MINERAL PROCESSING

016814

The Baker Mill start-up and early operation

W.W. MUIR
Mill Superintendent, Baker Mine
Du Pont of Canada Exploration Limited

94E/6E
(094E 026)
Chappelle

ABSTRACT

The Baker Mine used a 100-ton per day conventional cyanide leach mill located in northern British Columbia. A production decision was made in early 1980 and the mill was on line in April 1981.

Ore reserves had been estimated at 100,000 tons averaging 0.9 oz gold per ton and 18 oz silver per ton. Mineralization consists of electrum (Au/Ag) and acanthite (Ag_2S).

This paper will review plant design and focus on some problems associated with start-up and early operation with the intention of assisting other mine operators by indicating potential problem areas.

Introduction

This paper will discuss operating strategies employed at Baker which improved over-all mill performance. Also included are some ideas which could be steps toward a more efficient operation in a similar setting.

Actual dore production by mine life was 37,606 oz gold and 742,117 oz silver.

Figure 1 illustrates the relative stability attained over the mine life. Metal recoveries for gold and silver were 92% and 85% respectively.

Safety at Baker was of prime importance because of its remote location. Baker received the Mine Safety Association "Small Mines Award" in two of its three operating years.

At shutdown in late 1983 the mill had operated for over 850 days without a lost-time accident.

W.W. Muir

W.W. Muir was born and educated in British Columbia and graduated in chemical and metallurgical technology from B.C. Institute of Technology in 1970.

From 1971 to 1980 he worked at Cyprus Anvil Mining Corporation progressing from technician to plant metallurgist. In 1981 he joined DuPont of Canada Exploration Ltd., Baker Mines operations where he was involved with mill start-up and assumed mill superintendent's role in late 1981. In June 1984 he joined Giant Yellowknife Mines Limited, Salmita Division as mill superintendent, and in September 1984 he was appointed operations superintendent.

Bill is a member of CIM and Society of Engineering Technologists of B.C.

Keywords: Mineral processing, Plant design, Mill performance, Crushing, Grinding, Thickeners, Leaching, Filtration, Baker Mine.

Mill Circuit — General

The mill circuit illustrated in Figure 2 is basic in almost every sense.

Crushing

Ore trucked from underground and open-pit workings was dumped on a 12-inch slot grizzly over a 40-ton coarse ore bin. The bin was atop a reciprocating plate feeder for a 15 in. by 24 in. jaw crusher. The jaw crusher discharge was fed direct to a 3 ft standard head cone crusher set at ⅜ in. to ⅝ in. depending on moisture and fines content of the ore. Cone crusher discharge was fed directly to a 120-ton fine ore bin.

Prior to start-up, the design intention was not to supply heat to the crusher house; consequently, the building was not insulated and all around the coarse ore bin (COB) was wide open. Subsequently, it was thought worthwhile to heat the building for protection of equipment and operator comfort. The opening around the COB was enclosed, waste heat from a generator was ducted to the building and an oil heater was added for support. The two-year average for total precipitation was 32 in./yr, 14 ft of it in snowfall. Temperatures dipped to -40°C at the site.

Oversize rock on the grizzly was a problem. The surface crew initially used sledge hammers to break oversize. This was unsafe, labourious and very inefficient in terms of manhours and idle equipment while the grid was being cleared off. A mobile unit with a rock breaker attachment on a backhoe boom was rented and this was a genuine improvement. The machine with multiple attachments and a quick lock hydraulic coupler arrangement was used as a backhoe for exploration, trenching in the immediate mine site area, and as a forklift to handle pallets of reagents.

Moisture in the run of mine ore fed to the coarse ore bin ranged from 3% to 7%. This led to "hang ups" in the COB and a plate vibrator was installed on the sloping COB wall.

The reciprocating plate feeder at the foot of the COB yielded to the load and started to warp and crack. A conveyor return idler was positioned under the plate feeder to give much needed support.

Paper reviewed and approved for publication by the Canadian Mineral Processors Division of CIM.

