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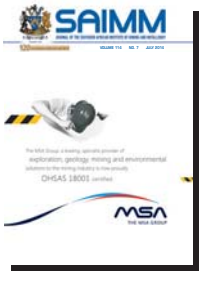

Journal Comment

Physical Beneficiation 2013 was the third conference of its kind, the first being the DMS and Gravity Concentration Conference held in 2006 and the second the Physical Beneficiation 2010 Conference. Physical beneficiation covers dense medium separation (DMS), gravity concentration, magnetic separation, electrostatic separation, and ore sorting –all processes that are widely used in the Southern African mining industry.

A total of 22 papers were presented during the two days of the conference, 19 and 20 November 2013, and covered a wide range of commodities and unit processes within the physical beneficiation field. Two keynote addresses set the stage for each day, with Lionel Falcon and Will Blair sharing their vast knowledge and experience in coal processing and DMS respectively. This issue of the Journal features 10 of the papers that were presented at this conference.

DMS is critical to the operation of coal and diamond beneficiation plants. It is also utilized in iron ore, manganese, chromite, and andalusite beneficiation as well as pre-concentration for platinum group metal (PGM)-bearing UG2 ore. DMS separates ore minerals from gangue on the basis of density difference, making use of a dense fluid that has a density between that of the ore minerals and the gangue. The feed to the DMS plant covers a wide size range, 125 mm to 0.5 mm, and initiatives are under way to extend this down to 200 µm. Separating densities vary from 1.35 for metallurgical coal to 3.8 for iron ore (haematite), but current research is aimed at extending the upper limit beyond 4.

Gravity concentration is by far the oldest form of mineral beneficiation, going back literally thousands of years. As the name implies, this involves the separation of ore minerals from gangue on the basis of density difference. Gravity concentration plays an important role in the processing of diamonds, coal, gold, titanium minerals (ilmenite and rutile), zircon, chromite, iron ore, and manganese. The equipment used in gravity concentration has in the past focused on jigs, spirals, cones, and shaking tables. However, hindered settling classifiers have recently become more prominent, particularly for coal and iron ore.

Magnetic separation is key to the success of DMS by ensuring the efficient recovery and recycling of the dense medium, which consists of magnetite or ferrosilicon. It also features in iron ore recovery circuits and the beneficiation of vanadium-bearing magnetite. Magnetic separation is important in the concentration and separation of ilmenite, rutile, and zircon, and is used for removing iron impurities from industrial minerals. As the name implies, this process entails separation of ore minerals from gangue on the basis of the difference in magnetic response. Low intensity magnetic separation (LIMS), which is used for the recovery of ferromagnetic minerals (such as magnetite), is well established and understood. Wet high intensity magnetic separation (WHIMS), which is used for the recovery of paramagnetic minerals (such as haematite) will assume greater importance in the near future, particularly for the beneficiation of fine (-1 mm) haematite.

Electrostatic separation is based on the separation of conducting minerals from non-conducting minerals. Although it has limited application, it is nevertheless an important unit operation in the concentration and separation of rutile (conducting) and zircon (non-conducting) in heavy mineral beneficiation circuits.

Ore sorting is still largely in the development phase. The presence of waste in ROM ore is unavoidable, and its removal before fine crushing and grinding can result in substantial benefits by increasing the grade of the feed to downstream operations while reducing the amount of material for processing. Ore sorting technology relies on distinguishing the differences in physical and or chemical properties between the ore and the waste, such as colour, density, magnetic response, or chemical composition. Its application promises significant results in the near future. A good example of the successful application of sorting is in diamond recovery, where diamonds are separated on the basis of their fluorescent properties when X rays are targeted on a stream of diamond and gangue particles.

A slight departure from the main theme was the inclusion of three papers on the contribution that screening can make to improving physical beneficiation performance. The first paper highlighted improved dense medium recovery, the second described the more efficient screening of ROM coal with a higher moisture content, and the third covered the use of fine screening of discard coal to recover a saleable product.

The conference was attended by 128 delegates represented mining companies, research institutes, academic institutions, and technology and equipment suppliers. Feedback was very positive, and there was strong interest in a repetition of the event in the future. The conference was generously supported by sponsorship from 14 companies, which further confirmed the significant interest shown in physical beneficiation by the mining industry.

M. Dworzanowski

Beneficiation is a term that has many different meanings within the mining industry and beyond. From an economic perspective, beneficiation relates to adding value to a mined raw material, or the transformation of mined ore into a higher value product that can be consumed locally or exported. In the mining industry, beneficiation is mainly used to describe metallurgical processes that are utilized to upgrade the mined raw material or 'run-of-mine' (ROM) ore.

When describing metallurgical facilities one encounters all sorts of terms – plant, beneficiation plant, treatment plant, washing plant, preparation plant, reduction plant, mill, concentrator, smelter, refinery. Let's see how much sense all these terms make in the context of beneficiation.

A 'plant' could describe any metallurgical facility and thus is far too vague to be of any use. A 'beneficiation plant' is generally used to describe a facility incorporating comminution (crushing and milling) and physical beneficiation (DMS, gravity concentration, magnetic separation, flotation). Beneficiation is a term used to describe any metallurgical process that results in some form of upgrading. Therefore, any metallurgical facility that incorporates some form of upgrading, which is the vast majority, can be termed a 'beneficiation plant'. The general application of 'beneficiation plant' is thus far too narrow and should rather be avoided. 'Treatment plant', as with a 'plant', could be used to describe any metallurgical facility. So this term is also far too vague and should be avoided. 'Washing plant' is generally used to refer to either a wet screening facility or a facility for upgrading coal. Although wet screening could be approximated to washing, the upgrading of coal certainly requires far more than just washing. However, coal metallurgists do like to use the term 'coal washability' to describe the amenability of coal to upgrading. Therefore, the description of coal beneficiation as 'washing' is not ideal, but convincing coal metallurgists to use other terminology will not be easy. Staying on the subject of coal beneficiation, the terms 'coal preparation plant' or 'coal handling and preparation plant' (CHPP) are also commonly employed to describe facilities for coal beneficiation. This is also a strange usage – are we referring to preparing the coal to be used in power stations or blast furnaces? It seems that coal metallurgists do not like the term 'beneficiation'.

In the past, a metallurgical facility for the extraction of gold from ROM ore was called a reduction plant. I have always considered this to be an unusual description. Does 'reduction' refer to the chemical term? In most cases, no, because the gold generally occurs in the metallic form in the ROM ore and is recovered in the metallic form. Does 'reduction' refer to reducing one ton of ROM ore to a few grams of gold metal? This seems more plausible. These days, the metallurgical facility is more commonly referred to as a 'gold plant'. Given the wide array of metallurgical processes used to produce the gold, 'gold beneficiation plant' would provide a more representative description. A 'mill' is another one of those terms that is vague and misleading when used to describe a metallurgical facility. It is sometimes used to denote a facility that incorporates comminution and physical beneficiation. However, extractive metallurgists will normally refer to a comminution device as a 'mill', and physical metallurgists use the term for a facility where steel is fabricated into different forms. Therefore, rather avoid the term 'mill'.

The term 'concentrator' is generally used to describe a metallurgical facility where comminution and physical beneficiation are used to transform ROM ore into a concentrate. As such, this is an appropriate term. The term 'smelter' generally refers to a metallurgical facility utilizing mainly pyrometallurgical processes to produce a matte, an alloy, or a metal. This is also a suitable term. The term 'refinery' is generally used to indicate a metallurgical facility that employs mainly hydrometallurgical processes to produce pure metals – also appropriate usage. While concentrators, smelters, and refineries are all beneficiation plants, these terms do nevertheless provide a representative description of the metallurgical facilities in question.

I do hope that this short review has provided some clarity around the terminology used to describe metallurgical facilities.

M. Dworzanowski
President, SAIMM

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South African National Committee on Tunnelling

Report back on the ITA 2014 General Assembly Fortieth Annual Meeting held at Iguassu Falls, Brazil

The International Tunnelling and Underground Space Association (ITA) held its fortieth annual meeting in Iguassu Falls, Brazil, from 9 to 14 May 2014, in conjunction with the World Tunnel Congress 2014 'Tunnels for a Better Life', organized by the ITA and the Brazilian Tunnelling Committee. More than 1500 persons participated in the conference. The Association registered 12 new Affiliate Members in the preceding year, which resulted in a total of 71 Member Nations and 285 Affiliate Members (taking into account some resignations). Fifty-two of the 71 Member Nations either participated or were represented in the General Assembly. Ron Tluczek, Chairman of SANCOT, represented South Africa on behalf of the SANCOT Committee.

A task force worked during the previous year to clarify the mission, the goals, and actions of the ITA. A new vision statement was defined for the Association, which was presented and approved at the General Assembly, as follows:

'ITA, the leading International organization promoting the use of tunnels and underground space through knowledge sharing and application of technology'.

Seven strategic goals were defined:

1. Consolidate / activate Member Nations – particularly newly joined Member Nations
2. Improve communication and functioning of Working Groups and Committees
3. Expand industry relations
4. Encourage knowledge sharing through education and training
5. Create and develop an ITA Young Members Group
6. Promote the use of underground space
7. Improve communication with Member Nations, industry, and public

The General Assembly voted on and approved the creation of a Young Member Group of the ITA, whose main goals are:

a) To provide a technical networking platform within the ITA for young professionals and students
b) To bridge the gap between generations and to network across all experience levels in the industry
c) To promote awareness of the tunnelling and underground space industry to new generations
d) To provide young professionals and students with a voice in the ITA, including the Working Groups
e) To look after the next generation of tunnelling professionals and to pass on the aims and ideals of the ITA.

The Open Session, which took place on 13 May, was dedicated to 'Underground Space and Natural Resources' with a special focus on mining. It was stated that mining operations are always complicated as they follow the

geology, and the use of ramps, access tunnels and tunnel boring machines (TBMs) is becoming more standard around the world. There are many opportunities and challenges, and both industry and the mining fraternity have to adapt to face these challenges. The experts who participated in the debate advocated mutual co-operation between the civil and mining industries. An arduous task, but one which could bring benefits to both parties.

Sergio Brito, President of BVP Engineering, spoke about the challenges faced by the Brazilian mining industry. He stated that Brazil needs to be more competitive at its copper mining projects, where the majority of the ore is excavated by means of open pit mining. Another challenge that Brazil will have to face is that of underground mining. Currently, about 80% of ore production is derived from open pit mining. A rapid change in this industry is expected over the next 20 to 30 years, due to the decrease in available ore at shallow levels, the need to preserve the environment, and the emergence of new technologies. It is estimated that 50% of the ore production will be obtained from underground mining within 20 years.

South African representatives participated in three Working Groups. Ron Tluczek participated in Working Group 2 (Research), Tony Boniface in Working Group 12 (Sprayed Concrete Use), and Monica Walnstein participated in Working Group 21 (Life Cycle Asset Management). With the assistance of Frank Stevens and Montso Lebitsa, Monica made a presentation to Working Group 21 on some aspects of life cycle asset management in South Africa.

Four reports were published in the previous year:

- 'Refuge Chambers' by ITA Working Group 5
- 'Guidelines on Monitoring Frequencies in Urban Tunnelling' by the ITAtech Activity Group on Monitoring
- 'Guidelines on best practices for segmental backfilling' by the ITAtech Activity Group on Excavation
- 'An engineering methodology for performance-based fire safety design of underground rail systems' by the ITA CUSUF Committee.

All these documents are available free of charge on the ITA website and available for comments.

The forthcoming annual meetings of the ITA General Assembly will be held at the following venues:

- *Dubrovnik, Croatia*: 22–28 May 2015, during the ITA-AITES WTC 2015 'Promoting Tunnelling in South East European Region'.
- *San Francisco, USA*: 22–28 April 2016, during the ITA-AITES WTC 2016 'Uniting our Industry'.
- *Bergen, Norway*: 9–16 June 2017, during the ITA-AITES WTC 2017 'Surface Problems – Underground Solutions'.

From Left to Right: Soren Eskesen (president of the ITA), Tarcisio Celestino (Organizer of the Brazilian World Conference and Vice President of the ITA), Monique Walnstein, Tony Boniface and wife Veronica, Ron Tluczek (SANCOT chairman) and wife Merryn

H.J. Tluczek

Professional registration with the Engineering Council of South Africa (ECSA)

Grow your business by getting more of your engineering staff professionally registered

The wheels of service delivery and the broader economy demand not just more engineering graduates, but more registered engineering professionals, able to take responsibility and make decisions independently, while ensuring the highest level of quality and safety. It is for this reason that organizations need to work towards developing all engineering staff towards professional registration with the Engineering Council of South Africa (ECSA).

Making staff more valuable

It is not difficult to see why professionally registered employees are more valuable to a company. Apart from the positive end-result (registration empowers your staff to sign off on projects), the learning that takes place includes a wide range of essential skills.

Candidates must master – and be able to demonstrate – that they can not only analyse engineering problems but also develop solutions. They must understand and apply advanced knowledge, including general principles and specific aspects of their jurisdiction and local area.

It is critical for businesses that staff must manage engineering activities and communicate clearly with others in the course of doing this. Taking a broad and forward view of what they are doing is also vital: they must develop forward thinking, and address the social, cultural, and environmental impacts of their projects – as well as the legal requirements and health and safety aspects.

ECSA also requires the candidates to demonstrate ethical conduct, and to take responsibility for the engineering decisions they make. All these demands raise the calibre of candidate in the workplace, and therefore uplift the profile of the company – instilling confidence in the employee's ability.

Better returns, lower risk

The skills learnt in the lead-up to professional registration create a strong foundation for the candidate that engineering companies can rely on for years to come in the servicing of their clients' needs– knowing that their staff are legally and professionally capable of tackling complex projects independently, or actively contributing to project teams. This means less time-consuming management, lower risks, and better returns.

Supporting development towards registration ensures that there is a continuous cycle of skills transfer and experience within the business – ensuring that the expertise can be built up over the years and effectively passed down from one generation to the next. How else is a business to remain sustainable and retain its competitive edge?

Developing the profession

The reality is that the engineering profession is one that is at risk, with a scarcity of skills and a dearth of professionals aged 35-55 years. The development of engineering graduates to the point of registration has therefore been identified as a national priority, which requires targets, policy, and funding.

