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MARVEL LOCH GOLD MINE'S EXHIBITION HEAP LEACH PROJECT

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ABSTRACT

In May of 1987, the Marvel Loch Gold mine began heap leaching high-clay, low-grade ores from its Exhibition Pit. The project was designed at 1500 tonnes per day with double-shifting allowing a 3000 tonne per day processing rate. The ore is crushed, agglomerated, and conveyor-stacked to 10 meters high on PVC-lined leach pads. Installation, startup and operating experiences are presented.

INTRODUCTION

On 16 May 1987, the Marvel Loch Gold Mine began processing high clay ore via heap leaching at their newly commissioned Exhibition Heap Leach Project at Marvel Loch, Western Australia. The first gold bar was poured on 5 June 1987 and recent production has averaged approximately 62 kilograms (2000 ounces) per 4-week period.

The process involves crushing to minus 25 millimeters in closed circuit, agglomeration with cement and barren solution, conveyor stacking on PVC-lined leach pads, and sprinkling with cyanide solution. Gold is recovered from solution in 1.8-tonne carbon absorption columns which are periodically transported to existing milling facilities for stripping.

State-of-the-art agglomeration and stacking equipment was designed and built specifically for this project. The project design was based on 1.6 million tonnes of reserves. Initial production rate was set at 1500 tonnes per day, but double shift operation of the crushing and agglomerating facilities has resulted in a processing rate averaging approximately 2500 tonnes per day.

HISTORY

Mining in the Marvel Loch area dates back to the turn of the century. A second surge of activity occurred in the mid-1930's when underground miners chased quartz veins accessed by Marvell Loch, Firelight, Boulder, and Exhibition shafts to depths of 130 meters. The most recent mining started in 1979 when the Marvel Loch shaft was reopened. Drilling of near-surface ore in what was to become the Marvel Loch Pit began in 1983.

Full-scale open pit production from the Marvel Loch pit spanned late 1984 to the end of 1986 – during this period, 400,000 tonnes of ore grading 3.5 grams per tonne and 700,000 tonnes of ore grading 1.5 grams per tonne were mined. The higher grade material was milled in the existing carbon-in-pulp plant while the lower grade portion of the oxidized ore was stockpiled.

As the Marvel Loch Pit was depleted, drilling commenced along strike to the north of the Exhibition shaft area. Although little low grade ore was defined in the reserve by RC drilling, it was strongly suspected that similar ratios of mill feed and low grade ore, as observed in the Marvel Loch Pit, would be realized. Mining commenced in mid-1986 and is continuing at present. The suspected low grade ore has been encountered and defined by intensive in-pit sampling. The ratio of mill feed to low grade ore of approximately 1:2 has continued in this pit and present production is about 60,000 tonnes per month.

As with the Marvel Loch pit ore, the high grade material was processed in the existing mill while the lower grade material was stockpiled. The presence of large quantities of low grade material led to the decision to construct a heap leaching facility with

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projected recoveries of 70 percent of the contained gold from ore averaging 1.6 grams per tonne.

GEOLOGY/MINERALOGY OF ORE

The ore bodies are located in the Southern Cross greenstone belt within an Achaean sequence of metamorphosed, interlayered basalts and sediments. The Exhibition open pit itself is located within a 70 to 80 meter wide shear zone oriented at 20 degrees to the sequence.

Primary mineralization is located in an en relais set hosted by metabasalts and carbonaceous pelitic to semi-peitic metasediments. Later pegmatites and faulting disrupt the ore system. Alternation of mineralization in the main zone consists of silicification, carbonization, and sulphidization. Some biotitization is also present.

The ore is severely oxidized from the surface to a depth of about 60 meters. The oxidation profile consists of three meters of red lateritic soils consisting of kaolin and hematite with minor quartz and dolomite. Below this, the rocks are strongly bleached and kaolinized.

Little gold appears in the first 10 meters of the bleached zone; however, strong supergene halos exist within the kaolin at greater depths. It is the supergene gold within the kaolin which constitutes the majority of the low grade heap leach feed. The presence of such large amounts of kaolinized material results in extremely difficult processing characteristics.

TEST WORK

Heap leach testing of low grade material consisted of both laboratory column tests and a 25,000-tonne trial heap leach test. These tests were conducted on ore from the adjacent Marvel Loch pit. This material was considered to be representative of the low-grade ore expected to be encountered in the upper 40 meters of the Exhibition pit. The results indicated that adequate gold recoveries to support a heap leaching operation could be attained from material crushed to minus 50 millimeters and agglomerated with 1.0 percent cement.

