PILOT PLANT TESTING OF ORES FOR AUTOGENOUS AND SEMI-AUTOGENOUS GRINDING

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ABSTRACT

Considerations and procedures for evaluating the amenability of ore to autogenous and semi-autogenous grinding by pilot plant testing, first presented at the SAG Conference 1989 in Vancouver, Canada, have been revised and expanded. Ball charge size and weight, removal of pebbles for crushing and recycle, or for pebble milling, changes in mill feed size distribution, and classification size are typical variables discussed.

The importance of secondary ball or pebble milling included as an integral part of the test circuit is discussed.
INTRODUCTION

Since the introduction of autogenous grinding in the nineteen fifties the acceptance and construction of autogenous grinding plants has been increasing steadily. Today most large plants, and an increasing number of medium size plants, find it economically advantageous to use these mills. In addition to these new mills, older conventional mills are either expanding with or converting to autogenous mills. The present trends are for even smaller plants to use autogenous mills and a rapidly increasing acceptance and use of pebble crushing as an integral part of the circuit. For the near future we see an increasing use of mathematical modelling for both the design and operation of grinding circuits.1

The resulting increase in the demand for testwork has been sufficient for Lakefield Research to expand its test facilities. The Cascade mill circuit, originally commissioned at Lakefield in 1957, has been modified several times; most recently in 1988. As more of our clients have been requesting semi-autogenous or autogenous milling for feed preparation for their flotation or other beneficiation pilot plant programs, Lakefield Research has commissioned a second parallel FAG/SAG test circuit around a recently acquired Nordberg test mill. This allows one mill circuit to be tied up for long periods, preparing feeds to programs designed to test other beneficiation operations, while maintaining capability to provide FAG/SAG test service on a demand basis. The Nordberg mill also provides capabilities for extraction of larger pebbles and testing higher length to diameter ratios.

The extraction of useful data, the accuracy of the measurements and the continuity of the operation are the prime responsibilities of the testing laboratory and its engineers. Every effort is made to conform to this within the testwork limitations.
With 32 years of testing experience we find the same issues arising continually and we hope that by addressing the following points some of these issues can be resolved:

1) considerations for the clients prior to setting up a test program - preparation
2) how a test program should be conducted - operation
3) looking at the results - interpretation
4) problems that can occur - pitfalls

1. PREPARATION

This section is best covered by presenting a series of questions that the client should consider in consultation with an engineering group, a private consultant and/or the testing facility.

1.1 What Information Do We Require From The Testwork?

The reasons for the testwork could be: a) preliminary examination of a small bulk sample to characterize the ore b) generation of engineering data c) comparison with conventional grinding with regards to power consumption or effect on metallurgy d) most power efficient development of the circuit configuration.

1.2 How Variable Is the Ore And What Ore is Available For the Testwork?

The question of what ore should be shipped for testwork is often a very difficult one to answer. The answer depends to a great extent on what the mine plans for the orebody are. If the ore is homogenous in nature then one overall composite is sufficient. If the orebody is not homogenous but the mine plan
calls for extensive blending facilities, then the bulk of the testwork can be carried out on an overall composite followed by brief tests on other specific samples from the orebody. If no blending is to be performed, then more extensive work on the major components should be considered with brief tests on other minor samples particularly if they are at the extremes of hardness or softness.

All of these possibilities may be rejected if availability of ore for testing is the deciding factor. Then the question should be asked as to whether now is the appropriate time to start investigations into the processing.

1.3 Are There Any Constraints On Plant Design?

This question applies particularly when modernizing or expanding existing plants. Available floor space may preclude the possibility of using a crusher in an ABC circuit. The available power supply may limit the motor size on the primary mill and thus change the balance between the primary mill and secondary mill or mills. Economic constraints on capital or operating costs may have some effect on the test program although more properly they should be dealt with in the subsequent feasibility study.

Metallurgy or reagent costs may dictate that only fully autogenous grinding, either single or two stage, is feasible. Shipping of steel to remote areas can also be prohibitively expensive dictating fully autogenous grinding.

1.4 Can The Grinding Circuit Be Designed To Improve The Metallurgy?

In some cases this could be considered under the previous section as a restraint on plant design. For instance a client may wish to remove coarse free gold at as early a stage as possible and may therefore stipulate a fixed
product size from the primary grind to be followed by some form of gravity recovery ahead of the secondary grinding circuit.