The SETAs are being encouraged to make discretionary funds available to support companies in their efforts to provide structured workplace experience to graduates. ECSA has been lobbying the Department of Higher Education and Training to set national targets, not only for graduation, but for candidacy programmes and to fund the candidate phase. Calls for expressions of interest from SETAs are increasingly reflecting support for graduate internships or candidate programmes.

Candidate programmes are no longer a 'nice-to-have' but are fast becoming key to addressing South Africa's ever-increasing demand for engineering professionals. Candidates are also quick to realize which organizations will invest in them and move there as soon as an opportunity arises.

Organizational commitment

Addressing the problem starts with an organization's commitment to fill their engineering workforce with registered professionals as well as supporting graduates to obtain registration and stay registered.

'Research has confirmed that companies that invest in structured training and mentoring of their graduates record higher levels of productivity from their graduates when compared with their counterparts whose training and mentoring is not structured,' says Edgar Sabela, Acting CEO of ECSA.

Organizations should start by registering a Commitment and Undertaking (C&U) with ECSA, ensure that their graduates register as Candidates, and provide them with appropriate projects, supervisors, and a mentor to oversee and monitor progress. The sooner companies invest in this process the quicker candidates can pick up the workload and take responsibility.

Registration with ECSA is just one step in the continuous development of engineering staff that will allow the business to retain them and plan ahead for succession. Knowing that there is a plan for their advancement also encourages younger staff to contribute fully to the business; if they regard themselves as part of the business's future, you can rely on their commitment.

T. Machimane
Strategic Services ECSA

Obituary

Lynne Wilson Pearson van den Bosch

Lynne van den Bosch, one of this country's most distinguished mining engineers, lived a full, productive and happy life. He died peacefully at his home in the retirement village of Highbrook, Lonehill on 3 June 2014 at the fine age of 93.

Lynne was the third of four children and the only son. He was born in Senekal in the Free State in 1921 to Eelco and Kate (nee Spilsbury) van den Bosch. His father was a bank manager in various towns, including Kestell and Frankfort, finishing his career in Heidelburg where Lynne completed his schooling. He was justifiably proud of being the first member of his immediate family to attend university. He graduated from the University of the Witwatersrand in 1942 with a degree in mining engineering.

Lynne began his lifelong career with Union Corporation in 1943 when he joined Marievale Gold Mine as a learner official. He spent the next 13 years rising through the ranks, working underground at the Marievale, Geduld, and East Geduld mines.

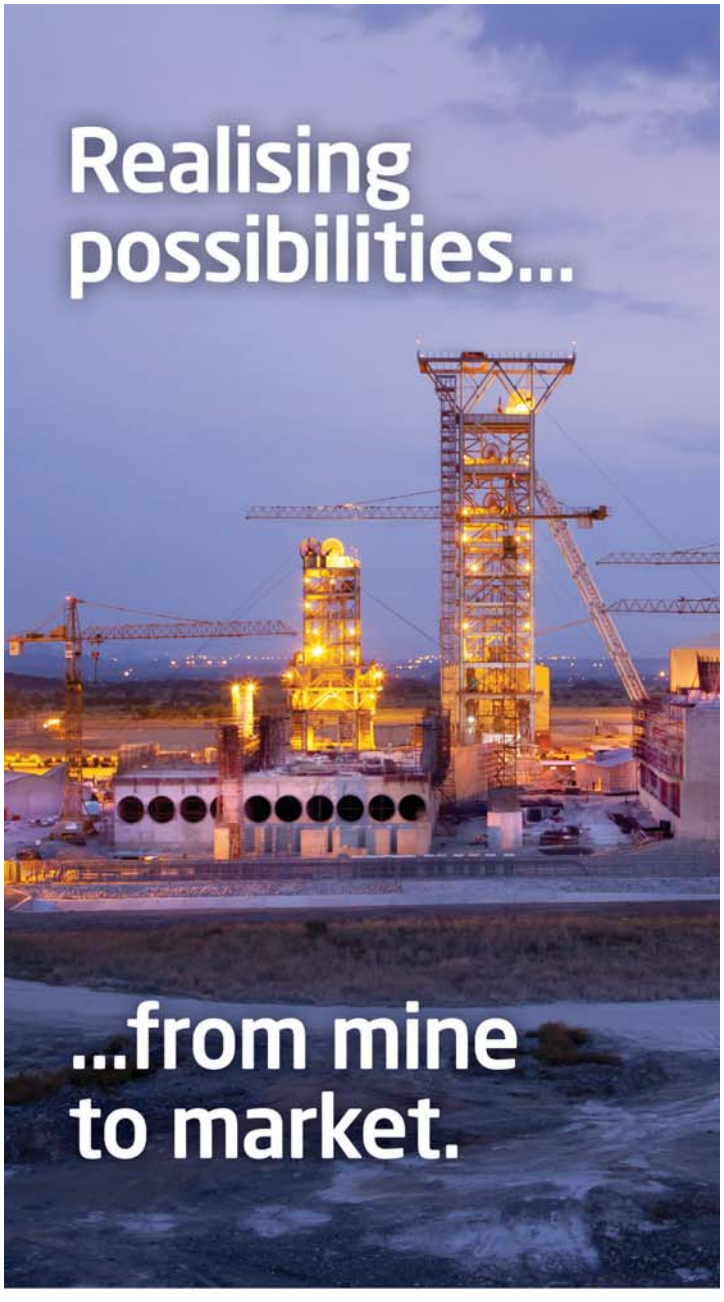
In 1946, he married Thelma Mary Kempthorne. They had two children, Bruce Eelco (born 1951) and Glynis Mary (born 1954). In 1956 Lynne was appointed mine manager underground at St Helena Gold Mine and the family moved from the Springs area to Welkom. In 1961 he took up a position in head office in Johannesburg as technical assistant. The family moved back onto the mines at Leslie in 1964, where Lynne was general manager. Just over a year later, in 1965, he came back to Johannesburg after promotion to consulting engineer. Promotions continued, and Lynne finished his Union Corporation career as the director in charge of all of the company's mines. He retired in 1983.

Lynne also served on the board of the nationally - important Chamber of Mines for many years, and he was elected and served as the Chamber's President twice, once in 1978 and again in 1982. He was a member of South Africa's premier professional mining association, the South African Institute for Mining and Metallurgy, from 1960 and was elected an Honorary Life Fellow in 1990.

These career achievements of this remarkable man tell only a small part of the story. Lynne was an Imposing individual – he was nearly 6 ft 2 in in the old units – with a somewhat stern demeanour which could be intimidating. But underneath this sometimes gruff exterior was a man of immense compassion and integrity. He was absolutely and unflinchingly honest. First and foremost, he was a family man. His devotion to his wife of 67 years, Thelma, was obvious to all. He was proud of both his children and especially proud that both graduated from his alma mater, Wits. He firmly believed that a good education was an important foundation for life. He provided financial support to all six of his grandchildren so that they could attend private schools. All six went on to university – five have graduated, one is still a student.

The most commonly used word to describe Lynne by the people he knew and met was 'gentleman'. Lynne was a true gentleman in the old-fashioned sense of the word. He cared deeply about people and their welfare. One demonstration of this from his working life was that he became an industry champion for mine safety. In his private life he was generous, not just to his immediate and extended family but also to many of the people with whom he came in contact who needed a helping hand. Those who knew him and loved him will forever remember Lynne Wilson Pearson van den Bosch with much fondness and affection.

Contrbuted by Bruce van den Bosch and Glynis Hood



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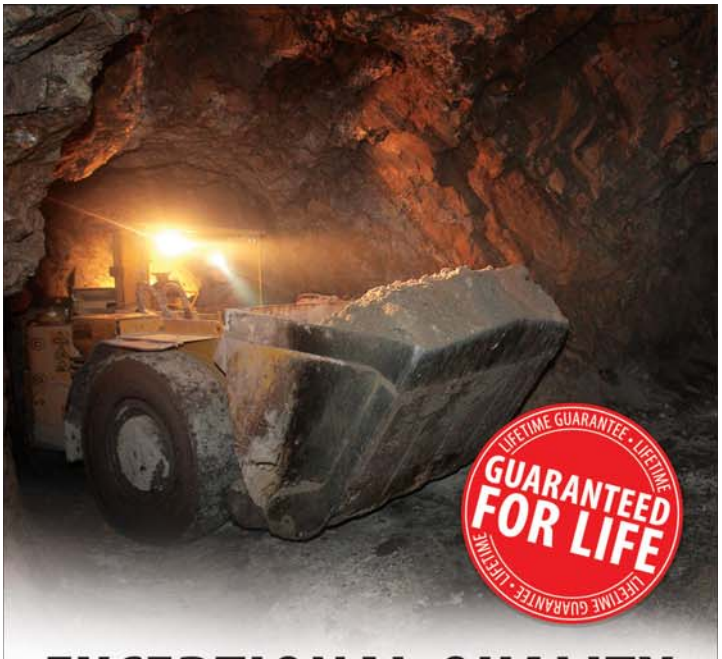

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PAPERS IN THIS EDITION

These papers have been refereed and edited according to internationally accepted standards and are accredited for rating purposes by the South African Department of Higher Education and Training

Physical Beneficiation

Production of pelletizing concentrates from Zandrivierspoort magnetite/haematite ore by magnetic separation
by P. Muthaphuli 505

A number of piloting campaigns were conducted to generate design data, validate the process flow sheet and demonstrate process performance for beneficiation of a low-grade magnetite-haematite ore. A product was obtained that compares favourably to other leading direct reduction (DR) pelletizing concentrates.

The Lamella High Shear Rate REFLUX™ Classifier
by T. Orupold, D. Starr, and T. Kenefick 511

This paper describes the commercial development of gravity separation of fine particles using a Lamella High Shear Rate REFLUX™ Classifier, with a specific focus on coal applications. Advances in design have led to a significant improvement in gravity separation performance.

The application of Baleen Filter microscreening technology at BECSA's South Export Plant
by S. Tshikhudo and V. Shikwambana 519

This paper describes the performance of the Balleen filter for the recovery of saleable fractions of coal from a washing plant effluent containing a nominally -150 μm material. It was found that the Baleen Filter could screen at an acceptable efficiency with a very sharp cut-point and better-than-predicted yields.

Positron emission particle tracking inside a laboratory batch jig
by W.P. Roux and N. Naudé 525

There is a need for better understanding of the jigging process to improve the recovery efficiency of finer, lower grade iron ore material. The technique of positron emission particle tracking (PEPT) was applied to study the motion of iron ore particles inside a laboratory batch jig. From the results, detailed information on the stratification rate of a particle was obtained, with adequate resolution to track the particle's movement through an individual pulse.

The art and science of dense medium selection
by J. Bosman. 529

Medium suspensions play an integral part in the successful application of dense medium separation for both static and dynamic separators. Owing to the non-Newtonian nature of medium suspensions, viscosity measurements cannot be used in existing models, and as such, medium selection remains an art based upon practical experience and indirect measurements of viscosity.

Current trends in the development of new or optimization of existing diamond processing plants, with focus on beneficiation
by P. van der Westhuyzen, W. Bouwer, and A. Jakins 537

This paper identifies some recent technology advances in diamond recovery and demonstrates how these could be considered as a replacement of, or in combination with, conventional technologies to arrive at an optimum techno-economic solution for the new generation of diamond processing plants.

A simple framework for developing a concept beneficiation flow sheet
by J. Rabe 547

This paper works through the metallurgical concept study process, from identifying suitable process routes to constructing a concept flow sheet and mass balance.

Maximizing haematite recovery within a fine and wide particle-size distribution using wet high-intensity magnetic separation
by M. Dworzanowski 559

Wet high-intensity magnetic separation (WHIMS) bench scale test work was conducted on a haematite-rich material that contained a 60% -10 μm fraction. The -200 +10 μm and -10 μm fractions were treated separately and together under varying WHIMS conditions. It was concluded that for a given concentrate grade, the mass yield obtained was greater when the total particle size distribution was treated.

Operation and performance of the Sishen jig plant
by H.A. Myburgh and A. Nortje 569

The beneficiation of stockpiled low-grade iron ore as part of the Sishen expansion project is described. During commissioning and ramp-up, a number of changes were made to optimize jigging performance. Based on improved knowledge of the operating principles the production target was achieved.

These papers will be available on the SAIMM website
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Production of pelletizing concentrates from Zandrivierspoort magnetite/haematite ore by magnetic separation

by P. Muthaphuli*

Synopsis

Kumba Iron Ore's Zandrivierspoort (ZRP) magnetite-haematite project aims to mine and beneficiate a magnetite resource with low contaminant levels to produce from 1 Mt/a to 2.5 Mt/a product, which will be either a concentrate, micro-pellets for direct reduction, or blast furnace pellets. Anglo American Technical Solutions Research conducted a number of piloting campaigns to validate the process design flow sheet, demonstrate process performance, and to generate data for use as a design basis. Secondary to this, the pilot plant was to produce approximately 7 t of magnetite-haematite concentrate with low enough SiO_2 to be used as feedstock for pelletizing studies.

The pilot plant achieved 34.2% mass yield, at 69.0% Fe, 2.2% SiO_2, from an IsaMill grind of 80% -45 μm. This product quality will compare favourably to other leading direct reduction (DR) pelletizing concentrates. However, two-stage comminution to 80% -75 μm and beneficiation of ZRP ore will produce a blast furnace (BF) quality pelletizing concentrate, indicative quality being 67.6% Fe and 4% SiO_2, at higher mass yields. An economic evaluation between producing a DR and a BF pelletizing concentrate will have to be conducted prior to finalizing the concentrate production flow sheet..

Keywords
Magnetic separation, iron ore, magnetite, haematite.

