Gravity Concentration

at W.M.C.'s

St Ives Gold Mines.

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INTRODUCTION

Western Mining Corporation Limited owns and operates the St Ives Gold Plant located near Kambalda, approximately 90 kilometres south of Kalgoorlie in Western Australia. Production commenced in March 1988 at a design capacity of approximately two million tonnes per year. The plant was quickly expanded as a result of new ore discoveries in the district and has been processing ore since 1993 at the rate of three million tonnes per year. The plant has recovered over 50 tonnes of gold since the operation began.

The plant was able to maintain acceptable recoveries throughout the plant expansion phases. Further gains in recovery were possible since there was still free gold in the tailings after approximately 24 hours of leaching. Testwork carried out on prospective orebodies showed that some contained considerable quantities of free visible gold, whileothers were seen to be semi-refractory using a conventional grind - leach circuit. Alternative treatment techniques were considered because a substantial amount of revenue was lost with the treatment of these orebodies using conventional milling techniques.

The basis for all the investigations was to maximise the gold recovery from each orebody while minimising the effective cost per ounce of gold produced.

PILOT PLANT TESTWORK OUTCOMES

Several treatment routes were explored prior to the selection of the Knelson concentrators within the gravity circuit. These included flotation, fine grinding and various gravity concentration methods. It was decided early in the project to concentrate on gravity concentration processes as they were the most appropriate forms of gold concentration for the existing circuit and orebodies. They offered the ability to run fully automatically, produce good recoveries of free gold at low cost, were environmentally more appropriate and incorporated a high degree of security.

Parameters found to be critical from the pilot plant testwork carried out on site were feed particle size, feed particle shape and minimum water fluidisation flowrate. The fluidisation flowrate was important because if this rate was too low the concentration rings would bog and little or no recovery would occur. Once the minimum flow rate was found, it was then a simple matter of ensuring that this became a critical operating parameter of the machine.

Particle size and shape were also very important to total recovery. As the feed size increased and the particles became more elongated total gold recovery decreased quite markedly because the large elongated particles (often bell wire or ball scats) bridged the concentration rings making it very difficult for efficient separation to take place. Hence, square aperture screen panels between two and four millimetres were thought to be most appropriate for the circuit.

Magnetic material in the concentrate was also found to be a problem when further downstream treatment was carried out. Approximately 20 percent of the concentrate collected consisted of magnetic material, mainly ball scats and bell wire. Intensive cyanidation of the concentrate with the magnetic material present proved to be unsuccessful. Recoveries of less than 35 percent into the solution phase were achieved after 24 hours of intensive leaching.

Separation of the gold from the concentrate using a standard WilfIey shaking table was either inefficient if the magnetic material was not removed before separation or difficult due to the large amount of magnetic material collected. Prior cleaning of the Knelson concentrate using a magnetic separator proved to be successful in improving the separation of gold on the shaking table.

JUSTIFICATION

Three major and two minor reasons were applied to justify the project.

1. Reduced costs due to a reduction in the amount of carbon needed to be stripped.
2. Increase in revenue due to improved recoveries.

3. Cost savings due to a reduction in cyanide usage in the CIP plant.

4. Improved adsorption kinetics.

5. A reduction in the amount of gold locked up in the circuit and losses due to tailings spikes.

A pre-feasibility study was carried out by the Metallurgical and Engineering teams at St Ives. The initial circuit proposal consisted of **a** screen prior to gravity separation, three 30 inch centre discharge Knelson concentrators, a magnetic separator, a Gemini 1000 table and all associated tanks, pumps, pipes and structural steel work. This study showed the plant cost to be approximately one million dollars.

Elution Savings.

Examination of tailings samples via diagnostic testwork had shown that significant quantities of gold were still free for leaching in the solids. The quantity changed over time, however it was common to see 15 percent of the gold in the tailings unleached. Free liberated gold particles in the tailings could be seen under a microscope. These particles should ordinarily have been recovered by gravity separation methods.

Plant testwork using a 12-inch Knelson concentrator showed that an average gold recovery of 37.5 percent was achievable over a wide range of plant conditions. If this recovery could be extended to a full plant operation then there would be the ability to save the equivalent amount in elution costs, as this gold would not have to be won using the elution circuit.