Fully autogenous grinding may be preferred to semi-autogenous grinding or vice-versa for a number of reasons, such as control of different product size distribution or iron from the grinding steel affecting the metallurgy or reagent consumption. With certain minerals such as graphite, liberation of the mineral grains at a coarse size is important in obtaining a premium grade product.

1.5 How Much and What Size Of Ore Should be Sent?

Since 1980 we have received as little as 20 tonnes of ore and as much as 640 tonnes. A normal minimum quantity of ore that we would recommend for characterization tests of a single ore type in a 1.8 m diameter Cascade mill would be 60 tonnes. This would allow about 7 or 8 shifts (8 hours) on an ore of average hardness. The low tonnage tests have been on six inch drill core where a limited quantity of this very expensive material has been available. The large tonnages have been for multiple ore types with extensive metallurgical testwork.

In each individual case the client will have to balance the complexity of the ore body, the ore availability, the need for metallurgical testwork and last, but not least, the availability of funds for the testwork.

The upper size limit for feeding to a 1.8 m diameter Cascade mill is 250 mm. A certain amount of oversize can be received but this must be broken down by sledge hammers or some other means and reblended with the rest of the sample. These oversize pieces frequently consist of the more competent rock type and removal from the feed sample can falsify the test data.
The size distribution of the feed should ideally be similar to that of a primary crusher product. However, unless fully autogenous grinding is being seriously considered this becomes less important as the amount of steel increases. For a semi-autogenous grind with 6% mill volume of steel, the weight ratio of steel to plus 75 mm pebbles typically would be about 5:1. In other words the effect of the steel begins to outweigh the effect of the larger pebbles. What should be watched for is excessive fines in the feed — fines being material finer than the primary classification size. This can occur with excessive handling of a soft ore or when the explosive to rock ratio is higher than normal, such as in development drifts.

On the other end of the scale, drill core invariably requires some crushing to produce an acceptable size distribution.

2. OPERATION

2.1 Feeding

Most feed samples are sized into the three size fractions -250 mm + 150 mm, -150 mm + 75 mm and -75 mm. Occasionally with well-weathered or very soft ore there is so little +150 mm material that other finer size fractions are used.

Increments of reconstituted ore are prepared in tubs in 100 kg lots and fed to the Cascade mill. This procedure ensures an even feed size distribution throughout the testwork.

The feed rate is determined by maintaining a fixed charge weight in the mill, normally chosen to give 25% to 27% of mill volume. The weight is determined by load cells under the mill. A digital readout of the weight is displayed by the
feed station to allow the operators to regulate their feed rate. The operators maintain a record of the time that each 100 kg batch is started and by this record our feed rate is determined. The mill charge level is checked occasionally prior to the sample period and again at the end of the sample period.

2.2 Flowsheet

Clients and consultants differ on their wishes to install a secondary grinding circuit; some preferring to obtain direct readings of the secondary power consumptions; others preferring to rely on mathematical calculations from a Bond ball mill work index and the size distribution of the primary mill circuit product. Since the only extra operating cost to clients is in the addition of three extra screen analyses, we usually recommend the inclusion of the secondary circuit. This point will be discussed again in the section on interpretation of the results.

A typical flowsheet is shown in Fig. 1. The pulp splitter on the Cascade mill (CM) screen undersize is used to maintain an approximately constant feed rate to the secondary grinding circuit. Either a screen or a cyclone can be used in the secondary grinding circuit. However, a screen has the advantage of giving a more definite size split and allows a better judgement when visually adjusting the pulp splitter to give a suitable circulating load. Additionally, plugging of a small cyclone when using a coarse primary classification can be a problem.
2.3 Data Collection

The circuit is considered to be in equilibrium when the feed rate, power consumption and circulating loads have all stabilized. Sample collection begins after equilibrium has been maintained for approximately an hour. Data is collected throughout the test run but is averaged and reported only for the sample period.

2.3.1 Power Calculations. The power calculations are made according to methods used by the Hardinge Company (now MPSI), Report No. 26-H2002, April 1958. Total power consumed is that registered by a Sangamo KYWP Demand Energy Meter. Gross power is the calculated power delivered to the chain drive from the output shaft of the speed reducer, and is equal to the total power multiplied by the overall efficiency of the motor, sheaves and speed reducer. The efficiency figures are taken from graphs prepared by Lakefield Research using a method involving the application of a Prony brake to the output shaft of the speed reducer. The efficiency of the chain drive cannot be determined. The total power is determined by measuring with a stopwatch the time interval during which the rotating disc in the meter makes a given number of revolutions. This is determined every 30 minutes.