Introduction

Kumba Iron Ore is looking to grow its production from the current 44 M/ta to 70 M/ta by 2019. Of this planned growth, approximately 6 Mt/a is scheduled to come from projects in South Africa's Limpopo Province. The Zandrivierspoort (ZRP) project aims to mine and beneficiate a magnetite resource with low contaminant levels to produce from 1 Mt/a up to 2.5 Mt/a product in the form of either a concentrate, micro-pellets for direct reduction, or blast furnace pellets. Anglo American Technical Solutions Research conducted piloting campaigns with a bulk sample obtained from the ZRP site. Kumba Iron Ore is currently designing a plant that will process the ZRP magnetite-haematite ore-body. Anglo American Technical Solutions Research conducted these piloting campaigns to validate the process design flow sheet, demonstrate process performance, and to generate data for use as design basis. Secondary to this, the pilot plant was to produce approximately 7 t of magnetite-haematite concentrate with low enough SiO_2 to be used as feedstock for pelletizing studies at Kumba's Value in Use Laboratory and elsewhere. The piloting campaign entailed crushing the run-of-mine to -1 mm prior to low-intensity magnetic separation (LIMS) and rare earth drum separation (REDS) for haematite scavenging, ball milling in closed circuit with a vibrating screen, cleaning of the ball mill circuit product through three stages of LIMS, IsaMilling the ball mill LIMS concentrate, and further re-cleaning of the IsaMill product through three LIMS stages. A combination of LIMS and REDS in different parts of the circuit was chosen to recover magnetite and liberated haematite respectively. Magnetite is ferromagnetic, thus it can be effectively recovered using LIMS (800–2000 G) while haematite is paramagnetic, thus requires high magnetic field intensities (>6000 G) (Wills and Napier Munn, 2006). This paper presents interpretations of the feed sample characterization, pilot plant mass balance results, and the implications for the proposed ZRP plant flow sheet.

Background

ZRP geological outline

The Zandrivierspoort prospecting right is located approximately 25 km northeast of Polokwane on the farm Zandrivierspoort 851 LS, in South Africa's Limpopo Province, as shown in Figure 1 (Kumba Iron Ore, 2013). The low-grade magnetite project mineral resource base is estimated at 476.1 Mt with 34.5% Fe (Kumba Iron Ore, 2013). Zandrivierspoort is a banded iron formation

* *Anglo American Mining and Technology, Technical Solutions Research.*

© *The Southern African Institute of Mining and Metallurgy, 2014. ISSN 2225-6253. This paper was first presented at the,* Physical Beneficiation 2013 Conference*, 19–21 November 2013, Misty Hills Country Hotel and Conference Centre Cradle of Humankind, Muldersdrift.*

Production of pelletizing concentrates from Zandrivierspoort magnetite/haematite ore

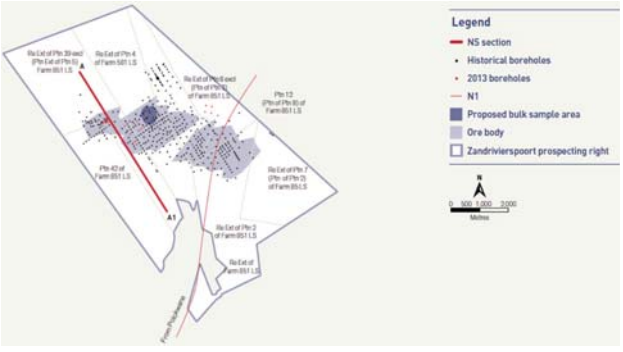

Figure 1—Zandrivierspoort prospecting right area north of Polokwane in Limpopo Province (Kumba Iron Ore, 2013)

(BIF) magnetite deposit in the palaeoproterozoic Rhenosterkoppies greenstone belt or Rhenosterkoppies fragment, which occurs to the northwest of the main, northeast-trending Pietersburg greenstone belt. Figure 2 represents a vertical profile (slice) intersecting the three-dimensional model of the Zandrivierspoort deposit (A-A1 red line in Figure 1), demonstrating the company's interpretation of the relationship between the orebodies, waste material, and local geology (Kumba Iron Ore, 2013).

Bulk sample preparation and characteristics

Kumba Iron Ore delivered approximately 38 t of run-of-mine (ROM) bulk sample at a 150 mm top size. This was reduced using a jaw crusher to 100% passing 70 mm. The jaw crusher product was further crushed to 100% passing 40 mm prior to comminution by high-pressure grinding roll (HPGR). The HPGR crushing was conducted at Mintek using a Koppern HPGR unit to produce 100% passing 1 mm product, which yields more than 60% liberation of the magnetite and haematite combined. On average, the bulk sample received contained 35.7% Fe with a 40:60 magnetite to haematite ratio. Figure 3 shows the bulk mineralogical composition of the bulk sample processed through the pilot plant.

Previous company internal data indicated that the average ZRP orebody contains approximately 40.8% magnetite with a 70:30 magnetite to haematite ratio (Kumba Iron Ore, 2013). The implications of the lower magnetite to haematite ratio of this bulk sample on the pilot plant mass yield will be highlighted and discussed under the mass balance results.

The Bond Ball Mill Work Index (BWI) was determined at a limiting screen of 75 μm. The BWI test was carried out in accordance with the modified Warren Spring Laboratory method. The BWI of this ZRP ore was measured at 17.6 kWh/t, which is in line with previous values obtained from other hardness testing carried out on ZRP drill cores at Anglo Research (van Drunick, 2007). A grind of 80% passing (P_{80}) 75 μm was chosen targeting a blast furnace pelletizing concentrate with 67% Fe and less than 5% SiO_2, while a 45 μm grind was targeting a direct reduction pelletizing concentrate with a much cleaner product containing 69% Fe and less than 2% SiO_2. Batch grinding and Davis tube tests at 1000 G (Murariu and Svoboda, 2003) had shown that these qualities were achievable at P_{80} of 75 μm and 45 μm.

Piloted flow sheets

High-SiO_2 concentrate production (blast furnace pelletizing concentrate)

Figure 4 illustrates the first phase of the piloting campaign, in which the circuit was operated with a cobbing and a REDS scavenger magnetic separator, the ball mill in closed circuit with a 75 μm screen, and the product of the ball mill circuit reporting to three stages of LIMS cleaning. The ZRP ore consists of magnetite and haematite, a significant amount of

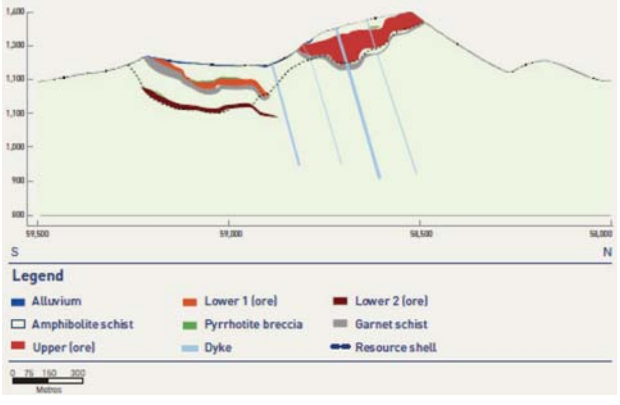

Figure 2—Southeast-northwest profile depicting the local geology through the Zandrivierspoort magnetite deposit (Kumba Iron Ore, 2013)

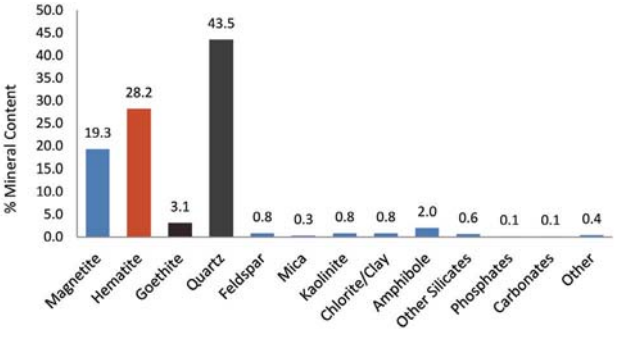

Figure 3—Bulk mineral composition of the ZRP bulk sample used for piloting

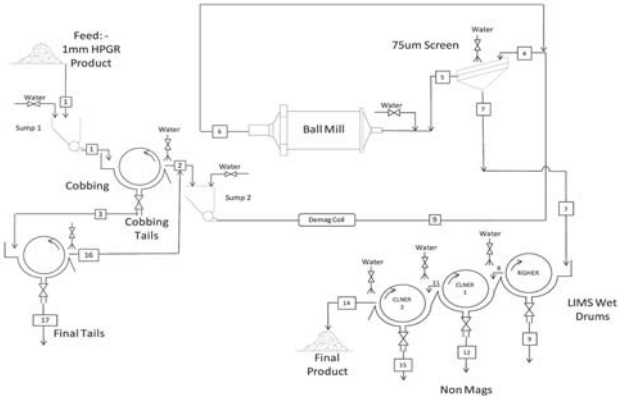

Figure 4—Flow sheet for production of a BF pelletizing concentrate

which has been shown to be intimately associated with the magnetite. The product of the REDS scavenger was combined with the cobbing product to feed the ball mill circuit.

The addition of a REDS scavenger improves yield due to the recovery of additional mass of fully liberated haematite which is not recovered across the cobbing LIMS because of its lower magnetic susceptibility.

Low-SiO_2 concentrate production (direct reduction pelletizing concentrate)

The blast furnace pelletizing concentrate was sent to the Anglo American Platinum Division Metallurgical Laboratory (DML) for IsaMilling using a 100 litre (M100 IsaMill) pilot plant unit with 75 kW installed power. Davis tube test work data showed that it was possible to reduce the SiO_2 content in the concentrate to around 2% from a grind of 80% -45 μm.

The IsaMill product was re-pulped using a ball mill to break lumps and allow for dilution prior to the three LIMS cleaning stages as illustrated in Figure 5. This circuit was run at very low feed densities as its primary objective was to produce a high-quality concentrate that is low in SiO_2. Thus it was critical to avoid SiO_2 entrainment, which would have been worsened by high feed densities.

Equipment specification

Ball mill and screen

Table I and Table II summarize the ball mill and screen specifications respectively. The ball mill had rubber liners. The screen was a retro-fit of a linear screen that was modified and fitted with a 75 μm woven wire cloth due to the lack of pilot Derrick screens in the country at the time of the test work. The screen deck was re-designed and fitted in-house.

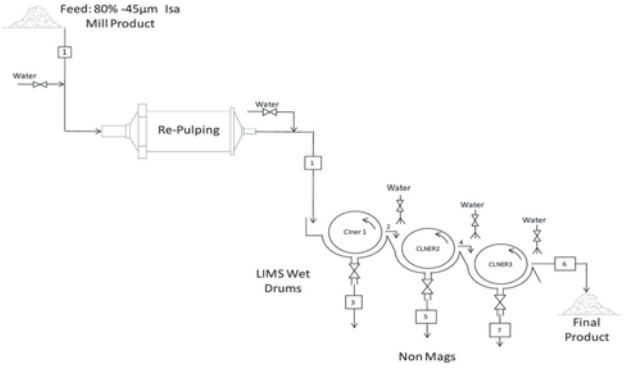

Figure 5—Re-cleaning of high-SiO_2 concentrate after IsaMilling in Rustenburg

Magnetic separator specifications

The four LIMS units were salvaged from an old magnetic separation plant and refurbished in-house with assistance from Eriez Magnetics. The REDS was hired from Multotec. Table III summarizes the magnetic separator specifications and operational settings employed. All the LIMS had a concurrent tank configuration, while the REDS had a counter-rotation configuration.

The drum-tank clearance for both the LIMS and REDS was as narrow as mechanically possible to ensure high magnetite and haematite recovery (Dworzanowski, 2010). For the REDS, the drop in magnetic field strength away from the drum surface is very significant, thus a need for very narrow drum-tank clearance.

Mass balance results and discussion

High-SiO_2 concentrate production

Primary LIMS cobbing

Blast furnace (high-SiO_2) concentrate production involved running the circuit as illustrated in Figure 4. The plant was fed at 714.5 kg/h at a head grade of 35.7% Fe. The average plant feed magnetite to haematite ratio was 1:1.5 (or 40.1% to 59.9%). Table IV summarizes the overall circuit mass balance. Mass balancing was conducted using the least-square error technique adapted from JKSimMet and

Table I

Summary of ball mill specifications

Ball mill	Value
Effective grinding length	1 200 mm
Internal diameter	960 mm
Ball diameter	40 mm
Steel load	35 %
Critical speed	70 %
Installed motor power	30 kW

Table II

Summary of ball mill screen specifications

Ball Mill Screen	Value
Screen length	2 100 mm
Screen width	96 mm
Screen panel aperture opening	75 μm

Table III

Summary of drum magnetic separator specifications

Wet drum magnetic separators	Cobbing LIMS	1st stage LIMS	2nd stage LIMS	3rd stage LIMS	Cobbing REDS
Drum diameter mm	915	915	915	915	600
Drum width mm	610	610	610	610	450
Drum-tank clearance mm	15	15	15	15	10
Drum-discharge lip clearance mm	15	15	15	15	10
Magnetic field strength at drum surface, G	1 500	1 500	1 000	1 000	7 000

JKSimFloat (JKTech, 2001). Mass balance adjusted flow rates and assays compared well with measured data from the piloting work, with standard deviations less than 5% (Table IV).

The LIMS cobbing stage operated at 4.03 m³/h per metre drum length. This stage achieved a mass yield of 57.5% with 1.47 times Fe upgrading. Overall mass rejection was 42.5%, which was expected given the significant liberation at a 1 mm top size. Even though the cobbing stage rejected 42.5% of the mass in the feed, the magnetite recovery across this stage was 98.4%, while the haematite recovery was 74.6%. Again, the high haematite losses confirm a need for haematite scavenging at a higher magnetic field strength from the cobbing tails.

Cobbing scavenger (REDS)

The cobbing tails were re-treated in the cobbing scavenger. Mass balance showed that this drum received 42.5% of the mass that was fed to the plant at 14.3% solids and slurry feed rate of 4.26 m³/h per metre drum length. The cobbing scavenger recovered 3.5% mass relative to plant feed while recovering 38.4% of the magnetite and 23.8% of the haematite in the cobbing tails. The concentrate from this scavenger contained 38.6% Fe with 44.7% SiO_2. Overall, the scavenger improved the haematite recovery from 74.6% to 80.6% before ball milling. Mineralogical analysis of the cobbing scavenger feed showed that only 4.2% of the haematite in this stream was fully liberated, with approximately 44.8% occurring with 80–95% liberation by surface area. The poor haematite liberation explains the low haematite recoveries across the cobbing scavenger.