The average cost of elutions at St Ives are less than $3,000 per strip. At a head grade of 3.57 g/t gold, one strip is done per day. If 37.5 percent of the gold is recovered via a gravity circuit then $300,000 per year in equivalent elution savings could be made.

Recovery Improvement.

Approximately 15 percent of the gold in the tailings was still available for conventional leaching methods. If this gold could be recovered via a gravity circuit then the following improvement in recovery could occur.

Assuming a head grade as stated above, a tailings grade of 0.24 g/t gold and a gold price of A$480/oz, a notional increase in revenue of $1.69 million oer vear could be made.

Leach Reagent Savings

Prior to the commissioning of the gravity circuit the free cyanide concentration in the leach slurry was 480 ppm. It was envisaged that if all of the coarse free gold was removed before leaching then it would be possible to reduce the free cyanide concentration while still maintaining good gold recovery. This section of the justification was extremely difficult to quantify and therefore an estimate of 10 percent savings was made based on an equivalent reduction in free cyanide concentration. The annual cyanide cost in the CIP plant is approximately $2,600,000. therefore giving an annualised saving of $260,000.

DESIGN CONSIDERATIONS

The first considerations taken when designing the plant were, what would be the appropriate size and capacity of the circuit and where should the circuit be located. The most concentrated slurry stream in the existing circuit is the cyclone underflow. The advantage of using this stream over others in the circuit is that it has a concentration ratio of gold of approximately seven to one. The stream can also be accessed high in the plant, allowing full opportunity to exploit the gravityfall of the slurry to the ground level.

Three Knelson concentrators were chosen to take advantage of the concentration ratio of the gold in the cyclone underflow slurry. Assays show that the gold travels around the grinding circuit approximately seven times. If a proportion greater than one seventh of the total feed can be processed, most of the gold entering the plant should pass through the gravity circuit.

Three concentrators operating at a nominal feed rate of 35 tonnes per hour satisfies the previous requirement. The water balance of the circuit also allows the water flow from the concentrators to counteract any additional water requirements to the grinding circuit.

A schematic drawing of the gravity circuit is shown in Figure 1.

