The Use of Process Simulation Methodology in Dc-bottlencecking of Production Lines

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ABSTRACT

In 2004, LKAB decided to start a basic engineering study in expanding its existing production lines at the Malmberget mine site. It was expected that some of the production lines would become bottlenecks with reference to capacity. In this study, the sorting plant is examined and the results show that several unit operations will be limiting under current operating conditions. An additional purpose with the study was to investigate how replacement of some machines would influence performance and if production plans for the coming years might be achieved.

Steady-state simulations using the software Modsim™ were extensively used to study process performance for different operating conditions, such as iron content and size distribution in run-of-mine ore. Model parameters were derived from pilot plant data and when possible from existing circuits. A complication for the simulations was the lack of relevant data and therefore some of the initial results were only indicative.

The result shows that installation of new screens would give a much better operation, but also new crushers would be an improvement. This, since they can be run with full crusher chamber and tighter CSS.

During 2007, most of the proposed remedies were implemented and the result is that the bottlenecks are gone, and the grades and material balances are even better than what was envisioned from the simulations.

Keywords: Cobbing; Crushing; Process simulation; Process design

INTRODUCTION

LKAB is today the only producer of iron ore products in Sweden, and the largest within the European Community. In 2007, LKAB produced about 24.7 million tonnes of iron ore products, of which 76% were pellets and the rest sinter fines. The demand for iron ore is anticipated to increase until 2010. Therefore, expansions in production are initiated with the intention to increase the total production of iron ore products to nearly 10 Mtpa at Malmberget.

LKAB started already in 2004 a basic engineering study to be the basis for further expansions of the surface operations at the Malmberget mine site. Results for the concentrator are published earlier (Tano, et al., 2006) and (Tano, Pålsson, Alatalo, 2006). In the current report, the cobbing plant is investigated, and the results
show that several unit processes and production lines would be limiting with the existing machinery. Therefore, one purpose of the study was to investigate how a change of machines would influence capacities and operations. Another one was to check if future production plans were viable.

The investigation are all done according to a common blueprint: a few tests on pilot scale of planned tests/investigations in existing production lines, built and calibrations of simulation models for single unit operations or parts of process circuits, validation of the simulations for known conditions, full simulations with an amalgamated flowsheet with varying input data. The drive for this approach was the need to minimise the time between idea and completion, to investigate the potential in the existing process, and to rapidly investigate or eliminate alternative layouts and operations. For the cobbled plant it is then an obvious goal to be able to do bottleneck analysis for varying feed rates and feed materials.

**THE COBBING PLANT**

The cobbled plant, cf. Fig. 5, at Malmberget treats ores from the Western field, hoisted in the Alliansen shaft, and from the Eastern field by the Vitafors shaft and conveyor system. The process is simulated with four feeds: magnetite ore from Vitafors that today contains some drifting rock; a separate stream with drifting and development rock since its influence is investigated; magnetite ore from Alliansen (Western field); H-PAR that is a magnetite fines emanating from the hematite cobbled process. It has three waste streams: gangue (30–200 mm) that is eliminated in the pre-treatment (GRAM) step; crusher tail (5–30 mm) that emanates from the crushing and the PAR line; fines tail (0–4 mm) that is the result of scavenging on the middlings from the FAR line. There are two main products: FAR that is the naturally occurring fine fraction of the ore and goes to the fines production; PAR comes from the coarse fraction of the ore and is the feedstock for the pellets production.

The first screen and separator makes up the pre-treatment or GRAM section. The screen was before 2006 a single large vibratory screen with screen aperture on average 30 mm. The large belt cobbing separator has drum diameter 1200 mm and is tasked with eliminating a coarse clean gangue tailings. The brittle magnetite ore breaks to a large extent in transport. This increases the iron grade in the fines from the screen and means that the large belt-cobbing separator has a comparatively easy task to eliminate coarse stones of gangue.

Fines from the GRAM screen and magnetic product from the GRAM separator constitute the feed to four parallel primary sections. Fines from the primary screens are upgraded on dry low-intensity 900 mm drum separators. Coarse from the primary screens goes to two parallel crusher sections. The crusher screens and crusher separators have the same physical dimensions as the primary ones.