Ball mill circuit

The LIMS cobbing stage and the cobbing REDS scavenger stage concentrates, together with the ball mill discharge, were fed to the screen. The combined stream feeding the screen represented 114.8% of the plant feed. Around 53.1% of the screen feed went to the screen undersize. The screen oversize/ball mill feed was measured at 384.4 kg/h. On average, the ball mill power draw was 7.55 kW, which translates to a 19.6 kWh/t power consumption. Figure 6 summarizes the particle size distribution (PSD) around the ball mill circuit, which was operated at 76.9% solids by mass discharge and 70% critical speed.

Due to the inefficiencies of an in-house built screen, the ball mill circuit product was much coarser at 66.8% passing 75 μm compared to the targeted 80% passing 75 μm. This poor grind resulted in poor liberation of magnetite from silica, which had an impact on the concentrate grade as will be discussed in the next section. However, the product from this campaign was scheduled for IsaMilling to 80% -45 μm prior to secondary cleaning.

Primary LIMS cleaning stage

The primary LIMS cleaning stage consisted of three drums in series; which were rougher, cleaner, and re-cleaner as shown on the flows sheet in Figure 4. This stage was fed by the ball mill screen undersize at 436.1 kg/h, representing 61.0% of the mass of the plant feed, with a 49.7% Fe grade.

The rougher LIMS was operated at 7.64 m³/h per metre drum length at 8.7% solids by mass. Overall, the rougher LIMS stage rejected 19.3% mass out of the 61.0% that was in

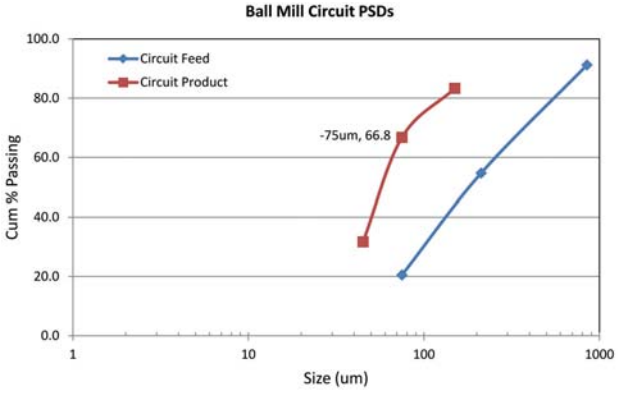

Figure 6—PSD of ball mill circuit feed and product

Table IV

Campaign 2 mass balance results

Stream name	Solids		Fe		SiO_2		Magnetite		Haematite		Al_2O_3	
	kg/h	% Distr	Grade	% Distr	Grade	% Distr	Grade	% Distr	Grade	% Distr	Grade	% Distr
Cobbing Feed	714.5	100.0	35.7	100.0	46.1	100.0	20.2	100.0	30.1	100.0	0.37	100.0
Cobbing Conc	410.8	57.5	52.3	84.4	23.1	28.8	34.5	98.4	39.1	74.6	0.22	34.7
Cobbing Tails	303.7	42.5	13.1	15.6	77.3	71.2	0.8	1.6	18.0	25.4	0.56	65.3
Scav Conc	25.3	3.5	38.6	3.8	44.7	3.4	3.5	0.6	51.5	6.0	0.24	2.3
Scav Tails	278.5	39.0	10.8	11.8	80.2	67.8	0.5	1.0	14.9	19.3	0.59	63.0
Comb Cobbing and Scav Conc	436.1	61.0	51.5	88.2	24.4	32.2	32.7	99.0	39.8	80.7	0.22	37.0
Screen Feed	820.5	114.8	50.7	163.1	25.9	64.5	32.2	183.7	39.1	148.9	0.19	58.9
Screen O/S	384.4	53.8	49.7	74.9	27.6	32.2	31.7	84.7	38.2	68.2	0.15	21.9
Ball Mill Product	384.4	53.8	49.7	74.9	27.6	32.2	31.7	84.7	38.2	68.2	0.15	21.9
Rougher Feed (Screen U/S)	436.1	61.0	51.5	88.2	24.4	32.2	32.7	99.0	39.8	80.7	0.22	37.0
Rougher Conc	298.5	41.8	63.8	74.8	9.0	8.1	46.8	97.1	42.8	59.4	0.11	12.5
Rougher Tails	137.6	19.3	24.8	13.4	57.7	24.1	2.0	1.9	33.4	21.3	0.47	24.5
Cleaner 1 Feed	298.5	41.8	63.8	74.8	9.0	8.1	46.8	97.1	42.8	59.4	0.11	12.5
Cleaner 1 Conc	289.6	40.5	64.4	73.2	8.3	7.2	48.2	96.9	42.3	56.9	0.10	11.4
Cleaner 1 Tails	8.9	1.2	44.4	1.6	33.2	0.9	3.8	0.2	59.6	2.5	0.32	1.1
Cleaner 2 Conc Feed	289.6	40.5	64.4	73.2	8.3	7.2	48.2	96.9	42.3	56.9	0.10	11.4
Cleaner 2 Conc (Final Conc)	281.	39.3	64.9	71.6	7.7	6.5	49.5	96.6	41.6	54.3	0.10	10.7
Cleaner 2 Tails	8.4	1.18	48.8	1.62	27.6	0.71	4.7	0.28	64.9	2.54	0.22	0.72

the feed, while achieving a 1.24 times Fe upgrading ratio. The rougher concentrate contained 63.8% Fe with 9.0% SiO_2. This stage achieved a 98.1% magnetite recovery and a 73.6% haematite recovery. Thus, relative to the plant feed, 1.9% magnetite and 21.3% haematite was lost from the rougher LIMS stage.

The primary cleaner LIMS received feed from the rougher LIMS. This drum rejected only 1.2% out of the 41.8% mass that constituted the feed, achieving 1.01 times Fe upgrading ratio. The SiO_2 grade across this stage dropped from 9.0% to 8.3%. This primary cleaner LIMS stage achieved 99.8% magnetite recovery and 95.8% haematite recovery. The high haematite recovery shows a strong mineral association between magnetite and haematite.

The primary recleaner LIMS treated the concentrate from the cleaner LIMS. This last stage upgraded the Fe content from 64.4% to 64.9% while rejecting 1.18% mass out of the 40.5% that fed the stage. This recleaner LIMS stage achieved 99.7% magnetite recovery and 95.5% haematite recovery. Based on these results, it is clear that LIMS scavenging will be required only on the rougher tails, which contains 1.9% of the ROM magnetite. Overall this campaign achieved a 39.3% mass yield at an Fe grade of 64.9%, with 7.7% SiO_2 and 0.10% Al_2O_3. The low Fe grade and high SiO_2 can be attributed to poor liberation as a result of a coarser grind. The final concentrate contained 49.5% magnetite and 41.6% haematite.

Low-SiO_2 concentrate production

IsaMilling of the high-SiO_2 concentrate

Part of the concentrate (approximately 10%) from campaign 1 was milled to 90% passing 45 μm. The M100 IsaMill was operated at 80% mill filling, 1.34 t/h throughput, and a feed SG of 1.40. Under these conditions the IsaMill specific energy was 22.6 kWh/t. The rest of the campaign 1 concentrate was milled at a reduced mill filling of 60%, operating at 1.82 t/h throughput with a feed SG of 1.46. With these conditions, the IsaMill specific energy was 13.6 kWh/t. The IsaMill achieved a grind of 80% -45 μm. Figure 7 shows the difference in grind under the two operating conditions.

Cleaning of the 80% -45 μm IsaMill product

This run involved cleaning of campaign 1 concentrate after IsaMilling. This circuit consisted of three stages of LIMS. Mineralogical analysis was conducted prior to this campaign to ascertain the degree of liberation and feasibility of producing a concentrate suitable for direct reduction quality pellets. Table V summarizes the mineralogical results with respect to mineral association by size on the IsaMill product.

Mass balance results as summarized in Table VI show that the three stages achieved 90.2% mass yield (34.2% relative to ROM) and reduced the SiO_2 content from 6.56% to 2.25%. The 9.8% mass rejection is in line with the indications from the mineral association information in Table V.

As per Table V, most of the haematite that was recovered from campaign 1 (up to primary LIMS at -75 μm) was still associated with the magnetite even after IsaMilling to 45 μm. This is further confirmed by the high haematite recovery of 89.2% as indicated in Table VI. Given the high recoveries of magnetite and haematite, undertaking a test work campaign for the scavenging of the secondary LIMS tailings after IsaMilling was not considered worthwhile.

The ZRP concentrate at 80% -45 μm compares well with other DR pellet concentrates due to its low impurities levels, as summarized in Table VII.

Conclusions

- The pilot plant achieved 34.2% mass yield, at 69.0% Fe, 2.25% SiO_2 from a grind of 80% -45 μm. The lower overall mass yield of 34.2% compared to the expected 40% is a result of the low magnetite and high haematite content in the bulk sample

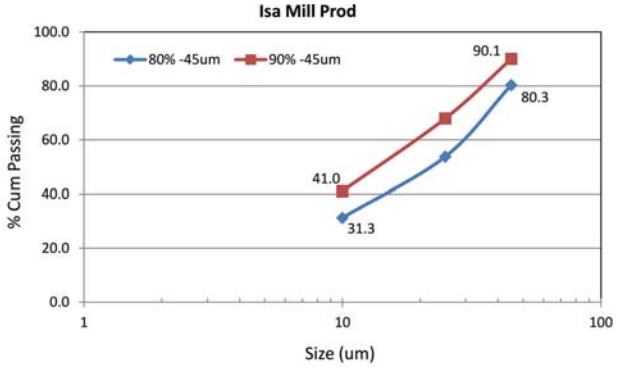

Figure 7—IsaMill product particle size distribution with different operating conditions

Table V

Mineral association data of the Isa Mill product size fractions at 80% -45 μm

Degree of liberation	ZRP final conc after Isa Milling				
	+45 μm	+25 μm	+10 μm	-10 μm	Total
Fully liberated haematite	-	-	-	0.3	0.3
Fully liberated magnetite	-	-	-	1.4	
Liberated silicates	4	2	0.7	2.6	9.3
Magnetite+haematite	13.3	22.4	19.1	16.5	
Magnetite+silicates	0.1	0	-	0.1	
Haematite+silicates	0.3	0.2	0.1	0.6	1.2
Magnetite+haematite+goethite	0.4	1.1	1.9	7.5	
Magnetite+haematite+silicates	1.1	0.5	0.3	1.1	
Barren	0.1	0.1	0.1	0.3	0.5
Other	0.4	0.3	0.4	1	2
Most likely to be rejected on cleaning					13.2

Table VI

Campaign 4 mass balance results

Stream name	Solids		Fe (Total)		SiO_2		Magnetite		Haematite		Al_2O_3	
	kg/h	% Distr	Grade	% Distr	Grade	% Distr	Grade	% Distr	Grade	% Distr	Grade	% Distr
Cleaner 1 Feed	260.5	100.0	65.8	100.0	6.56	100.0	50.5	100.0	40.2	100.0	0.13	100.0
Cleaner 1 Conc	241.3	92.6	68.7	96.8	2.56	36.1	54.3	99.5	40.5	93.4	0.08	61.3
Cleaner 1 Tails	19.2	7.36	28.6	3.2	56.9	63.9	3.10	0.45	36.2	6.62	0.66	38.7
Cleaner 2 Feed	241.3	92.6	68.7	96.8	2.56	36.1	54.3	99.5	40.5	93.4	0.08	61.3
Cleaner 2 Conc	235.7	90.5	69.0	94.9	2.27	31.3	55.3	99.1	39.8	89.6	0.08	57.0
Cleaner 2 Tails	5.67	2.18	58.2	1.9	14.6	4.86	10.29	0.44	69.8	3.78	0.25	4.26
Cleaner 3 Conc Feed	235.7	90.5	69.0	94.9	2.27	31.3	55.3	99.1	39.8	89.6	0.08	57.0
Final Conc	235.1	90.2	69.0	94.7	2.25	31.0	55.4	99.1	39.7	89.2	0.08	56.7
Cleaner 3 Tails	0.58	0.22	61.8	0.21	9.82	0.34	11.4	0.05	73.7	0.41	0.18	0.31

Table VII

ZRP concentrate specification at 80%-45 μm after IsaMilling

Product specification	Fe %	SiO_2 %	Al_2O_3 %	K_2O %	P %	Mn %	CaO %	MgO %	TiO_2 %	S %	V_2O_5 %	PbO %
Final concentrate (80% -45 μm)	69.0	2.25	0.08	0.01	0.012	0.13	0.01	0.04	0.01	0.008	0.00	0.006

- The ZRP concentrate is very low in impurities, containing 2.25% SiO_2, 0.07% Al_2O_3, 0.01% K_2O, 0.01% P, 0.13% Mn, and virtually no TiO_2 and V_2O_5
- The primary LIMS product could be sold as a blast furnace pelletizing concentrate or could be processed further. If so, the secondary LIMS product after IsaMilling could be sold as a direct reduction pelletizing concentrate
- The cobbing stage achieved a mass yield of 57.5% with a 1.41 times Fe upgrading ratio. The ROM mass rejection was high (42.5%), but this was due mainly to the high haematite content of the bulk sample
- Even though the cobbing stage rejected 42.5% of the mass fed, the magnetite recovery across this stage was very good at 98.4%, while the haematite recovery was low at 74.6%. These results suggest a need for haematite scavenging from the cobbing tails at a higher magnetic field strength
- The primary LIMS (rougher, cleaner, and re-cleaner) performance, with over 98% magnetite recovery, shows that a LIMS scavenger will be required only for rougher tails, which accounts for about 1.9% loss of the ROM magnetite and most of the lost haematite
- The REDS scavenger on cobbing tails recovered 38.4% magnetite and 23.8% haematite from the cobbing tails. Overall, the REDS scavenger improved the haematite recovery from 74.6% to 80.6% before ball milling
- Most of the haematite that was recovered after milling to 75 μm was found to be associated with magnetite, even after IsaMilling to P_{80} 45 μm. This was further confirmed by the high haematite recovery of 89.2% achieved during re-cleaning of IsaMilled concentrate
- The piloting work shows the full-scale plant should not differ significantly from the piloted flow sheet.

Recommendations

- The pilot plant LIMS, cobbing, and rougher, were not optimal but still produced good results in terms of magnetite recovery. The full-scale plant should use countercurrent units, not concurrent, and with higher drum surface magnetic field strength, 2000 G not 1500G.
- A trade-off study should be conducted to evaluate the economic feasibility of using wet high-intensity magnetic separation (WHIMS) or spirals for haematite recovery compared with REDS scavenger on cobbing tails.