However, as is often the case, the actual ore initially processed from the Exhibition pit did not have identical characteristics to the ores previously tested. This led to no small amount of anguish prior to startup, but small-scale testing conducted during commissioning of the plant resulted in alteration of the operating parameters to a crush size of minus 25 millimeters, addition of 2.0 percent cement, and agglomeration at a final moisture content of slightly over 20 percent. Gold recoveries of 70 to 75 percent were expected.

Based on the laboratory test results, the design leach time was selected to be a total of 90 days broken into two 45-day leach cycles. Leaching would take place in a counter-current fashion.

PAD DESIGN & INSTALLATION

Clearing of the leach site began in mid-1986, although the final layout of the project was not determined until October of that year. Clearing and stockpiling of the topsoil was scheduled at the convenience of the mining contractor.

The leach pad was designed with the stacked ore segregated into “modules” of 45 days production by intermediate berms. The pad site was divided into two sections, with natural drainage roughly cutting the area in half. Stacking would begin at the downslope end of the eastern half of the leach area. Eight modules would be constructed, each draining to a common collection channel running down the center of the leach area.

After the eastern half of the area had been filled with ore, the western half would be prepared in a mirror image fashion. A total of 16 modules would be contained in the area.

Initially, sufficient pad surface for 4 modules (roughly 48,000 square meters) was prepared. Pad preparation consisted of removal of topsoil, grading as required, and compaction of the exposed kaolinitic surface material followed by manual inspection of the surface for the presence of roots, twigs, and rocks. The berms were prepared by windrowing sand to the designed height and location using a grader. A layer of 0.46 millimeters PVC was then laid, glued, and anchored.

Pad preparation was hampered by the lack of available water for compaction during the dry summer months. This led to considerable delays in the program. High winds during the PVC installation period also resulted in some setbacks (you can’t have too many sandbags!).

Initially, labor for installation of the pads was supplied from the existing mill crews. However, it soon became evident that a dedicated “pad crew” would be required to maintain consistency in installation techniques and to keep to the proposed schedule. After installation of the PVC, a layer of non-woven geotextile was placed over the entire leach area. Perforated agricultural drain pipe (65 millimeter diameter) was placed up and down slope on 6 meter centers on the pad surface. A 300 to 400 millimeter thick layer of crushed/screened ore was placed on the pad where vehicular traffic was likely to occur (conveyors, loaders, etc.).

The gravel was sized at -25 millimeters / +6 millimeters to form a percolation base as well as to protect the PVC from puncturing during placement of the ore. Significant load bearing ability was also required to support the relatively high ground pressure of some of the equipment. Material used for this function was mined form the oxide/sulphide transition in the Marvel Loch pit. Gold content of the material averaged 1.35 grams per tonne and a 60 percent extraction was expected.

Initially, the production crushing plant was used to prepare the gravel, but a dedicated crushing plant has since been brought onsite to reduce the production losses during preparation of pad cover gravel. Fines from the screening operation are stockpiled for later agglomeration with the clayey ores.

PVC pipes are installed through the downslope berm of each module to allow exit of process solutions. These pipes are then connected to a distribution box which directs the solution to the appropriate process pond via large diameter collection pipes. The main collection pipes are located in a central ditch which connects all the heap modules to the ponds. This ditch was dug with a scraper and is lined with geotextile and PVC.

POND DESIGN & INSTALLATION

Three process ponds are required for a 2-stage heap leaching operation: pregnant, intermediate, and barren. During normal production, barren solution is pumped to an older module in the secondary leach stage. The intermediate solution exiting this module flows to the intermediate pond where it is pumped to the module in the primary leach stage (new ore). Solution exiting this module then flows to the pregnant pond before being processed in the carbon columns.

These ponds were sized to contain minimum process pump requirements plus a 24-hour rainfall event of 50 millimeters over 4 modules of ore. Each pond is identical in construction with a contained volume of 3,420 cubic meters each. They are interconnected with spoon drains so that all ponds fill before any overflow can occur.

In addition, a fourth pond was constructed with a contained volume of approximately 9,000 cubic meters. This overflow pond is used for emergency containment in the case of an excessive rainfall event. It also doubles as a storage capacity for makeup water during the dry months.