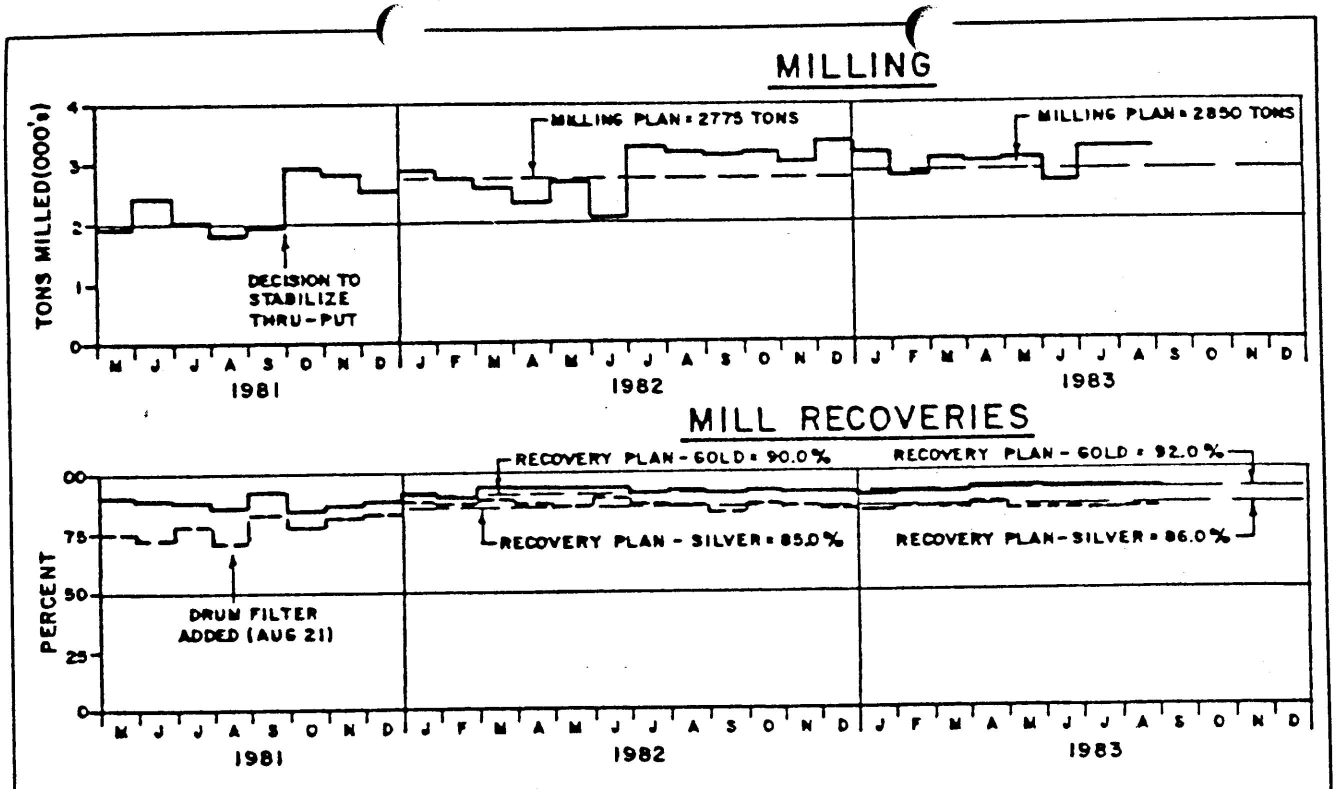

FIGURE 1. Relative performance stability attained over the mine life at Baker Mine.

As mentioned, there was no screen between the jaw and cone crusher. Cone crusher plugging was experienced; however, it was thought that it was more likely due to the flat rock box discharge chute beneath the cone causing poor flow characteristics beneath the cone. The rock box chute was replaced with a sloping sides chute. Wear plates were installed as well.

Cone crusher plug-ups were also experienced during winter when ore would slide back down partially ice-glazed conveyor feeding the fine ore bin. This filled the small pocket beneath the cone (and subsequently the cone). A tilt switch was put in the exiting skirt/dust control framework to warn of such incidents, no further plug-ups occurred.

Current crushing facilities have now been optimized to the point where crushing time is down to 2 to 2.5 hours per day, 50% of start-up experience.

Grinding

Grinding was accomplished with a 7 ft diameter by 10 ft long overflow ball mill; grinding media was 4 in. steel balls. Classification was done with a D10B cyclone. Initial grind targets of 85% -325 mesh were not attainable with a single stage grind. Filtration problems led to development of a coarser grind currently at 60% to 65% -325 mesh with little impact on dissolution of gold and silver values.

Missing from the grinding bay was an overhead crane. The best alternative for charging the mill was to use a hiab, a truck-mounted crane. Forty-five gallon drums containing one ton charges were placed in a frame and lifted above one of the ball mill doors. A makeshift, wooden chute was made to fit the door. With the 45 gallon drums at a 10- to 20-degree angle, the drum lids were cut off and the charge allowed to roll into the chute.

Consistent with all northern operations, fine ore bins presented freezing problems. The original slot feeders with chutes down to the conveyor belt and curved plates controlling feed were inappropriate. They were replaced with a feeder designed with an opening at the front to permit access with a pry bar and in the case of a hollowed-out fine ore bin, propane heat could be used to bring down severe hang-ups. Extreme caution was taken when using propane as when large amounts of ore would fall, the air gust would sometimes put out the flame. Therefore, operators would be in attendance at all times. A simple tilt switch was placed on the feed conveyor to indicate an empty belt.