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The Lamella High Shear Rate REFLUX™ Classifier

by T. Orupold*, D. Starr*, and T. Kenefick*

Synopsis

This paper covers the commercial development of gravity separation of fine particles using a Lamella High Shear Rate REFLUX™ Classifier (REFLUX™ Classifier), focusing primarily on coal applications. The REFLUX™ Classifier is a fluidized bed device that incorporates a system of closely spaced parallel inclined channels located above the fluidized bed. These channels make it possible to achieve a significant suppression of the effects of particle size, resulting in a highly effective separation on the basis of density. The improved gravity separation performance is shown to be remarkably high, with a significant reduction in the variation of separation density with particle size, and a significant reduction in the change in Ecart probable error (Ep) with size.

The first full commercial-sized units of the REFLUX™ Classifier were field-tested in late 2009 in coal applications. More recently, the technology has been applied in fine particle separation in minerals applications and there are a number of full-sized units operating in chrome applications in South Africa. Initially, pilot-scaled units (typically the RC™300) were trialled in iron ore, mineral sands, and manganese plants amongst other minerals, typically after other technologies failed to achieve the desired results. Currently a number of laboratories globally are carrying out more testing in minerals applications.

More than 50 RC™ units are now operating in coal and minerals applications. This paper introduces the REFLUX™ Classifier technology, identifies commercial applications, and gives some commercial results.

Keywords
gravity separation, classification, fluidized bed.

Introduction

In coal and mineral processing, gravity separation of fine particles (generally less than 2 mm in size) is normally carried out in water through the application of various gravity separation devices, including spirals and variations of upstream classifiers. The objective is to produce a separation on the basis of particle density, ideally with particles lower than the target density reporting as one flow stream and those higher in density reporting as a second flow stream. In practice, as the particle size range of the feed increases the overall separation efficiency declines significantly. When targeting a given product grade, this loss of efficiency leads directly to lower recovery of the valuable component. The efficient separation of a broad size range of particles on the basis of particle density in a low-density fluid such as water remains a considerable challenge.

There is a growing trend to replace existing technologies for beneficiating fine minerals and coal, nominally less than a few millimetres in size, with technologies based on fluidized beds. Generally, fluidized beds tend to be hydraulically limited and are constrained by feed conditions, and hence the feed needs to be as concentrated as possible to maximize throughput. However, the FLSmidth REFLUX™ Classifier achieves a much higher hydraulic capacity through the incorporation of parallel inclined plates in the fluidized bed housing, and therefore this unit requires a much smaller footprint within the plant.

The REFLUX™ Classifier (RC™) is an innovative device offering advantages in both gravity separation and particle size classification. The device combines a conventional fluidized bed with sets of parallel inclined plates, as shown in Figure 1. Feed slurry enters below the plates while fluidization water is introduced through a distribution plate in the base. The slurry feed moves downwards into the vessel, forming a bed of particles that is fluidized from below. High-density particles settle into the lower portion of the bed, and light and fine particles are transported upward, with the majority flow towards the lamellae. The high hydraulic load carries the suspension up into the parallel inclined lamella plates. Here slower settling particles, which are unable to settle against the fluidization water, emerge through the plates and report to the overflow. Faster settling particles drop out of suspension and onto the plates before sliding back to the zone below.

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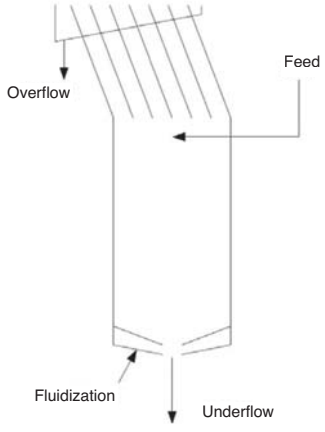

Figure 1—REFLUX™ Classifier schematic

At high bed concentrations in the reflux zone, this reject suspension provides an autogenous dense medium, allowing the separation to proceed largely on the basis of density. When the density of the fluidized bed exceeds the set-point value, a valve opens near the base of the unit and discharges some of the denser particles as an underflow stream.

The inclined channels, the geometry of which is defined by the plate length, L, perpendicular channel spacing, z, and angle of inclination with the horizontal, h (Figure 2), provide a significant hydraulic advantage over conventional fluidized beds, consistent with the well-known 'Boycott Effect' (Boycott, 1920; Ponder, 1925; Nakamura and Kuroda, 1937; Zhou *et al.;* 2006) showed that over the range h = 60–80 degrees the performance was optimal, hence all of the work since then has been based on an angle of inclination of 70 degrees.

Development of the REFLUX™ Classifier

As the fundamental research was funded by the Australian coal industry, all the initial testing and subsequent pilot and full-scale work were performed using Australian coals. Mineral testing started in South Africa many years after development of the first commercial units. All the development details in this paper are therefore based on work with Australian coals.

Following the laboratory and theoretical development (Galvin and Nguyentranlam, 2002; Galvin *et al.*, 2002a, b) of the REFLUX™ Classifier technology, the next step was to conduct performance testing of commercial-sized units.

In 2005 the performance of the first-generation full-scale REFLUX™ Classifier (RC™ 1800), supplied by Ludowici Australia, was evaluated using a coal and mineral matter feed nominally less than 2 mm in size. The unit was installed, commissioned, and studied at a Lower Hunter Valley coal preparation plant. The plates were inclined at an angle of 60 degrees to the horizontal and were spaced at 120 mm intervals.

In late 2008, further research conducted by Galvin (Galvin *et al.*, 2010) established that the efficiency could be significantly increased by reducing the lamella channel spacing. Based on this result, Ludowici redesigned the REFLUX™ Classifier, with the RC™2020 being the first of new design units.

In late 2009 the performance of a full-scale RC™2020, supplied by Ludowici Australia, was evaluated at a Central Queensland coal preparation plant. The inclined plates were inclined at an angle of 70 degrees to the horizontal and spaced at 6 mm intervals for this installation.

The results of the 2005 and 2009 tests were reported by Orupold *et al.* (2013). The RC™2020 exhibited significantly lower Ep values and a reduced D_{50} shift by size. The RC™2020 results are represented in Figure 3 and Figure 4.

Figure 3 shows a relatively moderate D_{50} shift with change in particle size when compared to other technologies.

Figure 4 shows that Ep values are still very low even at fine particle sizes.

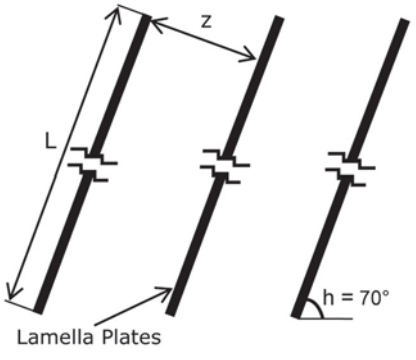

Figure 2—Lamella plate configuration

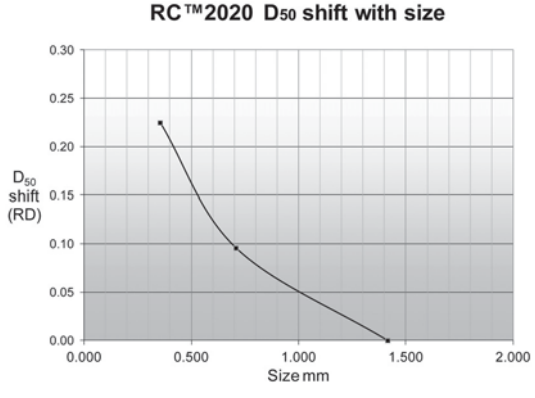

Figure 3—RC™2020 D_{50} shift by size for coal applications

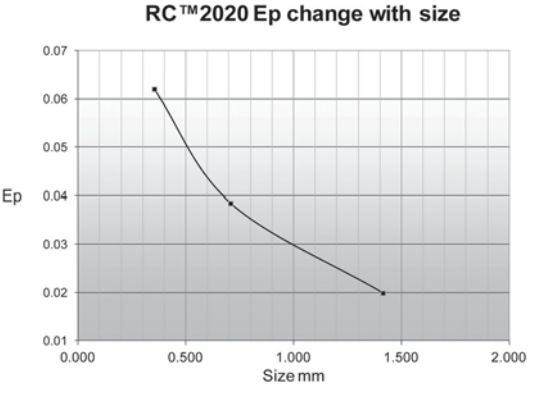

Figure 4—RC™2020 Ep change with particle size in coal applications

The RC™2020 then became the mainstay unit, with over 40 units sold globally in three years. The RC™2020 design has now been replaced by the RC™2000 (Figure 5).

The operational process capabilities of the RC™2020 and RC™2000 are generally equivalent. Based on site feedback from global locations, the RC™2020 was re-designed as the RC™2000 to incorporate various enhancements for improved internal feed preparation and distribution, and others for lamella channel protection, including internal oversize protection and de-aeration. The latter are additional safety features and do not preclude in-plant installation of oversize protection and de-aeration of the feed. The RC™2000 body is fully fabricated from 304 stainless steel with wear-resistant lining and parts in the higher wear sections.

Further enhancements to the channel design will allow the RC™2000 a throughput advantage of 15% over the similar sized RC™2020. FLSmidth, which acquired Ludowici Australia (Pty) Ltd in 2012, now produces commercial sized units from RC™850 to RC™3000, all based on the RC™2000 design. Also available for in-plant testing is the RC™300 pilot-scale unit, which is a pilot-scale unit designed to take the equivalent of one spiral start. It is 1/36th the size of a RC™2000 full-scale commercial unit, although it is roughly the same height. Many features are shared with the commercial units, including the lamella plates, pressure probes, and fluidizing nozzles. A number of underflow valve configurations are available to cater for various applications. Currently, more than 24 RC™300 units are in use in coal and minerals applications in six countries around the world. The RC™300 is shown in Figure 7.

Coal applications

The FLSmidth® RC™300 REFLUX™ Classifiers have been used extensively for onsite pilot-scale test work. RC™2020 and, more recently, RC™2000 units have been installed in greenfield and brownfield applications in Australia, New Zealand, China, India, the USA, and Mozambique. Larger RC™2350 units were commissioned in August 2013 in Australia, and RC™3000 classifiers are expected to be commissioned in Mozambique in 2014. A number of plants have in excess of six units.

Most applications are as primary separation devices, although a number have been installed as scavenger units to recover lost coal from other processes.

The REFLUX™ Classifier is targeted at the treatment of -2.00 mm coal and is effective down to 0.125 mm. However, as noted by Galvin *et al.* (2010), the REFLUX™ Classifier, in common with other gravity separation devices, can show a shift in cut-point by size. This effect is pronounced with the finer sizes. Therefore the generally targeted size fraction has a top-to-bottom size ratio of 8:1. In Australia this is generally -2.0 mm +0.250 mm. In the USA the units operate treating finer material (-1.00 mm +0.150 mm) (Ghosh *et al.*, 2012).

The first RC™2020 unit was installed as a trial in a Central Queensland coking coal plant in 2010. The results from the 8-hour acceptance run sampling are given in the Appendix. The cut-point for the targeted -2.0 mm +0.0250 mm fraction was RD 1.60 with an Ep of 0.07.

Figure 5—RC™2000 REFLUX™ classifiers factory test assembly prior to delivery

Figure 6—RC™2000 overflow launders on a coal application

Figure 7—Pilot-scale RC™300 REFLUX™ Classifier

In 2008 the Australia Coal Association Research Program (ACARP) published a report into extending the size fractions that the REFLUX™ Classifier could treat. The pilot RC600 used for this research successfully extended the top size to 8 mm. In 2014, a follow-up ACARP-funded project will look at extending the REFLUX™ Classifier range to a top size of 4 mm in a full-scale demonstration plant in the Hunter Valley, NSW, Australia. This research is independent of FLSmidth.

Minerals applications

The FLSmidth® RC™300 and RC™850 REFLUX™ Classifiers have been used extensively for onsite pilot-scale test work. A wide range of mineral applications have been tested, including iron ore, chromite, manganese, and mineral sands. The vast majority of the test data discussed in the following sections was obtained using the Lamella High Shear Rate REFLUX™ Classifier technology, utilizing 6 mm spacing between lamella channels.

Iron ore - Australia

A Western Australian iron ore bulk sample (nominally -1 mm +0.106 mm) was tested using an RC™300 REFLUX™ Classifier (RC™300). The material was a sample of a resource that contained a significant amount of naturally occurring (liberated) fine iron ore (-1 mm particles). The resource owners considered the finer fractions to be unrecoverable, as no other technology was successful in reducing the silicates and aluminates to a commercially viable level.

Heavy liquid separation (HLS) analysis of the sample had shown a separation at a specific gravity (SG) of 3.8 could achieve a yield of around 40%. The corresponding iron recovery would be around 60% and the combined silicates/aluminates would be less than 10%. This separation would produce a commercially viable concentrate.

As the resource owner was unable to perform an onsite test, a representative bulk sample was collected. A RC™300 pilot-scale unit was used in a simulated REFLUX™ Classifier circuit. The feed material was prepared to match the proposed feed circuit. However, due to the nature of the simulated circuit set-up and limited sample size, the test work conditions were not fully optimized and only several short 'sighter' tests and a single longer run were conducted.

During the test work the RC™300 was fed with 30–35% (w/w) slurry of the bulk ore sample and water at a flow rate of 4.5-5 m³/h. Several relative density (RD) set-points were investigated during the test work, while the fluidizing water flow rate was kept constant at 4.85 m³/h. The best separation produced an underflow stream consisting of 60% iron and less than 10% combined silicates/aluminates. Figure 8 shows the comparison of the HLS and RC™300 test data.

Although the yield was only around 19%, the product from the REFLUX™ Classifier would equate to a significant financial benefit to the resource owner. The additional iron recovery would be achieved without increased mining or liberation expenses. Also, a reduction in the size of downstream processing units handling the disposal of the -1 mm discard stream would be possible.

Commercial RC™2000 units will enter into production in Western Australia in the later part of 2014.

Iron ore – South Africa

In 2014 a full-scale RC™2020 unit was installed as a trial in a large South African iron ore beneficiation plant, replacing other gravity and magnetic devices. The unit was commissioned late March 2014, and from April it commenced operation as a commercial unit. Initially, the unit is to upgrade -1.00 mm +0.200 mm material to required specification. Results to April/May 2014 have indicated haematite recoveries in the order of 84%.

