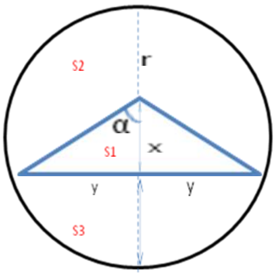
For safety purposes, milk-of-lime is dosed into the mill feed chute based on the pH within the mill. During maintenance periods, this allowed the pH within the mill to be increased and prevent the formation of hydro-cyanic gas from residual cyanide in the water within the mill. The operator monitored the pH levels measured with a handheld pH meter, after which the control room operator will open the lime addition valve until the pH levels are within the acceptable ranges (pH>10.5). The lime addition will be done prior to stopping the mill (Table 1.1).

Figure 1.5. Ball mill

New Liberty Plant Ball Load Measurement

Table 1.1. Ball mill measurement calculation

Distance from ball level to liner (r+x)				
	INLET	MIDDLE	OUTLET	**AVERAGE**
Lifter (cm)	376	376.4	371.2	**374.5**
Shell (cm)	386	385	382.7	**385**
h	10	8.6	11.5	
Blank length (Mill Diameter)				
Lifter-lifter (cm)	499	498.5	496.5	**498.0**
Shell- shell (cm)	517	515.5	513	**515.2**
h	9	8.5	8.25	



Volume calculation;

volume = $\pi.\ r^2.\ h = 146.76\ m^3$

Loaded area volume = $(\pi.\ r^2.\ h) * s^3$ / (Mill Diameter)

= 28.96

Mill ball level (cm) = 381.22

Ball mill diameter (cm) = ((493.3 * 2) +(509.7 * 4)) / 6 = 509

R = 5.094 m and r = 2.54 m

x	1.265
r2 = X2 + Y2	
y	2.210908422
Tan a - 1(a)	60.22340706
B	239.5531859

The mill loaded area will be calculated according to	
measured height above load (m)	1.282
measured mill Diameter(S)	20.38372457
$S = S_1 + S_2 + S_3$	
$S_1 = (X*2Y) / 2$	2.796799154
$S_2 = (\beta / 360) * S$	13.56385045
Ball cross-section area S3	4.023074971
Mill ball charge rate	
% Ball	**19.74**
Amount of ball in loaded areas = (loaded area volume) x (d) x (1 - porosity)	**137.4 ton**
Ball mill power draw	**2350 kW**
Ball porosity (%)	38
d_{Ball} **= 7.65 gr/cm³**	7.65

1.3.3. Verti mill and Components

Regrind Mill: The underflow from the regrind cyclone cluster is also collected in a launder and then direct-fed to the top of the regrind mill which operates in reverse closed circuit. The regrind mill (Figure 1.6) is a Metso Vertimill VM-1500 unit with an agitating screw suspended into the grinding chamber. The screw is supported by spherical roller bearings and driven by a fixed speed motor through a planetary gearbox. As the particles are milled, they rise to the top of the mill and overflow, gravity discharging to the primary cyclone classification cluster overflow tank for feed to the regrind cyclone cluster. Grinding media for the regrind mill is added manually via a steel ball hopper. 12 mm forged steel balls are used for the regrind duty. The mill has two modes namely: manual mode and automatic mode. These modes are selected via the HMI (Human Machine Interface). In manual mode each of the individual components can be started/stopped or open/closed via the local HMI. The mill mode can only be changed once the mill is stopped. In automatic mode, the mill is started and stopped via the HMI. When the mill is started in automatic mode the PLC (Programmable Logic Controllers) starts and stops the mill as per the start-up sequence and the shutdown sequence.

Figure 1.6. Verti mill

1.3.4. Separate Material by Means of a Cyclone

The basic principle of cyclone operation

Mil Cyclone cluster #1: The cyclone (Figure 1.7) feed stream at ±58% solids by mass is pumped to a cluster of hydro-cyclones, operated at ±85 kPa, monitored by valve and manually controlled by taking hydro-cyclones on/off-line to suit. Cyclone overflow is gravity fed to the cyclone stage #1 overflow tank, whilst the underflow will be collected in an underflow box and split into two streams; gravity concentration circuit feed and cyclone stage #1 underflow excess being re-cycled to the mill feed. The required split is achieved using a manual dart valve in the cyclone underflow box. It is very important to ensure that at least 2 cyclones are open when starting up the mill discharge pumps.

Mil Cyclone Cluster #2: The pressure of the feed stream to the regrind cyclone cluster has a bearing on the efficiency of the cyclone in terms of the desired cut-point. Seven of the cyclones in this cluster have manual feed isolation valves which will be open under normal operating conditions. The other three cyclones are fitted with auto-isolation valves which allow for a degree of pressure control in that number of cyclones online can be increased by opening the feed auto-isolation valves to decrease the feed pressure. Alternatively, the reverse also applies in that the number of cyclones online can be decreased by closing the auto-isolation valves, consequently increasing the feed pressure. The pressure set point is an input defined by the control room operator (Table 1.2).

Figure 1. 7. Cyclone

New Liberty Plant Cyclone efficiency calculation

Table 1.2. Cyclone efficiency calculation

d solid					%Solid	%Liquid	%solid/%liquid		
2.8				feed	63	37	0.587		
Cyc.1 over flow solid				under f.	81	19	0.235		
212.3				over f.	43	57	1.326		
	F x	0,587		=	0.235	***x U***	+	1.326	***x O***
U = F - O									
F x	0,738	=	1.091	***x U***					
U/F =	0,677	**67.67%**				Fq =	656.6488372		recirculation load
						Oq =	212.3		
						Uq =	444.3488372		209.30233
						Fq x	O/F	= Oq	
						Fq x	U/F	= Uq	

1.3.5. Falcon Gravity Concentrator components

Falcon *Gravity* Concentrator: The primary mill classification cyclone cluster underflow is collected in an underflow box and split into two streams: one stream forms the feed to gravity concentrator, while the second stream, excess to gravity concentration requirements, is returned to the mill feed chute. For initial plant construction, a single Falcon gravity concentrator (Figure 1.8) was installed, but allowance made for the possible future installation of a second gravity concentrator.

The gravity concentrator feed is pre-screened on a vibrating screen to remove oversize, +1.6 mm material not suited to the concentrator. The spray bars of the screen were used to increase screen efficiency and dilute the feed to the concentrator to fall within the range 65%-70% solids by mass. Further dilution of the concentrator feed with process water can be done in the screen underpan. No auto control loop for the water addition has been allowed for, and requires the operator to set the manual valve installed before the flow meter delivering the required dilution water. Water addition to the scalping screen is monitored to allow for an accurate water balance over the entire milling circuit. Screen oversize is directed back to the mill feed by an overflow chute, while the underflow gravitates to the gravity concentrator (Table 1.3 and Figure 1.9).

Figure 1.8. Falcon gravity concentrator

- New Liberty Plant Comparison of falcon efficiency on ball mill

Table 1.3. Ball mill falcon recovery data

		Tph	Grade	% Gravity	Recovery	Poured Gold oz.	Poured Gold kgs	Total Tonnage
2019	April	198	2.73	23	87.1	8141	253.218	113.469
	May	198	2.98	23	87.7	11886	369.702	132.911
	June	180	2.68	20	87.5	7998	248.770	105.771
	July	169	2.83	25.8	90.5	8677	269.889	101.513
	August	194	2.67	17	80.9	9901	307.961	120.323
	September	201	2.94	20	88.4	10656	331.444	131.807
	October	179	2.54	22	89.9	8952	278.443	124.986
	November	181	2.23	23	88.4	7727	240.341	124.488
	December	188	2.55	23	88.5	7874	244.913	112.339
2020	January	207	2.38	24	87.5	8254	256.732	122.600
	February	161	3.29	27	90.2	6842	212.814	77.282
	March	167	3.49	33	92.1	10771	335.021	108.273
	April	189	2.15	29	92.3	3882	120.746	53.797

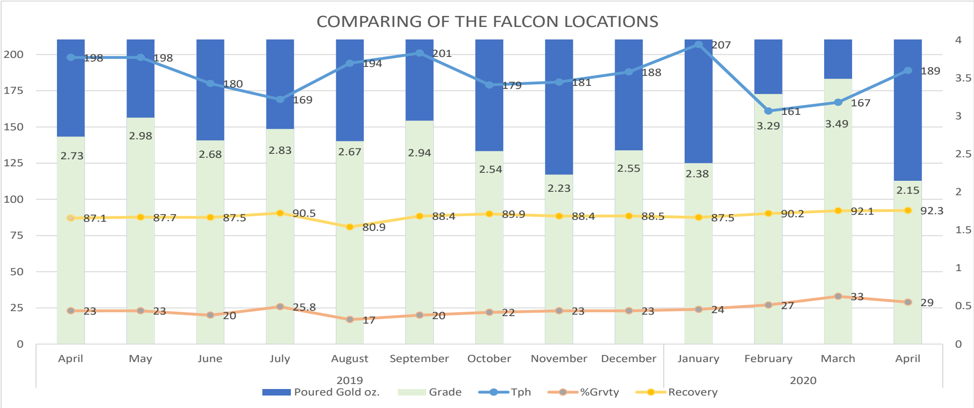


Figure 1.9. Comparing Falcon recovery

1.3.6. Grinding Circuit Mass Balance

Mass balance of the grinding circuit was presented. Primary mill feed is withdrawn from the mill feed stockpile via two variable speed belt feeders (120-FD-053/054) onto the mill feed conveyor (120-CV-057). The feed chute to each belt feeder is fitted with a low-level switch for belt protection purposes. The mill feed conveyor is fitted with a two-idler electromechanical weight meter (120-WT-091) to allow control of the mill feed rate by adjustment of the belt feeder drive speed setting.

The ball mill will be operated in closed circuit to produce a target grind of 80% passing 212 µm. The mill feed consists of fresh crushed ore, the primary classification cyclone cluster underflow, gravity concentrator scalping screen oversize and gravity concentrator tailings. The mill discharge density is controlled between 70-75% solids by mass by the addition of process water to the mill inlet. Process water flow control is affected by a flow meter and control valve on the process water line into the mill feed chute. The operator should take note of the isolation valves on the process water lines, ensuring that they are in the correct position during operation: The manual isolation valve ahead of the flow meter enables one to isolate the instrumentation should maintenance be carried out. This valve needs to be in the open position during normal operation.

The manual isolation valve installed across the control valve is used as by-pass should the control valve become dysfunctional. This valve should be closed during normal operation.

A single ball valve installed downstream of the control valve allows for water addition to the “dead box” area, ensuring no material build-up. This valve needs to be in the open position during normal operation.

Two scenarios exist when considering the amount of process water to be added yielding the required in mill density:

- Gravity Concentrator running
- Gravity Concentrator not running.

A ball hopper and hoist for the loading of forged 90mm steel balls into the mill has been allowed for. Slag from the gold room may also be returned to the milling circuit via this hopper and chute. Mill seal water drawn from the raw water header is added to the mill seal ring via valve. This valve needs to be in the open position during normal operation. A flow switch, installed on the raw water line is interlocked with the mill and will stop the mill after a pre-set period, reducing excessive wear on the seal ring. A grate discharge on the mill exit will hold larger solid particles in the mill while allowing the milled slurry to discharge onto a trommel screen. Scats material will be caught by the trommel screen and stored in a separate bunker for periodic removal by front-end loader. The slurry passing through the trommel screen gravitates into the 38 m^3 mill discharge sump.

Mill discharge sump slurry is pumped via a single delivery pipeline by an operating or standby variable speed pump to a cluster of four hydro-cyclones three operating and one on standby, depending on mill status, installed on a tower adjacent to the ball mill. The mill discharge pumps are capable of auto-changeover by means of automated isolation valves on the suction and delivery of each pump.

Gland seal water is supplied to each pump. Level in the mill discharge sump is controlled by level indication and adjustment of the variable speed setting on the duty pump.

It is very important for the field operator to ensure that the selected pump's drain valves are closed before selecting the duty pump. Once the duty pump has been selected, the field operator needs to open the standby pump's drain valves, draining the lines ensuring that the lines don't salt up. These drain valves should be CLOSED upon completion of drainage, and be ready for an auto change over if required.

The process water addition valve is also interlocked with the mill discharge sump level. If a level in the mill discharge sump is detected, the density control is overridden, and the valve opens 100% ensuring that the pumps are not run dry.

General mill area spillage accumulates in a spillage sump within the dedicated milling bunded area and is pumped back to the primary mill discharge sump by either the mill feed end spillage pump or the mill discharge end spillage pump. The spillage sump is designed to allow Bobcat-type front-end loader access for periodic removal of accumulated settled solids. A dedicated level switch to each pump is installed on the bund automatically starts the spillage pump when activated. The pump will automatically stop once a low current is detected.

The cyclone feed stream at ±58% solids by mass is pumped to a cluster of hydro-cyclones, operated at ±85kPa, monitored by automation and manually controlled by taking hydro-cyclones on/off-line to suit. Cyclone overflow is gravity fed to the cyclone stage #1 overflow tank, whilst the underflow will be collected in an underflow box and split into two streams; gravity concentration circuit feed and cyclone stage #1 underflow excess being re-cycled to the mill feed. The required split will be achieved using a manual dart valve in the cyclone underflow box.

It is very important to ensure that at least 2 cyclones are open when starting up the mill discharge pumps.

The primary mill classification cyclone cluster underflow is collected in an underflow box and split into two streams: one stream forms the feed to gravity concentrator, while the second stream, excess to gravity concentration requirements, is returned to the mill feed chute. For initial plant construction, a single Falcon gravity concentrator was be installed, but allowance has been made for the installation of a second gravity concentrator.

The gravity concentrator feed is pre-screened on a vibrating screen to remove oversize, +1.6 mm material not suited to the concentrator. The spray bars of the screen will be used to increase screen efficiency and dilute the feed to the concentrator to fall within the range 65%-70% solids by mass. Further dilution of the concentrator feed with process water can be done in the screen underpan. No auto control loop for the water addition has been allowed for, and requires the operator to set the manual valve installed before the flow meter delivering the required dilution water. Water addition to the scalping screen is monitored to allow for an accurate water balance over the entire milling circuit. Screen oversize will be directed back to the mill feed by an overflow chute, while the underflow gravitates to the gravity concentrator.

The Gravity Concentrator is fed with a controlled portion of the mill cyclone cluster #1 underflow, after passing the scalping screen as outlined above. The unit will be operated as a batch concentrate process and relies on the principle of centrifugal forces separating the heavy phase from the light phase by means of a rotating fluidized bowl. The heavy phase (containing the gold) is then discharged from the gravity concentrator as a concentrate and fed by gravity to the gold room. The tailings from the gravity concentrator are continually discharged to the mill feed chute by means of a gravity pipeline. The option to bypass the gravity concentrator feed to the mill feed chute exists, and will be used during concentrate discharge, enabling the scalping screen to operate continuously and ensuring a quick switch-over back to feed increasing concentrator utilisation.

Cyclone cluster no. 2 overflow is gravity fed to the Mill Linear Screen via the Linear Screen Feed Box. The cyclone cluster by-pass line and cyclone cluster drain valve is also fed to this box before discharging onto the linear screen. The screen overflow collects in the Trash Dewatering Screen Basket, whilst the underflow gravitates to the Pre-Leach Thickener Feed Box. Lime fed from the ring main system is dosed into the underflow stream controlled with a manual pinch valve. It is the field operator's duty to ensure that lime is constantly dosed at this point.

Screen spray water is drawn from the filtered water header and fitted with a pressure switch alarming on the SCADA if the line pressure is lost. The field operator needs to ensure that the manual valve is open during normal operation. A pressure indicator is installed on the spray water line for monitoring purposes only.

Instrument air fed from the instrument air header is used for the auto tracking system. Manual valve needs to be open during normal operation. A pressure indicator is installed on the air line for monitoring purposes only.

The underflow from the regrind cyclone cluster is also collected in a launder and then direct-fed to the top of the regrind mill which operates in reverse closed circuit. The regrind mill is a Metso Vertimill unit with an agitating screw suspended into the grinding chamber. The screw is supported by spherical roller bearings and driven by a fixed speed motor through a planetary gearbox. As the particles are milled, they rise to the top of the mill and overflow, gravity discharging to the primary cyclone classification cluster overflow tank for feed to the regrind cyclone cluster. Grinding media for the regrind mill is added manually via a steel ball hopper. 12 mm forged steel balls are used for the regrind duty. The mill has two modes namely: manual mode and automatic mode. These modes are selected via the HMI. In manual mode each of the individual components can be started/stopped or open/closed via the local HMI. The mill mode can only be changed once the mill is stopped. In automatic mode, the mill is started and stopped via the HMI. When the mill is started in automatic mode the PLC starts and stops the mill as per the start-up sequence and the shutdown sequence outlined.