A contaminated lube system on the ball mill feed end was a most serious problem. The slurry/oil mixture acted like a grinding compound and the insert bearing actually had 3/16 in of wear. The insert bearing (800 lb) was replaced without use of an overhead crane, with some difficulty. The feed end trunnion and oil sump are protected now as a light gauge metal flange was welded to the trunnion, encircling the feed spout so slurry cannot get near the oil sump face. The gap between the trunnion and oil sump (approximately ¼ inch) was filled with a stiff grease. Trunnion position checks, using the oil sump as a reference, are made regularly. Oil samples are collected and sent out for analyses.

All underground ore comes complete with its share of wood and plastic. Two traps were built, one for the ball mill discharge trommel screen, and a second finer screen to catch smaller tramp material contained in the cyclone overflow.

Thickening

The cyclone overflow at 35% solids was fed to a 28 ft diameter by 10 ft deep thickener. The overflow was fed to pregnant liquor storage tank. The underflow at 55% to 60% solids was pumped to first-stage agitation.

The thickener underflow was originally pumped directly to the No. 1 agitator. This feed line was inaccessible. For convenience and flexibility an access floor and splitter box were installed atop the agitator. Operators were now able to:

a) Operate a control valve atop the agitator and actually see how the pump was operating.
b) Sample the thickener underflow controlling densities, and
c) Be able to recycle slurries back to the thickener without additional piping off the underflow pump.

The thickener underflow pump was an SRL centrifugal pump. Discharge to the first agitator was throttled with a pinch valve and received considerable attention. The merits of a diaphragm pump here, in terms of metering capability with

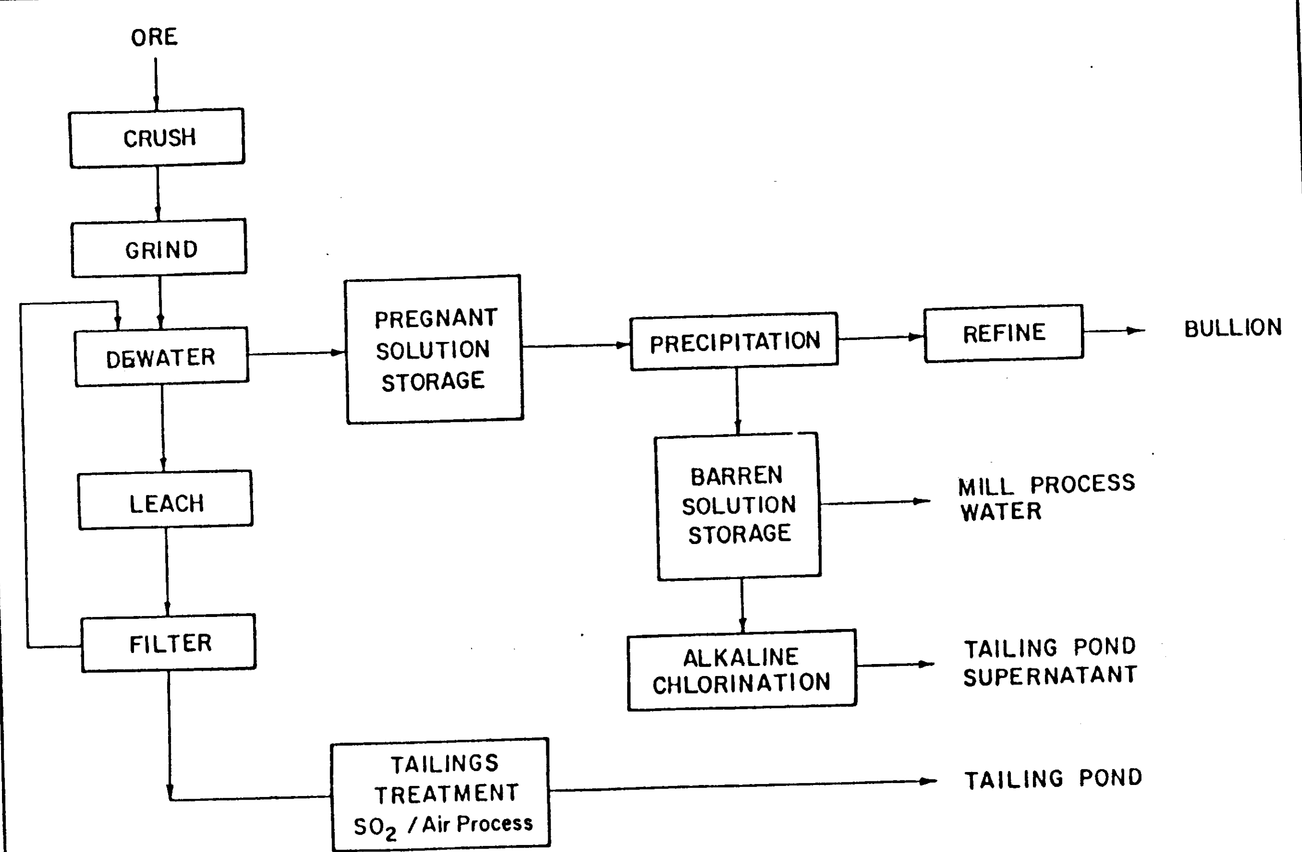

FIGURE 2. Baker Mine simplified flowsheet.