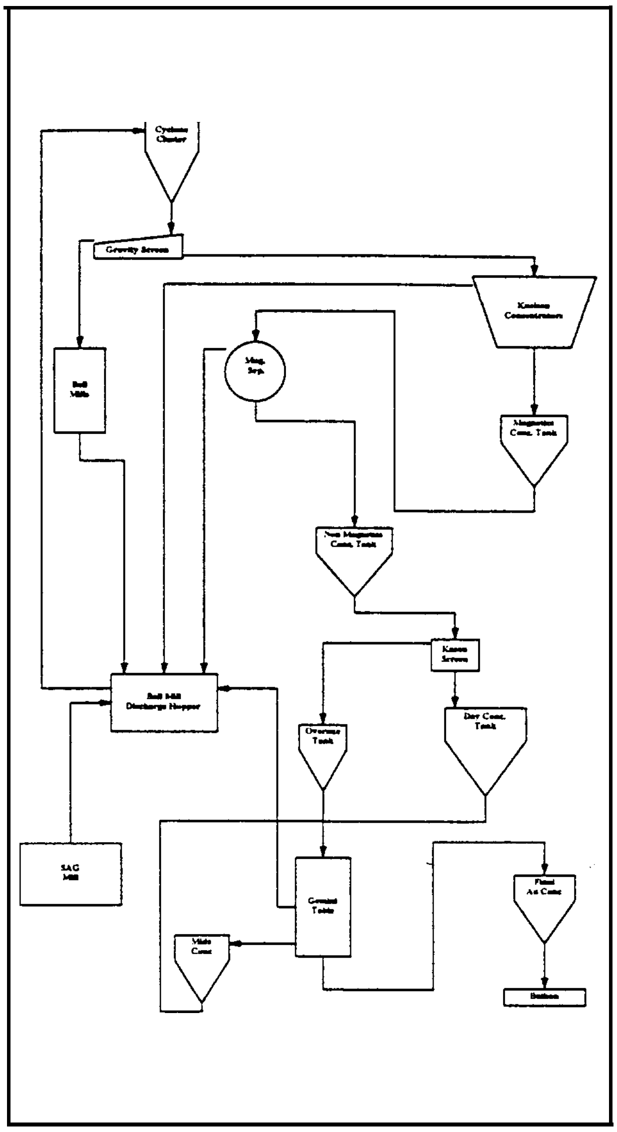


Figure 1 - St Ives Gravity Circuit Schematic

CIRCUIT DESIGN AND OPERATION

Feed System

Part of the underflow from the cyclone duster is fed to a vibrating screen before gravity concentration. The underflow passes through a dropout box located in the base of the cyclone underflow launder. The dropout box was designed to allow the heavy gold particles to fall by gravity to the pre-concentration screen, and the lighter particles to continue to the ball mill feed hopper. The dropout box is fitted with an auto dilution valve that allows both clearing of the slurry from the box during startups and dilution of the thick slurry so that screening efficiency is improved.

The slurry from the dropout box falls down onto a 1.2 by 3.0 metre Malco screen with polyurethane screen panels with two and four millimetre apertures.

Due to the finite height in the area encompassing the ball mills and cyclone tower, the screen underpan doubles as a feed distributor for the concentrators. **A** set of automatic control Linatex muscle valves and magnetic flow meters are bolted to the discharge pipes on the underpart to allow even distribution of the feed to the three concentrators. They also acted as isolation devices when the system is required to be shutdown.

The oversize from the screen passed over the end of the screen and into a sluice where the material was then fed to the ball mills via the ball mills feed hopper.

Knelson **Concentrators**

The three concentrators are located on the level below the pre-concentration screen and above the two ball mills. The machines are primarilymade of stainless steel for durability. The bowls are lined with polyurethane to withstand the abrasion of the slurry and all non rotating internal parts are covered with Linatex rubber.

The machines are fed with water at a controlled rate through a water jacket located around the rotating bowl. The bowl consists of ten concentric rings of increasing size, through which water is injected and concentrate retained. The slurry is fed to the machines down a centrally located feed pipe and discharged onto a distribution plate attached to the rotating bowl.

The slurry that passes over the top of the concentrator bowls, falls into a launder or directly into the ball mill discharge hopper. This diluted slurry then mixes with the ball mill discharge slurry and is again fed to the cyclone cluster for further sizing. The water injected into the bowls of the concentrators makes up approximately 90 percent of the water that would normally be added to the ball mill discharge hopper to control cyclone overflow density. Control of this water and the subsequent makeup water addition to the hopper is critical to the operation of the entire plant.

Magnetic Separation

Approximately 350 kilograms of concentrate are produced per day. These concentrates, which contain approximately 20 percent by weight, magnetic material, and 4% gold, are stored in a holding tank above the ball mill discharge hopper. The tank is designed to store approximately one tonne of concentrate. The tank's contents are cleaned daily for accounting and security purposes.

The magnetic separator is a high density concurrent wet separator made mainly ofstainless steel with polyethylene end flanges. The unit is designed to treat approximately one tonne per hour. As cleaning is carried out on a daily basis, its operation is usually limited to 30 minutes per day. Operation of the unit is carried out on a semi-automatic basis. All pumping, flushing and shutdowns are done automatically. Supervision of the feed flowrate to the machine is required.

The magnetic material passes over the end of the separator and falls into the ball mill discharge hopper. The non-magnetic concentrate falls into a storage tank below the magnetic separator. The non-magnetic concentrate tank is of the same dimensions as the magnetic concentrate tank.

The magnetic concentrate is pumped to the magnetic separator via a Warman 3/2 B-SC pump which is located below the magnetic concentrate tank. The flow rate of the slurry to the separator is controlled via two valves. One valve controls flow to the separator itself and the other valve controls the amount of material that is recycled back to the tank.

