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# PEBBLE GRINDING AT BETHLEHEM COPPER CORPORATION LTD.

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## Introduction

The operations of Bethlehem Copper are situated in the south-central interior of British Columbia at an elevation of 4900 feet above sea level. Main road and rail transportation systems pass through Ashcroft in the Thompson Valley, which is 28 miles by road from the minesite. The operation was the first low-grade open-pit porphyry copper mine in British Columbia.

The minerals are chiefly chalcopyrite and bornite, while the rock is principally feldspar and quartz. In the outer fringes of the orebodies some pyrite is present. Average grade of mill feed is 0.55% copper.

## History of Milling Operations

Production commenced in December 1962 at a rate of 3,000 tpd in a plant that comprised two-stage crushing followed by rod and ball milling. Production ratings were increased to 6,000, 10,000 and 12,000 tpd by the addition of a further stage of crushing and more rod and ball mills. When the last expansion was completed tonnage throughput exceeded the 12,000-ton designed rate and was stabilized at 15,500 tpd.

However, recovery did not meet expectations at the expanded rate because of coarse flotation feed. Capacity in the rod mill circuits exceeded the subsequent fine grinding capacity, resulting in poor liberation. Studies were initiated to improve recovery without decreasing throughput.

## Description of Operation Prior to Pebble Milling

Crushing with primary, secondary and a closed tertiary stage proceeds at a scheduled 20.5 shifts per week. Fine ore screened on 1/2" x 4" apertures is stored in a 30,000-ton bin ahead of two 12.5' x 15' rod mills. Rod mill discharge joins secondary ball mill (one 10.5' x 14' and one 11' x 14') discharge for classification in a single 50" cyclone which closes the secondary mills. Overflow from this cyclone is split between three third-stage ball mill-pump-cyclone circuits (two 11' x 14' and one 12.5' x 15'). Prior to addition of the two pebble mills the flotation feed was produced by this three-stage grinding system. Grinding criteria are shown in Table I.

Feasibility of Economic Improvement in Recovery

Laboratory grinding and flotation tests on plant flotation pulps indicated that tailing grades could be lowered to yield an additional 0.44 pounds of copper per ton by reducing the 80% passing size from 210 microns to 102 microns. Our calculations showed that the additional work input required to attain this improved liberation using ball mill grinding, while resulting in a profit, showed a return on investment that was unattractive.

A cheaper method of grinding was necessary and information was collected to assess the feasibility of pebble grinding. This study showed a higher capital cost, lower operating cost, and substantially increased revenue from copper contained in pebbles consumed. The calculated return on investment was attractive and arrangements were made to pilot the operation at the facility of a mill manufacturer to assure the reliability of the figures being utilized.

Samples of rougher tailing and primary crushed ore in the amount of 25 and 75 tons respectively were prepared for the grinding test. Pebbles sized between 3" x 3" and 1-3/4" x 1-3/4" were prepared from the primary crushed ore. The test was performed in a 4' x 8' grate discharge mill closed with a pump and cyclone. In one part of the test work pebble chips were removed as discharged from the mill. In the remainder of the work the chips were recycled to the feed of the mill along with cyclone overflow. New feed was introduced directly to the mill instead of to the pump sump. This was an expedient divergence from intended practice.

The pilot plant work confirmed the laboratory results and a decision was made to install the necessary equipment in April of 1971. The work was fully completed by June 1972. Budgeted cost was $2.7 million and actual cost was $100,000 less.

The calculated recovery of investment indicated that 60% would come from milling an additional 1,000 tpd of ore (500 tons of pebbles and 500 tons of rod mill feed) and 40% from improved copper recovery from tailings. The payback period was reduced to well below the three years used in the justification when actual throughput increased by 2,000 tpd and the recovery increase reached as high as 0.6 or more pounds per ton of ore.

The Installation

Pebbles are prepared from primary crushed ore on an 8' x 16', 2-1/3 deck primary screen. The upper deck cover consists of cast manganese grids with 1-3/4" x 1-3/4" openings for the first 12 feet, increasing to 3" x 3" openings in the last 4 feet. Material passing these larger openings drops to the 1/3 deck covered with grids the same as the first 12 feet of top deck so that pebbles are sized between 3" x 3" and 1-3/4" x 1-3/4". A motorized sliding gate diverts pebble production in excess of requirements to the secondary crusher.

Pebbles are conveyed on 18" conveyors to a 1,000-live-ton cylindrical storage bin. Load level is measured sonically and the signal transmitted to the crusher control room for readout. Pebbles are fed from this bin by vibrating feeders to 18" conveyors which in turn feed the two pebble mills.

The two 16.5' x 32' grate discharge pebble mills are driven by low-speed synchronous motors through air clutches and single helical gearing at 69% of critical speed. Babbitt bearings are hydrodynamically lubricated and the mill journals are lifted by manual pumps for starting.

The pulp discharges through grates with 1/2" x 1-3/16" slotted openings to three 16" x 16" pumps driven by 300 hp motors. Two pumps operate while the third serves as standby and is linked through ball valves for automatic switching of pumps. Pump lines discharge to D20B cyclopacs with underflow returning to the pebble mills and overflow going to rougher flotation.

Liners

In the original installation No. 8 pebble mill was equipped with rubber lining and grates, while No. 9 pebble mill had Ni-Hard throughout. Figure 1A illustrates the rubber liners installed in No. 8 pebble mill shell, and Figure 1B the present configuration using a modified shell lifter replacing initial lifters which lasted ten months. Figure 2 shows the Ni-Hard lining installed in No. 9 pebble mill shell and still in use. Removing and weighing clamping bars and plates indicates only a 7% weight loss in one year.