less dilution water, may have been an advantage in terms of consistent operation. Density control had a large impact on filter throughput. Dole valves were used on gland water lines to limit dilution.

Leaching

Leaching was carried out in four 16 ft by 16 ft dia. tanks in series. Low-pressure air was introduced beneath the bottom turbine. Retention time was a hypothetical 75 to 85 hours allowing for some solids build-up on the tank bottoms.

The Baker Mine ore was very amenable to cyanidation. Cyanidation lab tests were carried out at 25% solids. Thickener underflows of 55% to 60% are maintained with the existing design layout, yet dissolution curves remain practically the same. Dissolved oxygen content of in-plant slurries are 3.5 to 4.0 ppm; initial laboratory work did not indicate levels of dissolved oxygen in the cyanidation tests. The last concern of the laboratory should be to question "was the sample tested a legitimate indication of ore reserves?" It appears metallurgically, we were fortunate and a simple process remained simple. In terms of hydrometallurgical study, there are many variables to influence design. (Refer to reference paper[1] for the basics of plant design).

The agitators themselves are essentially a mixer. On lowering the tank levels, we have found as much as a two-foot thick wall of settled mud around the perimeter wall. In essence, useful volume was estimated at 70% of that installed reducing retention time from a hypothetical 75 to 85 hours down to 50 to 60 hours. Again, it was fortunate that the original testwork was only done over 60 hours. With this performance information on the agitators, would it be advantageous to consider the high-volume flotation machines as leaching equipment?

Cost comparison:

— existing leach tank complete \$20k/2000 ft^3
  = \$10/useable ft^3

— two 1275 flotation machines \$50k/2000 ft^3
  = \$25/useable ft^3

With the thick mud layer on the agitator sidewalls the draught tubes for incoming feed all proved useless as they were plugged off at the bottom. These draught tubes were cut from the intake side and placed on the outlet side, halfway down the agitator. They were kept clear with an air lift. This was done to minimize short-circuiting as was the original intention.

One small design feature with the agitators that proved bothersome was they were all at the same level. Flow through connecting piping was virtually flat and always had to be kept clean.

Filtration

The No. 4 leach tank slurry was pumped to a Dorr Oliver drum filter, added as an essential element in August 1981 to improve washing characteristics. Primary filter cake was repulped and pumped to a Denver horizontal belt filter. Barren solution was used as the displacement washing on both filters and the mid-repulping stage. The filtrate from both filters was fed back to the thickener where the rich solution overflows to the pregnant liquor storage tank. The secondary filter cake was repulped and pumped to the tailing pond.

The vacuum pump used a considerable amount of fresh water. A reclaim system was put in and water was re-used either as belt filter cloth wash spray and/or recycled to the vacuum pump.

The vacuum system had two distinct operating disadvantages. Filtrate lines to the receivers were all flat and plugged lines were experienced. This was happening when cloth tracking problems would arise with the belt filter, slurry being drawn into the receivers. The plugged lines were also linked with filtrate receiver pump performance. The root of the problem with the receiver pumps was not being able to observe filtrate pump discharge. Filtrate lines were routed directly to the foot of the thickener bridge and further piped under the

