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Addition of pebbles to a ball-mill to improve grinding efficiency – Part 2

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ABSTRACT

Nkwanyana and Loveday (2017) used batch grinding experiments in a 0.6 m diameter mill to test partial replacement of steel balls (37.5 mm) for secondary grinding, by partly rounded pebbles (19–75 mm) from a SAG mill. At the optimum pebble content of 25% by volume, a *25% saving in steel ball consumption and a 15% saving in energy consumption was achieved, with no change in productivity*. These very encouraging results were discussed with operators and designers of SABC circuits.

This paper compares the performance of a 25/75 pebble/ball (volume ratio) composite charge to ball-milling, under conditions typical for a ball-mill in SABC circuits, i.e. a large ball top size (75 mm), a high mill speed (75% of critical speed) and a coarse feed top size (19 mm). The volume of pulp in some experiments was increased to mimic pulp filling in an overflow discharge ball-mill. All the experiments in this phase were conducted using a silicate ore. The rate of production of material finer than 75 µm, when using a 25/75 mixture, was virtually identical to that of the corresponding ball-milling experiment. The average saving in energy consumption was about 13% and a saving in ball consumption of 25% applies.

It was noted that the replacement of balls by pebbles reduced the rate of grinding of coarse particles. This problem was solved by retaining the original quota of large balls, and replacing only smaller ball sizes. This method had a minimal effect on the rate production of fines (−75 µm) and makes composite secondary milling even more attractive, for reducing the cost of balls.

Concerns about accumulation of 'scats' (coarse particles) have also been analysed. The effect of scats on ball-mill productivity was tested and it was concluded that scats may not have an adverse effect on productivity in secondary ball-mills and composite mills, if large balls are present. The use of a trommel on the ball-mill is suggested to solve related problems in the classification circuit.

1. Introduction

Milling is the most expensive process in the mineral processing stage, with energy and grinding media (steel) consumption being the most expensive cost items. It is estimated that grinding media accounts for up to 40% of the milling costs (Powell et al., 2015a). Research on conventional milling circuits has focused mainly on the optimization of the first stage of milling, typically semi-autogenous grinding (SAG), and it was aimed at stabilizing throughput and reducing costs.

Nkwanyana and Loveday (2017) performed pilot-scale batch milling experiments to test a new method of grinding for secondary or regrind ball-mills. The method is referred to as 'composite secondary milling', or CSM. The method entails replacing a fraction of the volume of steel balls in ball-milling by critical-size pebbles emanating from a SAG mill, which are normally crushed and recycled. The method was tested for milling a silicate ore (F_{80} = 1.3 mm, BWI = 15 kWh/t) and a platinum ore (F_{80} = 0.150 mm, BWI = 21 kWh/t). The conditions were designed to simulate secondary grinding of a platinum ore. The size distribution of balls and pebbles simulated a seasoned charge, with a ball make-up size of 37.5 mm, and various ranges of pebble feed size. The mill was operated at 69% of critical speed. At an optimum pebble content of 25% by volume, the method reduced energy consumption by 15%, without a negative impact on the productivity and implied a 25% reduction in ball consumption.

Hard, moderately hard and medium hard ores ($30 < A*b < 56$) are conventionally processed in SAG-Ball mill-Crusher (SABC) circuits. The ball-mills in SABC circuits are operated at a relatively high speed (70–78% of critical speed), with large balls (50 mm to 80 mm), to cope with a coarse feed (a transfer size from the SAG mill typically larger than 10 mm) and to maximize power draw. Overflow ball-mills are often used and a mixture of ball sizes is used to top up the ball charge at regular intervals, e.g. the KCGM Mt Charlotte circuit in Australia used a mixture of 70 mm and 60 mm balls (Dupont, 2006), the Batu Hijau concentrator in Indonesia used a mixture of 65 mm and 80 mm balls

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(Burger et al., 2006) and the Kinross RPM operation in Brazil used a mixture of 38 mm and 70 mm balls (Tondo et al., 2006). Having proved the concept of composite secondary milling in the previous paper, it was decided that a comparison between ball-milling and CSM under these more extreme conditions (coarse feed, higher mill speed, large make-up ball sizes and an excess slurry filling) would provide a more robust test. The first part of this paper presents the experimental results under these conditions.

Particles larger than the aperture size of the trommel in the SAG circuit are perceived to be 'undesirable' in the ball-mill circuit. The ball-mill may lack sufficient impact energy to effect breakage of these particles, which are referred to as 'scats'. They may find their way into a ball-mill circuit as a result of wear of the trommel or due to a deliberate change of the aperture size, to increase SAG mill throughput. E.g. Geita gold mine in Tanzania reported scats with a P_{80} between 14 mm and 22 mm (Mwehonge, 2006). Scats cause the following problems: Firstly, if entrained in the slurry exiting the ball-mill, they increase wear in the cyclones and cyclone feed pumps, and increase the circulating load. The probability of scats being entrained in the exiting slurry depends on the slurry viscosity and the ball level. Secondly, they can build up in the mill and may reduce grinding efficiency. The pebbles recovered from a SAG mill trommel will typically be in the size range 15–75 mm. When the larger pebbles are used for composite milling, they will over time wear and become scats, but the remaining mass is a small fraction of the original mass. The second part of this paper addresses the concern expressed by some, that composite secondary milling will exacerbate an existing scats problem.