Due to the naturally occurring selective ore breakage, the fines from the primary screens consist largely of liberated magnetite particles up to 3 mm in size. They are easily concentrated and go by an autogenous crusher (Barmac Dupactor) for production of sinter fines and special products. Fines from the crusher screens have a larger proportion of mixed particles and go by a Duopactor to the concentrator's grindings sections for production of pellets feed.

The tonnages of the four feed materials entering the cobbled plant do vary:

1. Magnetite ore from the Vitafors shaft is the dominating feed, 11.4 Mt/a in 2003, equivalent to on average 1425 t/h. It has large swings in particle size distribution and head grades.
2. Rock consists of drifting and development rock, and is hoisted with the ore, but is in the simulations treated as a separate stream, since its influence is investigated. In the first simulations runs, very little was known about its particle size distributions and iron content if any.
3. Magnetite ore from the Alliansen shaft is a small amount, 0.5 Mt/a or 62.5 t/h on average. It is slightly coarser and contains more gangue than the other ores.
4. H-PAR is magnetite in a fines fraction that comes from the cobbled of hematite ores. The amount is circa 0.18 Mt/a or 23 t/h.

Of the materials, it is of course the magnetite ores from Vitafors that mostly influence the process, both on particle size and in grades. The latter is also tightly linked to the particle size. The rock gives mostly a dilution of the feed.
grades to the GRAM section and momentaneous load shocks to the GRAM screen. It also increases the load on the crusher sections. The other materials may act disturbing since they are not fed continuously.

Variations in particle size influences the feed grade since the ores break in a way that liberated magnetite grains to a large extent is found below 10 mm, cf Table 1 (Sundvall, 2004). Although the grades in the individual size fractions may vary slightly up and down, it is the feed particle size distributions (Fig. 1) that will change the head grades. From this figure, it is obvious that the elimination of coarse gangue material on the GRAM screen results in a much finer material entering the real cobbng process, i.e. the primary screen. Magnetite from the A-Klionsen shaft is from a particle size point of view similar to the material from feeder 8, while H-PAR from the hematite section is much finer.

<table>
<thead>
<tr>
<th>Fraction/mm</th>
<th>% Fe</th>
<th>Magnetite %</th>
<th>Gangue %</th>
</tr>
</thead>
<tbody>
<tr>
<td>&gt; 60</td>
<td>29.41</td>
<td>40.8</td>
<td>59.2</td>
</tr>
<tr>
<td>30–60</td>
<td>78.07</td>
<td>38.9</td>
<td>61.1</td>
</tr>
<tr>
<td>20–30</td>
<td>79.96</td>
<td>41.4</td>
<td>58.4</td>
</tr>
<tr>
<td>10–20</td>
<td>34.01</td>
<td>47.2</td>
<td>52.8</td>
</tr>
<tr>
<td>&lt; 10</td>
<td>50.28</td>
<td>69.8</td>
<td>30.2</td>
</tr>
</tbody>
</table>

Table 1 Composition of feed to GRAM section (feeder 8) (72 % Fe in magnetite)

All in all together, the variations in feed particle size distributions caused by segregation effects in storage bins, and varying amount of rock dilution, causes the distribution between primary separators and crusher separators to vary, but also gives uneven crusher loads. This is worsened by the fact that when coarser, and thereby more barren, feed is run, the feed rate is increased to maintain production of FAR and PAR. This leads to extra high loads on crushers and crusher screens.

METHODS – SIMULATIONS

Requirements

Computer simulations are done in MODSIM™ (King, 2003) ver 3.6.12. In these, a simplified description of the feed material is used. It is assumed to consist of only two components and two particle classes: magnetic with 91 % magnetite (equiv to 66 % Fe) density 4.7 g/cm³, gangue with 12 % magnetite (circa 9 % Fe) density 3.0 g/cm³. The reason for this simplification is that the simulation models for the low-intensity magnetic separators can only handle two components (magnetic – non-magnetic), and that there is no data available for the mixed particle distributions in the feed. The rock was initially assumed to consist of the non-magnetic component only.