Solution flowing into the intermediate or pregnant ponds passes through a Parshall flume immediately before entering the pond. This allows monitoring of solution flow rates and aids in assessment of the overall metallurgical balance.

The process ponds were constructed with 1:1 side slopes and lined with 0.46 millimeter PVC. In addition, a layer of geotextile was placed below the PVC to provide further protection from puncture. In hindsight, the pond walls are much too steep - operators scrambling up and down the pond sides during pump maintenance can easily cause pinhole leaks. In addition, thicker liners (minimum 0.75 millimeter material) with enhanced ultraviolet protection should have been employed to give added protection against puncture and long-term UV degradation.

PROCESS PLANT

Due to the presence of the existing milling operation, the process plant for the project was quite simple, consisting only of carbon columns, a carbon screen, and a reagent mixing sump.

Six specially designed carbon columns are employed for processing pregnant solution, and they can be operated in either fixed or fluidized bed mode. The columns are located on a concrete slab that drains to the barren pond. Solution pumped to the columns flows through one of two magnetic flow meters with instantaneous and totalized flow capacity.

Larger diameter (150 millimeter) rubber hoses are used to connect the columns in either a series or parallel arrangement. Victaulic fittings are used for relatively rapid connection and disconnection. Although connection of the hoses can be cumbersome, this was the simplest and cheapest method of connecting the columns while still maintaining desired flexibility in piping arrangements.

Normal operation uses a parallel arrangement consisting of one set of two columns and one set of three columns on-line at any time. A higher solution flow rate is processed in the set of three columns to reduce gold losses to barren solution. As the lead column in one of the sets of becomes loaded, it is removed from the circuit for stripping and the next column in line is “moved” to the front of the set. As stripped columns become available, they are placed in the last position on the appropriate set of columns.

Carbon loadings have been exceptionally good: 11,000 grams gold per tonne of carbon during the initial months of operation with present loadings in the 8,000 to 10,000 gram per tonne range. Given that preg concentration have averaged only about 1.0 ppm and that process water contains in the neighborhood of 200,000 ppm of dissolved solids, the continuation of this level of loadings is quite remarkable.

Solution exiting the last column in-line flows to a carbon catch screen consisting of a DSM screen mounted in a standard screen box. The box was modified from one which had been removed from service in the CIP plant – the sizing was quite fortunate. Any carbon washed out of the columns is caught on the screen and collected before eventually being returned to the adsorption columns. Barren solution flowing through the screen is directed to a concrete reagent mixing sump before overflowing into the barren pond. NaCN briquettes are added to the sump from drums by manual dumping.

When a batch of carbon is sufficiently loaded with gold, the column is transported to the existing mill for stripping. Initially, the carbon was stripped in situ in each adsorption column, but the large flow rate required by the column for proper solution distribution forced a change in stripping operation. At present, the carbon is educated from the adsorption column into specialized tanks for stripping and acid washing.

The carbon columns and the heaps are fed by Flygt 2150 submersible pumps mounted on a raft in each pond. Pump starters, check valves, bypass lines, and strainers are mounted at a pump station on the edge of each pond. These pumps were chosen to match existing pumps used in de-watering the underground workings and continue to provide low maintenance and easy startup (no priming).

CRUSHING & SCREENING

Early in the evaluation program it was recognized that the near-surface ore containing over 80 percent kaolin would lead to plugging problems in a conventional jaw/cone crushing circuit. However, ore material found deeper in the pit contained sufficient quartz mineralization to adversely affect operating costs in an impact-type crusher. The final decision to use a jaw/cone plant was based on the availability of the equipment.

The crushing plant itself consists of a belt feeder/hopper with grizzly, 760 millimeters by 1060 millimeters (30-inch by 42-inch) jaw crusher, and a 1295 millimeters (4 ¼-foot) Symons standard cone crusher (coarse bowl) operating in closed circuit with a 1830 millimeters by 4880 millimeters (6-foot by 16-foot) vibrating screen.

The system is generally too small for continuous production at 150 tonnes per hour on high clay material. As expected, severe plugging problems were encountered whenever the ore contained a large clay fraction (a significant portion of the time) or excessive moisture. This has been a continuing problem and actual production is chronically behind schedule throughput. Hindsight suggests that more time should have been spent locating an appropriately sized impact-type crushing system for use during the initial stages of operation.