Full results cannot be released at this time, but the user will publish the results at future SAIMM conferences. The user also is conducting research at other sites with RC™100 and RC™300 laboratory and pilot units.

Mintek also has a RC™100 laboratory-scale unit available for independent research and materials testing purposes.

Iron ore – Canada

Amariei *et al.* (2013) reported an investigation into spiral feed and spiral reject retreatment of Canadian iron ore using a Ludowici® RC™300. The pilot-scale tests were compared with sink-float (HLS) tests for two samples – Sample1, a spirals feed sample screened to -0.106 mm, and Sample2, a spirals reject stream 'as received', containing coarse silica (+0.106 mm).

Both samples were analysed using sink-float tests to establish a means of comparison for the streams produced from the RC™300.

The RC™300 was fed at approximately 40% of the maximum feed capacity. Several relative density (RD) set-points were investigated during the test work for both samples. The effect of fluidizing water rate was also investigated during the test work, although only on Sample1.

RC™300 processing of Sample1 produced an underflow stream consisting of 69.8% iron and 1.13% combined silica. This is an impressive result considering the feed stream contained 42.6% Fe and 30.7% SiO_2. The weight yield corresponded to 54.8%.

RC™300 processing of Sample2 produced an underflow stream consisting of 67.1% iron and 3.00% combined silica. Considering the sample originated from a waste stream containing 30.0% Fe and 55.6% SiO2, this recovery is an encouraging result for the REFLUX™ Classifier technology. The weight yield corresponded to 39.5%.

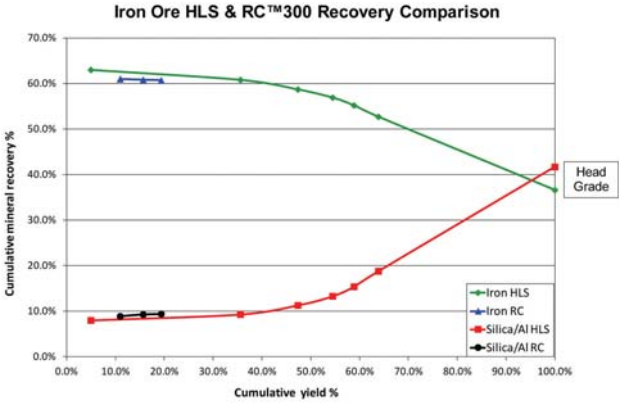

Figure 8—Australian iron ore – HLS compared to RC™300 data

Of particular interest is the comparison that Amariei *et al.* (2013) investigated between the RC™300 results and the sink-float tests for each feed stream. The RC™ performed very close to the accepted maximum achievable results.

Iron ore – other

The RC™300 has been (or is being) tested for beneficiation of fine iron ore at a number of other locations globally. The minerals being tested include haematite, goethite, and magnetite.

Mineral sands

The RC™300 has been utilized for pilot-scale test work on mineral sand deposits in South Africa, India, and Australia. Laboratory-scale test work has also been conducted in Australia. The majority of the tests have shown upgrades in heavy mineral content. Unfortunately, the data from the test work is usually 'commercial in confidence.' Table II shows results from mineral sand tests for two different head grades of heavy minerals (HM). The test work was performed at a feed rate of 30 t/m²/h, which would equate to 100 t/h of dry solids through a full-scale RC™2000 unit.

The test data for the results shown in Table I are summarized in the Appendix.

Although the test work conditions were not optimized, the recoveries of HM as shown in Table I represent good upgrading of the HM content in the concentrate or underflow stream. In the case of the low head grade material, the upgrade ratio was over 7 times.

The test work completed using the RC™300 strongly indicates that HM can be upgraded using a reduced number of units compared to competing technologies.

A commercial unit is now operating in a mineral sands application.

Chromite – South Africa

Onsite pilot-scale tests were successfully conducted in South Africa for an existing chromite operation in 2009 using an RC™600. The test work resulted in the resource owner installing six full-scale production RC™2020 REFLUX™ Classifiers for continuous commercial operation across four sites in South Africa. The resource owner has requested that plant performance data is not is not publicized.

Manganese

Onsite test work using the REFLUX™ Classifier technology has been undertaken on manganese, following on from laboratory and offsite pilot-scale test work. Due to the throughput requirements of the test work (production of a 5000 t sample for market testing), a small commercial RC™850 was used for this trial. The grade/yield characteristics for the RC™ product exceeded those of competing technologies. As with other results, the end user does not wish to be identified and publication of any results is not permitted.

Currently, the design of a full-scale plant with a number of RC™2000 units is being finalized and, subject to market conditions, a project go-ahead is expected in 2014.

Other minerals

The REFLUX™ Classifier technology has been tested using both laboratory and pilot-scale units on various minerals and in industrial applications. Unfortunately due to commercial sensitivities, details and data are not available for publication at this stage.

Conclusions

The RC™ is usually used to process -2 mm feeds where there is a difference in particle density between the valuable mineral and gangue.

The improvement in separation performance with the Laminar High Shear Rate REFLUX™ Classifier has been dramatic and has led to widespread adoption of the technology, which has been available commercially only since 2010, as the gravity separator of choice in coal applications, especially for cut-points below 1.80.

The REFLUX™ Classifier can treat coals at lower cut-points and at higher efficiencies, thereby allowing the production of lower ash products combined with improved yields.

The REFLUX™ Classifier has been installed in a number of minerals applications across the world. To date, all of the data is commercial in confidence; however, all the results show that the Lamella High Shear Rate REFLUX™ Classifier technology is well suited to minerals applications and in some instances can provide an excellent separation where other equipment struggles to meet grade and recovery targets.

FLSmidth is seeking to overcome the lack of publishable data by investing in laboratory and pilot-scale testing at its research facility in Salt Lake City, Utah. FLSmidth will also have the ability to test and recommend flow sheets for customer-specific applications.

The lack of specific data in this paper is to a large extent due to current global economic conditions, as all end users of this technology have embargoed publication of results and identification of installations as they try to achieve market advantage. The authors encourage potential end users to conduct their own test work on new minerals and industrial applications.

Acknowledgements

The authors gratefully acknowledge the financial support of

Table I

Mineral sands test work data for low and medium head grades of heavy minerals

Head grade	Feed (wt% HM)	Underflow (wt% HM)	Overflow (wt% HM)	HM recovery (%)
Low	6.6	44.4	0.8	92.0
Medium	22.0	53.6	1.0	96.5

ACARP for these projects, and the work and support from Professor Kevin Galvin and his team at the University of Newcastle. We thank Sedgman for their investment in the full-scale RC™2020 trial in Central Queensland.

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Appendix

Table II

Mineral sands assay by size for REFLUX™ classifier feed, underflow, and overflow. (Totals of the grades do not add up to 100%, trace minerals are not reported in these tables)

Feed	Fraction, µm							Total
	+425	-425 +300	-300 +212	-212 +150	-150 +106	-106 +75	-75	
TiO_2 (%)	0.0	2.1	3.5	19.9	33.5	17.0	0.0	18.9
ZrO_2 (%)	0.0	0.2	0.5	2.8	5.8	4.0	0.0	3.1
Fe_2O_3 (%)	0.0	1.5	2.3	12.7	20.9	8.4	0.0	11.7
Mn_3O_4 (%)	0.0	0.0	0.0	0.3	0.5	0.2	0.0	0.3
*SiO_2 (%)	0.0	93.9	93.4	62.5	36.1	65.8	0.0	63.9
$P2O_5$ (%)	0.0	0.0	0.0	0.1	0.3	0.5	0.0	0.2
Al_2O_3 (%)	0.0	0.7	0.8	0.7	0.8	1.6	0.0	0.8
*Note: Not all free silica								

Underflow	Fraction, µm							Total
	+425	-425 +300	-300 +212	-212 +150	-150 +106	-106 +75	-75	
TiO_2 (%)	0.0	2.7	18.8	49.2	50.0	45.4	0.0	45.9
ZrO_2 (%)	0.0	0.3	2.8	7.1	8.9	12.1	0.0	7.7
Fe_2O_3 (%)	0.0	1.9	11.1	31.1	31.5	21.8	0.0	28.4
Mn_3O_4 (%)	0.0	0.0	0.3	0.8	0.8	0.6	0.0	0.7
*SiO_2 (%)	0.0	94.4	65.8	9.4	5.9	12.8	0.0	14.4
P_2O_5 (%)	0.0	0.0	0.0	0.2	0.5	1.5	0.0	0.4
Al_2O_3 (%)	0.0	0.4	0.9	0.6	0.5	1.1	0.0	0.6
*Note: Not all free silica								

Overflow	Fraction, µm							Total
	+425	-425 +300	-300 +212	-212 +150	-150 +106	-106 +75	-75	
TiO_2 (%)	0.0	0.6	0.6	0.8	2.0	2.2	0.0	1.0
ZrO_2 (%)	0.0	0.1	0.0	0.0	0.0	0.0	0.0	0.0
Fe_2O_3 (%)	0.0	1.0	0.5	0.7	0.9	1.4	0.0	0.7
Mn_3O_4 (%)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
*SiO_2 (%)	0.0	97.1	97.0	97.0	93.3	90.5	0.0	96.0
P_2O_5 (%)	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Al_2O_3 (%)	0.0	0.8	0.8	0.9	1.6	1.9	0.0	1.0
*Note: Not all free silica								

Table III

Coal washability and sizing data for REFLUX™ Classifier feed, underflow, and overflow

Size analysis (mm)		Feed		Product		Reject	
Minus	Plus	Mass%	Ash%	Mass%	Ash%	Mass%	Ash%
3.00	2.00	1.52	33.22	0.76	5.39	2.85	57.09
2.00	1.00	30.49	19.84	31.04	5.14	30.67	60.85
1.00	0.50	35.16	19.98	35.62	6.73	33.48	66.38
0.50	0.25	21.78	23.22	21.34	6.58	20.70	69.33
0.25	0.13	11.05	31.09	11.24	12.93	12.31	75.14
		100.00	22.08	100.00	6.89	100.00	66.11
-3 mm +2 mm							
Relative density		**Feed**		**Product**		**Reject**	
Sinks	**Floats**	**Mass %**	**Ash%**	**Mass %**	**Ash%**	**Mass %**	**Ash%**
1.150	1.200	0.00	1.9	0.0	1.90	0.00	1.9
1.200	1.250	0.00	2.2	0.26	2.20	0.00	2.2
1.250	1.300	20.29	2.8	54.3	3.40	0.00	3.0
1.300	1.350	16.66	6.2	31.34	6.50	0.62	8.8
1.350	1.400	12.43	10.4	9.8	8.90	5.84	14.2
1.400	1.450	4.24	14.7	3.69	13.50	8.31	17.0
1.450	1.500	2.37	21.5	0.3	19.50	4.98	20.8
1.500	1.550	1.88	27.2	0.29	21.70	2.09	28.7
1.550	1.600	1.34	31.6	0.0	0.00	2.09	32.6
1.600	1.650	1.31	34.5	0.00	0.00	2.24	38.4
1.650	1.700	1.52	39.7	0.0	0.00	2.03	42.5
1.700	1.800	2.48	46.1	0.00	0.00	4.68	49.0
1.800	2.000	5.68	56.4	0.0	0.00	10.67	59.4
2.000	2.225	29.80	76.7	0.00	0.00	56.47	74.6
		100.00	33.22	100.00	5.39	100.00	57.09
-2 mm +1 mm							
Relative density		**Feed**		**Product**		**Reject**	
Sinks	**Floats**	**Mass**	**Ash**	**Mass**	**Ash**	**Mass**	**Ash**
1.150	1.200	0.00	1.9	0.0	1.90	0.00	1.9
1.200	1.250	0.59	1.4	0.04	2.10	0.00	6.7
1.250	1.300	34.88	2.2	53.0	2.00	0.03	7.6
1.300	1.350	21.94	5.8	26.49	6.30	0.09	17.6
1.350	1.400	10.83	9.9	10.4	9.50	0.96	14.9
1.400	1.450	4.46	14.8	8.84	13.00	2.88	21.2
1.450	1.500	2.36	18.9	0.9	19.50	6.01	23.2
1.500	1.550	1.48	26.6	0.19	21.70	3.82	29.0
1.550	1.600	1.16	30.9	0.0	29.30	3.36	32.3
1.600	1.650	0.97	35.4	0.01	33.00	2.80	37.0
1.650	1.700	0.99	39.8	0.0	35.60	2.05	41.4
1.700	1.800	1.55	45.6	0.01	42.30	4.87	48.4
1.800	2.000	2.86	55.9	0.0	50.90	8.54	58.1
2.000	2.225	15.93	74.2	0.03	77.70	64.60	73.2
		100.00	19.84	100.00	5.14	100.00	60.85
-1 mm + 0.5 mm							
Relative density		**Feed**	**Product**	**Reject**			
Sinks	**Floats**	**Mass**	**Ash**	**Mass**	**Ash**	**Mass**	**Ash**
1.150	1.200	0.02	1.9	0.0	1.90	0.00	1.9
1.200	1.250	1.55	0.9	0.11	1.60	0.00	5.7
1.250	1.300	40.10	2.7	52.1	3.90	0.06	6.1
1.300	1.350	18.42	5.7	23.75	5.90	0.07	14.8
1.350	1.400	8.95	10.3	9.6	9.40	0.08	14.8
1.400	1.450	3.98	14.8	9.95	13.10	0.18	21.6
1.450	1.500	2.08	20.2	2.5	20.60	1.14	26.4
1.500	1.550	1.41	25.4	1.19	25.20	1.58	32.3
1.550	1.600	1.17	30.1	0.5	29.40	3.00	33.3
1.600	1.650	0.97	34.6	0.14	33.00	3.23	37.7
1.650	1.700	0.94	39.4	0.0	35.60	2.48	41.0
1.700	1.800	1.26	45.1	0.01	42.30	4.30	46.5
1.800	2.000	2.38	54.9	0.0	50.90	7.56	57.3
2.000	2.225	16.77	75.2	0.06	76.50	76.32	73.3
		100.00	19.98	100.00	6.73	100.00	66.38
		100.00	31.09	100.00	12.93	100.00	75.14

Table III (continued)

