Ore size does affect direct reduction of
titaniferous magnetite

by M.M. Manamela* and P.C. Pistorius*

Paper written on project work carried out at the University of Pretoria in partial fulfilment of the requirements of the B Tech degree at the Tshwane University of Technology

Synopsis
This work tested whether ore particle size affects the carbon-based pre-reduction of titaniferous magnetite. Pre-reduction is performed at a temperature of some 1100°C in a rotary kiln, in the Highveld Steel process. Laboratory isothermal reduction measurements were performed with ore-coal mixtures at this temperature, to test the effect of ore particle size. The results show that size does limit the degree of reduction for particles larger than 15 mm.

Background
In the Highveld Steel process, titaniferous magnetite (which also contains a substantial amount of vanadium) is pre-reduced in a rotary kiln, using coal as reductant. The residence time in the furnace is some six hours, and about 50% metallization of the iron is achieved in the kiln. Pre-reduction serves to decrease the energy requirement during subsequent electric smelting of the charge. The extent of pre-reduction affects the energy requirement and carbon balance during electric smelting, and hence a stable and high degree of pre-reduction is advantageous.

Previous work indicated that the rate of pre-reduction is controlled by the coal gasification (Boudouard) reaction. However, this conclusion was based on laboratory measurements with ore particles of approximately 6 mm in diameter. The feed to the rotary kiln can contain pieces of ore that are up to 50 mm in diameter. For large ore particles, diffusion of the reduction gas (mainly CO) through the iron product layer into the particles—and the diffusion of the reaction product (CO₂) in the opposite direction—may limit the reduction rate. For full diffusion control, the effect of ore size is strong, with the time for a given degree of reduction being proportional to the square of the particle diameter.

Experimental work
A sample of titaniferous magnetite ore, from the feed to the rotary kiln, was received from Highveld Steel and Vanadium, together with a sample of the coal used as reductant. The ore samples were prepared for reduction experiments by screening the ore into ten size fractions (ranging from -8 mm to +70+50 mm), and the coal by devolatilizing at 900°C. Ore samples from the different size ranges were analysed by X-ray fluorescence (XRF) and X-ray diffraction (XRD), and polished sections examined by scanning electron microscopy with energy dispersive micro-analysis (SEM/EDS). These showed that all size fractions consisted largely of magnetite, with a slightly higher gangue content in the smaller ore particles.

Based on the analyses, and assuming that chromium and vanadium are present in the magnetite in only the trivalent form, and that all titanium is tetravalent, the average composition of the magnetite is approximated by Fe₇₃²⁺Fe₆₇³⁺Ti₀.₃₉⁴⁺Al₀.₁₃³⁺Mg₀.₀₉²⁺V₀.₀₆³⁺Cr₀.₀₁³⁺O₄.

Reduction experiments were performed by maintaining ore-coal mixtures at 1100°C in a vertical tube furnace, under an argon atmosphere. The mixtures were made up by using different ore size fractions, whereas all the coal particles were approximately 6 mm in diameter. The mass ratio of ore to coal in the mixture was 2.9:1. Each mixture was contained in an alumina crucible with an inner diameter of 50 mm; the crucible was suspended in the hot zone of the furnace for reduction. Following reduction for different lengths of time, the crucible was lifted from the hot zone, and cooled under argon. Coal particles were separated from reduced ore particles by hand, and the mass loss of the ore determined to give a measure of the degree of reduction. Most samples were reduced for up to 6 hours.

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Thermocouple-based temperature measurements close to the centre of a few samples showed that the sample reached the set furnace temperature within 1 hour, which is a short period compared with the total reduction time; this means that reduction was not limited by heat transfer into the packed bed under these experimental conditions.

### Results and discussion

#### Reduction mechanism

Examination of partially and fully reduced samples by XRD and SEM/EDS showed that reduction occurred in two steps. In the first step, the magnetite was converted to a mixture of wüstite ('FeO') and ülvospinel (with the approximate composition Fe$_2$TiO$_4$). This first reduction step appeared to occur uniformly throughout the ore particles, and is hence not significantly affected by particle size. Based on the original composition (as given above), this step involves a mass loss of 3.7% (relative to the original ore mass), and a percentage reduction (based on the decrease in the amount of oxygen associated with iron) of 18%.