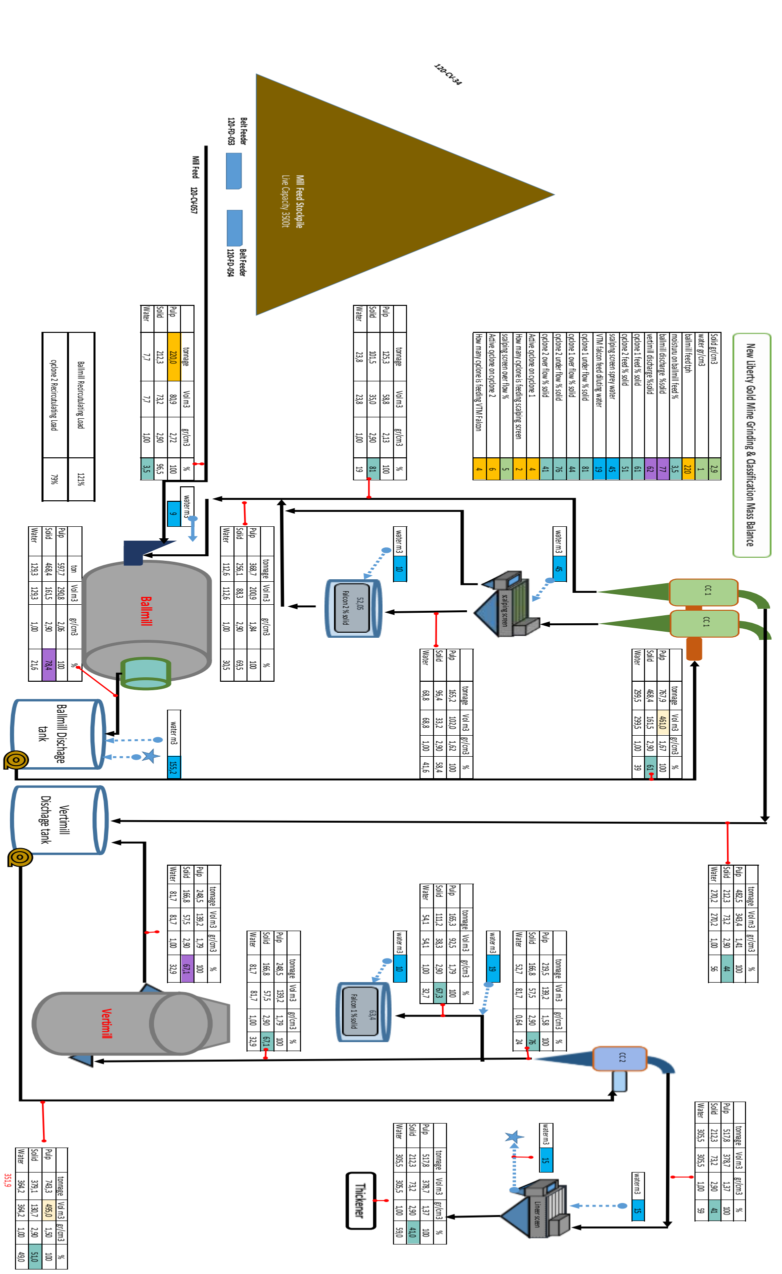


Figure 1.10. Mass balance grinding circuit

2. PREVIOUS STUDIES

McDougall and Hancock (1980), studied the CIP and CIL processes use granulated activated carbon that was mixed with the pulp, composed of ore and water, and after it has been loaded with aurocyanide ion, the rich activated carbon is pumped out into the adsorption circuit and sent to the elution process. The main difference between the CIL and CIP processes was that in the latter leaching and adsorption occur simultaneously, while in the former dissolution of the gold has already finished when adsorption starts.

Adams (1990), studied the kinetics of cyanide loss were found to be first order under any given set of conditions, allowing rate constants to be easily determined. At room temperature and in the absence of activated carbon, some cyanide is lost by hydrolysis to hydrogen cyanide, but this does not occur to any significant extent at pH values higher than about 10. A higher rate of hydrolysis occurs when air agitation is used. When activated carbon is present, an additional carbon-catalysed oxidation reaction is responsible for a fairly high loss of cyanide. The evidence indicates a reaction mechanism that consumes oxygen and produces cyanide ion. Some of the cyanide thus produced decomposes to form a mixture of ammonia, carbonate, and urea, depending on the solution conditions. Additional cyanide is lost as a result of the adsorption of sodium cyanide by the activated carbon. At high temperatures, an additional hydrolysis reaction, which involves the formation of ammonium format as an intermediate, occurs.

Reuter et al. (1991) presented a framework to simulate leaching, CIP and CIL plants. It is based on an object-oriented knowledge base system (KBS) used for dynamic simulation and fault diagnosis. Continuous models of the leaching, CIP and CIL processes were used as an example of the use of KBS. The system produced realistic simulations of experimental and published data.

Marsden and House (1992), presented The Chemistry of Gold Extraction bridges the gap between research and industry by emphasizing the practical applications of chemical principles and techniques. Covering what everyone in the gold extraction and processing industries should know: Historical Developments; Ore Deposits and Process Mineralogy; Process Selection; Principles of Gold Hydrometallurgy; Oxidative Pre-treatment; Leaching; Solution Purification and Concentration; Recovery; Surface Chemical Methods; Refining; Effluent Treatment; and Industrial Applications.

Schubert et al. (1993), A dynamic simulator of the leaching and adsorption cascades of a gold plant was developed and used to compare the effect of various disturbances on the overall efficiency of the plant. These comparisons show which disturbances most critically need to be controlled to optimize plant performance. The simulator was based on an existing gold plant. Dynamic and steady-state data were collected on the plant and were used in the modelling of the process. Simple dual-rate models were used to describe the leaching and adsorption processes. Mole-balances of various species take into account the changes of concentrations in the leach, and also the movement of pulp and carbon up and down the adsorption cascade. Fluctuations in the flow rate, density, and cyanide concentration of the feed to the leach have a negative effect on the efficiency of the leach, while carbon attrition was found to be an important factor in decreasing the efficiency of the carbon-in-pulp (CIP) section. Badly regulated transfers contribute towards the decreased efficiency of the CIP section, but overflows and carbon leakage have only a limited long-term effect. Various carbon profiles and transfer schemes were studied, and the results indicated that a slightly heavier loading at the bottom of the cascade is beneficial, while the type of transfer is relatively unimportant.

Aldrich et al. (1994), In this study the use of neural nets to model gold losses on a reduction plant and the consumption of an additive on a leach plant, as well as the pyrometallurgical processing of zinc and aluminium was discussed. The gold and leach plant models performed better than the multilinear regression models used on the plants, even where relatively few data were available. The neural networks used to model the recovery of lead and zinc from industrial flue dusts, process synthesis of zinc recovery plants and the processing of secondary aluminium in a rotary salt flux furnace produced realistic results that could be used by plant personnel to optimize their operations.

Annandale et al. (1996), have studied the interrelationship between mineral liberation and leaching behaviour of a gold ore is ill defined, mainly due to the complexity of both leaching and mineral liberation. A better understanding of this relationship could result in lower operating costs on gold extraction plants, since an increase in the efficiency of gold dissolution and a decrease in costs related to the crushing and grinding operations could be expected. In this investigation artificial neural nets were used to analyse diagnostic leaching data of gold ores obtained from South African gold mines. A self-organising neural net with a Kohonen layer was used to generate order preserving topological maps of the characteristics of both the unmilled and milled ores. The arrangement and shapes of these clusters could then be used to develop simple neural net models which were capable of predicting the degree of liberation more accurately than previously proposed models. Moreover, the neural net models were also capable of providing direct estimates of the reliability of their predictions by comparing new inputs with the data in their training bases.

Coetzee and Gray (1999), in this study Carbon-in-pulp (CIP) and carbon-in-leach (CIL) technology is applied successfully to recover gold from cyanided pulps. These plants are operated on a counter-current basis allowing for small concentrations of gold to be recovered at the expense of inefficiencies such as back-mixing, additional carbon breakage by interstage pumps and difficulties in carbon inventory management. In this study, linear regression techniques were utilised in order to study the adsorptive behaviour of carbon in gold bearing solutions with the aim of determining whether a partial co-current mode of operation will be feasible in CIP operations. It has been found that these empirical techniques are able to predict the adsorption rate of gold onto activated carbon and that the point of discontinuity of the constant rate adsorption period can accurately be determined. Thus, the behaviour of a single activated carbon particle can be predicted making use of standard linear regression techniques and using inputs such as, slurry density, rate of agitation, carbon loading and carbon and gold concentration. This information can then be utilised to maximise the use of the constant rate adsorption period by using a partial co-current mode of operation and reducing inefficiencies such as back-mixing and gold loss due to the abrasive action of interstage pumping on the adsorbent.

Rees and van Deventer (2001), in this study an overall batch model was developed to describe the simultaneous processes of leaching, preg-robbing and adsorption onto activated carbon. These processes were quantified for an oxide ore and three sulphide ores, which consisted of a flotation feed, a pyrite concentrate and a copper concentrate. The effect of the kinetics of preg-robbing on gold extraction with or without activated carbon was also closely examined for these ores. A variable order empirical Mintek-type model described the leaching, while the film diffusion equation with a Freundlich isotherm was used to describe both the preg-robbing at the ore surface and adsorption of aurocyanide onto activated carbon. The model developed fitted the observed experimental data well, and was used to decouple preg-robbing from leaching for the refractory ores and simulate the simultaneous leaching, adsorption and preg-robbing processes. Overall, the most important factor governing the extraction of gold was found to be the relative rates of leaching, adsorption and preg-robbing.

De Andrade Lima (2001), investigated a survey of the possibilities of the simulation techniques for improving the design and operation of gold processing plants. The procedure to build kinetic models that are used in simulators is first reviewed, and examples are given for cyanide gold dissolution and gold adsorption on carbon. Then an overview on the structures of the plant simulators is proposed, and the simulation equations formulated for a carbon-in-leach process. Finally, the potential of simulation methods is illustrated by four examples. The first one shows how it is possible to use a simulator to design a cascade of leaching tanks while optimizing an economic performance criterion. The second one illustrates the role of simulators for optimal tuning of the cyanide addition strategy in a leaching circuit. The third example focuses on the power of simulation to design carbon transfer strategies in a carbon-in-pulp process. Finally, the last example shows the role of on-line simulator for optimally supervising the automatic control of cyanide addition into leaching tanks.

De Andrade Lima et al. (2002), studied Carbon-in-leach and carbon-in-pulp are continuous processes that use activated carbon in a cascade of large agitated tanks. Which have been widely used to recover or concentrate precious metals in gold extraction plant. In the carbon-in-pulp process adsorption occurs after the leaching cascade section of the plant, and in the carbon-in-leach process leaching and adsorption occur simultaneously. In both processes the activated carbon is moved from one tank to another in counter current with the ore pulp until the recovery of the loaded carbon in the first tank. This study presented a dynamic model that describes, with minor changes, (lie carbon-in-leach, the carbon-in-pulp, and the gold leaching processes. The model is numerically solved and calibrated with experimental data from a plant and used to perform a study of the effect of the activated carbon transfer strategy on the performance of the adsorption section of the plant. Based oil the calculated values of the gold loss in the liquid and of the gold recovered in the loaded activated carbon that leaves the circuit, the results indicate that strategies in which a significant amount of activated carbon is held in the first tank and the contact time between the carbon and the Pulp is longer are the best carbon transfer strategies for these processes

Pornsawan Jongpaiboonkit (2003) studied the development and application of a dynamic model of gold adsorption onto activated carbon in gold processing. The primary aim of the model was to investigate different carbon management strategies of the Carbon in Pulp (CIP) process. This model is based on simple film-diffusion mass transfer and the Freundlich isotherm to describe the equilibrium between the gold in solution and gold adsorbed onto carbon. A major limitation in the development of a dynamic model is the availability of accurate plant data that tracks the dynamic behaviour of the plant. This limitation is overcome by using a pilot scale CIP gold processing plant to obtain such data. All operating parameters of this pilot plant can be manipulated and controlled to a greater degree than that of a full-scale plant. This enables a greater amount of operating data to be obtained and utilised. Two independent experiments were performed to build the model. The model was then used to optimise the operations of the pilot plant. The evaluation of the plant optimisation simulations was based on an objective function developed to quantitatively compare different simulated conditions.

De Andrade Lima and Hodouin (2005), The effect of particle size on the kinetics of gold cyanidation was investigated for a gold ore from the Abitibi region (Canada). Six size fractions representative of the plant operation were used to carry out the gold leaching and cyanide consumption kinetic tests. A two-level factorial experimental design was used for the gold leaching tests of the six size fractions. The free cyanide concentrations were selected as 260 and 650 mg/L, the dissolved oxygen concentrations as 8 and 40 mg/L, and the pH was maintained constant at 12. The gold concentration as a function of time was fitted to a lumped pseudo-homogeneous kinetic model, using a least-squares method. The results show that the gold dissolution kinetic reaction orders are approximately two for gold, one for free cyanide and a quarter for dissolved oxygen, and that the kinetic rate constant is a decreasing cubic function of the ore particle average diameter. In the cyanide consumption tests, conducted also for the six different size fractions, the pH was maintained at 12, the dissolved oxygen concentration was chosen as 8 mg/L, and the initial free cyanide concentration as 260 mg/L. The cyanide consumption lumped kinetic model shows that the reaction order is approximately three for free cyanide, and that the rate constant is a decreasing reciprocal square-root function of the ore particle size. Although finer grind of the ore enhances gold release, it considerably increases the cyanide consumption due to the liberation of the copper and iron bearing minerals, thus leading to a possible optimization of the ore size fraction that can simultaneously promote high gold recovery and acceptable cyanide consumption.

Bea Mine Technical report (2013), This Technical Report on the New Liberty Gold Mine (New Liberty, NLGM or the Project) within the Bea Mountain Mineral Development Agreement (Bea-MDA) property in Liberia, West Africa, has been compiled by SRK Consulting (UK) Ltd (SRK), for Avesoro Resources Inc. (Avesoro). Bea Mountain Mining Corporation (BMMC or the Company), which is a 100% owned subsidiary of Avesoro, has a 100% interest in the Bea-MDA. This report has been compiled by SRK, describes the Project in its current stage of development, presents SRK's opinions on the Mineral Resource and Mineral Reserve and current production forecast and presents an updated economic model and cash flow forecast which is based on the Life of Mine (LoM) plan currently in place

3. MATERIALS AND METHOD

3.1. Materials

3.1.1. The Metallurgical Laboratory

All the representative samples used in this study have been collected at the New Liberty Processing Plant and the fresh ore at the stockpile. The main purpose of the metallurgical laboratory is to ensure that the grinding circuits meets it parameter or targets, which include: the 80% of materials (ore) passing 75 microns to meet suitable cost reduction in extracting the concentrated mineral (Gold) using cyanide.

The purpose of the metallurgical lab is basically quality control (QC) and quality assurance (QA) of the entire gold processing plant.

Besides grinding, the metallurgical laboratory also does major test works on all of the reagents used on the processing plant to ensure all chemicals are mixed with the require percentage.

3.1.2. Simcil Simulation Tool

A model was developed of a CIL/CIP circuit for recovery of gold and it uses a simple two rate representation of the leaching kinetics. Parameters for the leaching kinetics may be simply obtained by sampling the feed to a circuit and the tailings from each tank and assaying the solids for both gold and silver to obtain the assay profiles of the solids down the tanks. The parameters in such a leaching model would be expected to be a function of the ore type, the grind, whether oxygen was used and the general chemical environment. These parameters are primary inputs to the model for any simulation.

Standard power law relationships are used to express the practical equilibrium between the gold and silver cyanides adsorbed on the active carbon and the tenor of the solutions. The parameters for this may be derived from solution and carbon assays for the same set of samples.

The model is constructed by writing a mass balance for gold in and out of each tank in the train and for the circuit overall. Other inputs to a simulation are the feed rate of ore, per cent solids of the slurry in the tanks, gold loadings of the input 'barren' carbon and the rate of movement of carbon through the circuit counter-current to the slurry. With appropriate practical parameters the simulation is able to reproduce typical tank profiles.

The model was used in a series of simulations to explore the influence of:

- leaching parameters,
- variation in ore feed rate and per cent solids of slurry,
- variation in rate of counter-current movement of active carbon,
- the number of tanks,
- tank size,
- tankage configuration,
- using some tanks for leaching only, and
- the impact of a thickener recycling a percentage of tailings solution to the head of the circuit.

The data used for calibrating the model and the baseline simulation was data for the six months of 2020 treating ore from the New Liberty open pit.

3.2. Methods

3.2.1. Particles Size Distribution

The main purpose of the metallurgical laboratory is to ensure that the grinding circuits meets it parameter or targets, which include: the 80% of materials (ore) passing 75 microns to meet suitable cost reduction in extracting the concentrated mineral (Gold) using cyanide.

Data Manipulation: The sample was collected from the Pre-Ox to investigate the influence of the size particles (Table 3.1). The weight of the sample retained on each sieve was calculated by the total weigh and below is the chart.

Table 3.1. Pre-Ox screen analysis

CLUSTER 2 CYCLONE OVERFLOW/PRE-OX			
Size (microns)	Mass (g)	Mass (%)	Passing (%)
+345	2.7	0.37	99.63
+300	1.23	0.17	99.47
+212	10.9	1.48	97.99
+180	10.65	1.44	96.54
+150	17.48	2.37	94.17
+106	63.03	8.55	85.63
+75	96.7	13,11	72.51
+63	34.85	4.73	67.78
+45	103.58	14.05	53.74
-45	396.21	53.74	
Total	737.33	100.00	
P80		93	microns
Interpolation			
Size (microns)	Passing (%)		
106	85.63		
93	80.00		
75	72.51		

Grinding Time (min)	D80 (mm)	% <0.037 mm (−400 mesh)	% <0.106 mm (−150 mesh)	% <0.212 mm (−65 mesh)
15	0.850	21	26	40
35	0.318	25	33	51
65	0.106	35	49	76

3.2.2. Adsorption Testing on Activated Carbon

The quality of activated carbon is of critical importance to the recovery of gold in CIL. Losses of gold due to poor carbon performance can be classified as follows:

- Gold adsorbed by preg. -robbing ore rather than carbon
- Gold adsorbed by carbon fines
- Gold in solution not adsorbed by carbon

The properties of activated carbon that are critical for plant performance are:

- Rate of gold adsorption (kinetic activity),
- Amount of gold loaded onto carbon (equilibrium capacity), and
- Resistance of the carbon to fines formation (attrition resistance).

Commercial products are supplied in specific size ranges and made from coconut shells or extruded carbon based on peat or coal. Quality of the Carbon made from coconut shells can vary with respect to activity and strength.