The non-magnetic concentrate is pumped to the gold room using a warman 3/2 C-AH pump. Operation of this pump is fully automatic and transfers are made only at the request of the Goldroom Supervisor. The discharged slurry from the tank is diluted with process water and pumped over a Kason screen in the goldroom. The entire transfer procedure takes approximately five minutes per day.

Gold Room

Concentrate Storage and Handling

Diluted non magnetic concentrate (approximately 15 percent solids) is pumped over a three-foot Kason K30 vibrating screen. The screen is fitted with a 1.5 millimetre woven wire screen cloth. Undersize concentrate from the screen falls into a day tank, which has a capacity of approximately four tonnes. The oversize from the Kason screen falls into a smaller tank.

The Kason screen was placed in the circuit to allow more efficient operation of the Gemini table. The Gemini table is rated as having maximum efficiency of recovering gold that is one millimetre or less in size. The screen separates the concentrate into the two products at an approximate ratio of 12 kilograms of undersize material to one kilogram of oversize.

The holding tanks are both fitted with Linaflow control valves. The control panel for the valves is situated next to the Gemini table. Control is carried out by the adjustment of air pressure regulators, that control the throat opening size of a diaphragm inside the valves.

Gemini Table and Tails Handling

The largest of the Gemini tables in the supply range was chosen for use within the circuit. The Gemini 1000 table has a rated capacity of 1000 pounds of alluvial feed per hour (~500 kg / hr). The feed rate is reduced as the feed particle size decreases.

The concentrate is fed onto the end of the table that slopes gently down and away on both sides. Middlings are collected near the end of the table and fall into one of four collection containers. These containers can then be used to feed the middlings onto the table for further upgrading once feeding of the primary concentrate from the day tank has been completed.

The concentrate collected at the end of the Gemini table can be direct smelted after nitric acid digestion of the gangue. The tailings from the table falls into a small hopper located below the Gemini table and is pumped to a secured sump pump outside the goldroom via a Warman 1.5/1 B-AH pump. The sump pump directs the tailings back to the ball milldischarge hopper for further treatment.

Security

Security was always a major concern when considering designs for the plant. The security system will not be outlined in this paper. Devices have been put in place that will not allow any access to any area of the concentration section of the circuit without full supervision.

The automatic Knelson concentrators were chosen for the fact that they required no physical attendance during their operation, except for a mechanical failure. All valves, pumps, screens and other equipment operate automatically. This is not done only from a security stand point but also from the aspect of continuity of the process and for reduced operator involvement.

COMMISSIONING

The first meeting to discuss the concept of the circuit design was held in November 1993 with the consultant design engineers. Early discussions showed that it could be possible to design, tender, construct and begin commissioning of the plant by the middle of May 1994. Tenders for the construction of the plant were called near the end of February 1994 after all drawings had been finalised The contract for construction of the plant was let in March. An eight-week construction period was agreed upon.

Work commenced in March and was completed within two months, on time and within budget. A further week of electrical and instrument tie-ins slightly held back commissioning. Commissioning of the new plant began at the end of May 1994.

Feed System

Commissioning of the feed system caused the most modifications to be made to the plant. The pre-concentration screen was commissioned using alternate rows of two and five millimetre square aperture panels. It was evident that the slope of the underpan base was too shallow. A water tundish was added to the top of the underpan that acted as a sluice, washing the solids to the bottom of the underpan. The slurry was also seen to be travelling too fast over the surface of the screen, causing very poor screen efficiency. The addition of dilution water to the feed together with the water sprays reduced this problem.

Many combinations of aperture sizes were tried on the screen. The final aperture size was a compromise between open area and down stream recovery. It was decided to keep the open area high so that the feed rate to the concentrators could be maximised. It has been seen that the greatest effect on recovery is the amount of tonnes treated by the gravity circuit.

Knelson Concentrators.

Operation of the concentrators went smoothly through commissioning. All programming interlocks proved to be successful. Initially, the slurry flow rate to each unit was attempted to be kept as even **as** possible. This strategy was later abandoned when it was seen that the concentrators could handle a wide range of feed rates. The resistance in the feed lines was enough to ensure that the concentrators were never overfed.