Feed to the crushing plant is stored on a large ore stockpile located adjacent to the plant. Haul distance is about 0.8km from the pit boundary and 70-tonne haul trucks are used for transporting the ore.

A front-end loader is used to feed the plant. Crushed ore product (screen undersize) is conveyed to a 60-tonne surge bin which feeds the agglomeration system. Sampling of the ore takes place at the discharge of the surge bin. Two to four samples per shift are collected for moisture analysis and gold assay determinations.

The crushing/agglomerating plant capacity was sized at 1500 tonnes per shift. Single shift production was instituted until leaching behavior was determined. Double shift processing commenced two weeks after initiation of leaching.

AGGLOMERATION & STACKING

Agglomeration of the ore is accomplished in a 2.5 meter diameter by 10 meter long rotating drum. The drum was custom manufactured for this project and incorporates design features allowing for considerable flexibility in setup and operation. Drum speed is 12.5 revolutions per minute and inclination can be varied from 6 to 10 degrees depending on the characteristics of the ore. A scraper bar mounted along the inside top of the drum eliminates buildup of material on the drum wall. A shroud on the feed end of the drum was installed at the request of the Mines Department to reduce dust emissions.

Ore is fed to the drum using a variable speed belt puling from the bottom to the surge bun. Throughput is controlled using the output from a belt scale located on the feed belt. Operators can set the drum feed to coincide with changes in throughput of the crushing system, thus maintaining a consistent federate to the drum.

Cement is used as a binding agent and is metered onto the ore on the drum feed belt. The cement is stored in two 50-tonne silos and is transferred from the storage silo to the live silo using a dedicated blower. A screw feeder is used to feed cement from the bottom of the live bin onto the drum feed belt. Speed of the screw is also controlled by the belt weightometer through a ratio controller. This allows for adjustment of the percent of cement added to the ore as ore characteristics change. The system requires frequent manual calibration to check that screw output at a given revolutions per minute is always known.

Barren solution is sprayed onto the ore in the drum through a system of spray nozzles mounted on the scraper bar in the drum. Solution addition rate is controlled manually although a solenoid valve shuts off flow to the drum if either power or ore feed is lost. A flow meter on the solution addition line is used to aid in operating control as well as for metallurgical balance purposes.

Drum product is conveyed to the heaps using a series of semi-permanent transfer conveyors and 18-meter long portable conveyors feeding a radial stacker. Because of the fragile nature of the agglomerated ore pellets, conveyor stacking was selected to reduce abusive handling of the pellets and to eliminate the need for vehicular traffic on top of the heap. In, fact, the additional rolling and mixing at transfer points appear to enhance pellet quality.

The portable conveyor units run down the center of the module under construction and retreat uphill as the material is stacked. As many as nine portable units are used when beginning construction of a new module. As construction progresses, the units are removed from the string.

The portable conveyors and stacker were all designed and constructed specifically for the Exhibition project based on equipment presently in use at heap leach projects in the US. However, several improvements were incorporated in the designs which greatly reduce the amount of time required to rearrange the conveyors after each move.

The radial stacker is 30 meters long with a 5-meter “stinger” conveyor mounted on the end. The stinger can extend or retract under control of the stacker operator and significantly reduces the number of conveyor relocations. The stacker can stack to a maximum height of 10 meters and is relocated after a 4.7 meter thick arc of pellets has been stacked.

The stacker is self-power to travel radially or in towing position. A standard truck bogie is used on the fed end and a towing bar attached to the bogie is used to steer the stacker feed hopper underneath the discharge chute of the portable conveyors feeding it. The individual portable conveyors are connected with tow bars and the entire string is lulled upslope simultaneously using a loader or dozer. Relocation time requirements are about 30 minutes.

Due to the nature of the stacking operation, the top surface of the heap is composed of a series of ridges and valleys. No effort is made to flatten or “grade” the top; however, a competent stacker operator can construct a surprisingly flat surface.

INITIAL OPERATING CONDITIONS

At the start of the operations, the ore was crushed to minus 25 millimeters and agglomerated with 2.2 percent cement and a final moisture content of about 21 percent. Cement dosages were approximately 10 percent higher than determined to be required in laboratory tests. The higher dosage was used as insurance to cover variations in feed auger output. Due to the reduction in clay content of the ore, the cement dosages were later scaled back in stages to the present 1.5 percent addition rate.