Calibration of sub-circuits

With such a large, although simplified, flowsheet it is not possible to calibrate and validate the whole scheme at once. Instead, it is divided into five major sub-circuits, which are calibrated against full-scale investigations: GRAM section; primary screen – separation – scavenger separation; crusher – screen – separation; FAR Duopactor; PAR Duopactor. Here, only the result for the crusher sub-circuit is reported in detail.
Crusher – screen – separation

The first simulations with a probabilistic screen model gave way too good screen results. The next attempt where the screen was simulated with a classification curve gave a reasonable fit to calibration data. Since the goal was to have a more advanced simulation that took into account the screen load, it was finally decided to simulate it as an over-loaded vibratory screen with just a single deck with the same physical dimensions as the lower-most deck in the Mo-

gensen sizer. In the second simulation round, however, the screen decks are treated independently. For the first simulation round, there were uncertainties in the measured crusher setting, the real CSS, and the measured circulating loads in the crusher circuits. The model parameters, therefore, had to be adapted intuitively. Despite that, the circuit simulations gave fair coincidence with measured data, cf Fig. 2 for flowsheet and Fig. 3 for particle size distributions.

![Flowsheet for crusher screen separation sub circuit](image)

Fig. 2 Flowsheet for crusher screen separation sub circuit

![Cumulative % smaller vs Particle size/mm](image)

Fig. 3 Particle size distributions in the crusher – screen – separation sub-circuit
The particle size distributions reveal a fundamental problem with this circuit – the discharge from crusher has in its steepest part approximately the same particle size as the cut-size of the screen. It means that a small change in crusher setting; a little more moisture in the ore and thereby worse screening efficiency; more rock in the feed to the cobbng plant; they may all cause large fluctuations in the circulating load.

Whole cobbing plant
When the five sub-circuits were sufficiently calibrated, they were merged into a large simulation model for the cobbing plant, cf Fig. 5. In this model, a large number of simulations with varying feed rates, particle sizes and grade distributions were run. It gave the worrying result the desired amount of finished products might be achieved for the next year, but not in the future.

RESULTS – DISCUSSION

First simulation run
The cases deemed to be most interesting were the production prognoses for 2006 (12 Mtpa magnetite ore and 0.9 Mtpa rock) and 2009 (15.7 Mtpa ore and 2.0 Mtpa rock). Simulated material flows in different parts of the process are compared with given or estimated maximum capacities for single units in a combined line and bar diagram, cf Fig. 4. The results showed that the needed throughput in the crushe sections would not be reached for 2009. The bottleneck sections were in order:

1. Primary screens were limiting, since they for moist and fine feed sent too much fine mate-

Fig. 4 Load and capacity diagram for the cobbing plant before 2006

rial with the coarse product. The same thing happened for the cruscher screens, resulting in a build-up of circulating load over the latter.

2. The screen in the GRAM section could not handle the 2009 material flow, and was mechanically and structurally worn out.

3. Cone crushers would 2009 get too large material flows.

4. The scavenging separator was over-loaded in situations where the gangue content of the feed is large.

Second simulation run
For the identified narrow sections, it now become necessary to hurry up with investments. As a first stage during 2006, the GRAM screen and the cruscher screens were replaced for new ones with higher capacity, cf Table 2. At the same time, several samplings are done on sensitive process parts and the feed material better characterised for grade and particle size. The supposedly barren rock is eg found to contain a fair amount of freed magnetite.

With partly new machinery and better simulation data a new round of simulations is initiated, where the sub-circuits are better calibrated. Now it is also of interest, to try to predict the effect of the already decided exchanges of crushers and primary screens that are scheduled for 2007.
Table 2 New machinery in the cobbing plant

<table>
<thead>
<tr>
<th>Unit</th>
<th>Type</th>
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<tbody>
<tr>
<td>GRAM screen</td>
<td>1 Shenker Banana, width 3.15 m, length 9.6 m.</td>
</tr>
<tr>
<td>Primary screens</td>
<td>4 Mogensen Sizer SEL2026-D2, 2 divergent decks, 2 pairs of screens with aperture 35mm and 16 mm.</td>
</tr>
<tr>
<td>Cone crushers</td>
<td>2 Sundvik H6800, MC outer mantle and D inner mantle. Stroke 32mm, 28mm or 24 mm, CSS 15-22 mm.</td>
</tr>
<tr>
<td>Crusher screens</td>
<td>2 Mogensen Sizer SEL2046, 4 pairs of screens with length 2400 mm and apertures 45mm, 35mm, 75mm and 19 mm.</td>
</tr>
</tbody>
</table>