	KC1	KC2	KC3	TOTAL
Ave. Feed Rate (m3 per hour)	38	38	35	37
Ave. Availability %	72.30	72.80	75.60	73.60
Ave. Tonnes Per Day Treated	644	650	642	1936
Ave. Water Diff. Pressure (kPa)	46	47	66	53
Ave. Bowl Pressure (kPa)	66	65	65	65
Ave. Plant Head Grade (g/t)				4.24
Ave. % of Total Feed Treated				21.7%
Percent Circuit Recovery				37.1%

Table 1. Knelson Concentrator Operating Statistics.

Table 1 shows the operating conditions and results since startup. It can be seen that the circuit has recovered 37.1% of all the gold introduced to the plant.

A study comparing the effect of tonnes of material treated and actual gold recovery from the gravity circuit was made. The effect of tonnes treated was tested in early October when the feed rate to each concentrator was increased and their utilisations maximised. It was seen that although the amount of gold to the plant remained relatively constant, the recovery improved in line with the increase in quantity of tonnes treated. The feed rate to each concentrator has been made a priority control tool since this was established.

Problems have been experienced with blockages of the fluidisation holes with the bowls. When the bowls were initially removed, a very hard concrete like deposit was found in the bottom three rings. The upper rings were generally clean. The outside of the bowl was covered in a very fine deposit of calcite or gypsum that was easily removed when washed in a dilute hydrochloric acid bath. After the scale was removed, the holes were all reamed using an oxy/acetylene lance tip cleaner. The entire procedure for cleaning the bowls takes approximately thirty minutes to remove the bowl, an hour and a half to clean the bowl and three quarters of an hour to replace the bowl.

It was noticed early in the commissioning of the concentrators that a single flushing cycle during the concentrate cleanup step was not sufficient. An extra flush cycle was added to try to break the crust that was forming at the bottom of the bowl and to allow thorough cleaning of the fluidisation holes to occur. This has proved to be partly successful.

Magnetic Separation

The magnetic separation of the primary concentrate has proven to be the most manual step in the process. The system is semi-automated as previously stated however close supervision of the magnetic separator must be made during its operation.

Flux	E/W Sponge	Gravity Concentrate
Borax	2	3
Nitre		1
Manganese Dioxide	1	
Soda Ash	0.5	1
Silica	0.5	1

Table 2. Flux Blend Comparisons.

The major problem that has occurred in this area is that the feed material has varied greatly in nature. The particle size has varied from ten millimetre pebbles to fines. The pebbles have blocked the impeller of the pump and sanded the lines. They have also scored the surface of the drum separator. The large rocks were carried over from spillages from the gravity screen. Wear on the two manual feed valves has also been a problem. The valves are both Saunders valves and when the bonnets wear out they quickly bog the pump and lines.

The magnetic separator has been extremely efficient in removing any magnetic material and removing many downstream processing problems. The magnetic material falls directly into the ball mill discharge hopper and in doing so releases the entrained gold for further recovery by the gravity circuit.

Concentrate Transport

Once the magnetics have been removed from the primary concentrate, the non-magnetic material fallsinto the non-magnetic concentrate tank. The procedure to transport the concentrate to the gold room is fully automatic.

This section of the process has run without any problems to date.

Gemini Table

The operation of upgrading the concentrate grade over the table is carried out manually. The feed rate is controlled via pressure regulators that control the open area of the Linaflow control valves. The table has proved to be an extremely effective tool in upgrading the concentrate. Once it was set up during the commissioning phase of the project, the table has run with only minor adjustments.

The table can produce a final grade concentrate (88% gold) in one pass from a feed grade of approximately 5% gold. The major problems that have occurred are that many hold down bolts and bump springs have broken due to excess vibration. Larger hold down bolts have since been used to secure the table.

Tabling occurs for approximately two hours per day. In that time the undersize material in the day tank is tabled over a ninety minute period. After that, the oversize material and the middlings are tabled. This process takes a further thirty minutes.

The small tailings pump below the Gemini table has worked well after some minor teething problems associated with the small size of the pump and therefore its inability to pump rocks.

Smelting.