Coal washability and sizing data for REFLUX™ Classifier feed, underflow, and overflow

-0.5 mm + 0.25 mm							
Relative density		Feed		Product		Reject	
Sinks	Floats	Mass	Ash	Mass	Ash	Mass	Ash
1.150	1.200	0.13	1.9	0.0	1.90	0.00	1.9
1.200	1.250	8.41	1.2	0.52	1.60	0.00	5.4
1.250	1.300	29.76	3.2	52.6	1.40	0.19	5.8
1.300	1.350	20.32	6.0	19.66	5.90	0.10	12.3
1.350	1.400	6.55	10.3	8.5	8.90	0.03	13.8
1.400	1.450	3.21	14.9	9.50	13.00	0.06	25.0
1.450	1.500	1.49	20.3	3.4	19.80	0.12	29.8
1.500	1.550	1.21	25.3	1.92	25.90	0.15	31.1
1.550	1.600	1.33	30.4	1.4	30.70	0.45	35.9
1.600	1.650	1.03	34.9	0.82	35.10	0.86	40.1
1.650	1.700	1.10	40.8	0.6	38.70	1.24	43.4
1.700	1.800	1.23	46.1	0.38	44.20	3.27	47.9
1.800	2.000	2.44	55.6	0.3	56.00	4.83	56.4
2.000	2.225	21.78	73.7	0.24	79.80	88.71	72.0
		100.00	23.22	100.00	6.58	100.00	69.33
-0.25 mm + 0.125 mm							
Relative density		Feed		Product		Reject	
Sinks	Floats	Mass	Ash	Mass	Ash	Mass	Ash
1.150	1.200	2.39	1.7	0.0	1.90	0.00	1.9
1.200	1.250	13.06	2.8	2.09	1.80	0.00	6.9
1.250	1.300	25.13	7.5	41.1	2.40	0.27	7.5
1.300	1.350	14.99	9.4	22.99	7.40	0.07	10.6
1.350	1.400	4.82	14.4	7.0	14.20	0.04	16.6
1.400	1.450	2.51	24.0	8.52	12.90	0.05	17.8
1.450	1.500	0.79	28.1	4.2	20.40	0.08	29.8
1.500	1.550	0.55	30.4	1.81	28.00	0.09	32.4
1.550	1.600	1.13	36.2	1.9	23.90	0.13	36.3
1.600	1.650	1.08	44.3	1.52	37.20	0.14	43.4
1.650	1.700	1.87	51.9	1.4	42.70	0.22	47.6
1.700	1.800	1.62	55.0	1.00	49.30	0.46	54.6
1.800	2.000	3.37	64.5	3.7	62.60	7.70	63.4
2.000	2.225	26.67	77.9	2.84	82.20	90.76	76.8
		100.00	31.09	100.00	12.93	100.00	75.14

The application of Baleen Filter microscreening technology at BECSA's South Export Plant

by S. Tshikhudo* and V. Shikwambana*

Synopsis

This paper outlines an investigation into the recovery of saleable fractions of coal from 'as-arising' South Export Plant effluent streams, using Baleen microscreening technology. South Export Plant, a subdivision of BHP Billiton Energy Coal SA (BECSA) Coal Processing, is a two-module plant treating 2000 t/h. The nominally -150 μm coal is untreated and is therefore passed from classifying cyclones to the thickeners for process water recovery. The thickened underflow is pumped into a series of slurry cells for further settling and recovery of supernatant water. The marginal quality, moisture content, and handlebility of this settled material renders it unsuitable for inclusion into saleable products and it is thus stockpiled and trucked to designated pits for disposal.

Over the years, stockpiling and trucking has become an overly expensive exercise. In an effort to recover some of this cost, a task team was assigned to investigate less costly options to process slurry across BECSA plants. Various technologies such as froth flotation, sieve bends, and Reflux Classifier were considered, although the results were generally not beneficial – this could be attributed to weathered/oxidized coal.

A decision was made to pursue an alternative approach by testing the suitability of the new 'Baleen Filter'. The concept is to screen out the higher-grade fraction (+50 μm) as saleable product and reject the finer fractions to the slurry ponds.

The Baleen Filter was found to effectively screen at an acceptable efficiency between 94% and 99.99%, with a very sharp cut-point (d_{50} and *Ep*). The actual yields from the screening results were better than the predicted yields in terms of both mass and energy as predicted from feedstock analysis.

Keywords

Fine coal, coal slurry, upgrading, screening, Baleen Filter, thickener feed.

Introduction

The South Export Plant of BHP Billiton Energy Coal SA (BECSA) is a two-module dense medium separation (DMS) plant designed to process 2200 t/h run-of-mine (ROM) coal with a nominal top size of 50 mm into a range of products for both the domestic and the export markets. Slimes from the plant consist of -150 μm particles, which constitute approximately 6% of the feed to the plant. This is initially dewatered using thickeners and then pumped to slimes dams for temporary storage and drying over a period of 7 months. The slimes are later reclaimed and hauled to designated pits for disposal. The process of slurry reclamation and disposal is cost-intensive, and in an effort to reduce the cost an initiative was undertaken to explore more cost-effective slurry handling techniques.

The quality and handlebility of the settled slurry render it unsuitable to be included in the saleable products in its unprocessed form. Various technologies such as the Baleen Filter, flotation, Reflux Classifier, and sieve bend were considered as techniques for upgrading the slurry to a saleable product. The Baleen Filter technology was considered on the basis of its novelty and feasibility compared to other operations. The idea around the use of the Baleen Filter is to recover material above a specified cut-point based on size that results in a saleable product and rejection of the screen undersize. Following extensive prefeasibility test work conducted on the thickener feed, a 50 μm screen aperture was found most suitable as a starting point for the pilot plant. A 2.78 m2 pilot plant was therefore erected to process a portion of the feed to the thickeners.

Project objective

The objective of the project was to evaluate the effectiveness and benefits of Baleen Filter technology in upgrading the thickener feed to qualities that will be suitable for either the domestic or the export market. This project was conducted by taking a limited number of samples over a period of time. It was designed to provide an initial view of the operational parameters of the Baleen Filter. In addition, the effectiveness of the screen was also investigated in terms of screening efficiency and sedimentation of the underflow and feed material.

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Baleen pilot plant overview

Figure 1 provides an overview of the Baleen Filter process. The feed is drawn from the thickener feed head box with a 250 mm diameter pipe connected to the Baleen feed box. The pipe is fitted with a manual valve to control feed rate to the Baleen Filter. The feed flows over the screen, which removes -50 μm particles to the underflow and recovers +50 μm particles in bags situated at the discharge end of the screen. Clarified water is used for pressurized sprays to facilitate the screening process by dislodging material from the screen and scraping off the oversize to the discharge. This is connected to a mobile spray rack that is pneumatically controlled and moves up and down the screen continuously. The underflow from the screen reports to the thickener sump, from where it is pumped to the slimes disposal tank. The bags containing the recovered oversize are allowed to dry over time and later shipped to Australia for further binderless coal briquetting testing.

The clarified water added to the sprays should be solids-free to prevent blockages of the sprays, as they are the central part of the Baleen separation process. A water filtration plant accompanies the Baleen Filter to remove all suspended solids, particulates, and scale-inducing constituents from the water prior to feeding to the sprays.

Operating principle

The Baleen Filter or micro-screening technology is based on a 'double-act' of high- or medium-pressure, low-volume sprays, one of which dislodges material caught by the filter media, while the other sweeps it away. As water flows through the filter, particles initially suspended in the water are left behind, but before they are allowed to accumulate, the 'double-act' sprays sweeps away the solids from the filter media into bags. A travelling spray boom self-cleans the static screen. The boom is pneumatically driven from its upper limit to the lower limit (Baleen Filters, 2011).

Experimental procedure

Equipment test

Measurements taken included:

- Amount of water consumed to aid with screening process (time taken to empty the particulate-free water holding tank)
- Spray boom cycle time (time taken to move the spray rags from one end to the other, cycles per minute)
- Flow rate into the Baleen Filter from the 250 mm HDPE pipe (measured with a Doppler flow meter).

Test methodology

The Baleen performance test was done at South Export laboratory according to ISO standard. A flow diagram of the sample preparation and analysis procedure is shown in Figure 4.

Table I

Baleen screen operational parameters

Parameter	OEM specification
Water pump	8 bars
Compressed air output	5 bars
Sprays	3.9 m^3/h
Spray boom	10 cycles/min
Medium flow - spray nozzle (15 bars)	0.0867 m^3/h per/nozzle
Number of bottom sprays	30 sprays
Number of upper sprays	15 sprays
Area of Baleen	3 m^2
Angle of repose	30°
Feed flowrate	50 m^3/h
Feed RD	1.01–1.04
Aperture size	50 μm

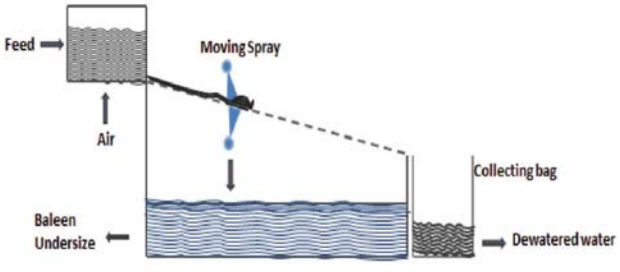

Figure 2—Baleen micro-screen cross-section

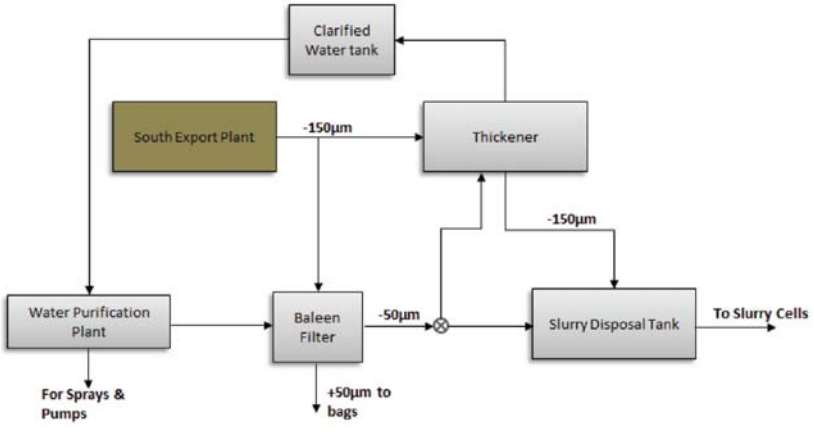

Figure 1—Baleen micro-screen flow diagram

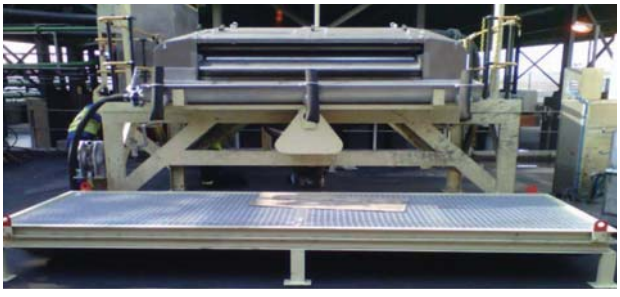

Figure 3—Baleen micro-screen

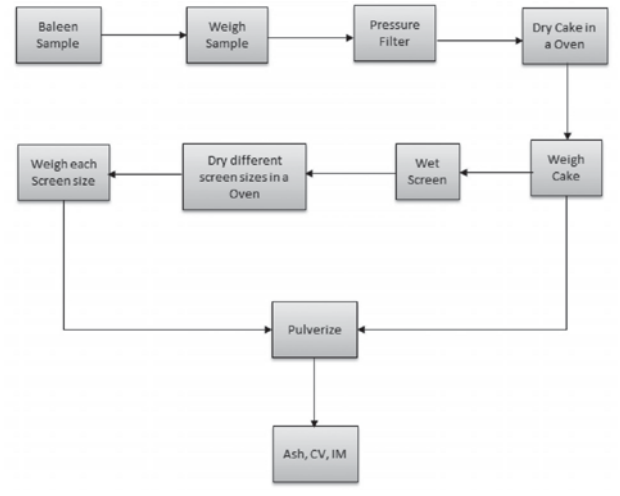

Figure 4—Sample preparation and analysis

Samples were taken from the Baleen feed, undersize, and oversize at 2-minute intervals for an hour over a number of days and composited. Samples were weighed, pressure filtered, dried in a 40°C oven, and again weighed. Filter cake was wet-screened on a test sieve at 300 μm, 212 μm, 150 μm, 106 μm, 63 μm, and 45 μm apertures using a vibrator sieve shaker, and also hand screened. The samples for quality analysis were then air-dried.

Settling test

The percentage solids samples were prepared from the thickener feed and Baleen undersize. Magna flocculant 919 with strength of 0.05% m/m was dosed into a 500 ml or 1000 ml slurry measuring cylinder. The required amount of flocculant was added as a split dose, by adding half of the amount and inverting the cylinder three times; the remaining flocculant was then added and the cylinder inverted twice then placed on a workbench. Settling rate was determined by measuring the time taken for the slurry to settle in the measuring cylinder. Optimal flocculant consumption was determined from the clarity as measured using a Ciba clarity wedge. Compaction or dewatering rates were determined for periods of 2 h, 8 h, and 24 h.

Results and discussion

Table II shows that the Baleen Filter using a 50 μm screen cloth upgraded the fines material from an ash content of 36.6% to an oversize material with an ash content of 21.09%.

Partition coefficient

The ash balance calculation (Equation [1]) was used to determine the mass split between the Baleen screen oversize and undersize material.

$$Yield = \left(\frac{x_{A(US)} - x_{A(F)}}{x_{A(US)} - x_{A(OS)}} \right) \times 100 \quad [1]$$

where
$x_A(US)$ = fractional ash composition in the undersize
$x_A(F)$ = fractional ash composition in the feed
$x_A(OS)$ = fractional ash composition in the oversize

$$= \left(\frac{0.5983 - 0.3664}{0.5983 - 0.2109} \right) \times 100$$

$$= 59.96\% \text{ (Oversize material)}$$

The size analysis of the screen undersize and oversize is shown in Table III.