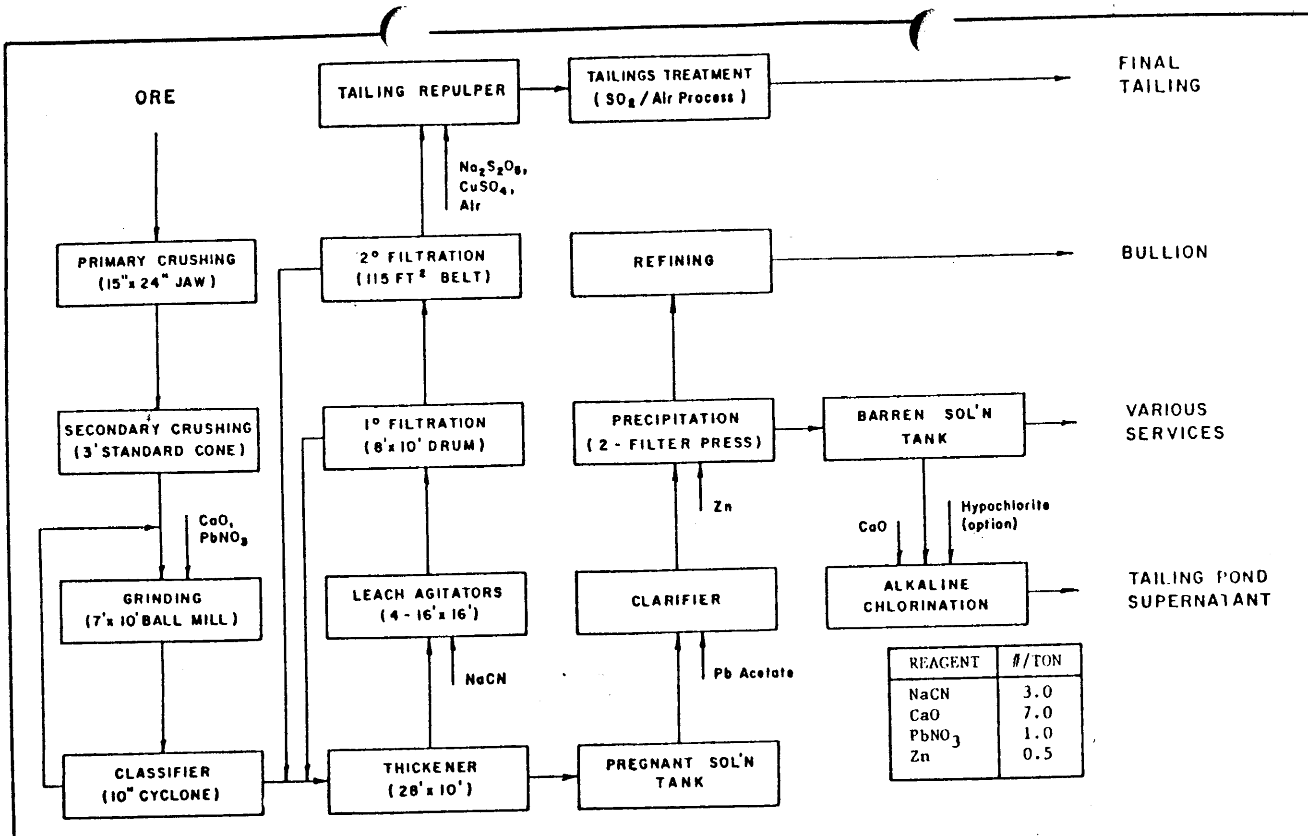

FIGURE 3. Baker Mine process flowsheet.

bridge to the thickener well. From both an operator and maintenance viewpoint, it was recommended that filtrate be piped to an accessible overflow box to view filtrate clarity and receiver pump performance.

Valving beneath filter feed splitter boxes was diaphragm type in a horizontal position. The valves were removed for the simpler red jacket type mounted vertically, virtually eliminating the chance for tramp material and/or slurry build-up.

Initial belt filter operation was troublesome. From day one, different cloths were tried to improve the belt filters operation. Most important cloth qualities are:

a) Over-all strength — limited stretching and width shrinkage.

b) Rigidity — the cloth has to be pliable enough to fit the transporter belt curbs to create a seal, yet stiff enough to have the tracking device work properly.

c) Porosity — once achieving a desired grind level, adopt a cfm rated cloth that will yield 150% design throughput. This pays off with extra retention time over vacuum box length — the longer the cake form time, the thicker the cake, the easier it was to optimize cake discharge.

d) Washability — the cloth should be able to be cleaned with a limited amount of water.

The belt filter had advantages over the drum filter, however, there are disadvantages. Reference paper[3] points out pros and cons of belt filters.

Considering the size of the belt filter installed (115 ft^2) and the area which it took up (approximately three times that of the existing drum which has 250 ft^2 of filtering area), and the location of Baker Mine, to install more conventional dewatering equipment. It would probably have been better.

Metallurgical results after adding the drum filter obviously were improved. Payback period on the drum filter was six months.

When comparing filter efficiency in terms of value recovery, the following now holds true:

— the drum filter recovery was 90%
— the belt filter recovery was 80% to 85%

The newest belt filter cloth performed well and recent focus was on putting this type of cloth on the drum.

Precipitation

Pregnant solution was pumped to a drop-leaf clarifier tank. Clarified solution was drawn under vacuum into a Merril Crowe tower to eliminate dissolved oxygen. Zinc dust was added to the tower discharge, which was then pumped through one of two filter presses where the gold/silver precipitate was collected. Gold/silver recovery was 99% plus. Barren solution was returned to the circuit for grinding make up water, washing, etc. Filter press precipitate was blown dry with high pressure air to approximately 40% H_2O.