The final concentrate collected from the Gemini table is added to a concentrated nitric acid bath. After the concentrate has been removed from the bath, it is washed and placed in a stainless steel tray to dry. Drying is carried out on top of the rotary furnace below an exhaust fume chute. The flux blend that is used when treating the gravity gold is slightly different to hat of the gold sponge from the electrowinning circuit.

The average grade of the bullion produced from the CIP circuit is approximately 88.5%, whereas the gravity bullion averages 94.5%.

Circuit Control

The entire circuit is controlled from the MOD 300 Distributed Control System. The main basis of concentrator control revolves around the control of the cyclone overflow density. Before the installation of the gravity circuit, cyclone overflow density was not automatically controlled but only monitored. The cyclone feed density was used for control as this was believed to be a good indicator of cyclone overflow density. Control was carried out by adding water to the ball mill discharge hopper according to the density measured just before the cyclone cluster.

Cyclone overflow density control was implemented just before the commissioning of the gravity circuit. Control is carried out by monitoring the density of the cyclone overflow as it is pumped out of the CIP surge tank. The measured density is compared against a set point and then ratioed against the new feed tonnage to the SAG mill.

A calculation of the actual flow of water needed to be added to the ball mill discharge hopper is made. This figure scans all water sources that are added to the ball mill discharge hopper, whether it comes from the concentrators or is new make-up water. A magnetic flowmeter and control valve were added to the process water line before the ball mill discharge hopper. The concentrators are all equipped with water flowmeters and control valves.

This new control strategy has been very successful. Not only has the water balance to the circuit been closely controlled, but the grindsize of the cyclone overflow slurry has improved by approximately 8%. Before the new water control strategy the cyclone overflow sizing was 65% passing 75 microns. This sizing has now improved to 70% passing 75 micron.

The water required at the ball mill discharge hopper is continually monitored and compared with the amount of water actually being supplied to the circuit. If the water required to be added to the circuit is less than that currently being added, (for more than thirty minutes), then the concentrator that has been running for the longest time will automatically shutdown and discharge its concentrate.

The concentrator is not allowed to restart until the water balance supplies at least one hundred litres per minute more than required to operate it. The concentrators operate on either a fully timed schedule, or as required by the water balance.

OPERATIONAL PERFORMANCE STATISTICS

The eight months of operation of the St Ives gravity circuit until the end of January 1995 has been analysed in this paper. The plant has in the main met all of the objectives that were set out in the initial justification over 12 months previously.

	Pre Gravity		
Data Set	Head Grade g/t Au	Tail Grade g/t Au	Recovery (%)
Less than 3.0	2.66	0.20	92.69
>=3 & <3.5	3.26	0.21	93.64
>=3.5 & <4.0	3.68	0.22	94.04
>=4 & <4.5	4.18	0.22	94.64
Greater than 4.5	4.64	0.24	94.93
	Post Gravity		
Data Set	Head Grade g/t Au	Tail Grade g/t Au	Recovery (%)
Less than 3.0	2.76	0.17	93.94
>=3 & <3.5	3.29	0.18	94.62
>=3.5 & <4.0	3.74	0.19	94.91
>=4 & <4.5	4.24	0.21	95.12
Greater than 4.5	5.26	0.21	95.97

Table 3 - Gravity Circuit Data GroupAnalysis

The data used for evaluating the effectiveness of the gravity circuit has been collected since August 1993. This data was retrieved from a Paradox for Windows Database introduced simultaneously. Eleven months of data from the start of August 1993 to just before commissioning at the end of May 1994 were grouped as the Pre Gravity data set. The second data set consisted of numbers from the start of June 1994 to the end of January 1995 and was called the Post Gravity data set.

A search of the entire data population was made, excluding days on which the daily feed tonnage was less than 8000 tonnes. This was done to eliminate the effect of residence time of leaching on the final tailings grade. The two sets of data were then grouped into a further five sets. These sets contained the plant head grades ordered in the following groups.

Group 1 Less than 3.0 grams per tonne.

Group 2	Greater than or equal to 3.0 and less than 3 .5 grams per tonne.
Group 3	Greater than or equal to 3.5 and less than 4.0 grams per tonne.
Group 4	Greater than or equal to 4.0 and less than 4.5 grams per tonne.
Group 5	Greater than 4.5 grams per tonne.