Usually, several carbon activity tests are carried out using different types of carbon or carbon at various degrees of fouling, together with a control test, which is freshly re-activated carbon. ν Other testing typically requires to characterize new (first use) carbon or existing plant carbons are:

- Attrition tests
- Size analysis
- Hardness
- Elution (desorption)

3.2.3. Leach optimization test

3.2.3.1. Leach test at pH 10.50 and Different Cyanide dosages

The main aim of this test was to compare the recovery on fresh ore when keeping pH parameter at similar condition and changing cyanide dosage.

As a result, the sample preparation and bottle roll test were conducted as follows:

- Live sample was collected from Pre-Ox discharge.
- The Solids percentage, Dissolved oxygen and pH were observed and noted.
- Sample was stirred with laboratory agitator and 2 bottles were marked with cyanide dosage required.
- Four litres of sample were transferred into each bottle.
- A head grade and gold deportation samples (1 and 5 litre respectively) were collected.
- Particle size analysis and gold deportation analysis were respectively conducted from the two fractions of the sample.
- Lime was added to raise the pH to 10.50 from each test bottle.
- Bottle marked with 0.70 kg/ton and 1.20 kg/ton were added with cyanideaccording to their respective labels.
- After the test, samples were filtered, filtrate was captured, and the residues washed twice to remove any traces of leach liquor.
- The samples were sent for gold analysis at the assay laboratory.

3.2.3.2. Leach test on fresh ore at pH 9.50 and 10.00 and 1 kg/t NaCN

The main aim of this test was to evaluate the recovery on fresh ore when changing pH parameter at similar cyanide dosage.

As a result, the sample preparation and bottle roll test were conducted as follows:

- The sample was screen to minus 300 µm.
- Sample was homogenized by mixing and rolling the under 300 µm in a drum.
- Sample was split using jones riffle splitter at laboratory to17 fractions.
- Particle size analysis and gold deportation analysis were respectively conducted from the two fractions of the sample.
- Two dry samples were pulped to 45% solids.
- Lime was added to raise the pH to 9.50 and 10.00, respectively.
- The two samples were pre-oxidised for 4 hours with oxygen.
- DO's were noted and lime was added to readjust the pH to 9.50 and 10.00 respectively.
- 1 kg/t of cyanide was added on each sample.
- After the test, samples were filtered, filtrate was captured and the residues washed twice to remove any traces of leach liquor.
- The samples were sent for gold analysis at the assay laboratory.

3.2.4. Leach Residence time

Measure kinetics of gold cyanide adsorption on carbon at selected pulp density in small scale batch test. Fit adsorption data to adsorption equation and generate constants k and K.

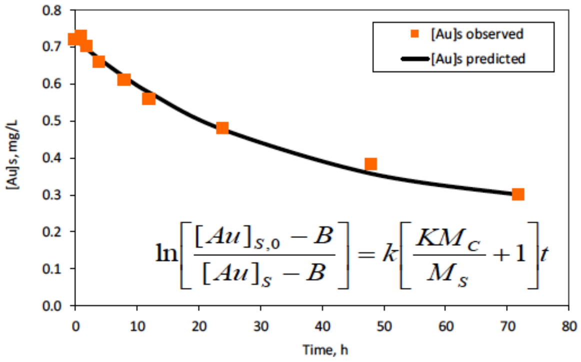


Figure 3.1. leach adsorption

3.2.5. Simulation Model development

The treatment circuits at the Bea Mine will treat new ores from the Ndablama and the New Liberty underground deposits. The treatment rates, amount of gravity gold. leach feed grades and extraction rates will vary over time. There will be two identical comminutions, leaching an CIL plants. However, it is proposed that that there will be a common elution andelectrowinning circuit for the two plants. Questions have been raised as to whether the current elution circuit will be able to cater for changes made to the operating conditions in the new circuits. In order to access the impact of some of these changes on the elution circuit it was proposed the elution modelling and simulation be undertaken. The models and elution simulation were first calibrated (baseline simulation) using current operating data and then a suite of simulations were undertaken with varying inputs.

The outcomes of the modelling and simulations are summarized in the following table (Table 3.2).

Table 3.2. Executive summary

Model Scenario	Treatment Rates	Leach Rate Constant	CIL Feed Grade	Elutions	Solution Loss	Carbon Loading	Carbon Inventory	Gold Inventory	Recovery
	dtph	(k)	(g/t)	(per day/week)	(ppm)	g/t	(t)	(kg)	(%)
1 Baseline -Currenr Circuit data	180	2.00	1.53	1.15/8.1	0.006	997	47.3	12.7	83.8
2 Base-Line -Reduced elution rate x2	180	2.00	1.53	0.58/4.1	0.008	1959	47.3	25.3	83.7
3 Head 3g/t; gravity 30%, tail 0.21g/t	180	2.00	2.10	0.5/3.5	0.01	3300	47.3	41.1	89.3
4 Head 3g/t; gravity 40%, tail 0.21g/t	180	2.00	1.80	0.5/3.5	0.009	2750	47.3	35	87.7
5 Head 3g/t; gravity 50%, tail 0.21g/t	180	2.00	1.80	0.5/3.5	0.009	2250	47.3	29.6	85.3
6 Scenario 3 with leach rate k-1.0, tail 0.21 g/t	180	1.00	2.10	0.5/3.5	0.013	3300	47.3	44.6	89.3
7 Leach feed grade 2.6g/t. k=1, tail 0.21g/t	180	1.00	2.60	0.5/3.5	0.014	4150	47.3	54.1	91.3
8 Leach feed grade 2.6g/t. k=2, tail 0.21g/t	180	2.00	2.60	0.5/3.5	0.011	4150	47.3	50.1	91.3
9 Leach feed grade 2.38g/t, tail 0.24g/t, barren 173g/t	212	1.00	2.38	0.51/3.6	0.029	4359	47.3	66.8	88.5
10 Leach feed grade 2.38g/t, tail 0.24g/t, barren 100 g/t	212	1.00	2.38	0.51/3.5	0.024	4359	47.3	66.4	88.8
11 Leach feed grade 1.88g/t, tail 0.19g/t, barren 153g/t	212	1.00	1.88	0.46/3.2	0.028	3816	47.3	61.6	88.3
12 Leach feed grade 1.88g/t, tail 0.19g/t, barren 100 g/t	212	1.00	1.88	0.46/3.2	0.024	3816	47.3	60	88.6
13 Leach feed grade 1.2g/t, tail 0.12g/t, barren 98g/t	212	1.00	1.20	0.46/3.2	0.02	2442	47.3	41.8	88.0
14 Leach feed grade 1.2g/t, tail 0.12g/t, barren 50g/t	212	1.00	1.20	0.46/3.2	0.017	2442	47.3	49.3	88.3

- **Operating Data for first four months of 2020**

The data used for calibrating the models and the baseline simulation was data for the first six months of 2020 treating ore from the New Liberty open pit.

- **Calibration**

The equations that describe the CIL were calibrated with the daily data for March, April, May and June from a plant that treats about 180 t/h of ore. Samples were simultaneously taken at the entrance and outlet of each tank in the leaching and adsorption section of the plant every day. After filtration of the sampled pulp, the liquid phase was analysed. The pH was measured with a standard pH meter, the cyanide concentration was determined by titration with silver nitrate and rhodamine, and the concentration of gold was measured by atomic absorption spectrophotometry. Every two hours, samples of the pulp were taken and filtered to measure the concentration of carbon in the tanks and their gold content was measured by fire assay and atomic absorption spectrophotometry.

The assumptions made for the simulation model include:

- Feed Rate
- Solid's grade
- Pulp density
- Solids and solution Tails
- Loaded carbon
- Barren carbon
- Carbon inventor
- Elution Column

To build this simulation model, three aspects were considered: the rate of adsorption, the mixing in the tanks and the exchange of mass between the tanks.

The first aspect involves using an appropriate rate of adsorption. The second aspect involves the implicit assumption of the model that all the tanks were well mixed. The third and final aspect was the configuration of the process and the description of the mass balance equations describing the interaction of the pulp and carbon flow between the CIL tanks. All the parameters for this relationship were derived from plant data. The model was constructed by writing a mass balance for gold in and out of each tank in the train and for the circuit overall. Other inputs to the simulation were the federate of ore, per cent solids of the slurry in the tanks, gold loadings of the input 'barren' carbon and the rate of movement of carbon through the circuit counter current to the slurry (Table 3.3).

Table 3.3. Plant calibration data

Feed Rate:	180.5 tph
Feed	2.2 g/t
Solids Grade:	1.530 g/t
Solution Grade:	0.000 ppm
Pulp Density:	45.00 %
Solids Tails	0.215 g/t
Solution Tails	0.010 ppm
Tank Volumes:	1000
Loadend Carbon:	1104 g/t
Barren Carbon:	35 g/t
Carbon Inventory:	47.3 t
Elution Column:	5 t
Stripping Frequency:	8.22 /week
Ore SG	2,90
Water SG	1

Carbon Concentration g/l					
Tank	June	May	April	March	Average
1	6.0	3.5	7	4.5	*5,3*
2	6.5	9.5	7	5	*7*
3	5.5	4	5	4	*4.6*
4	5	5	5	7	*5.5*
5	11	10.5	5	8,5	*8.8*
6	14	15.5	17	18	*16.1*
Inventory (t)	*48*	*48*	*46*	*47*	47,3
Solids Profiles, ppm					
Tank	June	May	April	March	Average
1	0.23	0.47	0.56	0.47	*0.432*
2	0.14	0.33	0.44	0.36	*0.316*
3	0.12	0.28	0.39	0.30	*0.272*
4	0.12	0.27	0.36	0.29	*0.261*
5	0.11	0.26	0.31	0.27	*0.238*
6	0.11	0.25	0.30	0.24	*0.225*
Solution Profiles,ppm					
Tank	June	May	April	March	Average
1	0.206	0.39	0.201	0.21	*0.251*
2	0.045	0.09	0.153	0.14	*0.105*
3	0.023	0.04	0.057	0.08	*0.051*
4	0.020	0.02	0.042	0.03	*0.028*
5	0.010	0.01		0.01	*0.011*
6	0.010	0.01	0.01	0.01	*0.01*
Carbon Grade, ppm					
Tank	June	May	April	March	Average
1		1075	722	875	891
2	258	406	298	537	375
3	231	274	252	336	273
4	103	128	144	177	138
5	65	106	119	125	104
6	40	64	62	63	57

Note significant difference in carbon loading (grade) between Tank 1 from the GIC data (891 g/t) and the loaded carbon grade from the elution data (1104 g/t). Mass balancing shows that using the lower number results in a predicted large number of elution; using the lower number predicts fewer elutions than actual. Numbers were averaged to give a loaded carbon value of 997 g/t.

This data (Table 3.3.) was used to calibrate SIMCIL for the circuit.

3.2.5.1. Mass balance from SIMCIL

Values enter the circuit primarily contained in the ground ore fed to the circuit. Water being recycled to the circuit may contain values especially water returned from a thickener which is intended to recover unused cyanide and values which would otherwise be lost in solution in the tailings. Regenerated carbon may also contain some residual values. The mass balance for the circuit and for any tank may be written as follows:

$$G(h-t)=C(L_{out}-L_{in})+G\frac{(1-p)}{p}(s_{out}-s_{in}) \tag{3.1}$$

Where; G is the feed rate of ore, h is the head grade of the feed to the entity (circuit or tank), t is the tailings grade, p is the per cent solids in the slurry and s is the solution concentration. L_{out} and s_{out} are related by the adsorption equation. t is calculated progressively down the bank, the tailings from any tank becoming the head to the next tank. For the circuit overall the initial head and t the tails from the final tank.

These equations were programmed in Excel, solution outputs from one tank being the input to the next and the carbon output from one tank being the carbon input to the preceding tank. The number and function (leach only or leach and adsorption) of the tanks is set in any version of the program and the size may be input. Normal inputs apart from the leaching and adsorption parameters were the feed rate of ore, the per cent solids in the slurry, the carbon advance rate and the residual loadings of the regenerated feed carbon. If the effect of return of water containing values being recycled from a thickener was being explored then a further input was the fraction of water in final tails which was recycled. A direct analytical solution of the complex equation set is not readily obtained, so the circuit was generally solved by progressive approximation of the solution concentration of the final tails. Alternatively, the final solution concentration for either gold could be chosen and one of the other operating variables changed to give the selected concentration.

- **Leach and carbon factors**

Ideally the sampling of solution and carbon would be taken immediately prior to moving the carbon on to the next tank up the train. In this way the carbon has had maximum opportunity to equilibrate fully to the solution concentration in a tank. In practice because of head grade variations, the solution concentration in a tank does not remain steady and timing sampling to correspond to carbon movement is difficult to coordinate. However, if the approach of the carbon to equilibrium loading in any tank occurs at a reasonable rate (e.g., within a few hours) then there may be quite a big window for satisfactory samples to be obtained.

As concentrations of the gold and silver cyanides in solution (max. order of 40 ppm) and on the carbon (< 3%) are relatively low, the adsorption of gold and silver on active carbon in the model has been expressed by the Freundlich equation (Treybal, 1955):

$$\boldsymbol{F = A.S^N} \tag{3.2}$$

Where; L is the carbon loading, s is the solution concentration of gold in ppm and A [mg Au kg soln^b / (mg Au^b kg carbon)] and N are the adsorption constants. The model assumes that gold adsorb independently and that there is no interaction at typical maximum loadings. The model also assumes that adsorption and desorption may both occur in the circuit provided the carbon has adequate residence time in each tank. Given carbon residence times in each tank typically of the order of at least ten hours this is observed in practice. Hence the loading on the carbon removed from the circuit is considered to be in equilibrium with the solution in each tank regardless of the solution profile and especially with the solution in the last tank and the tank from which the final loaded carbon is removed. As will be seen this proves to be important when significant solution losses occur in circuit tails.

The following shows the input requirements for the simulator

Table 3.4. The input requirements for the simulator

What is the leach rate constant(k1) t/h.g?	2.00
What is the Au_e value?	0.20
What is the adsorption rate constant /h? (k)	0.015
What is the adsorption equilibrium constant (K or Freundlich 'a')?	13,500
What is the Freundlich exponent 'b'?	1.00

The current ore has a fast-leaching rate, k = 2, and the carbon activity is high.

3.2.5.2. Modelled circuit profiles for current circuit data

- **Modelling Scenario 1 Baseline**

Table 3.5. Scenario-1

Stage	Solution	Carbon	Carbon	Ore	Gold Extraction	Inventory Au kgs			
	Au (g/t)	Au (g/t)	g/L	Au (g/)	(%)	Solution	Solids	Carbon	Total
Feed				1.53					
CIL 1	0.241	997	5.3	0.56	44	0.51	0.344	5.2841	6.138
CIL 2	0.113	535	7	0.37	41	0.30	0.227	3.745	4.272
CIL 3	0.058	253	4.6	0.30	27	0.22	0.184	1.1638	1.568
CIL 4	0.027	143	5.5	0.27	19	0.17	0.166	0.7865	1.122
CIL 5	0.011	86	8.8	0.25	12	0.14	0.154	0.7568	1.050
CIL 6	0.006	57	16.1	0.24	7	0.12	0.147	0.9177	1.185
Total						1.46	1.222	12.654	**15.336**

Note: Predicted elution rate 8.05 elutions/week

Note that we would normally target the solution loss when fitting a circuit. However, it appears that the reported solution may be below the detection limit and is therefore not exactly known.

- **Comparing modelled profile and average plant profiles**

The comparison of actual plant versus model prediction is shown in the following three figures.

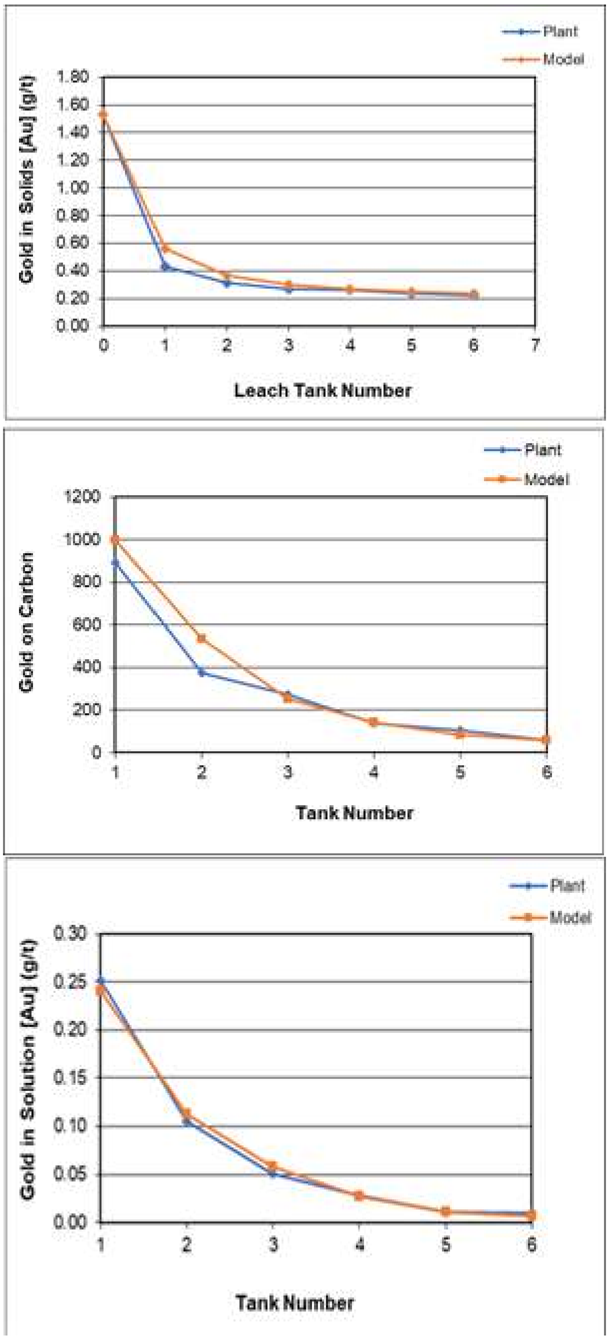


Figure 3.2. The comparisons profile

3.2.5.3. Modelling Scenario 2:

The simulation is the same as the baseline except the elution is carried out every second day (Table 3.6).