Security around the filter presses was by chain link fence; however, this was found impractical. Operators needed access to the weir tank to cut samples, check flow levels through the press and control valving to the presses. Therefore, weir tank placement and control valving should be far enough away from the presses to allow a fence to be built around the presses alone.

Leakage from the filter presses represented an accountability problem. As the tonnage through the presses was 750 ton/day, a one gallon/minute leak represented a 1% error in metal production. The flow might range from 2 to 5 g/m, therefore, a tray was prefabricated to catch the leak and was fed into the weir tank.

Measurement of flow through the weir tank was initially designed to be done with a level indicator/recorder. Accountability and instrument performance were inconsistent. A simple measuring stick put beside the weir notch gave impressive results. Solution operators kept the level steady at 5 in. and remarkably, variance in "call" and actual production was less than 1%. This was quite good as $^1/_{16}$ in. difference at 5 in. represents a 3% variance in volume.

Removing precipitate from the presses was awkward and messy when transferring precipitate to drums. Paper was laid down around the tray to reclaim spillage. However, in future design, engineers might consider a tray with a light rail wheel

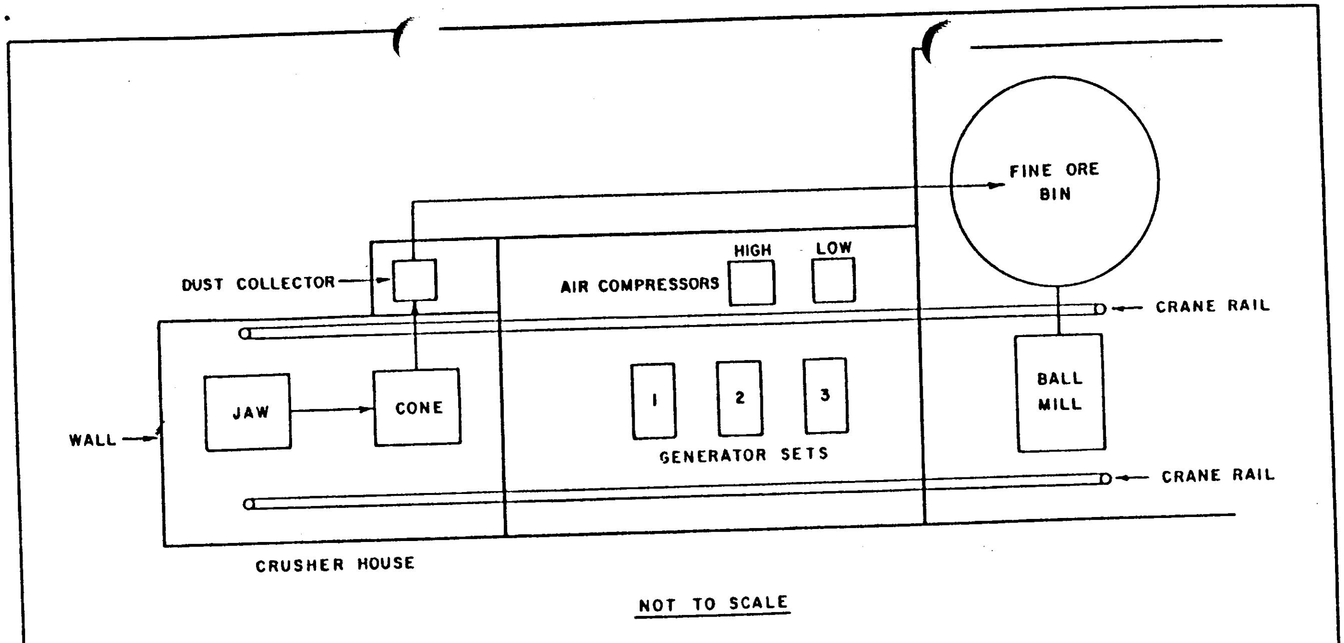

FIGURE 4. Baker Mine overhead crane layout.

on it. Then the tray as well could simply be wheeled to a lock-up holding area.

Over time, filter press feed lines became choked with precipitate build-up. Feed line piping was all threaded pipe and made for lengthy checks on piping take-downs. Piping was replaced with victaulic-type fittings.

Press paper preparation was made easier by building a "prep table". With template and drill, paper for one press could be ready in ten minutes.

Painting of clarifier leaves was a tedious, time-consuming job. A never-used acid wash tank was converted to a filter medium dip tank. The filter medium was added to water kept in suspension with a light air sparge operators then dipped the leaf in the tank under vacuum and had a "painted" leaf in 30 seconds.