The reason for comparing groups of data rather than simply the overall average for the periods was to find if the gravity circuit was effective over a wide range of head grades and if the effectiveness was consistent across all groups.

Data Set	Pre Gravity	Post Gravity	Saving
Tonnes	2,550,708	2,030,301	
Grade (g/t Au)	3.29	4.22	
Recovery (%)	93.93	95.24	
Cyanide (kg/t)	0.711	0.590	0.121
Cyanide (kg)	1,814,371	1,197,909	
Strips	340	246	
Kg Gold per Strip	23.13	33.20`	30.27%
Notional Carbon Loading (g/t)	3,300	4,743	

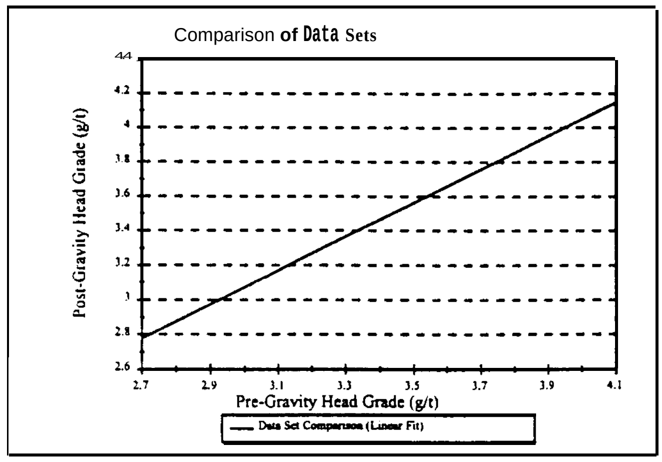
Table 4 - Comparison of Plant Performances

Elution Savings.

The savings in the elution plant were estimated by comparing the relative amount of gold recovered over an extended period (Table 4).

The Pre Gravity data set showed that 340 strips were carried out on carbon at an average loading of 3,300 g/t gold equating to a recovered gold quantity of 23.13 kg per strip. The Post Gravity data set showed that 246 strips had been carried out. The notional amount of gold recovered from the plant over this period was 33.2 kg per equivalent strip. If the carbon were stripped every 23.13 kg of gold that entered the plant, then the equivalent number of strips would be 353.

This then equates to a saving of 107 strips or 30.3% of the total stripping costs.



Graph **1** - Data Set Comparisons.

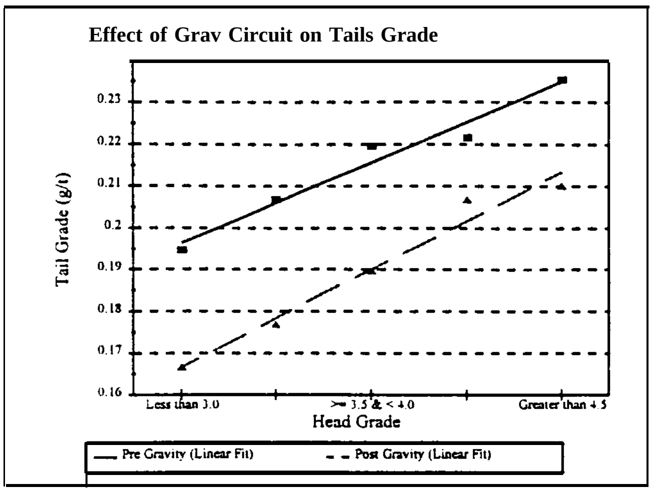
It was assumed in the initial project justification that the amount of gold recovered in the gravity circuit would be equivalent to the savings made in the elution circuit. The justification said that 37.5% of the total gold added to the plant would be recovered in the gravity circuit with an equivalent elution saving. However, the average gold recovery from the gravity circuit has been 37.1% (and improving), while the elution saving has only been 30.2%.

Recovery Improvement.

The recovery from the data sets is shown in Table 3. The data has then been linearly regressed and the lines of best fit have been plotted and shown in Graphs 1 to 3.