The no-load power is measured in the same way but with the mill operating empty.

Net power consumption is equal to gross power minus no-load power. It represents the net power delivered to the output shaft of the speed reducer, and includes power consumed by the chain drive as well as the mill bearings.

A Sangamo SC2 Thermal Graphic Demand Meter is used to record the kW input to the Cascade mill on a strip chart recorder. The recorder has a full scale
deflection of 1.0kW, which multiplied by the power transformer times current transformer ratio, equals 25. The meter has an accuracy of ± 1%. The strip chart recorder is recommended to observe and correct fluctuations in power readings. Temperature changes caused by open windows and doors in cold weather can have a significant effect. Inadequate lubrication will give false readings even though bearings may not be overheating.

2.3.2. Pulp Densities. Samples of the mill discharges are taken every 30 minutes during the operating period. A direct reading Marcy pulp balance is employed for the purpose. Specific gravities are determined on the products from the first two or three tests and percent solids calculated as the data is entered into the computer.

2.3.3. Circulating Loads. Timed weight samples of the CM screen oversize are also taken at 30 minute intervals. Since normal operation is with a protecting screen (13 mm aperture) and a classification screen, two samples will be taken. If pebble ports are open a third product, the trommel oversize, may also be included. A moisture determination is made once per shift on the combined recycle product.

Pebble ports can block easily and can be detected by changing circulating loads.

With the secondary mill it is usually a visual estimate although occasionally the BM screen oversize has been collected as a timed weight sample.

2.3.4. Feed Rate. The feed rate is calculated by taking the exact time taken to feed a given number of 100 kg batches. The series of batches chosen are those most closely representing the sample period. A moisture correction is then made to give a dry feed rate.
2.3.3. Sampling. For each test the following pulp streams are cut at 15 to 30 minute intervals depending on the length of the sample period:

<table>
<thead>
<tr>
<th>CM discharge</th>
<th>BM discharge</th>
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<tbody>
<tr>
<td>CM screen oversize</td>
<td>BM screen oversize</td>
</tr>
<tr>
<td>CM screen undersize</td>
<td>BM screen undersize</td>
</tr>
</tbody>
</table>

Screen analyses of these samples allow calculation of the work indices, the BM circulating load and, if confirmation is required, the CM circulating load.

2.4 Test Program

A typical starting series of tests would use a primary classification size of 6 mesh and would vary the % mill volume of steel from 0% to 3%, 6% and 9% or from 0% to 4% and 8%. For the Cascade mill 1% mill volume of steel is equivalent to 62 kg. The mill charge is normally removed and sized after the fully autogenous grind and at least one of these other tests. The information gained from these size analyses from the circulating loads and from the power consumptions would then suggest the direction of the next series of tests, e.g. change of primary classification size, change of ball size or further increase in the CM steel charge. Removal of pebbles for crushing, if being investigated, would develop from the second series of tests.

3. INTERPRETATION

3.1 Fully-autogenous Grinding

The interpretation of the results is an ongoing process beginning with the first fully autogenous test. The ultimate feed rate for fully autogenous
grinding has varied from as low as 230 kg/hr to as high as 1200 kg/hr with the most frequent rate being 500 to 600 kg/hr. The rate is dependent to some degree on the classification size but is even more dependent on the density of the major pebble forming fraction of the ore. The 1200 kg/hr was achieved with an ore containing 45% pyrite with a relatively fine screen size of 10 mesh. The feed material contained 65% plus 75 mm material and was not particularly friable. The rate at which equilibrium of the feed rate is achieved is indicative of the rate of build-up of pebble forming material in the mill. A very long period of build-up indicates the presence of a very small proportion of very hard ore in the sample. Equilibrium has been reached from as little as 3 hours in a quartzite gold ore up to as long as 260 hours on a Canadian iron ore. Unless fully autogenous grinding is being considered as a serious possibility, then one 8 hour shift is usually considered sufficient time to give a good indication of operating results.