Tailing Disposal and Pollution Control

Secondary filter cake solids and entrained cyanide-laden water discharged directly to the tailing repulper. Vacuum seal water first reclaimed and used for washing the belt filter cloth was then pumped to the tailing repulp tank. The tailing at 33% to 35% solids was pumped 2500 feet to an impoundment dam designed to settle solids and to leak pond supernatant via a coarse rock chimney in the dyke wall. The main concern with tailing disposal was obtaining acceptable levels of cyanide in the impoundment supernatant.

Start-up conditions surrounding the use of the alkaline chlorination circuit for cyanide destruction were a bit vague. However, once understanding of the chemical reaction, its hazards, and optimization steps taken, relatively good results were obtained in treating the barren bleed. The one drawback associated with alkaline chlorination was its inability to work in a slurry. Our solution to this problem was to pipe the treated barren with residual chlorine levels directly to the tailing pond where it could mix with the supernatant.

In early 1983, a new method of cyanide destruction was tried. The process uses sulphur dioxide and air[(2)]. Tests at Baker confirmed the effectiveness of the process. The new method was more efficient as slurries can be treated; it was much more economic (easily 50% less than alkaline chlorination) and it also destroys the ferro cyano complexes where alkaline chlorination does not. Operator preference working with the metabisulphite (SO_2 source) was high compared to using hypochlorite. Table 1 details Baker experience.

Tailing pond life was to be for 100,000 tons. Because of the relatively low pond volume, considerable time was spent on calculating volume used per ton of slurry. Interestingly, volume filled in the winter was considerably more than projected. Deposition of tailings on the hillside within the confines of the pond was necessary to gain on volume lost from ice lenses. During the summer months it was also possible to estimate volume gained due to ice melt. Actual findings were for winter 0.4 ton/yd^3 and in summer 1.2 ton/yd^3.

Refining Practice

The filter presses that collected the precipitate were first air blown for 8 hours. Average percent moisture was nearly 40%. On receiving assays of precious and base metals and silica, the precipitate was weighed into lots, mixed with flux and smelted in two Wabi furnaces. The charge was smelted for an average 2 hours. Final dore bars assay 950 to 980 "fine". The furnace lining was Greenpak AP90 and life was considered excellent at 500 to 600 hours.

Refining duties were handled by two people, management was present for all "button" and bar pours.

The refinery was located in the corner of the mill. It would be recommended that this be a separate building both for security and ventilation reasons. An 18,000 cfm fan was necessary to assure the fume was cleared of the refinery room and not allowed to filter into the mill complex. The stack/hood arrangement over the furnaces was too small and not of real value, mainly because of the necessity to turn furnaces when refining.

Reagent Consumption/Handling

Although illustrated in two feasibility studies, lime addition via a slurry ring main was not the set-up at start-up. Two dry feeders in place were taken out of service and a tank, lines, etc. put on line shortly after start-up.

For cyanide, a metering pump located on the bottom floor proved inadequate. The metering pump was replaced with a hose pump which fed a head tank. A more compact metering pump was put atop the head tank for cyanide addition. This was appproprïate as well, as the operator could see the cyanide pump action.

There were no provisions for handling materials except for one hoist well for the hypochlorite. Four electric hoists were added, one for handling steel and lead nitrate in the grinding bay, a second for cyanide. Two electric hoists also replaced chain falls for the clarifier leaf handling and the initial hypochlorite hoists.

When materials were airfreighted, all reagents were stored at

TABLE 1. Results of alkaline chlorination at Baker Mine*

Case 1

Cyanide (ppm)	Barren Solution raw	Barren Solution treated
Total Cyanide	1020	4.75

Operating Conditions:
Treating 20 tons barren solution/day representing approximately 40 pounds cyanide destroyed.

Case 2

Cyanide (ppm)	Tailing Solution raw	Tailing Solution treated
Total Cyanide	350	30.0

Operation Conditions:
Treating 230 tons tailing solution/day representing approximately 165 pounds cyanide destroyed.

Results with SO_2/Air Process of Baker Mine*

Cyanide (ppm)	Tailing Solution raw	Tailing Solution treated
Total Cyanide	213	0.1

Operating Conditions:
Treating 230 tons tailing solution/day representing approximately 110 pounds cyanide destroyed. Note the SO_2/Air process was very effective. In-plant adjustments to the water balance led to less cyanide having to be destroyed and further economic savings.

* Refer to Ref. 4.

the air strip. On completion of the freight haul, reagents were brought to the mill site in monthly consumption lots. For control of the smaller lots, these would be brought up altogether. A cold storage lean-to was built to aid storage against the sometimes severe wet and cold conditions.

Should a small mill have a section dedicated to reagent make up? Because of the current spread-out system, the numerous hoisting points, and three separate ventilation systems initially with troublesome roof exhausts, it would be worthwhile to cost out such an idea. The grinding operator could easily be responsible for reagent make up — one exhaust system for dust and fume, one area to clean up.

