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NEWMONT GOLD'S NO. 2 MILL

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Carlin, Nevada

Newmont Gold Company, formerly Carlin Gold Mining Company, is a publicly owned corporation, 95% owned by Newmont Mining Corporation. Newmont Gold Company has operated gold recovery operations North of Carlin, Nevada, since 1965. Development of the Gold Quarry ore body, 19 kilometers south of the original mine, resulted in the decision to construct gold recovery and ancillary facilities in what is referred to as the Southern Area.

Plant No. 2, located 11 kilometers North of Carlin, Nevada, was designed to treat 6350 tonnes per day of oxidized gold bearing ore assaying 2.5 grams per tonne of Au at an average recovery of 87%. Flowsheet development conducted by Newmont personnel resulted in the decision to construct a carbon-in-pulp recovery circuit preceded by two-stage grinding. Detailed engineering was done at Toronto by Bechtel of Canada beginning on January 1, 1984. Actual construction began on March 1, 1984, and was completed in June, 1985. First feed was run on June 22, 1985.

Plant No. 2 reached its design throughput on a monthly basis in October, 1985. However, several design problems became evident as the plant was operated. All of these will be discussed later. Current throughput rates are 7710 tonnes per day, and an expansion to 8164 tonnes per day is anticipated during 1987.

The Plant No. 2 flowsheet, equipment, and the design problems mentioned previously are discussed by unit operation. A flowsheet is presented in Figures I and II.

Crushing

Ore is received from the mine and dumped into coarse ore stockpiles sorted by hardness and by grade. The ore is then fed with a 992C loader through a stationary grizzly with 0.61 m square openings into a 250 tonne feed hopper. Oversized material is broken with a Tramac hydraulic rock breaker.

A 1.52 m wide hydraulically driven pan feeder discharges ore from the feed bin to a Simplicity 1.52 m vibrating grizzly with 15.2 cm slotted openings. Oversize material is broken in a 1.06 m x 1.22 m Traylor jaw crusher set at 15.2 cm, while undersize is discharged onto a conveyor belt by a 1.22 m hydraulically driven pan feeder. Crushing plant product is then conveyed on 1.22 m conveyors to a 1.22 m stacking conveyor which discharges into a 10200 live tonne crushed ore stockpile. The primary crushing facility is designed to operate on a two shift/day, seven day/week basis, with one shift reserved for maintenance. Nominal tonnage rate is 810 tonnes per hour.

Crushed Ore Reclaim and Grinding

Testwork on Gold Quarry ore performed at the Mineral Processing Systems Inc. pilot plant in York, Pennsylvania, indicated that either autogenous or semi-autogenous primary grinding would suffice at Plant No. 2. Autogenous grinding was selected on the basis of lower maintenance costs and fewer operating problems. However, in order to reduce the problems caused by critical size material in the primary mill, it was decided to use an autogenous mill/crushing circuit rather than purely autogenous grinding.

Two hydraulically driven variable speed pan feeders discharge crushed ore from the stockpile onto the 1.07 m mill feed conveyor. Minus 9.5 mm pebble lime is fed directly to the mill feed conveyor from a 129 tonne lime storage bin by a variable speed screw conveyor at a nominal rate of 1.6 kg/tonne. Primary grinding is done in an 8.54 m diameter by 3.05 m M.P.S.I. mill equipped with a 2610 kw wound rotor motor. The primary mill discharges through 6.35 cm slotted grates to a 2.44 m x 6.1 m vibrating screen equipped with 9.5 mm x 25.4 mm slotted polyurethane panels. Screen oversize is crushed in a 2.13 m Symons shorthead cone crusher operated at a closed-side setting of 9.5 mm. Crusher product is recirculated to the primary mill.

Screen undersize is pumped by a Warman 25.4 cm x 30.5 cm steel lined pump to the cyclone feed sump. Cyclone feed is pumped by a Warman 35.6 cm x 30.5 cm steel lined pump to a bank of seven 66 cm Krebs hydrocyclones, four of which operate under normal conditions. Cyclone underflow is split in half and is fed to two 4 m x 5.5 m Marcy overflow ball mills operating in parallel. These mills operate with a 45% charge of 7.62 cm balls and are equipped with clutch driven 1305 kw motors. Ball mill discharge flows by gravity to the cyclone feed sump.

Cyclone overflow is discharged to four 1.5 m x 4.3 m Kinergy trash screens equipped with 28 mesh polyurethane screen panels. Screen oversize is returned to the secondary grinding circuit, while undersize is pumped to two 12 m x 12 m surge tanks.

Grinding circuit product averages 58% minus 200 mesh and, average power draw is approximately 15 KWH per ton.

The major design problem in the grinding area was screen sizing. The grinding circuit trash screens were, as designed, of insufficient capacity to handle design flows. The factors contributing to the shortfall were a lack of appreciation for the highly viscous nature of the slurries and a lack of experience in calculating area requirements for the polyurethane screen panels. These problems are being addressed by the installation of a linear screen to gain additional capacity.