In the second reduction step, the wüstite which formed during the first step was reduced to metallic iron. For the larger ore particles, a product layer formed in the outer regions of the particles, with unreduced cores (of wüstite and ülvospinel) in the centre. This is the expected morphology if gaseous diffusion through the product layer limits the rate of reduction. Full reduction of the wüstite to metallic iron implies a total mass loss (relative to the original ore mass) of 15%, a percentage reduction (based on oxygen associated with iron) of 72%, and a percentage metallization (that is, the percentage of the iron which is present in the metallic form) of 66%.

No reduction of the ülvospinel occurred even after extended exposure of the samples at 1100°C with unreacted carbon present in the samples. This implies that the maximum degree of metallization that is achievable during pre-reduction is limited by the Fe:Ti ratio in the ore: with a higher titanium content in the ore, more iron would be captured in ülvospinel, with a correspondingly lower maximum degree of metallization. For the ore used in this work, the Fe:Ti ratio was approximately 6.0. This means that, out of every 6 moles of iron that are present in the original ore, approximately 2 moles would be captured in ülvospinel (Fe$_2$TiO$_4$) and a maximum of 4 moles of iron could be reduced to the metal—a maximum percentage metallization of 67% (this is slightly different from the value of 66% which is quoted above, since the latter value was calculated by taking into account the presence of cations other than Fe$^{2+}$ and Ti$^{4+}$ in the spinel).

This reduction sequence is summarized schematically in Figure 1. Photomicrographs of partially and fully reduced samples are shown in Figures 2 and 3.

#### Effect of ore particle size

The observation of a reaction product layer around partially reduced cores suggests that gaseous diffusion limited the reduction rate, which implies that ore particle size should affect reduction. This is confirmed by the results in Figure 4, which give the mass loss of the ore as a function of size. (The sizes plotted in this figure are the averages of the sieved size ranges, which were used in the different runs.) Clearly, larger ore particles show significantly lower mass losses (which correspond to lower degrees of reduction).

Based on the discussion on the reduction mechanism, the maximum mass loss—for full reduction to metallic iron and ülvospinel—is some 15%. However, the smaller ore particles showed larger mass losses, of up to 18.5% (Figure 4), even though no reduction of iron from the ülvospinel could be detected by electron microscopy. The likely reason for the larger mass losses of the smaller-sized fully reduced particles is that their composition was slightly different: the molar ratio Fe:Ti in the smallest size range was 6.2, which is larger than for the larger size range. This means that less iron is captured by the ülvospinel, and hence a larger degree of reduction is possible.

### Conclusion

Carbon-based direct reduction of titaniferous magnetite does...
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depend on particle size: for the typical residence time and kiln temperature of 6 hours at 1100°C, ore particles larger than 15mm in diameter are not fully reduced. The degree of reduction is also limited by the formation of ülvospinel,

\[ \text{Fe}^+ + \text{ülvospinel} \]

\[ \text{"FeO"} + \text{ülvospinel} \]

which captures some of the iron in an unreducible form; ülvospinel formation limits the maximum achievable metallization to some 66%.

Acknowledgements
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References
Tin market expected to be in deficit until at least 2006

New report analyses tin supply and demand worldwide

In 2004, tin performed the best of the six base metals traded on the LME, with prices increasing by almost 150% between August 2002 and May 2004, according to a new report from market analyst Roskill. Tin’s market strength is underpinned by increasing demand, stagnant supply and low inventories, meaning that the market is expected to be in deficit until the end of 2005, with a shortfall of around 15 kt in 2004.

The Economics of Tin (8th Edition, 2004) explains that the fundamentals of the tin market point to continued strength well into 2006, with the price remaining between US$ 8 000 and US$ 9 500/t. There is, however, an argument for considerably higher prices should world economies continue to grow at or about the rates seen in 2004, and tin supply struggles to keep pace.

Possible revival of tin mining in Malaysia and Thailand

The long period of low tin prices resulted in prolonged under-investment and a decline in the tin mining industries of Malaysia and Thailand, although both countries still have considerable refining capacity. The sharp decline in refined tin production in both of these countries in 2003, probably resulted from the restrictions placed on concentrate exports from Indonesia in 2002.

Since these restrictions on unofficial Indonesian exports have been put in place, companies such as Malaysia Smelting Corporation (MSC) have started to acquire interests in Australian and Indonesian tin mining companies in order to secure future supplies of concentrates. However, Malaysia itself is home to some of the largest known tin deposits, representing about 11% of the world total. Therefore, should the increasing price justify it, there are sufficient resources to sustain a significant expansion of the Malaysian tin mining industry.