Table 3.6. Scenario-2

Stage	Solution	Carbon	Carbon	Ore	Gold Extraction	Inventory Au kgs			
	Au (g/t)	Au (g/t)	g/L	Au(g/T)	(%)	Solution	Solids	Carbon	Total
Feed				2.1					
CIL 1	0.519	3300	5.3	0.62	40	0.405	0.381	17.49	18.276
CIL 2	0.277	1901	7	0.36	44	0.216	0.221	13.307	13.744
CIL 3	0.144	908	4.6	0.28	35	0.112	0.172	4.1768	4.461
CIL 4	0.066	463	5.5	0.24	29	0.052	0.147	2.5465	2.745
CIL 5	0.027	225	8.8	0.22	21	0.021	0.135	1.98	2.136
CIL 6	0.01	103	16.1	0.21	13	0.008	0.129	1.6583	1.795
Total						0.815	1.185	41.159	**43.158**

3.2.5.4 Modelling Scenario 3

For this scenario the feed tonnage is 180 dtph, feed grade to the milling circuit is 3 g Au/t and gravity gold recovery is 30%. All other parameters input parameters are the same as Scenario 2 (Table 3.7).

Table 3.7. Scenario-3

Stage	Solution	Carbon	Carbon	Ore	Gold Extraction	Inventory Au kgs			
	Au (g/t)	Au (g/t)	g/L	Au(g/T)	(%)	Solution	Solids	Carbon	Total
Feed				1.53					
CIL 1	0.321	1959	5.3	0.57	37	0.251	0.350	10.3827	10.983
CIL 2	0.174	1146	7	0.37	39	0.136	0.227	8.022	8.385
CIL 3	0.093	561	4.6	0.3	29	0.073	0.184	2.5806	2.837
CIL 4	0.044	298	5.5	0.27	22	0.034	0.166	1.639	1.839
CIL 5	0.018	155	8.8	0.25	15	0.014	0.154	1.364	1.532
CIL 6	0.009	80	16.1	0.24	9	0.007	0.147	1.288	1.442
Total						0.515	1.228	25.276	**27.019**

3.2.5.5. Modelling Scenario 4

For this scenario the feed tonnage is 180 dtph, feed grade to the milling circuit is 3 g Au/t and gravity gold recovery is 40%. All other parameters input parameters are the same as Scenario 2 (Table 3.8).

Table 3.8. Scenario-4

Stage	Solution	Carbon	Carbon	Ore	Gold Extraction	Inventory Au kgs			
	Au (g/t)	Au (g/t)	g/L	Au (g/T)	(%)	Solution	Solids	Carbon	Total
Feed				1.8					
CIL 1	0.430	2750	5.3	0.58	38	0.336	0.356	14.575	15.267
CIL 2	0.235	1615	7.0	0.35	42	0.184	0.215	11.305	11.703
CIL 3	0.124	787	4.6	0.28	33	0.097	0.172	3.6202	3.889
CIL 4	0.058	409	5.5	0.24	27	0.045	0.147	2.2495	2.442
CIL 5	0.024	203	8.8	0.22	20	0.019	0.135	1.7864	1.940
CIL 6	0.009	96	16.1	0.21	12	0.007	0.129	1.5456	1.682
Total						0.687	1.154	35.082	**36.923**

3.2.5.6. Modelling Scenario 5

For this scenario the feed tonnage is 180 dtph, feed grade to the milling circuit is 3 g Au/t and gravity gold recovery is 50%. All other parameters input parameters are the same as Scenario 2 (Table 3.9)

Table 3.9. Scenario-5

Stage	Solution	Carbon	Carbon	Ore	Gold Extraction	Inventory Au kgs			
	Au (g/t)	Au (g/t)	g/L	Au (g/T)	(%)	Solution	Solids	Carbon	Total
Feed				1.5					
CIL 1	0.346	2250	5.3	0.54	36	0.270	0.332	11.925	12.527
CIL 2	0.195	1359	7	0.34	40	0.152	0.209	9.513	9.874
CIL 3	0.106	682	4.6	0.27	31	0.083	0.166	3.1372	3.386
CIL 4	0.051	363	5.5	0.24	25	0.040	0.147	1.9965	2.184
CIL 5	0.022	185	8.8	0.22	18	0.017	0.135	1.628	1.780
CIL 6	0.009	90	16.1	0.21	11	0.007	0.129	1.449	1.585
Total						0.569	1.117	29.649	**31.336**

3.2.5.7. Modelling Scenario 6

For this scenario the feed tonnage is 180 dtph, feed grade to the milling circuit is 3 g Au/t and gravity gold recovery is 30% and gold leach rate is slower at k = 1. All other parameters input parameters are the same as Scenario 3 (Table 3.10).

Table 3.10. Scenario-6

Stage	Solution	Carbon	Carbon	Ore	Gold Extraction	Inventory Au kgs			
	Au (g/t)	Au (g/t)	g/L	Au (g/T)	(%)	Solution	Solids	Carbon	Total
Feed				2.1					
CIL 1	0.497	3300	5.3	0.74	36	0.388	0.454	17.490	18.333
CIL 2	0.294	2066	7	0.43	41	0.230	0.264	14.462	14.956
CIL 3	0.165	1066	4.6	0.32	35	0.129	0.196	4.9036	5.229
CIL 4	0.081	573	5.5	0.26	31	0.063	0.160	3.1515	3.374
CIL 5	0.034	288	8.8	0.23	25	0.027	0.141	2.5344	2.702
CIL 6	0.013	129	16.1	0.21	17	0.010	0.129	2.0769	2.216
Total						0.847	1.345	44.618	**46.810**

3.2.5.8. Modelling Scenario 7

For this scenario the feed tonnage is 180 dtph, feed grade to the leach CIL circuit is 2.6 g Au/t and leach rate of k-1 other input parameters are the same as Scenario 2 (Table 3.11).

Table 3.11. Scenario-7

Stage	Solution	Carbon	Carbon	Ore	Gold Extraction	Inventory Au kgs			
	Au (g/t)	Au (g/t)	g/L	Au (g/T)	(%)	Solution	Solids	Carbon	Total
Feed				2.6					
CIL 1	0.634	4150	5.3	0.83	38	0.495	0.510	21.995	23.000
CIL 2	0.360	2514	7	0.46	44	0.281	0.282	17.598	18.162
CIL 3	0.196	1254	4.6	0.33	37	0.153	0.203	5.7684	6.124
CIL 4	0.093	657	5.5	0.26	33	0.073	0.160	3.6135	3.846
CIL 5	0.038	321	8.8	0.23	27	0.030	0.141	2.8248	2.996
CIL 6	0.014	140	16.1	0.21	18	0.011	0.129	2.254	2.394
Total						1.043	1.424	54.054	**56.521**

3.2.5.9. Modelling Scenario 8

For this scenario the feed tonnage is 180 dtph, feed grade to the leach CIL circuit is 2.6 g Au/t and leach rate of k-2, other input parameters are the same as Scenario 2 (Table 3.12).

Table 3.12. Scenario-8

Stage	Solution	Carbon	Carbon	Ore	Gold Extraction	Inventory Au kgs			
	Au (g/t)	Au (g/t)	g/L	Au (g/T)	(%)	Solution	Solids	Carbon	Total
Feed				2.6					
CIL 1	0.66	4150	5.3	0.69	43	0.515	0.424	21.995	22.934
CIL 2	0.34	2316	7	0.38	47	0.266	0.233	16.212	16.711
CIL 3	0.172	1072	4.6	0.29	38	0.134	0.178	4.9312	5.244
CIL 4	0.077	533	5.5	0.25	31	0.060	0.154	2.9315	3.145
CIL 5	0.03	252	8.8	0.23	23	0.023	0.141	2.2176	2.382
CIL 6	0.011	112	16.1	0.21	14	0.009	0.129	1.8032	1.941
Total						1.007	1.259	50.091	**52.357**

3.2.5.10. Modelling Scenario 9

For this scenario the feed tonnage is 212 dtph, feed grade to the leach CIL circuit is 2.28 g Au/t andleach rate of k-1, leach solid tail is 0.24 g Au/t and elution efficiency is 96% (eluate tail 171 g Au/t) (Table 3.13).

Table 3.13. Scenario-9

Stage	Solution	Carbon	Carbon	Ore	Gold Extraction	Inventory Au kgs			
	Au (g/t)	Au (g/t)	g/L	Au(g/T)	(%)	Solution	Solids	Carbon	Total
Feed				2.28					
CIL 1	0.631	4359	5.3	0.86	32	0.493	0.528	23.1027	24.124
CIL 2	0.406	2938	7	0.5	39	0.317	0.307	20.566	21.190
CIL 3	0.245	1656	4.6	0.36	33	0.191	0.221	7.6176	8.030
CIL 4	0.131	981	5.5	0.3	31	0.102	0.184	5.3955	5.682
CIL 5	0.053	563	8.8	0.26	26	0.041	0.160	4.9544	5.155
CIL 6	0.029	318	16.1	0.24	19	0.023	0.147	5.1198	5.290
Total						1.168	1.547	66.756	**69.471**

3.2.5.11. Modelling Scenario 10

For this scenario the feed tonnage is 212 dtph, feed grade to the leach CIL circuit is 2.28 g Au/t and leach rate of k-1, leach solid tail is 0.24 g Au/t and elution efficiency is 98% (eluate tail 100 g Au/t). As a rule, elution efficiency increases with higher loading on the carbon (Table 3.14).

Table 3.14. Scenario-10

Stage	Solution	Carbon	Carbon	Ore	Gold Extraction	Inventory Au kgs			
	Au (g/t)	Au (g/t)	g/L	Au (g/T)	(%)	Solution	Solids	Carbon	Total
Feed				2.28					
CIL 1	0.631	4359	5.3	0.86	32	0.493	0.528	23.1027	24.124
CIL 2	0.405	2918	7	0.5	39	0.316	0.307	20.426	21.049
CIL 3	0.242	1612	4.6	0.36	34	0.189	0.221	7.4152	7.825
CIL 4	0.174	923	5.5	0.3	31	0.136	0.184	5.0765	5.397
CIL 5	0.057	497	8.8	0.26	27	0.045	0.160	4.3736	4.578
CIL 6	0.024	246	16.1	0.24	19	0.019	0.147	3.9606	4.127
Total						1.197	1.547	64.355	**67.099**

3.2.5.12. Modelling Scenario 11

For this scenario the feed tonnage is 212 dtph, feed grade to the leach CIL circuit is 1.88 g Au/t andleach rate of k-1, elution, leach solid tail is 0.19 g Au/t and elution efficiency is 96% (eluate tail 153 g Au/t) (Table 3.15).

Table 3.15. Scenario-11

Stage	Solution	Carbon	Carbon	Ore	Gold Extraction	Inventory Au kgs			
	Au (g/t)	Au (g/t)	g/L	Au (g/T)	(%)	Solution	Solids	Carbon	Total
Feed				1.88					
CIL 1	0.519	3816	5.3	0.72	28	0.405	0.442	20.2248	21.072
CIL 2	0.357	2710	7	0.42	37	0.279	0.258	18.97	19.507
CIL 3	0.225	1594	4.6	0.3	33	0.176	0.184	7.3324	7.692
CIL 4	0.125	964	5.5	0.24	32	0.098	0.147	5.302	5.547
CIL 5	0.061	555	8.8	0.21	29	0.048	0.129	4.884	5.061
CIL 6	0.028	303	16.1	0.19	22	0.022	0.117	4.8783	5.017
Total						1.027	1.277	61.592	**63.896**

3.2.5.13. Modelling Scenario 12

For this scenario the feed tonnage is 212 dtph, feed grade to the leach CIL circuit is 1.88 g Au/t and leach rate of k-1, leach solid tail is 0.19 g Au/t and elution efficiency is 97% (eluate tail 100 g Au/t). As a rule, elution efficiency increases with higher loading on the carbon.

Table 3.16. Scenario-12

Stage	Solution	Carbon	Carbon	Ore	Gold Extraction	Inventory Au kgs			
	Au (g/t)	Au (g/t)	g/L	Au (g/T)	%	Solution	Solids	Carbon	Total
Feed				1.88					
CIL 1	0.519	3816	5.3	0.72	28	0.405	0.442	20.2248	21.072
CIL 2	0.357	2710	7	0.42	37	0.279	0.258	18.97	19.507
CIL 3	0.225	1594	4.6	0.30	33	0.176	0.184	7.3324	7.692
CIL 4	0.125	964	5.5	0.24	32	0.098	0.147	5.302	5.547
CIL 5	0.061	555	8.8	0.21	29	0.048	0.129	4.884	5.061
CIL 6	0.028	308	16.1	0.19	22	0.022	0.117	4.9588	5.097
Total						1.027	1.277	61.672	**63.976**

3.2.5.14. Modelling Scenario 13

For this scenario the feed tonnage is 212 dtph, feed grade to the leach CIL circuit is 1.20 gAu/t andleach rate of k-1, elution, leach solid tail is 0.12 gAu/t and elution efficiency is 96% (eluate tail 98 gAu/t)

Table 3.17. Scenario-13

Stage	Solution	Carbon	Carbon	Ore	Gold Extraction	Inventory Au kgs			
	Au (g/t)	Au (g/t)	g/L	Au(g/T)	%	Solution	Solids	Carbon	Total
Feed				1.2					
CIL 1	0.321	2442	5.3	0.51	25	0.251	0.313	12.9426	13.506
CIL 2	0.235	1815	7	0.31	35	0.184	0.190	12.705	13.079
CIL 3	0.154	1098	4.6	0.22	32	0.120	0.135	5.0508	5.306
CIL 4	0.087	662	5.5	0.17	32	0.068	0.104	3.641	3.813
CIL 5	0.041	363	8.8	0.14	31	0.032	0.086	3.1944	3.312
CIL 6	0.017	170	16.1	0.12	26	0.013	0.074	2.737	2.824
Total						0.668	0.903	40.271	**41.841**

3.2.5.15. Modelling Scenario 14

For this scenario the feed tonnage is 212 dtph, feed grade to the leach CIL circuit is 1.20 g Au/t andleach rate of k-1, elution, leach solid tail is 0.12 g Au/t and elution efficiency is 96% (eluate tail 50 gAu/t)

Table 3.18. Scenario-14

Stage	Solution	Carbon	Carbon	Ore	Gold Extraction	Inventory Au kgs			
	Au (g/t)	Au (g/t)	g/L	Au(g/T)	%	Solution	Solids	Carbon	Total
Feed				1.2					
CIL 1	0.321	2442	5.3	0.51	25	0.51	0.313	12.9426	13.766
CIL 2	0.235	1815	7	0.31	35	0.30	0.190	12.705	13.195
CIL 3	0.154	1098	4.6	0.22	32	0.22	0.135	5.0508	5.406
CIL 4	0.087	662	5.5	0.17	32	0.17	0.104	3.641	3.915
CIL 5	0.041	363	8.8	0.14	31	0.14	0.086	3.1944	3.420
CIL 6	0.017	170	16.1	0.12	26	0.12	0.074	2.737	2.931
Total						1.46	0.903	40.271	**42.633**

4. RESULTS AND DISCUSSION

The results open from the leach and the simulation was discussed in this chapter.

4.1. Adsorption Testing on Activated Carbon

The following testing report identify the testing phases completed on carbon from four different companies (JACOBI, CEYCARD, GOLDPLUS and HYCARD). The test performance goals were to identify between variety of carbon products that we received from different company that have a good adsorption record. The test procedure consists to mix 2 g activated carbon in 500 ml solution of NaCN and during each 1-3-5-7-9-11 minutes 10 ml solution was removed from the 500 ml solution for each test Figure 4.1).

Table 4.1. Carbon adsorption in ppm

	JACOBI	CEYCARB	GOLD PLUS	HYCARD
1	1.790	1.844	1.778	1.769
3	1.764	1.684	1.639	1.662
5	1.638	1.501	1.511	1.569
7	1.499	1.341	1.387	1.461
9	1.409	1.210	1.223	1.365
11	1.333	1.133	1.126	1.240

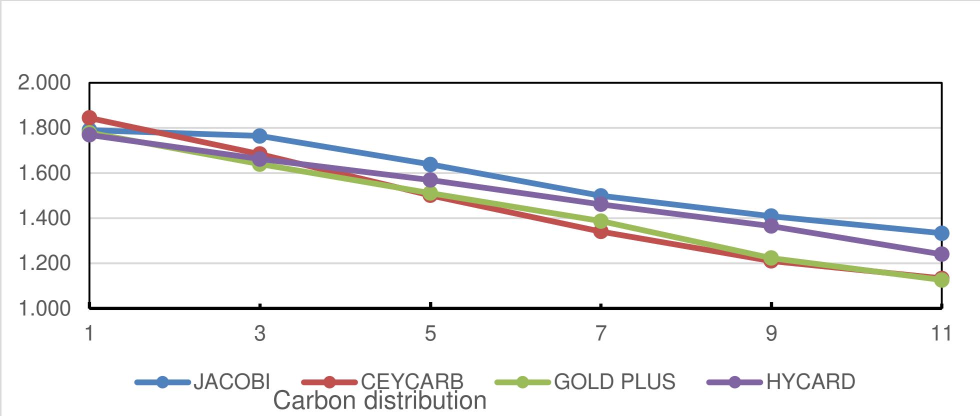


Figure 4.1. Carbon distribution from different company

According to the result obtained, at the first minute CEYCARD presented the high absorption but at the end of the analysis it was identified that GOLDPLUS has the best adsorption record as its solution contain less amount of gold. It was chosen for the leach test.