3.2 Variation Of Steel Charge At One Classification Size.

Various relationships can occur. A typical series of results using a 10 mesh screen for classification is shown in Figs. 2, 3, 4 and 5 determined for three ore zones with differing degrees of oxidation all from the same Peruvian Cu deposit.

As one would expect, the feed rate increases and the net power consumption decreases with increasing steel load. The circulating load is increasing with increased steel.

In the second set of results on a Au ore from B.C., using a 6 mesh screen, similar curves are obtained for feed rate and net power consumption, but in this case the circulating load decreases with increasing steel load.
Experience has shown that maximum efficiency is achieved in an autogenous mill with circulating loads of between 50% and 100%. In the first case shown above only the soft ore is within this range at 9% steel (Fig. 4). A change of screen to 14 mesh would probably bring the other ores within the range also.

In the second case, the circulating load decreases with increasing steel load (Fig. 8). Grinding efficiency was improved in this case by the crushing of pebbles. Interest in using single stage grinding in this plant resulted in the testing of a cyclone in closed circuit with the Cascade mill. With increases of the circulating load to levels usually associated with ball mills (200% to 400%) the work index for single stage grinding was the same as that achieved with a 10 mesh primary classification and two stages of grinding.

3.3 Two Stage Grinding

The graphs shown in Fig. 10 demonstrate the type of relationship we would like to develop in every test series. In fact this is not always developed for a number of reasons; the total power consumption may not show a minimum point but may trend towards a primary classification size that is impractical; ore or monetary constraints may limit the number of tests or other factors may indicate other avenues of investigation.

Rod mill and ball mill work indices from the standard Bond grindability tests are often good indications of whether optimum circuit configuration has been achieved. Normally, when the ball mill work index is higher than the rod mill work index, the overall operating work index will approach the ball mill work index. Occasionally the ore will show a higher rod mill work index and, in these cases, the overall operating work indices are closer to the rod mill figure.
### TABLE NO. 1 COMPARISON OF WORK INDICES

<table>
<thead>
<tr>
<th></th>
<th>Bond</th>
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<th></th>
<th>Operating</th>
</tr>
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<tbody>
<tr>
<td></td>
<td>RM</td>
<td>BM</td>
<td>CM</td>
<td>BM</td>
</tr>
<tr>
<td>Ore 1</td>
<td>18.0</td>
<td>14.9</td>
<td>28.7</td>
<td>14.6</td>
</tr>
<tr>
<td>Ore 2</td>
<td>13.1</td>
<td>14.7</td>
<td>14.1</td>
<td>14.4</td>
</tr>
<tr>
<td>Ore 3</td>
<td>-</td>
<td>12.0</td>
<td>52.7</td>
<td>3.1</td>
</tr>
<tr>
<td>Ore 4</td>
<td>-</td>
<td>12.0</td>
<td>15.2</td>
<td>6.7</td>
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</table>

When the overall operating work index is lower than the Bond work index (Ore 4) this is an indication that the Bond work index is misleading. This is not uncommon with ores with high clay, mica or fibrous mineral content where the dry grinding in the grindability tests results in packing and poor grinding conditions.

### 3.4 Cascade Mill Charge Size Distributions

The mill charge can be defined in two ways:

1. the total ore content of the mill when shut down under operating conditions,

or

2. the total ore content less that size fraction below the classification size for the Cascade mill circuit. (i.e. if 6 mesh screen is being used for a particular test the mill charge would be defined as all the plus 6 mesh material retained in the mill).
Since slimes and other fine material are often lost when unloading the mill or screening the charge we prefer to use the latter definition.

Figure 11 shows the changes occurring in a mill charge during the testwork development on a sample of gabbro Au ore from Alaska. The fully autogenous test shows a very high proportion of plus 2" material. With 8% steel, using 3" and 4" steel balls, the amount of fines produced increased considerably. The distribution was then greatly improved by changing the steel to 4" and 5" balls. A further small improvement was found by reducing the amount of plus 6" material in the feed. Finally, by changing to a coarser screen the amount of fines in the distribution is reduced.

The effect of ball size on the grinding efficiency can be quite significant as can be seen in Figure 13. The addition of 5" balls to the charge for this Canadian gold ore resulted in a decrease in operating work index from about 15 to 12.

A diabase Cu ore from Arizona (Fig.12) showed a tendency to break up very readily along fracture planes in the ore producing a concentration of pebbles in the 1/2" to 1" size range. With very few large pebbles present high levels of grinding steel were required. The testwork showed that similar throughputs and overall power consumptions were obtained either by using 12% mill volume of steel or by using 8% mill volume of steel and crushing the 1/2" to 3" pebbles to minus 1/2".