Large increase in Chinese demand

China is by far the largest market for tin, and accounts for the bulk of the increase in world demand since 2002. In 2003, Chinese consumption had risen to 24% of the world total, followed by that in the USA (14%) and Japan (9%). Chinese tin consumption is expected to total 80 kt in 2004, reducing exports still further in the absence of any major new production. Production of tin in China increased from around 20 kt to 112kt between 1985 and 2000 to become the major producer, during a period when production in most other countries was declining. Production declined in 2002 as the Chinese government enforced its environmental and safety guidelines, which resulted in some twenty thousand small workings being closed. Output recovered in 2003 to 100 kt, but the gap between concentrate production and demand smelters is growing, and the deficit in 2004 was reported to be about 40 kt of tin-in-concentrates.

Solder is now the largest end-use application for tin

The largest end-use application for tin is now in solders, accounting for 36% of total consumption. Tins use in solder, particularly in China where solder alone accounts for 10% of world tin consumption, has grown very rapidly in recent years with the boom in consumer electrical appliances and electronics.

This demand has been greatly enhanced by the move to lead-free solders, backed up by legislation in Europe, Japan and China. Solders that typically contain about 63% tin are being replaced chiefly with solders containing over 95% tin, creating a 35% increase in tin demand (allowing for weight differences) for the same task. The conversion was only about 15% complete in 2004, but is expected to be almost total in Japan and Europe by the end of 2006.

Decline in use of tin in tinplate

The coating of steel for tinplate was once the major end-use application of tin, but now probably ranks after tin chemicals as the third largest use. The tinplate market declined in the 1970s and 1980s, mainly in the USA, but has now stabilized at around 56 kt or 18% of total world consumption.

The single largest use for tinplate, in beverage cans, was taken over completely by aluminium in the USA, though in Europe, Japan and China it still has a major share in the beverage canning market. In addition to aluminium, tinplate competes with plastics and glass in packaging, but is still the dominant material in food and non-food canning. The ease of recycling steel cans has helped to maintain their market share. However, the coating of tin in tinplate has got progressively thinner, and now only makes up about 0.25% by weight of a 33 cl beverage can weighing 22 g.

The Economics of Tin (8th edition, 2004) is available at £1950/US$3900/EUR3415 from Roskill Information Services Ltd, 27a Leopold Road, London SW19 7BB, England. Tel: +44 20 8944 0066. Fax: +44 20 8947 9568. E-mail: info@roskill.co.uk ◆
Bord and pillar applications on the platinum mine: T-cut mining method

by C.C. Oosthuizen*

Paper written on project work carried out in partial fulfilment of Technikon Practical Training

Synopsis

Frank shaft is mining both Merensky reef and UG2 reef. The Merensky orebody is almost depleted and this was the reason for starting to mine the UG2 ore reserves. It was decided to mine the UG2 orebody with trackless equipment using a bord and pillar mining method. Frank shaft has been mining the UG2 orebody for nearly 5 years. The T-cut mining method was introduced to see if it could improve the grade of the run of mine (ROM) as well the extraction percentage.

Introduction

This report follows an in-depth investigation into the T-cut mining method. This investigation was carried out to determine the optimal mining method to increase the run of mine (ROM) grade and extraction percentages in the bord and pillar mining of the UG2 orebody on Anglo Platinum mines. The site of investigation was Frank 1 shaft on Rustenburg Platinum Mine (RPM).

Problem statement

From the middle to the end of 2002, with the sharp weakening of the rand against the major currencies (up to R14/$), major capital expenditures were made by the mine. These expenditures include buying very expensive trackless equipment for mining the UG2 orebody. At this time it seemed to be a very good investment. Due to the sudden strengthening of the rand against the major currencies, the profit margins decreased. From the beginning to the middle of 2003 a good investment turned into a total loss. As the profit margin decreased and the rand strengthened the UG2 pay limit also increased, from 3.1 g/t to 3.4 g/t for platinum group metals (PGM).

Facing the facts of HIV/AIDS in South Africa, Anglo Platinum has carried out an intensified investigation into this problem. The results were shocking, showing that and alarming number of the current workforce is infected with HIV/AIDS. This means that in the next 5 to 10 years Anglo Platinum will lose a large number of its current workforce.

The world’s demand for platinum group metals (PGM) has been identified by Anglo Platinum. The world’s leader in platinum, Anglo Platinum wants to increase its lead in this market. Anglo Platinum announced that it would increase its platinum production to 2.9 million ounces per annum in 2006.1

Objectives

This is an in-depth investigation into the T-cut mining method. The outcome of this investigation is to see if there is a solution to the current problems in the problem statement. The management of Frank 1 shaft decided on the T-Cut mining method because the current bord and pillar development method don not have to be altered. The objective of the T-cut mining method is to increase the UG2 run of mine (ROM) grade to 3.4 g/t, to increase the current productivity of the workforce and to increase the extraction percentage of the UG2 orebody.