4.2. Leach optimization test

4.2.1. Leach test at pH 10.50 and Different Cyanide dosages

Sample Pre-Ox discharge

Table 4.2. Pre-Ox feed sample particle size analysis

PARTICLE SIZE ANALYSIS			
Size (microns)	Mass (g)	Mass (%)	(%) Passing
+150	59.33	10.13	89.87
+106	55.87	9.54	80.33
+75	69.69	11.90	68.44
+45	93.84	16.02	52.42
-45	30.08	52.42	
Total	585.81	100.00	
P80		105	microns

From Table 4.2, it was observed that the sample was coarse because the percentage passing 75 microns screen was 68.44%. As the grinding circuit Ball mill and Verti mill in closed circuit with the with a cyclone cluster provided a grinding of 80% passing 75 microns, the percentage passing should be between 75-80% in the Pre-Ox at the same time. It must also be highlighted that splitting sample when it is slurry does introduce certain degree of bias.

Gold deportation

Table 4.3. Gold deportation

SIZE BY SIZE GOLD RECOVERY					
		0.7 kg/ton NaCN		**1.2 kg/ton NaCN**	
Size (microns)	Au (g/t) Head	Au (g/t)	(%) Recovery	Au (g/t)	(%) Recovery
+150	1.17	0.55	52.99	0.52	55.56
+106	1.36	0.52	61.76	0.61	55.15
+75	1.47	0.47	68.03	0.57	61.22
+45	1.08	0.46	57.41	0.59	45.37
-45	1.22	0.41	66.39	0.45	63.11
Average	1.26	0.48	61.32	0.55	56.08

The above gold deportation Table 4.3 observed that the gold is associated with the coarse particles when looking at the head grade. The highest recovery is associated with the minus 45 microns. Leaching takes place in coarser material is possible when the coarse material has an exposed gold. The recovery across all size fraction for 0.70 kg/ton and 1.20 kg/ton NaCN dosage was below 70%. The average head grade is below 2 g/t this is due to the fact that the sample was not conventionally divided.

Sample: Pre-Ox discharge grade comparison

It was observed that sampling and dividing a wet sample introduces a degree of bias to the whole fraction as seen in Table 4.4. It was observed that the size-by-size average grade is differs with the assayed head grade. The head sample and average size fraction grades but close to the latter by 14.20%.

Table 4.4. Grade comparison

Head Grade Comparison		
Assayed Head	Calculated Grade (average)	Average size fraction grade
g/t	g/t	g/t
2.06	2.4	1.27

Leach summary results

The below data in Table 4.5 demonstrated that comparing the two tests with respect to recovery it was observed that the recovery of leach at 0.70 kg/ton in comparison to 1.20 kg/ton yields 81.27 % and 78.77% respectively. Test conducted at 1.20 kg/ton had a cyanide consumption of 0.96 kg/ton when comparing it to 0.37 kg/ton of test at 0.70 kg/ton NaCN dosage. The DO's before and after leach test were below 6.00 ppm.

Table 4.5. Leach summary results

	TEST 1	TEST 2
Description	NaCN 0.70 kg/ton	NaCN 1.20 kg/ton
Dry Density g/cm3	2.8	2.8
Pre-Oxidation time (hours)	4	4
Leach Time (hrs)	24	24
Sample mass (g)	2686	2686
solution Volume (ml)	3153.00	3153.00
% Solids (after 24hrs)	46.0	46.0
DO's After Pre-Oxidation (ppm)	5.88	5.88
pH Monitoring		
pH before NaCN addition	10.50	10.50
Final pH	9.90	10.04
Reagents added		
NaCN added (kg/t)	0.70	1.20
Reagent Residuals		
Free CN ppm	166.31	436.70
residual g/t	0.33	0.24
NaCN consumed(kg/ton)	0.37	0.96
DO's after 24 hours leach (ppm)	5.96	5.02
WAD Generated	248.00	479.85
GOLD		
Head g/t	2.40	2.40
Residue g/t (Solids)	0.45	0.51
Solution g/t	1.495	1.621
% Recovery (Solids)	81.27	78.77
% Accountability	91.80	100.45

As seen in Table 4.5, comparing the two tests with respect to recovery, it was observed that the recovery of gold in the leach solution at 0.70 kg/ton-ore NaCN in comparison to 1.20 kg/ton-ore NaCN yields 81.27% and 78.77%, respectively. Test conducted at 1.20 kg/ton-ore had a cyanide consumption of 0.96 kg/ton-ore when comparing it to 0.37 kg/ton-ore of test at 0.70 kg/ton-ore NaCN dosage. The DO’s before and after leach tests were below 6.00 ppm.

It can be concluded that:

- Leaching at 0.70 kg/ton-ore NaCN yield better results according to data.
- Pre-Oxidation was not sufficient on the test samples.

- Size by size analysis indicated that most of the gold leached was exposed to cyanide as seen in Table 4.2-4.3.
- The sample was coarse P80 of 105 microns and as a regard conclusiveevidence was lacking to make undoubted conclusion.
- Sample after splitting introduced bias hence grades were not within 5% confidence level to each other.
- Cyanide consumption was lower than the main plant trends.

4.2.2. Leach test on fresh ore at pH 9.50 and 10.00 and 1kg/t NaCN

Feed sample

Table 4.6. Feed sample particle size analysis

Size (microns)	Mass (g)	Mass (%)	(%) Passing
+180	59.63	12.79	87.21
+150	25.49	5.47	81.75
+106	71.36	15.30	66.45
+75	58.94	12.64	53.81
+63	16.86	3.62	50.19
-63	234.08	50.19	
Total	466.36	100.00	
P80		145.60 microns	

This feed sample particle size analysis resulted that the sample is coarse because the % passing 75 microns screen is 53.81% (Table 4.6). In gold Metallurgy depending on the ore body type, the liberation at least should be measured at 75-80% passing 75 microns to 75-80% passing 45 microns. This will intern ensure that if there is any gold locked in either as nugget and carriers will be exposed and therefore will dissolved when in contact with the lixiviant.

Gold deportation

The liberation argument presented from Table 4.6 is valid and true, gold in the test sample is associated with coarser material. It is observed that the average grade of the +106 microns from the Table 4.7 is 3.84 g/t. while on the +75 and minus 75 microns fractions gold grade is 4.59 g/t and 3.95 g/t, respectively.

Table 4.7. Gold deportation

Size (microns)	Gold in Fractions (g/t)
+180	2.17
+150	3.08
+106	3.84
+75	4.59
-75	3.95

Test summary

In the summary test (Table 4.8) below, it is noted that the gold recovery was 80.30% and 81.79% respectively for test at pH 9.50 and pH 10.00. NaCN consumption was respectively 0.67 and 0.56 kg/ton for test at pH 9.50 and pH 10.00. The test was conducted on a sample that is coarse and contained Gravity Recoverable Gold (GRG), this is the reason the residue is higher than when test is conducted from Pre-Oxidation tank.

Table 4.8. Test summary

Description	pH= 9.50	pH= 10.00
Dry Density g/cm3	2.8	2.8
Pre-Oxidation time (hours)	4	4
Leach Time (hours)	24	24
Sample mass(g)	984.56	1088.04
solution Volume(ml)	1203.00	1330.00
% Solids (after 24hrs)	45.00	45.00
DO's After Pre-Oxidation (ppm)	20.98	20.19
pH Monitoring		
pH before NaCN addition	9.50	10.00
Final pH	9.57	9.73
Reagents added		
NaCN added (kg/t)	1.00	1.00
Reagent Residuals		
Free CN ppm	291.69	285.25
NaCN consumed(kg/ton)	0.67	0.56
DO's after 24 hours leach (ppm)	5.40	6.04
GOLD		
Head g/t	3.350	3.350
Residue g/t (Solids)	0.66	0.61
Solution g/t	1.803	2.046
% Recovery (Solids)	80.30	81.79
% Accountability	85.46	92.87

Gold Deportation

Table 4.9. Size by size recovery

GOLD DEPORTATION FOR TEST AT PH 9.50 AND 10.00						SIZE BY SIZE RECOVERY (%)	
Head Sample	Results	Tails (pH=9.50)	Results	Tails (pH=10.00)	Results	pH=9.50	pH=10.00
Size (microns)	Au (g/t)	Size (microns)	Au (g/t)	Size (microns)	Au (g/t)	Recovery	Recovery
+106	3.03	+105	0.96	+105	0.67	68.32	77.89
+75	4.59	+75	0.83	+75	0.76	81.92	83.44
-75	3.95	-75	0.58	-75	0.49	85.32	87.59
AVERAGE	3.86	AVERAGE	0.79	AVERAGE	0.64	78.5	83.0

Table 4.9 presented size by size recovery to look closely at the tail's characteristics compared to the feed sample. From the table it is observed that the recovery on the +106 microns is 68.32% and 77.89% of test at pH 9.50 and pH 10.00 respectively. At -75 microns the recoveries are 85.32% and 87.59 % respectively.

It has been examined that after the test the +106, and +75 microns there is a certain degree of locked gold. Above all the gravity recoverable gold when present on a sample will always have a slow reaction kinetics and therefore residence time is essential during leaching. The test residue is high due to locked gold inside coarse material. The cyanide consumption is lower as compared to the main plant trends due to the fact that gold that was available was not in contact with cyanide.

From the Table 4.9, it is observed that after the test the +106, and +75 microns there is a certain degree of locked gold. Above all the gravity recoverable gold when present on a sample will always have a slow reaction kinetics and therefore residence time is essential during leaching. The test residue is high due to locked gold inside coarse material. The cyanide consumption is lower as compared to the main plant trends due to the fact that gold that was available was not in contact with cyanide.

It can be concluded that:

- Leaching of fresh ore at a pH of 10.00 yields higher recovery when comparing to leaching at a pH of 9.50
- At pH of 10.00 the cyanide consumption is minimal in comparison to the consumption when pH is at 9.50.
- Size by size analysis indicated that most of the gold leached was exposed to cyanide.
- The sample was too coarse to conclude the test comparison.
- Cyanide consumption was lower than the main plant trends.

1. Leach test on Pre-Ox 12 HRS and 24 HRS samples

Table 4.10. Leach on 12 hrs and 24 hrs Pre-Ox sample

Description	TEST STATUS	TEST STATUS	TEST STATUS	TEST STATUS
	TEST 1	TEST 2	TEST 3	TEST 4
	PRE-OX 12 HRS	PRE-OX 24 HRS	TANK 1 12 HRS	TANK 1 24 HRS
Dry Density g/cm^3	2.8	2.8	2.8	2.8
Pre-conditioned time	0	0	0	0
Leach Time (hrs)	12	24	12	24
Mass sample added (grams)	419.73	411.73	496.46	471.86
H2O added (ml)	513	503	607	577
PULP % Solids	45	45	45	45
pH Monitoring				
pH before NaCN addition	9.72	9.72	9.72	9.72
Final pH	10.38	10.30	10.10	10.09
Reagents added				
CaO ppm	302	302	302	302
NaCN ppm	1000	1000	1000	1000
Carbon concentration	0	0	0	0
Reagent Residuals				
NaCN ppm	379.27	262.00	249.51	76.00
NaCN ppm Consumed	620.73	738.00	750.49	924.00
Free cyanide	201.74	132.72	132.72	46.71
GOLD				
Head g/t	2.480	2.480	2.760	2.760
Residue g/t (Solids)	0.40	0.39	0.37	0.35
Solution g/t	1.725	2.080	1.972	1.849
Carbon g/t				
Recovered Grade (solids) g/t	2.080	2.090	2.390	2.410
% Recovery (Solids)	83.87	84.27	86.59	87.32
Recovered Grade (products) g/t	1.725	2.08	1.972	1.849
Theoretical Head	2.125	2.470	2.342	2.199
% Accountability	85.69	99.60	84.86	79.67
% Theoretical Recovery	81.18	84.21	84.20	84.08
% Grade Comparison (Solids vs Theoretical)	96.79	99.92	97.24	96.29

COMMENTS

As seen in Table 4.10;

- Most of the leaching took place well within the first 12 hrs
- Looking at the 24 hours leach tests it is evident that some of the remaining leachable gold is locked due to grind issues
- Comparing the recalculated grade with the solid grade, it is observed that they are well within 5% confidence level variance
- Sodium Cyanide consumption is below 1000 g/t (ppm) across all the tests

4.3. Leach Residence time

Gold leach extraction versus residence time results for the master composite for different grind sizes are shown in Table 4.11. The gold leach extraction versus residence time results for the variability tests (which were conducted at a target grind of 80% passing 74, 60 and 42 microns) are presented in Figure 4.2.

Table 4.11. P80 size recovery

	P80 = 74 µm	P80 = 60 µm	P80 = 42 µm
Time (Hrs.)	Gold Extraction (%)		
0	60.2	53.7	57.4
2	83.5	85.6	87.3
4	85	87.2	90
8	86.3	89.5	93.5
16	84.5	90	95.4
24	91.3	92	94.8

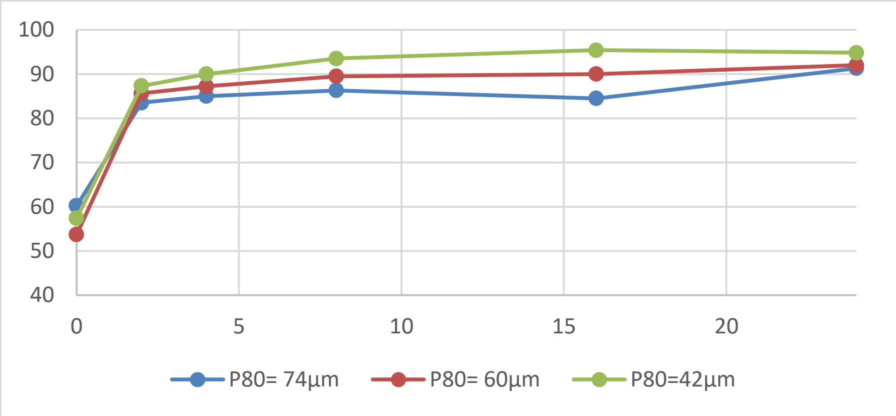


Figure 4.2. P80 size recovery distribution

Figure 4.2 demonstrated that the leaching is completed relatively quickly, over 80% extractions in the first 4 hours and is essentially complete between 8 and 16 hours depending on the grind size. As the residence time of the existing six tank CIL circuit is over 21 hours at 146 tph and 19 hours at 200 tph the leach extraction should not be an issue with an optimised circuit.

4.3. Optimization and simulation of carbon:

4.3.1. Mass balance from SIMCIL

Table 4.12. Mass balance from SIMCIL

Stage	Total Gold In g/h	Total Gold Out g/h	Gold In - Gold Out g/h	Relative difference between In & Out (%)
Tank 1	367.7	376.3	-8.6	-2.3
Tank 2	199.6	172.4	27.2	15.8
Tank 3	114.6	126.6	-12.0	-9.5
Tank 4	85.4	86.8	-1.4	-1.6
Tank 5	67.0	71.1	-4.1	-5.8
Tank 6	54.3	57.6	-3.3	-5.8
OVERALL	284.7	285.2	-0.5	-0.2
NOTE: You can review and change values for: Model parameter options; Plant feed parameters; Model type and Carbon data			**Colour key for relative difference values**	
			<±5%	
			>±5% <±10%	
			>±10% <±20%	
			>±20% <±30%	
			>±30%	

Mass balance is generally good. Note the overall balance difference of -0.2%, indicating that the loaded carbon value of 997 g/t and the elution frequency agrees very well with the gold being removed from the circuit.

4.3.2. Modelled circuit profiles and simulation of various scenarios

From SIMCIL data on comparing the first baseline modelled profile and average plant profile, gold on carbon (not full GIC of gold on carbon/solution/solids) but of course almost all the gold is on the carbon. The full GIC is calculated and shown in the model. The elution frequency column provides elutions/day and elutions/week.

For the second scenario when the elution is carried out every second day, the gold inventory Au increase but the gold solution loss increases as well. At this stage it can be observe that when we increased the feed grade, the GIC increase numbers increased simultaneously.

Locking all scenarios, with the respect to carbon distribution. We generally find that even carbon distribution (same mass of carbon mass in each tank) gives a solution loss similar to or better than uneven profiles and has the advantage of simplicity.

In this case the circuit is apparently working so well that modelling an even (7.88 g/l) showed no benefit or only small. For example, for the scenario 8 the elution loss dropped from 0.011 to 0.10, however, the gold inventory increased to 62 the gold inventory increased to 62 kg.

For the current circuit, the operations seem to maintain a steady profile (based on GIC profiles), being high carbon in the last tank and similar concentrations in remaining tanks, which is also important. Large variations in carbon distribution, particularly allowing a tank to drop a very low carbon concentration and can result in elevated solution losses.

These observations are particularly relevant when a circuit is operating at maximum capacity. As noted previously, with the apparent high carbon activity this circuit can accommodate a good degree of circuit variation and still perform well.

5. CONCLUSION AND RECOMMENDATIONS

5.1. Conclusion

A simulation model of CIL process in New Liberty Gold Processing plant has been developed and a dynamic model that describes gold leaching and adsorption on activated carbon is presented. The kinetic equations for modelling and simulation of CIL tanks in SIMCIL were collected from the four months (March - June 2020) plant data. This dynamic model is used to study the variation in gold concentration in the solution and on carbon. The calibration results obtained by the model are compared with the existing literature and are found to be close agreement.