An approximate average recovery improvement of 0.92% is seen across the board. The actual data does fluctuate but the spread of recoveries once linearised is seen to be quite even. The spreads in the tailings grade improvement is seen to reduce as the head grade increases. This is in part due to the slight separation of average head grades for the Group 5 set of data points. The average reduction in tails grade as taken from the linearised Graph 2 is 0.03 g/t gold.

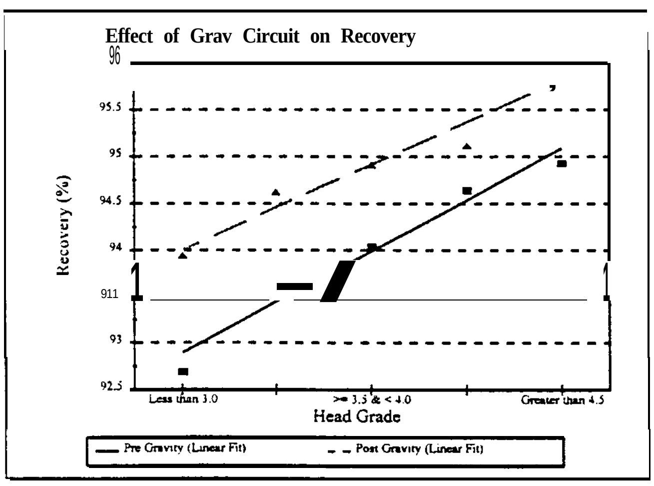
The annual increase in revenue since the commissioning of the gravity circuit is therefore approximately $1.272 million at an average gold price of A$515/oz.



Graph 2 - Effect of the Gravity Circuit on Tails Grade.

Cyanide Reagent Savings.

Table 3 shows that the cyanide usage in the CIP section of the plant has fallen from 0.711 kg/t before the installation of the gravity circuit to 0.590 kg/t since its installation. This represents a usage reduction of 0.12 1 kg/t and an annual cost saving of over $350,000.



Graph 3 - Effect of the Gravity Circuit on Total Recovery.

Leach Kinetics.

The performance of the adsorption section of the CIP plant has met or surpassed all expectations. The average grade of the tailings solution before the gravity circuit installation was 0.016 ppm gold for a head grade of 3.28 g/t gold. The solution tailings grade has fallen to 0.012 ppm gold (4.24 g/t gold head grade) since the installation of the gravity circuit. This is a 28% reduction in gold lost in the solution phase alone.

The CIP circuit variability has also fallen significantly. Prior to the installation of the gravity circuit the standard deviation (SD) of the head grade was 0.56 g/t gold and the SD of the tails was 0.04 g/t gold (total) and 0.012 ppm gold in the solution phase. Since the installation of the gravity circuit, the SD of the head grade has been larger at 0.93 g/t gold but the SD of the tails was only 0.035 g/t gold (total) and 0.006 ppm gold for the solution phase. This is proof of the reduction in variability in the tailings due to the installation the gravity circuit.

It can be seen that the reduction in concentration of free cyanide in the leach circuit has not affected recovery. This was not so prior to the installation of the gravity circuit as the higher free cyanide concentrations were required to help reduce tailings spikes due to spikes in the head grade.

Operating Costs.

Operating costs of the plant have been significantly higher than were predicted in the project justification. The justification estimated that the operating cost each month would be $4,000. The average operating costs of the circuit, including commissioning has been $15,000.

Production Cost of Gold.

It is very difficult to compare the production cost of gold per ounce as there are too many unrelated factors influencing the costs. The production cost of gold has fallen twenty percent however the head grade has also risen by twenty percent over the same period.

The total saving or increase in revenue as outlined above has been substantial.

Annualised Savings / Increase in Revenue to Date:

		(Millions)
1.	Elution Plant Saving	$0.4
2.	Increased Revenue from Improved Recovery	$1.2
3.	CIP Cyanide Saving	$0.4
4.	Operating Costs	($0.2)
	TOTAL	**$1.8**

These figures show that the approximate payback period of the project to be seven months.

ACKNOWLEDGMENTS

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