Initial stacking height was meant to be 6 meters, with an increase in stack height to 8 meters and ultimately 10 meters, if the behavior of the ore was acceptable. However, the absence of a tape measure during initial construction resulted in an 8-meter stack height for the first several thousand tonnes.

Sprinkler piping was placed on the heap as it was constructed and leaching initiated as soon as sufficient leach surface was available. Behavior of the initial section of the first module was quite acceptable and the stack height was increased incrementally from 8 meter to 9.5 meters then to the present 10 meters.

Since the pellets are quite fragile, shade cloth material was placed on top of the heaps before the leach piping was installed. This material protects the pellets from the full force of the sprinkler droplets falling onto the surface. No ponding has been observed, thus indicating that both pellet quality and pellet protection are adequate.

Sprinklers were installed on a 6 meter square grid pattern. Senninger No. 8 Wobbler sprinklers were used initially but other sizes and grid patterns have been evaluated over the year.

When sufficient pregnant solution became available (about 48 hours after start of leaching), flow through the first carbon column was initiated. As more sprinklers were placed on line and flow picked up, solution flow rate and the number of carbon columns in the adsorption circuit were increased. The first carbon column was fully loaded within 20 days of initiating sprinkling.

PERFORMANCE OF FIRST MODULE

The first ore module contained 80,500 dry tonnes of ore. Average grade of the material based on daily crusher grab sample assays was 1.61 grams per tonne (total combined gold = 129,600 grams). Based on solution assays, gold recovered from the module was 117,000 grams for an indicated recovery of 90.3 percent.

At the end of the leaching and washing cycles (138 days total), an auger was used to obtain tailings samples of the module. From a total of 10 samples, the average grade was a 0.16 grams per tonne (range – nil to 0.34 grams per tonne). This resulted in a final recovery of 90 percent.

Performance of this magnitude from a heap leach is unusual, with heap leach recoveries generally in the 65 to 80 percent range. The performance of recent Exhibition modules has yielded indicated recoveries ranging from 75 to 85 percent, and the operators estimate that the average recovery since startup is running at about 80 percent. The drop in recovery is due to the changes in the ore feed characteristics (degree of oxidation) with depth in the pit.

Pregnant solution grade from the first module peaked at 2.99 ppm Au during the first two weeks of operating. Average off-flow solution grade over the leach period was 0.48 ppm. Average leach solution grade during this period was 0.05 ppm Au.

CAPITAL & OPERTATING COSTS

Approximate capital costs for complete installation of the project are given as follows:

Table 1 - Installation Capital Costs (Approximate)

INSTALLATION TASKS	DOLLARS
Crushing/Agglomeration/Stacking Equipment (Contractor Supplied)	$2,000,000
Site Preparation & Plan	$1,000,000
Water Supply Installation	$700,000
TOTAL INSTALLED COSTS	**$3,700,000**

Heap leach operating costs (excluding mining) per dry tonne of ore processed for the initial 12 months of operation are given in Table 2.

These operating Costs are high by industry heap leaching standards. The contractor fee is relatively high and reflects the capital payback charges for crushing, agglomeration, and stacking equipment. Reagent costs are also high as might be expected. Mining costs were not available at the time of writing. However, operations that augment milling operations with heap leaching of lower grade material assign no mining costs to the heap leach feed since the material had to be mind and stockpiled in any case.

Table 2 - Heap Leach Operating Costs*

OPERATIONS	US$ PER TONNE
Crush/Agglomerate/Stack (Contract)	$4.70
Reagent Costs	$3.18
Labor Costs	$0.36
Power Costs	$0.10
Pad Costs	$2.25
General/Administrative	-
Stripping Costs	$0.17
TOTAL INSTALLED COSTS	**$10.76**

**excluding mining*

PRESENT STATUS

As of 23 March 1988, the Exhibition heap Leach has processed 571,700 dry tonnes of ore. Average grade over recent periods has fallen to 1.3 grams per tonne while recoveries average 80 percent. As of this writing, production costs for the entire operation are running about $300.00 per recovered ounce.

SUMMARY

The Exhibition Heap Leach Project is a prime example of the technical advances in heap leach processing over the past several years. The extremely high clay content and attendant characteristics of the ore would have eliminated heap leaching as a processing option before the development of agglomeration techniques. The agglomerating and stacking systems as well as the carbon adsorption recovery columns represent state-of-the-art equipment design and have contributed greatly to the technical success of the operation.

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