The grinding ore size, oxygen concentration, pH and cyanide concentration are critical for the gold leaching. Decrease in the grinding size (below 100 microns) and an increase in oxygen concentration and cyanide concentration resulted in an increase in the efficiency of the process. However, the improvement in the efficiency is limited in case of oxygen concentration due to the low concentration of cyanide or vice versa. The leach optimization was conducted in 3 stages which are: Leach test at pH 10.5 and different cyanide dosage, from this test it can be concluded that:

- Leaching at 700 g/ton yield better results according to data.
- Pre-Oxidation was not sufficient on the test sample.
- Size by size analysis indicated that most of the gold leached was exposed to cyanide.
- The sample was coarse P80 of 105 microns and as a regard conclusiveevidence was lacking to make undoubted conclusion.
- Sample after splitting introduced bias hence grades were not within 5%confidence level to each other.

- Cyanide consumption was lower than the main plant trends.

The second test was on fresh ore at different pH (9.50 and 10.00) and 1 kg/t NaCN. The conclusion from this test was:

- Leaching of fresh ore at a pH of 10.00 yields higher recovery when comparing to leaching at a pH of 9.50
- At pH of 10.00 the cyanide consumption is minimal in comparison to the consumption when pH is at 9.50.
- Size by size analysis indicated that most of the gold leached was exposed to cyanide.
- The sample was too coarse to conclude the test comparison.

Cyanide consumption was lower than the main plant trends.

The last test was conducted on Pre-Ox sample in different hours (12 HRS and 24 HRS). It was observed that most of the leaching took place well within the first 12 hrs. Looking at the 24 hours leach tests it is evident that some of the remaining leachable gold is locked due to grind issues. Comparing the recalculated grade with the solid grade, it is observed that they are well within 5% confidence level variance. Sodium Cyanide consumption is below 1000 g/t (ppm) across all the tests.

A simulation model for the current circuit, the operations seem to maintain a steady profile (based on GIC profiles), being high carbon in the last tank and similar concentrations in remaining tanks, which is also important. Large variations in carbon distribution, particularly allowing a tank to drop a very low carbon concentration and can result in elevated solution losses.

Locking all scenarios, with the respect to carbon distribution. We generally find that even carbon distribution (same mass of carbon mass in each tank) gives a solution loss similar to or better than uneven profiles and has the advantage of simplicity.

These observations are particularly relevant when a circuit is operating at maximum capacity. As noted previously, with the apparent high carbon activity this circuit can accommodate a good degree of circuit variation and still perform well.

5.2. Recommendations

The data applications presented in this study could be extended for the further by generating more data. Furthermore, other variables could be added such as agitation performance, pulp density for carbon suspension, carbon cycle time and the carbon transfer method for the optimization.

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CURRICULUM VITAE

Cheick Ahmed Tidiane SOUMAH, was born in Conakry the capital city of Guinea. He attended both Quranic and French elementary schools in Conakry. At the age of 19 he went to Adana, the city of Turkey, to continue his education. He graduated from the Mining Engineering Department in 2019 at the University of Çukurova. Afterwards he started working in Turkey and Africa for various mining companies. He speaks French, English and Turkish.

APPENDIX

Relevant Daily Production Numbers

DAILY JAN 2020

Monthly	MILLING CIRCUIT				GRAVITY	PRE OX		CIL 1			CIL 6		TAIL		RECOVERY		SHIPPED		TOTAL PRODUCED
	Capacity		Run	Grade	DAILY GRAVITY Au	Au Solids	Leached gold	Carbon Load PCH	Lıquıd Au	Solıd Au	Lıquid Au	Solid Au	Carbn PCT	Tailings Au	CIL REC	OVERALL Rec	ELUTION PROD. Au	GRAVITY PROD. Au	TOTAL Au
UNİT	Tpd	Tph	hrs	g/t	ounces	ppm	oun	ppm	ppm	ppm	ppm	ppm	ppm	Oun	%	%	ounces		
				2,25		1.73													
1.Oca	5014	209	24.00	2.26	68	1.60	257.93	1710	0.43	0.42	0.010	0.24	32	41	86.1	88.30%			
2.Oca	5050	210	24.00	2.21	56	1.58	256.51	1597	0.45	0.44	0.010	0.25	29	43	84.4	87.90%			
3.Oca	4921	205	24.00	2.08	76	1.80	284.79	1303	0.60	0.65	0.010	0.24	39	40	84.8	88.10%			
4.Oca	4931	205	24.00	2.79	57	1.61	255.23	1210	0.47	0.43	0.010	0.28	27	47	84.4	85.50%			
5.Oca	3827	206	18.55	2.42	36	1.98	243.60	1340	0.27	0.56	0.010	0.30	29	39	81.4	88.40%	1788.19	432.01	2220.2
6.Oca	4687	206	22.78	2.62	62	2.04	307.38	1745	0.46	0.58	0.010	0.27	26	44	86.4	87.90%			
7.Oca	4931	206	23.90	2.60	67	1.69	267.95	1227	0.48	0.68	0.010	0.25	31	42	87.7	89.60%			
8.Oca	5006	209	24.00	1.97	48	1.92	309.04	1432	0.63	0.46	0.010	0.27	29	46	84.0	88.80%			
9.Oca	5035	210	24.00	2.99	42	1.98	320.52	1677	0.46	0.45	0.010	0.24	32	41	87.5	86.70%			
10.Oca	4984	208	24.00	2.31	46	1.47	235.55	1565	0.58	0.55	0.010	0.40	35	67	79.8	85.90%			
11.Oca	5033	210	24.00	1.55	49	1.65	266.98	1695	0.40	0.37	0.010	0.31	52	53	78.9	85.60%			
12.Oca	5053	211	24.00	2.11	30	1.44	233.96	1492	0.41	0.36	0.010	0.26	48	45	84.2	81.80%	1599.14	314.42	1913.56
13.Oca	4375	207	21.09	2.04	47	1.43	201.13	1418	0.45	0.55	0.010	0.27	30	40	81.3	86.10%			
14.Oca	4250	206	20.59	2.50	16	1.75	239.10	1231	0.35	0.46	0.010	0.29	29	42	79.7	84.70%			
15.Oca	5020	209	24.00	1.77	51	1.68	271.16	1182	0.40	0.51	0.010	0.28	30	48	84.0	87.90%			
16.Oca	5012	209	24.00	1.67	34	1.44	232.04	1138	0.40	0.61	0.010	0.29	44	49	82.7	82.30%			
17.Oca	2885	206	14.00	1.72	22	1.16	107.61	1213	0.39	0.45	0.010	0.26	25	26	81.9	83.10%			
18.Oca	4868	203	24.00	1.75	43	1.44	225.35	1272	0.25	0.29	0.010	0.27	72	45	76.7	83.00%			
19.Oca	800	200	4.00	1.57	26	1.71	43.98	1326	0.41	0.38	0.010	0.24	25	7	83.3	85.00%	1327.68	204.37	1532.05
20.Oca	3925	188	20.83	1.96	26	1.50	189.27	951	0.28	0.39	0.010	0.21	29	28	87.7	85.20%			
21.Oca	4605	192	24.00	1.48	34	1.44	213.19	1017	0.34	0.27	0.010	0.22	30	35	85.3	87.60%			
22.Oca	4502	197	22.83	2.04	32	1.38	199.75	1248	0.35	0.27	0.010	0.21	42	33	85.4	84.30%			
23.Oca	4499	187	24.00	2.06	47	1.77	256.00	1125	0.38	0.35	0.010	0.26	27	40	81.2	86.20%			
24.Oca	4354	189	23.00	2.10	42	1.77	247.77	1044	0.39	0.37	0.010	0.27	28	40	84.7	85.80%			
25.Oca	4403	193	22.87	2.08	49	1.59	225.62	1010	0.43	0.34	0.010	0.28	28	41	84.4	85.80%			
26.Oca	4506	188	24.00	2.19	21	1.80	260.76	1339	0.17	0.50	0.010	0.26	25	40	83.6	86.30%	1.133.81	223.91	1357.72
27.Oca	4489	187	24.00	2.26	63	2.18	314.66	1353	0.15	0.40	0.010	0.25	23	38	86.1	87.60%			
28.Oca	4509	188	24.00	2.40	46	2.05	297.18	1003	0.12	0.48	0.010	0.30	72	46	86.2	85.70%			
29.Oca	4212	175	24.00	2.85	87	2.05	277.60	1388	0.25	0.48	0.010	0.30	27	43	85.4	86.60%			
30.Oca	4133	172	24.00	2.38	53	2.11	280.34	1602	0.19	0.57	0.010	0.30	26	42	85.4	88.70%			
31.Oca	4134	172	24.00	2.27	50	2.10	279.09	1418	0.44	0.60	0.010	0.26	33	37	87.7	88.10%			
Overall	**137950**	199	694.44	2,18	**1427**	1.71	**7601.04**	1331	0.38	0.46	0.01	0.27	34	1266	84,3%	86,9%	5848.81	1174.7	7023.51

Average	**4450**	**199**	**22**	**2.16**	**46**	**1.71**	**245**	**1331**	**0.38**	**0.46**	**0.010**	**0.269**	**34**	**41**	**8395%**	**0.86**
SD	**830**	**12**	**4**	**0.38**	**17**	**0.26**	**57.0**	**225**	**0.12**	**0.11**	**0.000**	**0.036**	**12**	**10**	**274%**	**0.02**
Max	**5053**	**211**	**24**	**2.99**	**87**	**2.18**	**321**	**1745**	**0.63**	**0.68**	**0.010**	**0.400**	**72**	**67**	**8770%**	**0.90**
Min	**800**	**172**	**4**	**1.48**	**16**	**1.16**	**44**	**951**	**0.12**	**0.27**	**0.010**	**0.210**	**23**	**7**	**7670%**	**0.82**

Feb-20

Monthly Process

	MILLING CIRCUIT				GRAVITY	PRE OX			CIL 1		CIL 6	TAIL			RECOVERY		SHIPPED		TOTAL PRODUCED
	Capacity		Run	Grade	DAILY GRAVITY Au	DO	Au Solids	Leached gold	Carbon Load PCH	Solıd Au	Lıquid Au	Solid Au	Carbn PCT	Tailings Au	CIL REC	OVERALL Rec	ELUTION PROD. AU	GRAVITY PROD. Au	TOTAL Au
UNİT	Tpd	Tph	hrs	g/t	ounces	ppm	ppm	oun	ppm	ppm	ppm	ppm	ppm	Oun	%		ounces		
				2.27			2.1												
1.Şub	4082	170	24	1.93	64	21.6	1.55	203.43	1175	0.49	0.010	0.25	19	35	88.1	88.00%			
2.Şub	4107	171	24	2.04	26	20.7	1.69	223.17	1233	0.44	0.010	0.2	59	28	87.1	88.50%	1559,89	340,08	1899,98
3.Şub	4076	170	24	1.86	67	21.2	1.21	158.58	1174	0.47	0.010	0.2	34	28	88.2	89.10%			
4.Şub	3774	172	22	1.69	69	20.1	1.4	169.88	1151	0.27	0.010	0.16	0	21	86.8	90.20%			
5.Şub	4249	177	24	1.37	40	20.2	1.19	162.58	1162	0.36	0.010	0.17	28	25	87.9	88.60%			
6.Şub	4130	172	24	1.7	40	19.7	1.31	173.94	1043	0.36	0.010	0.16	28	23	86.6	86.70%			
7.Şub	4104	171	24	2.39	36	23.1	1.58	208.48	958	0.34	0.010	0.18	26	26	86.3	88.10%			
8.Şub	4111	171	24	2.69	64	18.2	2.04	269.63	1097	0.38	0.010	0.2	29	28	87.3	90.70%			
9.Şub	4041	168	24	1.71	23	24	1.52	197.48	1260	0.39	0.010	0.15	113	22	92.6	93.60%	1208,51	274,95	1483,46
10.Şub	3867	161	24	2.08	73	24.8	1.48	184.01	1364	0.78	0.010	0.25	26	33	83.6	84.10%			
11.Şub	3892	162	24	2.36	55	25	1.52	190.2	1265	0.29	0.010	0.21	28	28	85.8	88.80%			
12.Şub	3833	160	24	1.99	55	24.4	1.46	179.91	1127	0.33	0.010	0.25	28	33	83.6	88.50%			
13.Şub	3861	161	24	1.84	39	25.1	1.4	173.8	1260	0.34	0.010	0.18	23	24	87.7	89.80%			
14.Şub	3878	162	24	1.5	33	25.4	1.49	185.76	1113	0.34	0.010	0.2	25	27	85.7	87.90%			
15.Şub	3792	158	24	1.94	32	24.1	1.47	179.22	836	0.3	0.010	0.21	28	28	85.9	84.50%			
16.Şub	3818	159	24	1.69	24	23.6	1.58	193.95	901	0.36	0.010	0.2	23	26	86.4	88.50%	1260,67	283,33	1544,01
17.Şub	1171	154	7.58	1.83	53	30.2	1.35	50.82	814	0.37	0.010	0.2	22	8	87.3	86.80%			
18.Şub	3419	158	21.63	2.5	20	24.4	1.86	204.48	1059	0.35	0.010	0.23	28	27	83	86.20%			
19.Şub	3798	158	24	2.12	43	25.6	1.57	191.71	927	0.34	0.010	0.16	39	21	91.4	92.70%			
20.Şub	4531	189	24	1.79	39	21.8	1.88	273.86	1206	0.42	0.010	0.19	33	30	87.9	90.00%			
21.Şub	4943	206	24	1.79	42	18.4	1.3	206.61	951	0.34	0.010	0.21	60	36	88.8	87.00%			
22.Şub	5062	211	24	1.89	55	19.3	1.36	221.32	1055	0.34	0.010	0.2	68	35	84.6	87.60%			
23.Şub	5059	211	24	1.82	26	20.1	1.3	211.45	953	0.4	0.010	0.17	34	30	87.5	89.80%	940,27	243,22	1183,49
24.Şub	5079	212	24	2.26	53	20.4	1.75	285.78	972	0.5	0.010	0.26	27	45	80	84.50%			
25.Şub	4972	207	24	1.67	53	17.6	1.32	211.02	1370	0.33	0.010	0.25	34	42	85.7	87.90%			
26.Şub	4974	207	24	1.58	63	20.8	1.31	209.49	1122	0.5	0.010	0.21	30	36	84.1	86.10%			
27.Şub	3244	207	15.67	2.05	39	21.8	1.43	149.13	1178	0.49	0.010	0.19	26	21	85.5	86.60%			
28.Şub	5000	208	24	2.23	36	19.8	1.52	244.37	1278	0.4	0.010	0.21	22	36	85.3	88.70%			
29.Şub	4995	208	24	2.21	50	16.5	1.68	269.82	1207	0.48	0.010	0.23	32	40	84.9	88.70%			
Overall	**119865**	180	666.88	1.95	**1311**	22	1.5	**5783.88**	1136	0.4	0.01	0.2	34	844	86.50%	88.70%	4969,35	1141,58	6110,93

Average	**4133**	**17923**	**1.95**	**45**	**22.01.50**	**199.44**	**1111**	**0.40**	**0.0100.20**	**34**	**29**
SD	**785**	**213**	**0.31**	**15**	**3.00.20**	**45.40**	**149**	**0.10**	**0.0000.03**	**20**	**8**
Max	**5079**	**21224**	**2.69**	**73**	**30.22.04**	**285.78**	**1370**	**0.78**	**0.0100.26**	**113**	**45**
Min	**1171**	**1548**	**1.37**	**20**	**16.51.19**	**50.82**	**814**	**0.27**	**0.0100.15**	**0**	**8**

Right-hand values: 86.4, 2.4, 92.6, 80.0 (**Average**, **SD**, **Max**, **Min** respectively).