The plant was designed to accomodate conversion to semi-autogenous grinding. Although the plant met design throughput rates as built, a decision to expand to 8164 tonnes per day resulted in the decision to convert to semi-autogenous grinding. In order to permit continued operation of the cone crusher, a magnetic head pulley was installed on the screen oversize conveyor and two belt magnets were positioned above the cone crusher feed conveyor to remove balls from the crusher feed. Also, ball charging facilities were installed. This conversion has proven successful and has resulted in significantly increased throughput. Currently, the mill is being operated with a 7% ball charge.

Thickening, Leaching, and Carbon-in-Pulp

Grinding in cyanide containing solution results in dissolution of approximately 60% of the contained gold in the grinding circuit.

Discharge from the grinding area surge tanks is pumped to a 25.9 m Enviroclear high-rate thickener. Mass flow is measured prior to the thickener to provide a mill feed tonnage figure for metallurgical accounting and the slurry is sampled for assay of solids and solution.

Thickener overflow solution is pumped to a series of five carbon adsorption columns operated in a cascade system. Each column contains approximately two tonnes of activated carbon. Carbon is transferred countercurrently to solution flow by Sala recessed impeller vertical pumps. Loaded carbon is screened on a 20 mesh Tyler 0.61 m x 1.8 m vibrating screen and stored in a loaded carbon storage bin for further treatment. A 1.2 m x 2.4 m vibrating screen following the last carbon column removes any carbon which might overflow the carbon columns. Barren solution from the carbon adsorption circuit is pumped to the mill solution tank for reuse.

Thickener underflow at approximately 45% solids is pumped to a six stage agitated leach circuit. The leach tanks are 11 m x 11 m and provide 12 hours of residence time. Air is sparged into the tanks to provide oxidation.

Leach circuit discharge is pumped to a six stage carbon-in-pulp circuit. This circuit consists of six 9.75 m x 10.4 m tanks, each equipped with 10 kw agitators with dual A-310 high efficiency impellers. Interstage screening is provided by modified E.P.A.C. launder screens with air lifts to the launders. Figure III shows the screen arrangement. Screening is at 20 mesh, and six screen panels in each tank provide a unit flow rate through the screens of 60 $m^3/hr/m^2$ of screen area. The screen surfaces are swept on both sides by 14.4 m^3/min (per tank) of 1.1 kg/cm^2 air produced by 190 kw Hoffman blowers. The air is introduced through 25.4 mm vertical pipes, 3 per screen side, terminating at the screen base. Air on the upstream side cleans carbon particles from the screen surface, while the air on the downstream side produces a lifting effect to aid slurry flow into the launders. This air lift effect and the use of submerged launders reduces the drop between C.I.P. stages.

Total C.I.P. residence time is 8 hours. Average screen life using the standard square opening screen cloth has been approximately three weeks. Recent testwork using wedge-wire screens indicates significant improvement in screen life may be achievable.

Carbon concentrations are maintained at 10 g/l of 6 x 16 mesh high activity carbon. Carbon transfer is done on a continuous basis using vertical recessed impeller pumps. The loaded carbon from C.I.P. is pumped over a 0.61 m x 1.8 m Tyler vibrating screen equipped with a 20 mesh wire deck. Slurry is returned to No. 1 C.I.P. tank and carbon is discharged to the loaded carbon storage bin. Stripped, reactivated carbon is continuously added to No. 6 C.I.P. tank using a variable speed screw feeder and a horizontal recessed impeller pump.

Typical C.I.P. carbon and gold profiles are presented in Table I below.

Tank No.	1	2	3	4	5	6
Carbon Contained (Tonnes)	6.6	6.6	6.6	6.6	6.6	6.6
Au Conc kg/T	154 oz/ton 5.3	4.4	3.5	2.6	1.8	0.9

C.I.P. CIRCUIT
CARBON CONCENTRATION AND LOADING PROFILES

Tailings

Carbon-in-pulp circuit tailings flow across two 1.5 m x 4.9 m horizontal vibrating screens operating in parallel. These screens, equipped with 28 mesh polyurethane decks, remove any activated carbon which may have passed through the interstage screens. Screen undersize is sampled using a two stage automatic sampler and pumped to the tails pond.

The original design included only one carbon safety screen. A second unit was installed immediately after start-up to provide additional capacity.

Tailings pumping is accomplished using two trains of 25.4 cm x 25.4 cm rubberlined horizontal slurry pumps. In each train, the second stage is a variable speed unit with pump speed controlled by sump level. Each train pumps tailings through a 30.5 cm high density polyethylene pipeline to the tails pond.