Geology

The UG2 orebody comprises a thick chromatite seam between a footwall of coarse crystalline pyroxenitic pegmatite and a hangingwall of pyroxenite. Figure 1 shows the detail of the local stratigraphy with minimum and maximum thickness derived from drill holes. Figure 2 also shows a generalized stratigraphic section with added photographs to clarify the actual underground appearance of the horizons.

Of note in these sections is the ‘stringer parting’ 23 cm above the main seam. This thickness varies from a few centimeters to many tens of centimeters. Mining is unlikely to undercut this, resulting in its inclusion in the evaluation of the mineable orebody.

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The UG2 chromitite is unmistakable in this area and is easily distinguished from the leader by both its thickness and the different footwall lithologies.

The orebody is irregular and randomly disturbed by potholes, rolls and undulations. Departing from any academic definitions of these features, the following practical descriptions are given and their usage is encouraged for the purpose of clear communication:

➤ **Potholes**—these will include any drastic change in the stratigraphic position of the main seam (where it suddenly cuts through its regular footwall rocks) or a change in its elevation that cannot be easily negotiated by a 1.8 m development end. This change in elevation will often be associated with a thinning of the seam, or ‘pinching’

➤ **Rolls**—these will include any change in elevation of the main seam, the amplitude of which is generally greater than the seam thickness. Rolls will generally not be contained within a 1.8 m development end and will cause minor reef losses. Thinning or ‘pinching’ of the seam may not occur, causing the waste thickness to vary considerably with the magnitude of the rolling

➤ **Undulations**—these can be likened to gentle waves, the amplitude of which is generally less than or equal to the seam thickness in that area. Generally, no reef loss is incurred if mining is suitably reactive.

**Mining method**

**Description**

The T-cut mining method is basically a partial pillar extraction method. The pillar (13 m x 13 m) created by the bord and pillar development is being stoped (only the UG2 reef) out for 2 m from the roadways on all sides, to leave an effective pillar of 9 m x 9 m. This method creates the required, higher extraction percentage and higher Run of mine ROM grade.

**Mining layout**

The mining layout is exactly the same as that of the normal bord and pillar layout (see Figure 3). The bords are 4 m wide and pillar sizes are 13 m x 13 m, thus creating centres of 17 m. The stopping width in the development phase is 1.8 m (0.2 m hanging wall, 0.8 m UG2 reef and 0.8 m footwall). The pillar that is created is then stoped out from the roadways for 2 m. The stopping width is only 1 m (0.2 m hanging wall and 0.8 m UG2 reef). When the T-cut is finished, an effective pillar of 9 m x 9 m is left behind. This creates a higher extraction percentage.

**Equipment**

The T-cut mining method does not require much capital. Only two trackless machines are required for the T-cut mining method. Firstly, a diesel-powered Tamrock EJC 115 load haul...
A dumper (LHD) is required. The EJC 115 has a loading capacity of 2.3 m$^3$—or ± 5 tons and a weight of almost 15 tons. The EJC 115 is used to load the broken ore and transport it to the nearest tipping point.

Secondly, an electrical powered Tamrock Micro Scoop E100 is required. The Micro Scoop E100 is a multifunctional trackless machine; it can be used as a LHD or a drill rig, depending on the conversion. The Micro Scoop E100 in the T-cut mining method is used as a drill rig. The Micro Scoop E100 is capable of drilling 15, 2.1 m holes per hour. A 40 kW electrical motor powers it, which drives a series of hydraulic pumps, which is used for tramming and drilling.

**Labour**

The problem that is been creating by HIV/AIDS, a rapid depleting workforce, is a very serious problem in South-African mines. The T-cut mining method creates a partial solution for this depleting workforce, by optimizing labour efficiency.

- 1 x miner
- 1 x LHD operator
- 2 x drill rig operators (day shift, night shift)
- 1 x miner assistance (blasting operations)
- 5 x stope timbers

**Drilling**

The drilling is done with the Micro Scoop E100. The hole must be drilled 90° to the local panel. The burdens are 0.5 m × 0.5 m, with an extra hole in the centre (see Figure 4). The extra hole is to ensure a full 2 m advance. The Micro Scoop is able to drill 15 (2.1 m effective) holes an hour, with a 38 mm diameter.