Daily March 2020

	MILLING CIRCUIT					GRAVITY	PRE OX		CIL 1			CIL 6		TAIL		RECOVERY		SHIPPED		TOTAL PRODUCED	TOTAL SHIPPED
	Capacity		Run	Grade	Milled Gold	DAILY GRAVITY Au	Au Solids	Leached gold	Liquid Au	Carbon Load PCH	Solid Au	Liquid Au	Solid Au	Carbn PCT	Tailings	CIL REC	OVERALL Rec	ELUTION PROD. Au	GRAVITY PROD. Au	TOTAL Au	SHIPPED Au
UNİT	Tpd	Tph	hrs	g/t	Ounces	ounces	ppm	oun	ppm	ppm	ppm	ppm	ppm	ppm	Oun	%	%	ounces			ounces
				2.21			1.68														
1.Mar	4973	207	24.0	2.97	475	76	2.42	386.93	0.4	1446	0.31	0.010	0.21	23	36	87.5	89.50%				
2.Mar	5078	212	24.0	2.38	389	48	2.31	377.12	0.59	1500	0.6	0.010	0.28	76	48	88.4	89.80%	1705.57	354.93	2060.5	2060.5
3.Mar	2007	206	9.8	2.27	147	95	2.04	131.66	0.43	1629	0.37	0.010	0.24	38	17	89.6	89.00%				
4.Mar	2112	205	10.3	1.6	109	22	1.78	120.86	0	1209	0	0.010	0.21	29	15	89.7	89.80%				
5.Mar	4483	206	21.8	2.03	293	41	1.78	256.56	0.26	1078	0.3	0.010	0.21	33	32	88.2	85.50%				
6.Mar	5033	210	24.0	1.61	261	53	1.59	257.31	0.35	1219	0.34	0.010	0.23	79	40	87.1	87.60%				
7.Mar	5008	209	24.0	2.32	374	40	2.02	325.23	0.38	1228	0.46	0.010	0.2	32	34	87.4	86.20%				
8.Mar	4986	208	24.0	2.17	348	49	1.9	304.61	0.47	1182	0.54	0.010	0.25	28	42	87.6	88.30%				
9.Mar	4116	203	20.3	2.38	315	24	1.79	236.89	0.55	1533	0.47	0.010	0.28	25	39	85.3	86.10%	1429.37	272.44	1701.82	1701.82
10.Mar	2507	207	12.1	1.97	159	68	1.54	124.11	0.44	1401	0.42	0.010	0.23	29	20	87.2	89.40%				
11.Mar	4919	205	24.0	1.84	291	60	1.48	234.04	0.42	1312	0.4	0.010	0.26	41	43	83.1	85.70%				
12.Mar	759	196	3.9	2.49	61	47	1.11	27.08	0.49	1243	0.39	0.010	0.22	24	6	85.1	86.80%				
13.Mar	3468	211	16.5	1.76	196	0	1.86	207.4	0.19	1175	0.2	0.010	0.16	24	20	85.6	92.70%				
14.Mar	4903	204	24.0	1.56	246	30	1.09	171.83	0.35	929	0.31	0.010	0.2	22	34	89.2	87.40%				
15.Mar	4949	206	24.0	1.8	286	48	1.1	175.03	0.3	956	0.25	0.010	0.21	25	36	80.7	85.10%				
16.Mar	3594	203	17.7	2.16	250	21	1.34	154.86	0.44	860	0.37	0.010	0.2	21	25	81.8	87.60%	1292.62	227.04	1519.66	1519.66
17.Mar	4994	208	24.0	2.42	389	57	1.79	287.38	0.38	741	0.44	0.010	0.24	21	41	82.1	87.80%				
18.Mar	4806	200	24.0	2.93	453	62	1.43	220.95	0.43	1116	0.39	0.010	0.23	26	38	87.2	89.60%				
19.Mar	465	159	2.9	3.7	55	64	2.51	37.49	0.47	1314	0.7	0.010	0.29	23	5	79.7	89.30%				
20.Mar	4695	207	22.7	2.52	380	14	1.72	259.6	0.32	1198	0.35	0.010	0.31	28	49	87.6	91.00%				
21.Mar	4447	209	21.3	2.72	389	51	2.02	288.81	0.47	1306	0.58	0.010	0.28	59	42	83.7	88.00%				
22.Mar	2838	203	14.0	2.68	245	50	1.83	166.99	0.53	1336	0.49	0.010	0.27	54	26	86.6	89.20%				
23.Mar	1631	198	8.3	2.26	119	22	2.19	114.84	0.53	1473	0.52	0.010	0.26	30	14	85.8	89.50%				
24.Mar	5033	210	24.0	2.99	484	39	2.28	368.94	0.33	1498	0.34	0.010	0.22	20	38	90	89.30%				
25.Mar	5078	212	24.0	2.92	477	58	2.26	368.99	0.48	1215	0.6	0.010	0.26	92	45	88.6	90.60%				
26.Mar	5053	211	24.0	2.03	330	55	1.9	308.66	0.53	1406	0.46	0.010	0.28	23	48	87.6	89.60%				
27.Mar	5038	210	24.0	2.03	329	63	1.65	267.28	0.41	1538	0.5	0.010	0.24	39	41	87.4	87.10%				
28.Mar	5020	209	24.0	1.7	274	48	1.38	222.73	0.53	1308	0.51	0.010	0.29	31	49	82.4	84.60%				
29.Mar	5020	209	24.0	1.57	253	43	1.21	195.3	0.34	1389	0.43	0.010	0.24	38	41	82.6	84.60%				
30.Mar	5006	209	24.0	1.53	246	40	1.05	169	0.36	1540	0.41	0.010	0.22	26	38	81.8	84.60%				
31.Mar	4979	207	24.0	1.32	211	43	1.26	201.7	0.35	1595	0.37	0.010	0.18	23	31	82.9	86.80%				
Overall	**126.999**	207	613.4	2.16	**8.830**	**1.431**	1.71	**6970.19**	0.42	1286	0.43	0.01	0.24	35	1034	**86.00%**	**88.30%**	4427.56	854.42	5281.98	**5281.98**
Average	**4097**	**205**			**19.8**	**2.21**	**285**	**46**				**1.73**	**224.84**	**0.40**		**1286**	**0.41**	**0.010**	**0.24**	**35 33**	**85.8**
SD	**1398**	**9**			**6.6**	**0.55**	**118**	**19**				**0.42**	**93.47**	**0.12**		**219**	**0.13**	**0.000**	**0.04**	**18 12**	**2.9**
Max	**5078**	**212**			**24.0**	**3.70**	**484**	**95**				**2.51**	**386.93**	**0.59**		**1629**	**0.70**	**0.010**	**0.31**	**92 49**	**90.0**
Min	**465**	**159**			**2.9**	**1.32**	**55**	**0**				**1.05**	**27.08**	**0.00**		**741**	**0.00**	**0.010**	**0.16**	**20 5**	**79.7**

Daily April 2020

	MILLING CIRCUIT					GRAVITY	PRE OX		CIL 1			CIL 6		TAIL		RECOVERY		POURED		TOTAL PRODUCED	TOTAL SHIPPED
	Capacity		Run	Grade	Milled Gold	DAILY GRAVITY Au	Au Solids	Leached gold	Liquid Au	Carbon Load PCH	Solid Au	Liquid Au	Solid Au	Carbn PCT	Tailings Au	CIL REC	OVERALL Rec	ELUTION PROD. Au	GRAVITY PROD. Au	TOTAL Au	SHIPPED Au
UNİT	Tpd	Tph	hrs	g/t	Ounces	ounces	ppm	oun	ppm	ppm	ppm	ppm	ppm	ppm	Oun	%	%	ounces			ounces
				1,32			1,26														
1.Nis	4286	203	21.07	1.92	265	31	1.24	170.87	0.29	889	0.49	0.010	0.22	24	33	82.5	81.60%	3374.86	712.36	4087.22	
2.Nis						58				1064	0	0.000	0	19		100	100.00%	138.17		138.17	
3.Nis																					
4.Nis																					
5.Nis																					
6.Nis																					
7.Nis							1.24			1119				32			#DIV/0!				
8.Nis	2251	204	11.03	1.57	114		1.24	89.72		1182		0.010	0.11	28	9	91.1	#DIV/0!				4087.22
9.Nis	4713	204	23.07	1.75	265	23	1.66	251.53	0.3	1199	0.34	0.010	0.11	45	19	91.1	91.60%				
10.Nis	5041	210	24	2.05	332	42	1.76	285.27	0.46	1174	0.51	0.010	0.22	30	38	86.7	86.10%				
11.Nis	5066	211	24	1.82	296	44	1.39	226.42	0.46	1157	0.52	0.010	0.29	23	50	83.5	84.80%				
12.Nis	5008	209	24	1.84	296	33	2.06	331.71	0.42	1024	0.36	0.010	0.28	21	47	79.9	83.40%				
13.Nis	4976	207	24	1.77	283	39	1.8	287.97	0.52	1065	0.39	0.010	0.28	77	47	86.4	83.60%				
14.Nis	5021	209	24	2.18	352	39	1.69	272.81	0.43	1381	0.41	0.010	0.22	37	38	87.8	86.30%				
15.Nis	4938	206	24	1.89	300	35	2.29	363.53	0.52	1260	0.5	0.010	0.28	26	47	83.4	86.10%				
16.Nis	4988	208	24	2	321	42	1.58	253.36	0.57	1258	0.44	0.010	0.28	51	47	87.8	84.00%				
17.Nis	4632	206	22.47	1.54	229	31	1.76	262.11	0.42	1068	0.44	0.010	0.26	63	41	83.5	85.90%				
18.Nis		202	16.67	1.67	181	32	1.25	135.48	0.4	875	0.37	0.010	0.25	19	29	85.8	82.30%				
19.Nis																	100.00%				
20.Nis																		2163.16	408.71	2571.87	
21.Nis																					2710.04
22.Nis																					
23.Nis	4239	200	21.17	1.31	179		1.04	141.74	0	711	0	0.010	0.17	21	25	86.4					
24.Nis	4983	208	24	1.34	215	23	1.21	193.84	0.22	805	0.27	0.010	0.14	22	25	86.5	87.60%				
25.Nis	4959	207	24	1.32	210	25	1.26	200.89	0.37	1057	0.45	0.010	0.21		36	82.6	82.70%				
26.Nis	4847	202	24	1.4	218	28	1.41	219.71	0.16		0.37	0.010	0.21		35	83.3	82.40%				
27.Nis	1444	195	7.39	1.94	90	26	0.8	37.14	0.22		0.42	0.010	0.22		11	84.4	82.70%				
28.Nis												0.010				100	98.80%				
29.Nis	3200	208	15.38	2.63	271		2	205.74	0		0	0.010	0.16		18	80	#DIV/0!				
30.Nis	5003	208	24	3.19	513	47	2.45	394.07	0.33		0.49	0.010	0.23		39	88.5	90.40%				
Overall	**82.965**	206	402.25	1.85	**4930**	**629**	1.62	**4323.91**	0.38	1032	0.42	0.01	0.22	32	634	**86.60%**	**87.10%**	5676.19	1121.07	6797.26	**6797.26**

Average	**4422**	**206**	**21.2**	**1.85**	**259**	**35**	**1.56**	**227.6**	**0.34**	**1076**	**0.36**	**0.010**	**0.21**	**34**	**33**	**86.7**
SD 1051		**4**	**4.9**	**0.46**	**93**	**10**	**0.43**	**89.8**	**0.17**	**175**	**0.17**	**0.002**	**0.07**	**17**	**13**	**5.4**
Max	**5066**	**211**	**24.0**	**3.19**	**513**	**58**	**2.45**	**394.1**	**0.57**	**1381**	**0.52**	**0.010**	**0.29**	**77**	**50**	**100.0**
Min 1444		**195**	**7.4**	**1.31**	**90**	**23**	**0.80**	**37.1**	**0.00**	**711**	**0.00**	**0.000**	**0.00**	**19**	**9**	**79.9**

Daily May-2020

	MILLING CIRCUIT					GRAVITY	PRE OX		CIL 1			CIL 6		TAIL		RECOVERY		POURED		TOTAL PRODUCED	TOTAL SHIPPED
	Capacity		Run	Grade	Milled Gold	DAILY GRAVITY Au	Au Solids	Leached gold	Lıquıd Au	Carbon Load PCH	Solıd Au	Lıquid Au	Solid Au	Carbn PCT	Tailings Au	CIL REC	OVERALL Rec	ELUTION PROD. Au	GRAVITY PROD. Au	TOTAL Au	SHIPPED Au
UNİT	Tpd	Tph	hrs	g/t	Ounces	ounces	ppm	oun	ppm	ppm	ppm	ppm	ppm	ppm	Oun	%	%	ounces			ounces
				3,19			2,45														
1.May	5065	211	24	1.86	303	59	2.56	416.92	0.52	1055	0.65	0.01	0.26	27	45	89.4	91.10%				
2.May	5033	210	24	2.62	424	15	1.53	247.58	0.34	1019	0.5	0.01	0.25	64	43	90.2	85.40%	1040.13	244.92	1285.05	
3.May	5010	209	24	3.36	541	76	2.94	473.53	0.59	889	0.68	0.01	0.22	24	38	85.6	90.70%				1285.05
4.May	3254	202	16.08	2.08	218	82	2.72	284.55	0.88	989	0.98	0.01	0.37	21	40	87.4	88.30%				
5.May	4355	210	20.75	1.94	272	25	1.76	246.44	0.4	1586	0.44	0.01	0.24	28	36	91.2	87.40%				
6.May	750	224	3.34	0.75	18	22	1.76	42.43	0.45	1478	0.44	0.01	0.22	30	6	87.5	87.50%				
7.May	4995	208	24	0.82	132	16	0.96	154.16	0.3	1239	0.17	0.01	0.12	312	22	93.2	81.00%				
8.May	4914	206	23.87	1	158	21	0.89	140.62	0.26	1452	0.28	0.01	0.14	96	24	85.4	80.20%				
9.May	4978	207	24	1.5	240	26	0.8	128.03	0.22	1054	0.19	0.01	0.12	43	22	86.5	85.80%				
10.May	3210	204	15.72	2.6	268	42	2.07	213.61	0.35	810	0.48	0.01	0.12	27	14	85	90.50%				
11.May	4904	207	23.67	2.25	355	49	1.65	260.16	0.39	1352	0.41	0.01	0.29	52	48	86	88.00%				
12.May	4947	206	24	1.59	253	45	1.84	292.62	0.41	1102	0.68	0.01	0.28	42	47	83	86.60%				
13.May	4341	207	21	1.54	215	43	1.48	206.56	0.3	1340	0.5	0.01	0.25	24	37	86.4	82.90%				
14.May	4301	210	20.5	1.95	270	34	1.65	228.16	0.32	985	0.4	0.01	0.2	25	30	86.5	85.60%				
15.May	5076	211	24	2.08	339	50	1.53	249.68	0.38	1275	0.43	0.01	0.21	39	37	87.3	88.10%				
16.May	5043	210	24	1.47	238	55	1.23	199.44	0.41	1122	0.58	0.01	0.24	37	41	84.3	87.40%				
17.May	4997	208	24	1.34	215	52	0.95	152.64	0.35	1051	0.43	0.01	0.21	34	36	82.9	84.20%				
18.May	4253	177	24	1.21	165	23	1.15	157.27	0.35	1241	0.47	0.01	0.18	26	27	81.1	84.90%				
19.May	3265	202	16.2	1.01	106	49	0.68	71.38	0.21	1011	0.34	0.01	0.15	31	17	87	85.70%				
20.May	5060	211	24	0.98	159	29	0.98	159.43	0.17	1035	0.22	0.01	0.12	33	22	82.4	85.90%				
21.May	5010	209	24	1.27	205	27	0.74	119.19	0.23	1005	0.21	0.01	0.13	21	23	86.7	84.40%				
22.May	5002	208	24	1.51	243	33	1.38	221.93	0.28	1010	0.27	0.01	0.17	28	30	77	84.80%				
23.May	5043	210	24	1.54	250	33	1.16	188.07	0.4	820	0.34	0.01	0.28	28	47	79.7	80.00%				
24.May	4869	203	24	1.68	263	49	1.18	184.72	0.32	840	0.46	0.01	0.18	28	31	84.5	86.90%				
25.May	4509	188	24	1.88	273	48	1.54	223.25	0.32	804	0.46	0.01	0.24	25	37	79.7	84.40%				
26.May	4678	195	24	2.44	367	39	1.71	257.18	0.39	830	0.52	0.01	0.28	18	44	81.8	83.90%				
27.May	4355	181	24	2.5	350	33	2.26	316.41	0.51	820	0.63	0.01	0.3	23	44	82.5	86.80%				
28.May	4992	208	24	2.64	424	33	2.21	354.71	0.55	981	0.68	0.01	0.45	25	75	80.1	81.10%				
29.May	4264	178	24	1.99	273	33	1.9	260.47	0.49	926	0.53	0.01	0.37	24	53	83.3	85.10%				
30.May	1197	161	7.43	2.01	77	26	1.76	67.73	0.56	1070	0.59	0.01	0.34	22	14	82.1	81.80%				
31.May	4096	171	24	1.68	221	0	1.59	209.41	0.54	930	0.55	0.01	0.28	35	39	84.1	85.00%				
Overall	**135.766**	202	672.56	1.79	**7834**	**1168**	1.54	**6728.27**	0.39	1068	0.47	0.01	0.23	42	1065	**84.90%**	**86.40%**	1040.13	244.92	1285.05	**1285.05**

			22			38			0.39			0.23			
Average	**4380**	**202**		**1.78**	**253**		**1.57**	**217.04**		**1068**	**0.470.010**		**42**	**34**	**84.8**
SD	**1063**	**14**	**5**	**0.61**	**107**	**17**	**0.58**	**94.51**	**0.14**	**210**	**0.170.000**	**0.08**	**52**	**14**	**3.6**
Max	**5076**	**224**	**24**	**3.36**	**541**	**82**	**2.94**	**473.53**	**0.88**	**1586**	**0.980.010**	**0.45**	**312**	**75**	**93.2**
Min	**750**	**161**	**3**	**0.75**	**18**	**0**	**0.68**	**42.43**	**0.17**	**804**	**0.170.010**	**0.12**	**18**	**6**	**77.0**