The Plant No. 2 tailings dam is a 61 m high, 600 m long, 36 m wide (at the crest) structure constructed across the James

Creek basin. Locally available material, along with pre-mine strip, were used to build an impervious structure equipped with an internal wick designed to maintain a low phreatic line and to prevent uncontrolled discharge. Estimated capacity of the impoundment basin is 60 million tonnes.

In order to reduce fresh water consumption and to maintain low solution levels in the tailings pond, solution is decanted from the pond and reused in the milling circuit. Barge mounted vertical pumps return the solution to the plant.

Approximately 60% of the solution sent to the tailings pond is returned to the plant for reuse. Fresh water for plant makeup and for use in ancillary facilities is supplied by four wells located within 2.4 km of the plant.

The original tails system design had two pump/line systems. Each was to be capable of carrying the design flows. While the design was done properly, the systems were designed with a very small safety allowance. The inability of the operators to control pulp density at exactly the design level necessitated that both lines be operated. While this situation has, at times, caused problems, the cost of a new line and pumps is, at present, not justified.

Precious Metal Recovery

Loaded carbon from C.I.P. and from the carbon adsorption circuit is transferred in four ton batches from the loaded carbon storage bin to a 1.5 m x 5.8 m fiberglass acid wash vessel for removal of calcium carbonate scale. Two percent hydrochloric acid solution is pumped through the carbon bed at a rate of two bed volumes per hour, or 0.3 m^3/min. Washing continues until a pH of less than 2.0 can be maintained in the discharge solution. At this point, the tank is drained and a 1% sodium hydroxide solution is pumped through the carbon bed to neutralize any residual acid.

After neutralization, the carbon is water washed and transferred to one of two 1.5 m x 6.1 m stainless steel strip vessels. Strip solution containing 0.3% NaCN and 2% NaOH is pumped upflow through the carbon bed at a flow of 0.17 m^3/min, a temperature of 135°C, and a pressure of 2.8 kg/cm^2. Two single pass plate and frame heat exchangers with pregnant solution and 141°C boiler water, respectively, on their hot sides are used to bring the strip solution to its operating temperature. Cooled pregnant eluate at 85°C and containing approximately 93 g/T Au is split and enters three of four 3.59 m^3 electrowinning cells. Each cell is equipped with eighteen 0.91 m x 0.91 m p.v.c. cathode baskets and eighteen stainless steel anodes. Each cathode basket contains 1.4 to 1.8 kg pounds of steel wool. The cells are operated at 3.2 volts D.C. and 280 amperes.

Stripping time averages approximately 20 hours and stripped carbon contains an average of 156 g/T Au. The circuit was designed for a strip cycle of eight hours. This rate has never been achieved. It is believed that the actual capacity of the circuit is approximately 35% of design.

Stripped carbon is dried and reactivated in a 225 kg/hr diesel fired vertical regeneration kiln at approximately 677°C. Regenerated carbon is water quenched, screened at 20 mesh on a 0.6 m x 1.8 m vibrating screen to remove carbon fines, and stored until required in the adsorption circuits. New carbon is added to the circuit as required following 8 hours of attrition, conditioning, and screening to remove fines. Carbon use to date has averaged 36 g/T of ore treated.

Research to develop a solution to the strip problem is underway. Two major items have been identified as likely causes:

1. The height to diameter ratio of the strip vessels is less than five to one. It is likely that this low ratio along with the low flow rate in the vessel is resulting in channelling in the carbon bed.

   Also, superficial velocity in the strip vessel is very low.

2. Scale is forming throughout the strip circuit piping causing further flow reductions and high system operating pressures.

   The presence of silica in the strip solutions is contributing to the scale formation.

Refining

Loaded steel wool is removed from the electrowinning cells and acid digested in a sulfuric acid solution to remove base metals. The acid digestion sludge is then retorted to remove mercury, fluxed, and melted in a 225 kg per hour induction furnace. The resultant product is a Dore bullion containing 84% gold, 13% silver, and 3% impurities.

Operating Control

A Fisher Provox system provides process control, while a Gould Modicon programmable controller, which interfaces with the Provox system, provides control of discrete functions such as interlock, time delays, and start-stop control. The Provox system provides automatic control of all process variables in the ore reclaim, grinding, thickening, leaching, C.I.P., carbon handling, and tailings areas of the plant. More than 20 graphic displays are available on two C.R.T.'s. Additionally, operating reports, alarm reports, and graphic printouts are available.

As previously noted, Plant No. 2 was started-up on June 22, 1985. Design throughput of 6350 tonnes per calendar day at 88% availability, or 7257 tonnes per operating day, was reached during the fourth month of operation. Average throughput through 1986 has been 7067 tonnes per calendar day at 89.2% availability, or 7923 tonnes per operating day.

Newmont Gold's Plant No. 2 produced its first Dore bar on July 24, 1985. To date, some 6488 kilograms Au have been produced, and the facility is producing at rates well above design.

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