4. PITFALLS

Many of these have been listed before in a paper originating from Lakefield Research but are well worth repeating here along with a number of updates.
They can be embarrassing or dangerous at the time to the company, the engineers or the operators but in retrospect many are amusing to look back on.

4.1 The Sample

After treating 600 tons of ore it was found that the sample, taken from a convenient outcrop, only represented 9% of the total orebody and was the most refractory section.....

80 tons of copper-zinc ore were shipped in a railroad car, on top of 5 cm of zinc concentrate.....

Another ore was shipped in a railroad car used for shipping scrap metal shavings soaked in oil.....

5 railroad cars passed through a blizzard and freezing rain. The ore arrived deep frozen in the form of 100-ton ice cubes.....

250 mm top size could mean desk size to a geologist.....

Large pieces of rock had drill holes right through them, some of which contained unblasted dynamite sticks.....

Another frozen ice cube in a large dump truck. The ore wouldn't budge even when the hydraulic lift was fully raised. Unfortunately the truck was not quite on level ground and the whole load fell over sideways.....
4.2 Operation

"Missing tubs" on the operators records, although not frequent, can happen when the operator forgets to record the completion time for a tub addition. If obvious this can be corrected later but it is not always obvious.

"Non-missing tubs". After a seemingly too frequent occurrence of "missing tubs", which the operators swore they were not mis-recording, we found that a 25 mm to 30 mm layer of fine material was packing onto the shell liners, reaching a critical thickness and then sloughing off causing changes in the load cell readings. Our apologies to the operators.

Large rock pieces can block the feed chute and the power will drop rapidly. As a rule an operator will follow each large rock up the feed belt and make sure the rock passes through the chute. A thin steel rod is used for poking the ore into the mill. A rigid rod mill rod was once used, caught, and threw the feed chute right off the platform. Fortunately the operator was on the right side of the chute.

A tar sand sample was sent to Lakefield to confirm some dry grinding testwork performed at MPSI. Whether our sample contained slightly more tar or the temperature was a little warmer we don't know, but our Cascade mill rapidly changed into a solid flywheel.

4.3 Scheduling

The loader required to fill the transport truck went through the ice.....

The containers selected for the ore shipment were rejected by the port authorities as being too slightly built for the heavy ore. After replacement,
the stevedores went on strike. The crane broke. Finally the boat that was to travel direct had to make detours because of the strike to discharge the different cargoes it had loaded to fill the hold. The massive sulphide ore was in transit for 5 months, two of which were in the tropics.....

The railcar loaded with 80 tons of ore had a hole, 14 tons arrived.....

The wrong outcrop was cleared and the overburden dumped right on top of the outcrop that was eventually sampled.....

A truck containing platinum ore stopped at the U.S./Canada border and the trucker declared he was carrying plutonium ore.....

REFERENCES


Figure 1: Typical Flowsheet
Fig 2: Variation of Feed Rate with Steel Charge-Peruvian Copper ore

![Graph showing variation of feed rate with steel charge.]

- Sample C
- Sample B
- Sample A

Fig. 3: Variation of Net Power Consumption with Steel Charge-Peruvian Copper ore

![Graph showing variation of net power consumption with steel charge.]

- Sample C
- Sample B
- Sample A
Fig. 4: Variation of Circulating Load with Steel Charge-Peruvian Copper ore

Fig. 5: Variation of Net Power Consumption with K80 of Product-Peruvian Copper ore
Fig. 6: Variation of Feed Rate with Steel Charge-B.C. Gold ore

Fig. 7: Variation of Net Power Consumption with Steel Charge-B.C. Gold ore
Fig. 8: Variation of Circulating Load with Steel Charge-B.C. Gold ore

Fig. 9: Variation of Net Power Consumption with K80 of the Product-B.C. Gold ore
Fig. 13: Primary Mill and Total Net Power Consumption

Primary Classification Size, microns

% Circulating Load-Primary Mill

Net Power Consumption, kWh

Primary NPC

Total NPC

C.L.
Fig. 11: Mill Charge Size Distributions

Fig. 12: Mill Charge Size Distributions
Fig. 13: Variation of Net Power Consumption with Product Size