Fig. 5 is shown a total simulation, with all the new machinery, for production according to the 2007 plan. Here, each primary section is fed 425 t/h. Limiting in this flowsheet is a high circulating load over the crushers, which are close to maximum capacity. The crusher CSS is raised to 15 mm, due to a more efficient crushing in the new units. From the scheme, it is also evident that the new GRAM screen (type Banana) is simulated as composed of five smaller screens, and that all screen decks in the Mogensen sizers are now accounted for in the simulation. To reach even higher capacity, the circulating load must be lowered, and that can be done in several ways: lowering of CSS, increase aperture on the lower-most crusher screen decks, or run the crushers choke fed. The latter is an absolute requirement to reach 500 t/h and primary section.

Validation

One interesting aspect of this exercise is what is the effect of the change of crushers on its own? A few comparative simulations are presented in Fig. 6. The curve for Crush2005 is for crusher models and with parameters from the first simulation round. The curve NewChoke is from a crusher model with adaptation to data from a full-scale test run in a H68000 crusher at the Kiruna mine with Malmberget ore. New-Trickle comes from the same model and data, but with the fraction of fines generation per crusher stroke lowered to better fit the old curves. The feed to the crushers is hardly affected by the modifications. It is not until the
crushers (with screens) are exchanged that there is an altered discharge curve from the crushers, with the choke fed crusher generating much more material below 10 mm and in turn generating an easier task for the crusher screens.

During autumn 2007, samplings are made in the new circuits to check whether the changes have given the desired results. In Fig. 6, there is a verification for a large CSS. It is, however, not entirely representative since it was run when the ore held an unusual amount of fine magnetite. A cautious interpretation is that the simulations are slightly of the target, but they have errored on the safe side. In the sampling at hand, the circulation load around a crusher is circa 360 t/h with CSS 19 mm. The feed to a primary section is 425 t/h, and that is the same as in Fig. 5.


**CONCLUSIONS**

This contribution has shown that is possible, by applying a systematic approach, to identify potentially critical sub-circuits and units. It is also possible to “play around” with different scenarios to check how the load is changing with feed particle size. This is especially important for a friable ore in combination with coarse hard development rock that in combination will produce large swings in feed particle size. For the simulations, it must be said that most simulation models today still are very primitive. It is only the screen models that give fairly true capacity relationships. Here, a lot is wished for, especially in improvements of models for magnetic separation.

The rock production, after installation of new crushers and screens, is lower than anticipated with at most 195 t/h compared to the planned and simulated 250 t/h. This is another explanation, why the crusher circulating loads are lower than the simulated for the same CSS.

After the installation of new machinery, some practical problems were encountered that were not possible to simulate for:

- The change to longer screen decks has given much better screen capacity and less sensitivity to blinding caused by unwanted material from the mine.
- An unexpected bottleneck that occurred after the installation of new crushers is that the load on the conveyors below the crushers has to be limited to 450 t/h, instead of the dimensioned 500 t/h. The reason is that the hold-up volume in the new H6800 units is larger than in the old Symons crushers. This means that larger volumes of material are dumped on the conveyor belts as a result of emergency shutdowns, with a stuck conveyor as the consequence.
- Another surprise is that wear on the crusher mantles is so much larger for the new H6800. Nowadays a crusher needs remantling 5-6 times pro year compared once a year for the old Symons crushers. The cause of the huge wear is a slanted load, and that the crushers are trickle fed. This will be remediated with installation of buffer bins during spring 2008.
The simulations had only been academic exercises of dubious value if not the concentrator personnel had critically reviewed the results during the project, and contributed valuable advice and justified criticism. It also made the simulations to have some extra margins, which show in Fig. 6 as errors on the safe side.

REFERENCES