2. Experimental

Experiments were carried out using a pilot scale batch mill with a diameter of 0.6 m and a length of 0.5 m. Removable metal lifters (8), approximately 54 mm high, with 90 degrees face angle are fitted inside the mill. The internal surface of the mill is rubber-lined, giving an internal diameter of 0.57 m. The mill is attached to a drive shaft at one end, which is driven by a motor and a variable speed gearbox assembly. The torque driving the rotation of the drum is monitored online and the average value was used to estimate the mill power draw. The experimental procedure has been described in the previous paper (Nkwanyana and Loveday, 2017), which requires duplication of all experiments.

The authors were given access to plant survey data from a large SABC circuit, which is processing gold ore. This data was used to establish the experimental conditions. The fines which were used to feed the ball-mill were chosen to simulate the hardness of gold ore. Initially crusher fines were obtained from a local quarry. However, the variability of the % −75 µm after preliminary ball-milling tests was unacceptable, and this was attributed to a variable clay content in this ore. It was therefore decided to make up an artificial feed, consisting mainly of high purity bagged silica sand of size less than 4.75 mm, and a relatively small amount of larger particles (−19 + 4.75 mm) from the quarry. The work index of silica sand (about 15 kWh/t) is comparable to that of gold ores from Witwatersrand (15.81 kWh/t) (Levin, 1989). Fig. 2.1 shows a comparison of the ball mill feed used in the experiments and the data from the plant.

The pebbles used for the experiments were mostly gold ore waste rocks, with a few from the local quarry (to make up for the coarsest size fraction). The pebbles were rounded since they have been used in previous autogenous grinding experiments (Pillay and Loveday, 2015). Experiments have shown that pebbles from the two sources have essentially the same hardness (Loveday (2004).

The size distribution of steel balls simulated a seasoned load, based on the chosen plant survey data, where a mixture of make-up ball sizes (53 mm, 65 mm and 75 mm) was used to maintain ball level. The size distribution of the seasoned load was calculated, by assuming that the mass distribution of the make-up balls was 50%, 25% and 25%, respectively. The seasoned pebble size distribution was calculated, using the size distribution of pebbles discharged by the plant SAG mill. The conditions common to most tests are summarised in Table 2.1. Table 2.2 shows the seasoned charge size distributions. The method used to calculate a seasoned size distribution is explained in the previous paper.

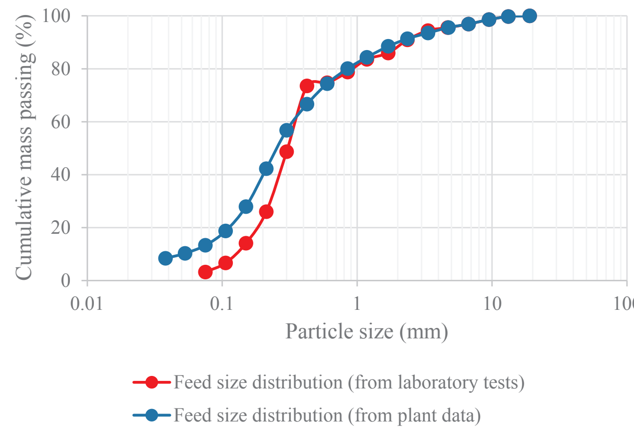

Fig. 2.1. Ball-mill feed particle size distribution.

Table 2.1
Operating conditions.

Mill rotational speed	75% of critical speed (*Nc*)
Feed (fines) top size	19 mm
F_{80}	1 mm
Static charge volume	33%
Slurry filling (U)	1, 1.53
Slurry solids concentration	65–68% wt.
Ore density	2650 kg/m^3

Table 2.2
Seasoned charge size distributions.

Size range (mm)	Steel balls (mass %)	Pebbles (mass %)
−75.00 + 65.00	13.26	7.54
−65.00 + 53.00	24.33	22.36
−53.00 + 37.50	47.76	37.48
−37.50 + 26.50	11.25	21.27
−26.50 + 19.00	2.59	9.06
−19.00 + 13.20	0.81	2.29

The previous paper demonstrated that the efficiency of grinding (kWh/t new −75 µm) was insensitive to changes in the (static) charge volume, as the results obtained using 30% and 40% charge volumes were similar. It was therefore decided to use a charge volume of 33%, which is within the typical charge volume range of 30% to 35% for large diameter overflow ball-mills (Kawatra, 2006). Mulenga and Moys (2014) found that the rate of production of fines was optimized when the voids within the static charge were filled completely and no slurry pool was visible. Assuming that the porosity of the static charge was 0.4, as assumed by many researchers (e.g. Tangsathitkulchai, 2003; Mulenga and Moys, 2014), the optimum slurry volume was calculated (i.e. 33% × 0.4 = 13.2% of mill volume). The ratio of the volume of slurry to the volume of voidage in the static charge was defined by Mulenga and Moys (2014) as *U*. The optimum production rate of fines is thus obtained at *U* = 1 and this condition was used in the first set of experiments.

Large overflow ball-mills allow a slurry pool to develop in the mill up to a total volume filling of 40%, to allow for discharge through the trunnion (Kawatra, 2006). Hence the volume of pulp required to simulate an overflow ball-mill was (13.2 + 7 = 20.2% of mill volume). This was used in the second set of experiments and the corresponding