Daily June-2020

	MILLING CIRCUIT					GRAVITY	PRE OX		CIL 1			CIL 6		TAIL		RECOVERY		POURED		TOTAL PRODUCED
	Capacity		Run	Grade	Milled Gold	DAILY GRAVITY Au	Au Solids	eached gold	Lıquid Au	Carbo nLoad PCH	Solıd Au	Lıquid Au	Solid Au	Carbn PCT	Tailings Au	CIL REC	OVERALL Rec	ELUTION PROD. Au	GRAVITY PROD. Au	TOTAL Au
UNİT	Tpd	Tph	hrs	g/t	Ounces	ounces	ppm	oun	ppm	ppm	ppm	ppm	ppm	ppm	Oun	%	%	ounces		
				1.68			1.59													
1	4112	171	24.0	1.35	178	53	1.01	134	0.478	1217	0.37	0.010	0.28	32	39	82.4	0.820	2188	406	2595
2	4143	173	24.0	0.86	115	24	0.85	113	0.373	893	0.30	0.010	0.19	20	27	81.2	0.843			
3	4647	194	24.0	0.85	127	28	0.60	90	0.247	737	0.25	0.010	0.13	19	22	84.7	0.823			
4	2998	207	14.5	1.15	111	34	0.82	79	0.209	728	0.24	0.010	0.11	23	12	81.7	0.844			
5	4807	200	24.0	0.93	144	22	0.84	130	0.176	748	0.24	0.010	0.10	27	17	87.8	0.894			
6	2898	196	14.8	0.86	80	22	0.74	69	0.176	530	0.24	0.010	0.10	24	11	88.1	0.868			
7	5051	210	24.0	0.77	125	23	0.85	138	0.228	627	0.15	0.010	0.11	24	20	85.1	0.846			
8	5027	209	24.0	1.31	212	22	0.91	147	0.217	710	0.21	0.010	0.10	32	18	88.2	0.841			
9	5004	208	24.0	0.97	156	28	0.89	143	0.315	713	0.30	0.010	0.15	19	27	83.5	0.868			
10	4529	189	24.0	1.70	248	25	1.01	147	0.248	625	0.28	0.010	0.15	21	24	83.1	0.822			
11	4617	210	22.0	1.99	295	42	1.35	200	0.344	655	0.36	0.010	0.17	18	27	83.2	0.887			
12	5055	211	24.0	1.02	166	35	1.11	180	0.377	942	0.31	0.010	0.20	25	35	85.2	0.888			
13	5024	209	24.0	2.41	389	44	1.37	221	0.267	1080	0.26	0.010	0.13	27	23	88.3	0.851			
14	1864	202	9.3	2.88	173	38	1.81	108	0.517	857	0.54	0.010	0.13	20	9	90.5	0.937			
15	5025	209	24.0	3.39	548	50	2.59	418	0.432	735	0.42	0.010	0.22	28	38	87.8	0.916	1990	441	2431
16	5006	209	24.0	2.91	468	67	2.50	402	0.730	1045	0.79	0.010	0.33	18	56	87.3	0.896			
17	5035	210	24.0	2.59	419	58	2.31	374	0.555	1201	0.51	0.010	0.26	27	45	89.6	0.903			
18	5043	210	24.0	2.35	381	79	2.01	326	0.548	1767	0.49	0.010	0.25	26	43	89.2	0.895			
19	5036	210	24.0	2.13	345	54	1.48	240	0.521	1465	0.37	0.010	0.23	50	40	88.6	0.893			
20	5028	209	24.0	1.65	267	53	1.52	246	0.470	1789	0.39	0.010	0.19	40	33	87.2	0.900			
21	4968	207	24.0	1.76	281	37	1.43	228	0.434	1855	0.34	0.010	0.22	25	38	85.5	0.853			
22	5001	208	24.0	1.69	272	46	1.29	207	0.405	1283	0.34	0.010	0.18	26	31	87.4	0.885			
23	3771	204	18.5	1.50	182	55	1.37	166	0.384	1549	0.31	0.010	0.17	23	22	86.8	0.886			
24	5034	210	24.0	1.43	231	45	0.96	155	0.389	12787	0.53	0.010	0.17	38	30	87.6	0.872			
25	505	202	2.5	1.27	21	28	1.28	21	0.457	1336	0.25	0.010	0.16	50	3	83.3	0.872			
26	4203	202	20.8	1.64	222	23	1.47	199	0.415	1141	0.24	0.010	0.11	27	17	91.4	0.896			
27	3014	208	14.5	1.23	119	51	1.26	122	0.318	1058	0.28	0.010	0.19	21	20	87.1	0.870			
28	4977	207	24.0	1.10	176	25	0.82	131	0.201	938	0.24	0.010	0.15	23	26	88.1	0.860			
29	5044	210	24.0	1.47	238	38	1.14	185	0.252	1090	0.22	0.010	0.13	27	24	84.1	0.861			
30	5055	211	24.0	1.04	169	33	1.02	166	0.323	965	0.30	0.010	0.15	62	27	86.8	0.883			
Overall	131521	204	644.8	1.62	6857	1182	1.30	5487	0.367	1421	0.34	0.010	0.17	29	803	86.4	0.883	4178	847	5025

	4384	**204**	**21**	**1.61**	**229**	**39**	**1.29**	**182.89**	**0.37**	**1436**	**0.34**	**0.010**	**0.17**	**28**	**27**	**86.4**
Average	**1102**	**10**	**5**	**0.69**	**121**	**15**	**0.51**	**94.81**	**0.13**	**2175**	**0.13**	**0.000**	**0.06**	**10**	**12**	**2.6**
SD	**5055**	**211**	**24**	**3.39**	**548**	**79**	**2.59**	**418.45**	**0.73**	**12787**	**0.79**	**0.010**	**0.33**	**62**	**56**	**91.4**
Max																
Min	**505**	**171**		**30.77**	**21**	**22**	**0.60**	**20.78**	**0.18**	**530**	**0.15**	**0.010**	**0.10**	**18**	**3**	**81.2**

7.1. Relevant Gold-In- Circuit Numbers

JANUARY 2020

	% Solids	Solids	Liquid	Density	Volume	Solids	Liquid	Gold	Carbon			Gold	
		Au g/t	ppm		m3	tonnes	m3	Kg	g / Litre	g/t	Tonnes	Kg	Kg
Preox tank	45	1,920	0.01	1.41	1.000	633	774	1.22				0	1.22
CIL1	44	0.49	0.374	1.39	1.000	614	781	0.59	5	1150	5	5.75	6.34
CIL2	44	0.36	0.114	1.39	1.000	614	781	0.31	4.5	713	4.5	3.21	3.52
CIL3	44	0.27	0.037	1.39	1.000	614	781	0.19	7	305	7	2.14	2.33
CIL4	44	0.27	0.015	1.39	1.000	614	781	0.18	8.5	145	8.5	1.23	1.41
CIL5	44	0.24	0.01	1.39	1.000	614	781	0.16	6.5	133	6.5	0.87	1.02
CIL6	44	0.23	0.01	1.39	1.000	614	781	0.15	22	68	22	1.5	1.65
Total												**14.7**	**17.49**

FEBRUARY 2020

	% Solids	Solids	Liquid	Density	Volume	Solids	Liquid	Gold	Carbon			Gold	
		Au g/t	ppm		m3	tonnes	m3	Kg	g / Litre	g/t	Tonnes	Kg	Kg
Preox tank	46	2,600	0.01	1.42	1.000	653	767	1.71				0	1.71
CIL1	45	0.69	0.612	1.41	1.000	633	774	0.91	4.5	1408	4.5	6.33	7.25
CIL2	45	0.44	0.152	1.41	1.000	633	774	0.4	7	608	7	4.25	4.65
CIL3	45	0.4	0.076	1.41	1.000	633	774	0.31	4	402	4	1.61	1.92
CIL4	45	0.34	0.036	1.41	1.000	633	774	0.24	4.5	208	4.5	0.94	1.18
CIL5	45	0.31	0.013	1.41	1.000	633	774	0.21	6	121	6	0.72	0.93
CIL6	44	0.27	0.01	1.39	1.000	614	781	0.17	17	63	17	1.08	1.25
Total												14.93	18.89

MARCH 2020

	% Solids	Solids	Liquid	Density	Volume	Solids	Liquid	Gold	Carbon			Gold	
		Au g/t	ppm		m3	tonnes	m3	Kg	g / Litre	g/t	Tonnes	Kg	Kg
Preox tank	46	1,073	0.015	1.42	1.000	653	767	0.71				0	0.71
CIL1	45	0.467	0.211	1.41	1.000	633	774	0.46	4.5	875	4.5	3.94	4.4
CIL2	44	0.355	0.136	1.39	1.000	614	781	0.32	5	537	5	2.68	3.01
CIL3	44	0.298	0.083	1.39	1.000	614	781	0.25	4	336	4	1.34	1.59
CIL4	44	0.294	0.029	1.39	1.000	614	781	0.2	7	177	7	1.24	1.44
CIL5	44	0.273	0.012	1.39	1.000	614	781	0.18	8.5	125	8.5	1.06	1.24
CIL6	44	0.239	0.01	1.39	1.000	614	781	0.15	18	63	18	1.14	1.29
Total												**11.4**	**13.68**

APRIL 2020

	% Solids	Solids	Liquid	Density	Volume	Solids	Liquid	Gold	Carbon			Gold	
		Au g/t	ppm		m3	tonnes	m3	Kg	g / Litre	g/t	Tonnes	Kg	Kg
Preox tank	47	1,290	0.059	1.43	1.000	673	759	0.91				0	0.91
CIL1	46	0.56	0.201	1.42	1.000	653	767	0.52	7	722	7	5.06	5.58
CIL2	46	0.44	0.153	1.42	1.000	653	767	0.4	7	298	7	2.08	2.49
CIL3	46	0.39	0.057	1.42	1.000	653	767	0.3	5	252	5	1.26	1.56
CIL4	45	0.36	0.042	1.41	1.000	633	774	0.26	5	144	5	0.72	0.98
CIL5	45	0.31	0.032	1.41	1.000	633	774	0.22	5	119	5	0.59	0.81
CIL6	46	0.3	0.01	1.42	1.000	653	767	0.2	17	62	17	1.05	1.25
Total												**10.76**	**13.58**

MAY 2020

	% Solids	Solids	Liquid	Density	Volume	Solids	Liquid	Gold	Carbon			Gold	
		Au g/t	ppm		m3	tonnes	m3	Kg	g / Litre	g/t	Tonnes	Kg	Kg
Preox tank	46	1,010	0.059	1.42	1.000	653	767	0.7				0	0.7
CIL1	45	0.47	0.387	1.41	1.000	633	774	0.6	3.5	1075	3.5	3.76	4.36
CIL2	45	0.33	0.086	1.41	1.000	633	774	0.28	9.5	406	9.5	3.86	4.13
CIL3	45	0.28	0.039	1.41	1.000	633	774	0.21	4	274	4	1.1	1.3
CIL4	45	0.27	0.02	1.41	1.000	633	774	0.19	5	128	5	0.64	0.82
CIL5	44	0.26	0.01	1.39	1.000	614	781	0.17	10.5	106	10.5	1.12	1.29
CIL6	45	0.25	0.01	1.41	1.000	633	774	0.17	15.5	64	15.5	0.99	1.15
Total												**11.47**	**13.75**

JUNE 2020

	% Solids	Solids	Liquid	Density	Volume	Solids	Liquid	Gold	Carbon			Gold	
		Au g/t	ppm		m3	tonnes	m3	Kg	g / Litre	g/t	Tonnes	Kg	Kg
Preox tank	46	0.68	0.01	1.42	1.000	653	767	0.45				0	0.45
CIL1	45	0.23	0.206	1.41	1.000	633	774	0.31	6	660	6	3.96	4.27
CIL2	45	0.14	0.045	1.41	1.000	633	774	0.12	6.5	258	6.5	1.68	1.8
CIL3	45	0.12	0.023	1.41	1.000	633	774	0.09	5.5	231	5.5	1.27	1.36
CIL4	45	0.12	0.02	1.41	1.000	633	774	0.09	5	103	5	0.52	0.61
CIL5	44	0.11	0.01	1.39	1.000	614	781	0.08	11	65	11	0.71	0.79
CIL6	45	0.11	0.01	1.41	1.000	633	774	0.08	14	40	14	0.56	0.64
Total												**8.7**	**9.92**

7.2.Relevant Elution Numbers

Jan-20			
Code	PCH (ppm)	PCT (ppm)	arrenSol. Au (ppm)
01-01	1,710	32	3.9
01-02	1,597	29	10
01-03	1,303	39	9.4
01-04	1,210	27	9.3
01-05	1,340	29	4.8
01-06	1,745	26	9
01-07	1,227	31	9.4
01-08	1,432	29	5.7
01-09	1,677	32	5.3
01-10	1,565	35	16.5
01-11	1,695	52	8.5
01-12	1,492	48	9.9
01-13	1,418	30	1.8
01-14	1,231	29	7.6
01-15	1,182	30	7.1
01-16	1,138	44	3.9
01-17	1,213	25	3.4
01-18	1,272	72	5.8
01-19	1,326	25	7.400
01-20	951	29	13.900
01-21	1,017	30	3.2
01-22	1,248	42	10.8
01-23	1,125	27	4.4
01-24	1,044	28	3.4
01-25	1,010	28	11.5
01-26	1,339	25	7.4
01-27	1,353	23	11.8
01-28	1,003	72	3.9
01-29	1,388	27	7.8
01-30	1,602	26	4.7
01-31	1,418	33	12.3
01-32	1384	32	9.1
01-33	1398	30	8.8
01-34	1640	26	9.7
01-35	1236	34	9.9
01-36	1,175	32	3.7

Code	**PCH (ppm)**	**PCT (ppm)**	**arrenSol. Au (ppm)**
02-01	1,233	59	3.6
02-02	1,174	34	9.1
02-03	1,151	34	5.4
02-04	1,162	28	3.4
02-05	1,043	28	10.1
02-06	958	26	8.6
02-07	1,097	29	4.5
02-08	1,260	113	5.8
02-09	1,364	26	9.6
02-10	1,265	28	5.1
02-11	1,127	28	3.3
02-12	1,260	23	4
02-13	1,113	25	8.4
02-14	836	28	8.4
02-15	901	23	7.4
02-16	814	22	5
02-17	1,059	28	3
02-18	927	39	4.7
02-19	1,206	33	3
02-20	951	60	10.7
02-21	1,055	68	12.5
02-22	953	34	10.5
02-23	972	27	8.9
02-24	1,370	34	10.8
02-25	1,122	30	10.5
02-26	1,178	26	6.6
02-27	1,278	22	5.5
02-28	1,207	32	10.4
02-29	1,086	24	6.8
02-30	1,922	28	8.3
02-31	1165	30	7.4
02-32	1128	25	13.1
02-33	1165	31	12.3
02-34	1445	23	7.1

Mar-20			
Code	**PCH (ppm)**	**PCT (ppm)**	**arrenSol. Au (ppm)**
03-01	1,500	76	8.7
03-02	1,629	38	7.4
03-03	1,209	29	3.3
03-04	1,078	33	5.4
03-05	1,219	79	4.4
03-06	1,228	32	9.6
03-07	1,182	28	2.7
03-08	1,533	25	8.2
03-09	1,401	29	2.5
03-10	1,312	41	2.3
03-11	1,243	24	10.5
03-12	1,175	24	5.9
03-13	929	22	12.8
03-14	956	25	4.6
03-15	860	21	5.3
03-16	741	21	2.8
03-17	1,116	26	3.4
03-18	1,314	23	9.12
03-19	1,198	28	7.2
03-20	1,306	59	4.6
03-21	1,336	54	7.2
03-22	1,473	30	2.2
03-23	1,498	20	5.7
03-24	1,215	92	4.3
03-25	1,406	23	3.2
03-26	1,538	39	5.4
03-27	1,308	31	9.4
03-28	1,389	38	2.2
03-29	1,540	26	3.4
03-30	1,595	23	3.1
03-31	1212	27	3.4
03-32	1048	44	4.7
03-33	1319	23	3.2
03-34	890	24	2.3
03-35	1064	21	6.9

May-20			
Code	PCH (ppm)	PCT (ppm)	arren Sol. Au (ppm)
05-01	989	21	7.3
05-02	1,586	28	6.2
05-03	1,478	30	8.1
05-04	1,239	312	7
05-05	1,452	96	11
05-06	1,054	43	5.8
05-07	810	27	10
05-08	1,352	52	9.2
05-09	1,102	42	8.7
05-10	1,340	24	9.7
05-11	985	25	10
05-12	1,275	39	8
05-13	1,122	37	1.3
05-14	1,051	34	12
05-15	1,241	26	2.6
05-16	1,011	31	7
05-17	1,035	33	13.7
05-18	1,005	21	2.8
05-19	1,010	28	6.3
05-20	820	28	3.4
05-21	840	28	5.5
05-22	804	25	5.6
05-23	830	18	7.6
05-24	820	23	7.4
05-25	981	25	9
05-26	926	24	3.5
05-27	1,070	22	3.4
05-28	930	35	5.3
05-29	1274	34	6.9
05-30	1308	24	12
05-31	1251	42	13
05-32	1,213	24	12.5
05-33	1,109	24	13.4
05-34	1085	25	5.5
05-35	1070	31	2.1
05-36	1152	31	11.9

Apr-20			
Code	PCH (ppm)	PCT (ppm)	arren Sol. Au (ppm)
04-01	1,119	32	7.5
04-02	1,182	28	8
04-03	1,199	45	4.9
04-04	1,174	30	4.8
04-05	1,157	23	4.6
04-06	1,024	21	9.4
04-07	1,065	77	2.8
04-08	1,381	37	3.9
04-09	1,260	26	3.5
04-10	1,258	51	4.3
04-11	1,068	63	9.7
04-12	875	19	1.3
04-13	811	23	2.8
04-14	914	19	3.7
04-15	760	32	5.6
04-16	905	37	6.5
04-17	711	21	3.5
04-18	805	22	4.1
04-19	1,057	79	2.9
04-20	1055	26	9.4
04-21	1019	64	5.2
04-22	889	23	4.7

Jun-20			
Code	PCH (ppm)	PCT (ppm)	arren Sol. Au (ppm)
06.01	1,217	32	3.03
06.02	893	20	1.2
06.03	737	19	
06.04	728	23	7.7
06.05	748	27	7.8
06.06	530	24	5.4
06.07	627	24	5.8
06.08	710	32	4.5
06.09	713	19	3.3
06.10	625	21	9.3
06.11	655	18	7.5
06.12	942	25	17.6
06.13	1,080	27	7.3
06.14	857	20	5.3
06.15	735	28	2.9
06.16	1,045	18	34.2
06.17	1,201	27	10.7
06.18	1,767	26	14.6
06.19	1,465	50	10.9
06.20	1,789	40	11.5
06.21	1,855	25	14.9
06.22	1,283	26	19
06.23	1,549	23	15.4
06.24	12,787	38	5.7
06.25	1,336	50	
06.26	1,141	27	9.7
06.27	1,058	21	6.5
06.28	938	23	2.5
06.29	1,090	27	2.3
06.30	965	62	1.5
06.31	972	68	
06.32	915	25	2.4
06.33	820	30	6.9
06.34	779	39	4.9
06.35	723	44	14.1
06.36	769	18	14.4
06.37	637	30	1.6