Reagent addition points/consumption is given in Figure 3.

Air Supply

Low-pressure air was supplied to the leach agitators from one unit. High-pressure air was supplied by two units for ball mill clutch, instrument air, filter press blow periods, crusher house dust collector and shop use. One unit was left on while the other was left in an auto mode for back-up.

Originally, there was only one high-pressure air compressor. The rated volume was too low for all services and so another compressor was added.

All three units are located beneath the fine ore bin. A dust free environment should be considered and the units should also be enclosed to dampen high noise levels.

Mill General

Power to the mill was supplied by three 600 kva diesel electric generating sets. One unit was on standby and rotated on line every two days.

Generating set placement/installation was planned poorly. The generator sets[3] were each enclosed in cubicles and the cubicles were placed with the main access door butted up to the mill building, essentially leaving minimum access to the units. Both engine and generator failure were experienced and "fortunately" each failure was on an outside generator set. The one engine block had to be brought through a side access door. The generator failure was accessible with a hiab. Again earlier feasibility studies had the generator sets turned 90 degrees with a heat reclaim system which should have been adhered to without question.

Even with the current set-up, a waste heat reclaim system was put in.

The MCC room was located by the fine ore bin. The MCC room was sealed and pressurized.

Efforts to be energy conscious were followed up with ceiling fans in the mill building. Ceiling temperatures were noticeably lower.

Ventilation fans for different reagents had roof exhausts. These were abandoned and wall exhaust systems were installed in preference for easier year-round access.

Fresh water/fire lines came into the mill between the tailing tank and the alkaline chlorination system — an already cluttered area. There was no reason for this; a more suitable position being along the same wall but at a more convenient location.

As mentioned in the grinding section discussion, there was no crane in the mill, but a 5-ton crane was installed in the crusher house. From a maintenance standpoint, during emergency repair or overhaul, a crane would be a big plus in working with the generator sets as well as over the ball mill. Based on Baker experience, the idea of designing overhead crane rails to service the crusher house and a service house enclosing the high noise compressors and the generator sets, and the ball mill seems an attractive option for crane utilization (Fig. 4).

During the winter, swelling of SRL pump liners posed an operations problem. It was traced to surface crews coating ore truck boxes with diesel fuel to prevent build-up.

Assay Laboratory Facilities

Although quite adequate now, the assay lab at Baker went through two expansions. The fact that the existing lab was adequate does not hold well in terms of value. It was believed designed trailer packages for a laboratory and bucking room would be advantageous in terms of safety, occupational health, and a quality work place. Resale value of the complete packages would be high.

Assay techniques in the precious metals industry must of course be of the highest quality (it does not take long for the slightest differences to add up to tens of thousands of dollars). It was deemed important that both buyer and seller visit each other's facilities, discuss technique and share the idiosyncrasies of the business.

Personnel

The staff complement is as follows:

Mill Superintendent	1
Assistant Mill Superintendent	1
Assayer	1
Assistant Assayer/First Air Attendant/Time Clerk	1
	4

The mill complement is as follows:

Crusher Operators	2	— Double as surface heavy equipment operators
Grinding Operators	4	
Solution Operators	4	
Swing Man	1	— 1 week solution operator 1 week grinding operator
Refinery Man	1	
Millwrights	2	
Electricians	2	— Cover the property
	16	

Original shift schedule was a 3-week-in 1-week-out rotation. With most employees concerned about frequency home, a 2-week-in 1-week-out schedule was developed and sanctioned by the work force. This step toward better morale and better efficiency cut labour costs by over 10%.

With improvements in the crusher section and time spent per day at the crusher down to 2 to 3 hours, it was decided to have the crusher operator a more versatile person. After some attrition, crusher operators hired usually held credentials in heavy equipment operations and/or air brakes ticket to double as surface operating personnel.

If the reagent mixing was a central area, the grinding operator, with only the belt feeder and ball mill to look after, could easily look after a reagent mix section.

REFERENCES

1. LORETTO, JOHN C., Where Did the Gold & Silver Go? Proceedings, 14th Annual Meeting Canadian Mineral Processors, p. 388.
2. DEVUYST, E., ETHEL, V., and BORBELY, G., New Method for Cyanide Destruction in Gold Mill Effluents and Tailing Slurries, Proceedings, 14th Annual Meeting Canadian Processors, p. 336.
3. VON MICHAELIS, HANS, Belt Filtration in Extractive Metallurgy, Annual Hydrometallurgical Conference of CIM on Solid-Liquid Separation, October 18 - 21, 1981.
4. MUIR, W., An Overview of Operation and Tailing Disposal Focussing on Alkaline Chlorination of Cyanide Bearing Mill Effluents, Alkaline Chlorination for Gold Mill Operators Seminar, May 26, 27, 1982.