Rubber head liners in No. 8 pebble mill required a replacement of lifters after one year of operation. The 1" lift wore off the Ni-Hard head liners in No. 9 pebble mill in eight to nine months and was corrected by bolting on separate castings 3" in height.

Grates

No. 8 pebble mill started with rubber grates and these have given no problems. The initial Ni-Hard grates in No. 9 pebble mill were subject to considerable blinding. This problem rapidly became worse as cleaning operations spalled the inner face of the grate, making chips wedge faster than ever. The slot size was 3/8", with a 2-1/2 degree taper.

A Cr-Mo grate was installed with double the taper and although results were better they were not satisfactory. Lower power draw and spillage out the feed end of the mill led to a decision to install a duplicate rubber grate in No. 9 pebble mill as quickly as possible.

Operation

Cyclone overflows from the third-stage ball mills are joined and split between the operating mill pump boxes. Normally, four or five cyclones are operating per mill with one of the five on each cyclopac started and stopped by pneumatic valves on signals received from the pump sump level. In this way the sump level is controlled over a 2' range. A preferable method of sump level control would involve automated water addition, but reclaim water capacity and constraints on flotation dilution dictate otherwise at present.

Cyclone feed pressure ranges from 15 to 20 psi. Overflow density range is 34% to 39% solids, and underflow density range is 68% to 72% solids. The spigot product is diluted with water to 65% solids before it enters the pebble mill.

The mill power draw is recorded in megawatts on a strip chart which guides the operator on pebble addition. Pebble conveyors run continuously while the vibrating feeders operate on a timed interval. Control of the times and the feed vibration intensity is left to the operator, who makes alterations to feed rate to maintain power draught at maximum and pebble load as close to 50% as possible. Grinding criteria for the pebble mill circuit are shown in Table II.

Modifications to Circuit

From the beginning a major point of concern has been to get maximum power draw from the pebble mills. Motors are 2,750 hp at 0.8 leading power factor with a 1.15 service factor. A considerable reserve exists in the motors. If it could be drawn it could be used for higher throughput, improved grind, or reducing ball mill horsepower and ball consumption, or any combination of the three.

Mill specifications called for full motor power draw at 69% of critical speed, 65% pulp solids, 40% pebble charge, and a throughput up to 1,000 tons per hour per mill. According to calculations, power drawn could reach 3,000 hp. The 40% pebble charge was selected as a point on the load power curve with sufficient slope to make control of pebble load simple. A speed of 69% of critical was recommended as a reasonable level for ensuring pebble survival, and 65% solids a good figure to allow coating of pebbles and efficient grinding conditions. Throughput of 1,000 tons per hour allowed for 340 tons of new feed and 660 tons circulating load.

Initial operation required six to seven cyclones operating at pressures of 25 to 30 psi and power draws of about 2,000 hp. Rates of wear of 4-1/2" vortex finders and 3-1/2" apex valves were excessive, with lives of only 2 to 3 weeks. To improve power draw the pebble loads were boosted to 50% and have been kept there since.

It was noted that as the mills pumped themselves out during shutdowns, the power draw increased significantly. It was decided to reduce circulating load by the combination of increasing vortex finder size to 6" and decreasing pump speed to lower cyclone pressure to 15 to 20 psi. Apex valve size remained the same. These changes brought power draw up to approximately 2,750 hp with lower pulp levels in the mills and cyclone wear reduced to reasonable levels.

Subsequent discussion on ways to improve performance led to the suggestion that the space at the periphery between the grate and the pan lifters was insufficient to allow free flow of pulp to the discharge cone. To check this, the complete grate assembly in one mill was shimmed out into the mill three inches, doubling the space in the area considered to be most important for low pulp level operation. Power consumption increased less than 2%. A reserve of about 10% remains to be utilized.

Effect of Changing Pebble Top Size

The top size of pebbles was increased by replacing the 3" x 3" openings with 4" x 4". Work index values were calculated at both pebble sizes and showed an increase from 18.5 to 20.1 at the larger size. Further test work using a top size of 2-1/2" x 2-1/2" is currently being evaluated.

Operating Cost

The year-to-date cost for pebble mill operation is about 6¢ per ton. A considerably longer period of time must elapse before meaningful figures reflecting liner replacement will become available.

TABLE I THREE-STAGE ROD MILL-BALL MILL GRINDING

| | KWH/T | K80 In (Microns) | K80 Out (Microns) | Wi * |
|---|---|---|---|---|
| Rod Mills | 3.29 | 9920 | 1800 | 24.5 |
| First Ball Mills | 2.57 | 1800 | 897 | 24.3 |
| Second Ball Mills | 4.69 | 897 | 210 | 13.3 |
| Total Circuit | 10.55 | 9920 | 210 | 17.6 |

* Wi values have not been corrected for mill diameter.

TABLE II QUATERNARY STAGE PEBBLE MILL GRINDING

| Mesh | Feed | Product |
|---|---|---|
| +48 | 14.2 | - |
| +65 | 12.6 | 3.7 |
| +100 | 10.3 | 7.8 |
| +150 | 10.1 | 12.8 |
| +200 | 8.3 | 11.3 |
| +325 | 9.3 | 13.7 |
| -325 | 35.2 | 50.7 |
| KWH/T, Total product | | 5.82 |
| -200 mesh produced | | 29.4 |
| K80, Microns | 254 | 118 |
| Bond Wi, not corrected for mill diameter | | 20.1 |