**Blasting**

The blasting operation is outsourced to Sasol Mining Explosives (SMX), which is in charge of all the blasting.
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This operation is carried out with a mixing and pumping plant, mounted on a Bord Longyear multi purpose vehicle (MPV). The explosives are gassed emulsions. The initiation is done by an EZ-Drifter detonator and a 15 mg Stinger Booster (15 mg PETN). The millisecond delay between the detonators creates a through blast.

Cleaning

Due to the through blast created by the millisecond delay of the detonators, only a small amount (±10%) of the ore is left in the panel. This ore is then blasted out with a Bestline water jet into the roadway. The ore in the roadway is then loaded by the Tamrock LHD and tipped onto the Buffalo- or Continental feeders. The cleaning is only done on the day shift.

Support

The pillar is used as the primary support. Secondary support in the T-cut mining method is to install ~250 mm pencil elongate. The pencil elongate are then pre-stressed to 5 kN (5 ton) with 4” Jackpots. The support pattern is to install the pencil elongate 1.5 m apart and 1 m from the panel face (see Figure 5).

Production results

The T-cut mining only started on the 1 May 2003 at Frank #1. From the first month a large improvement on the ROM grade could be seen. In July only 452 m² could be delivered, due to a major breakdown on the Micro Scoop. As seen from the Table I, the ROM grade also decreased in July.
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Problems
Overbreaking in the development phase is a major problem. When the development breaks over into the pillars, these pillars become too small to mine with the T-cut method due to the pillar strength and pillar safety factor, and are left as is. This then influences the ROM grade. Geological losses, such as rolling reef and potholes within the pillar, leave the pillar unmineable. The geological losses make up 23% of T-cut pillars lost. Breakdowns are also one of the major problems, and the effects can be seen in the July 2003 results in Table I.

Comparison
Table II compares the results of conventional stoping, normal bord and pillar, and bord and pillar with the T-cut mining method, within the UG2 orebody.

Conclusion
The slightly higher cost per ton than for bord and pillar is a very good trade-off for the higher ROM grade and extraction percentage. The high productivity is a very good solution for a depleting workforce. The T-cut mining method is surely the answer to all the problems in the problem statement. The average ROM grade is improved to 3.69 g/t, productivity increased to 108.6 m²/person/month and the extraction percentage increased to 71%. T-cut mining is surely a must in UG2 bord and pillar.

References
Surveying department Frank 1 shaft.
Geology department Frank 1 shaft.
Planning department Frank 1 shaft.

Table I
Production results

<table>
<thead>
<tr>
<th>Month (2003)</th>
<th>Area mined (m²)</th>
<th>ROM grade (g/t)</th>
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<tbody>
<tr>
<td>February</td>
<td>0</td>
<td>3.18</td>
</tr>
<tr>
<td>March</td>
<td>849</td>
<td>3.41</td>
</tr>
<tr>
<td>April</td>
<td>972</td>
<td>3.49</td>
</tr>
<tr>
<td>May</td>
<td>1017</td>
<td>3.61</td>
</tr>
<tr>
<td>June</td>
<td>1195</td>
<td>3.79</td>
</tr>
<tr>
<td>July **</td>
<td>452</td>
<td>3.24</td>
</tr>
<tr>
<td>August</td>
<td>1261</td>
<td>3.84</td>
</tr>
<tr>
<td>September</td>
<td>1278</td>
<td>3.89</td>
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<td>1293</td>
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<td>December</td>
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<tr>
<td>Average</td>
<td>1086</td>
<td>3.69</td>
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Table II
Comparison of mining methods

<table>
<thead>
<tr>
<th>System</th>
<th>Grade (g/t)</th>
<th>Extraction (%)</th>
<th>Cost/ton (R)</th>
<th>Productivity</th>
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<tbody>
<tr>
<td>Conventional</td>
<td>3.98</td>
<td>84</td>
<td>176.12</td>
<td>46 m²/person/month</td>
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<tr>
<td>Bord and pillar</td>
<td>3.10</td>
<td>42</td>
<td>147.00</td>
<td>N/A</td>
</tr>
<tr>
<td>Bord and pillar with T-cut</td>
<td>3.69</td>
<td>71</td>
<td>149.59</td>
<td>108.6 m² person/month</td>
</tr>
</tbody>
</table>

Figure 5—Pencil elongate support pattern

Prestressed Pencil Elongate

1.5m

4m