Table II

Screen product qualities

Test	Feed (% ash)	Oversize (% ash)	Undersize (% ash)
1	33.08	22.9	59.8
2	35.65	18.41	66.2
3	33.44	21.97	38.42
4	40.6	20.4	59.9
Average	**36.64**	**21.09**	**59.8**
STDEV	**3.46**	**1.97**	**3.66**

Table III

Sizing data for the Baleen screen oversize and undersize particles

Aperture size, μm	Mean size, μm	Oversize		Undersize		Calculated feed	Partition coefficient
		% Mass	Mass in sample (g)	% Mass	Mass in sample (g)		
+300	387.3	3.3	2.0	0.0	0.02	2.0	0.99
+212	252.2	7.4	4.4	0.0	0.01	4.5	1.00
+150	178.3	14.8	8.9	0.1	0.05	8.9	0.99
+106	126.1	26.3	15.8	0.3	0.14	15.9	0.99
+63	81.7	23.6	14.1	0.6	0.23	14.4	0.98
+45	53	14.1	8.5	0.7	0.27	8.7	0.97
-45	33.5	10.5	6.3	98.2	39.33	45.6	0.14
		100	59.96	100.0	40.04		

The partition coefficient may now be plotted on semi-log paper as shown in Figure 5.

It can be seen that:

The separation size

$$d_{50} = 42\mu m \text{ (tromp cut)},$$

$$d_{75} = 47.9\mu m, \text{ and } d_{25} = 36.1\mu m$$

The efficiency,

$$EP = \left(\frac{d_{74} - d_{25}}{2}\right) = \left(\frac{0.0479 \text{ mm} - 0.0361 \text{ mm}}{2}\right) = 0.0059 \quad [2]$$

The imperfection

$$I = \frac{E_P}{d_{50}} = \left(\frac{d_{75} - d_{25}}{2d_{50}}\right) = 0.140 \quad [3]$$

The Baleen Filter is fitted with a 50 μm aperture size screen cloth, and it was found that the screen is operating at a cut size of 42 μm.

Misplaced material

Figure 6 shows the average size distribution of the feed, undersize, and oversize. It can be seen that at a d_{50} of 42 μm, the misplaced particles to the undersize and oversize are 1.9% and 8.1% respectively.

Overall efficiency

The mass balance around the screen is derived in terms of feed, undersize, and oversize:

$$m_{S(F)} = m_{S(O)} + m_{S(U)} \quad [4]$$

where:

$m_{S(F)}$ = mass of solids in the feed

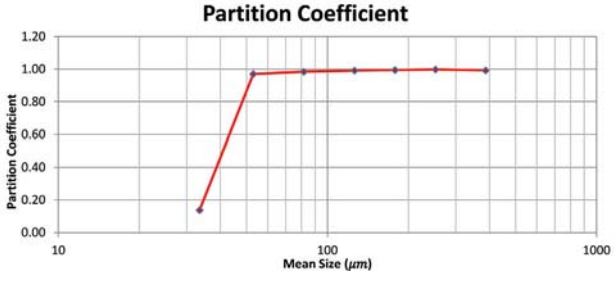

Figure 5—Tromp curve of Baleen screen data from Table III

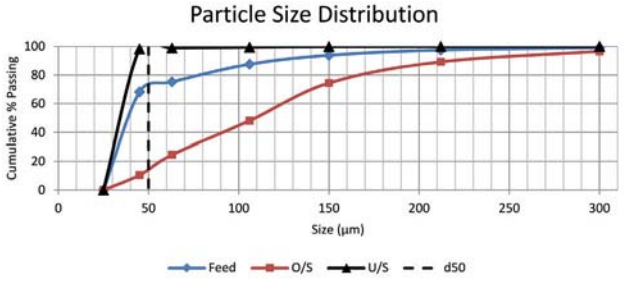

Figure 6—Particle size distribution of feed, oversize, and undersize

$m_{S(O)}$ = mass of solids in the oversize

$m_{S(U)}$ = mass of solids in the underflow

The composition of the undersize particles in the feed, oversize, and undersize streams is given by:

$$m_{S(F)} \times x_{U(F)} = m_{S(O)} \times x_{U(O)} + m_{S(U)} \times x_{U(U)} \quad [5]$$

where:

$x_{U(F)}$ = fraction of undersize in the feed

$x_{U(O)}$ = fraction of undersize in the oversize

$x_{U(U)}$ = fraction of undersize in the undersize

Substituting Equation [4] into Equation [5]:

$$\therefore \frac{m_{S(U)}}{m_{S(F)}} = \frac{x_{U(O)} - x_{U(F)}}{x_{U(O)} - x_{U(U)}} \quad [6]$$

'Osborne considered the efficiency of a square aperture screen as the ratio of the amount that actually passes through the screen to the amount that should pass through the screen' (Gupta and Yan, 2006). The screen efficiency (Table IV) is determined as follows:

$$E = \left(\frac{m_{S(U)}}{m_{S(F)} x_{U(F)}}\right) \times 100 \quad [7]$$

Substituting Equation [6] into Equation [7]:

$$\therefore E = \frac{100}{x_{U(F)}}\left(\frac{x_{U(O)} - x_{U(F)}}{x_{U(O)} - x_{U(U)}}\right) \quad [8]$$

$$\therefore E = \frac{100}{0.7015}\left(\frac{0.1438 - 0.7015}{0.1438 - 0.9840}\right)$$

$$\therefore E = 94.62\%$$

Feed particle size distribution

The project is focused on the recovery of fines material using particle size as the selection criterion. From Table V it can be extrapolated or interpolated that fines material at an ash content of 28% can be recovered at a cut size of 35.75 μm at an oversize yield of 63.34%. Alternatively, an ash content of 20 % can be achieved at a screen cut size of 49.5 μm at an oversize yield of 30.03%. However, the partition coefficient of the oversize and undersize shows that the screen was operating at a cut size of 41.6 μm (Figure 5).

The effect of the Baleen screen on thickener operation

One of the primary determinants of settling of material in a thickener is the open area. Installing the Baleen Filter results in a reduction of the solids flow rate to the thickener

Table IV

Summary of Baleen efficiency

Test	Efficiency	d_{50}	Actual yield
1	90.43	40.15	72.41
2	95.29	42.09	63.93
3	102.65	47.00	30.27
4	93.97	43.46	48.88
Average	95.59	43.17	53.87

Table V

Baleen screen feed particle size distribution

Aperture size (μm)	Mass fraction (g)	% mass	Cum. % mass	% ash	Cum. % ash
300	8.5	0.9	0.9	9.1	9.1
212	15.6	1.6	2.5	10.2	9.8
150	35.1	3.7	6.2	13.4	11.9
106	59.7	6.2	12.4	16.0	13.9
63	117.6	12.3	24.7	22.6	18.2
45	67.8	7.1	31.8	29.0	20.6
-45	651.8	68.2	100.0	44.1	36.6
Total	956.1	100.0			

(measured in tons per hour) of 59.96%, which is likely to enhance the performance of the current thickeners due to the increase in the open area available for settling.

Figure 7 and 8 show the results of settling tests on a number of samples taken from the thickener feed and the Baleen Filter undersize material. The percentage solids content of the samples was 2.51%, and the tests were carried out at a flocculant dosage of 30 g/t.

It can be seen that the settling rates of the thickener feed vary from sample to sample, whereas more consistent results were obtained from the Baleen Filter undersize material. However the thickener feed material settles at a faster rate than the Baleen Filter undersize.

Although the results indicate that the installation of the Baleen Filter might have an adverse impact on the rate of settling of the solids, it is important to note that the total mass of solids sent to the thickener would be reduced by 59.96%, and the increased open area thus created in the thickener would aid the settling process.

Dewatering rate

Figure 9 clearly shows that the feed reached its maximum after 4 hours, while the Baleen undersize size has not reached stability after 24 hours. It is expected that the -150 μm fraction has greater percentange of pores than -50 μm slurry. Currently a 7-month sedimentation period is required to dewater the thickener underflow. The impact of the reduction in the amount of solids in the slurry ponds by 59.96%, and the removal of the +50 μm fractions from these solids will need to be investigated further.

Conclusion

It can be concluded that the Baleen Filter is capable of upgrading the thickener feed material for the recovery of higher quality fine material based on size, despite the limited sample data used in this trial. The current trials, which are continuing for approximately 5 months, support this conclusion. The Baleen Filter is capable of treating micro-particles at a screening efficiency of 95.59%. The installed pilot Baleen Filter screen yielded efficiencies similar to larger aperture commercial screens. The installation of the Baleen screen will reduce the amount of solids in the thickener by 59.96%, which could have a positive impact on the operation of the thickeners, particularly when treating Seam 4 material, (4 seam contains a high amount of weathered coal which give rise to a high volume of fines) undersize material. The results clearly show that there will be a reduction in the amount of solids feeding the slurry cells. The overall impact of the removal of the +50 um fraction needs to be explored further.

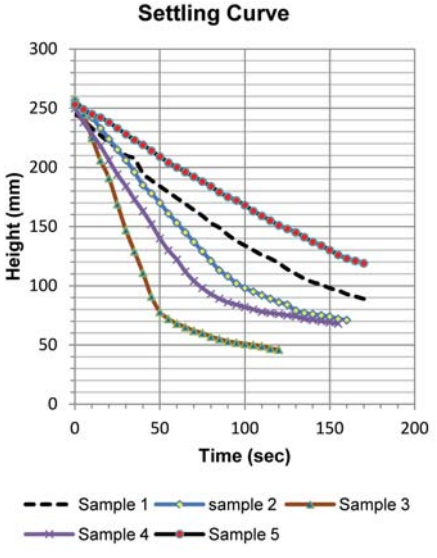

Figure 7—Thickener feed settling results

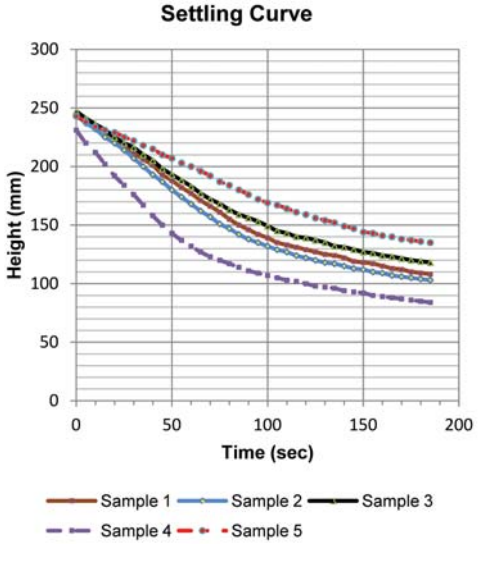

Figure 8—Baleen screen undersize settling results

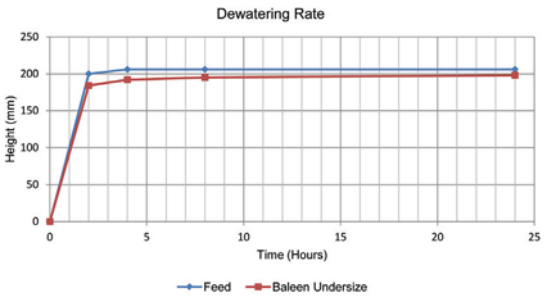

Figure 9—Dewatering rate

Acknowledgements

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Positron emission particle tracking inside a laboratory batch jig

by W.P. Roux* and N. Naudé*

Synopsis

Owing to decreasing high-grade ore reserves, there is a need for better understanding of the jigging process to improve the recovery efficiency of finer, lower grade material. The use of positron emission particle tracking (PEPT) was examined as a technique to study the motion of iron ore particles inside a laboratory batch jig. PEPT is a non-invasive method that can provide three-dimensional kinetic data on a particle in laboratory-scale processing units and has been successfully used to study mills, hydrocyclones, and flotation. Experiments were conducted to determine whether PEPT would be a viable technique to study iron ore jigging and what valuable information could be obtained. The results indicated that detailed information on the stratification rate of a particle could be obtained, with adequate resolution to track the particle's movement through an individual jig pulse.

Keywords
positron emission particle tracking, PEPT, jigging, gravity separation, modelling.

Introduction

Stratification through the depth of a jig bed is a result of the differential settling of particles under the influence of gravity. The main parameters that influence the stratification behaviour are the pulse cycle and feed properties (Mukherjee and Mishra, 2006). There have been many attempts to predict the performance of jigs over the years, but due to the complex interactions between the different parameters most of these models and theories provide only insight into the jigging process rather than predictive results.

Most of the theories are verified using empirical data based on the feed and product of the jig and not on the movement of the material inside the jigging chamber. This, however, is not sufficient to fully understand the jigging process and a deeper study of the movement of the particles inside a jig is required. Tracking of individual particles in a jig is therefore important. Only two experimental techniques have been used in the past: optical high-speed camera (Kuang *et al.*, 2004) and positron emission particle tracking (PEPT) (Williams *et al.*, 1998). The disadvantage of the optical techniques is that an artificial transparent sample is used, while with PEPT almost any type of jig feed can be used. PEPT shows significant promise as a research tool in the mineral processing industry. Using a radioactive tracer, PEPT allows for the tracking of a single particle inside a closed system without interfering with the process. PEPT has been successfully used for describing mineral processing systems such as mills (Bbosa *et al.*, 2010), hydrocyclones (Chang *et al.*, 2011), and flotation cells (Waters *et al.*, 2008), and the technique is gaining momentum as a research option.

PEPT makes use of a radio-isotope tracer that decays through the beta-plus mechanism and emits a positron, the positive counterpart of an electron. When an electron collides with a positron, it is annihilated, releasing two back-to-back 511 keV γ-rays, 180° apart within ±0.3° (Parker and McNeil, 1995). When a particle that emits these gamma rays is placed inside a cylindrical array of detectors, its position can be determined by extrapolating lines from the points where the gamma rays are detected and then finding the positions where these line cross (Figure 1).

PEPT monitors the behaviour of a single particle inside a jig. Real-life scenarios can be emulated and the movement of tracers with different shape, size, and density can be compared under a range of operating conditions. The initial objective of the investigation was to see whether suitable results can be obtained from the PEPT technique when testing an existing iron ore jig feed.

A batch jig was used with a cylindrical jigging chamber with an inside diameter of 160 mm. The pulse was generated by a PowerRod Linear Actuator (PRA), which offers better control compared to air cylinders by making use of a magnetic drive to propel the cylinder rod.

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Positron emission particle tracking inside a laboratory batch jig

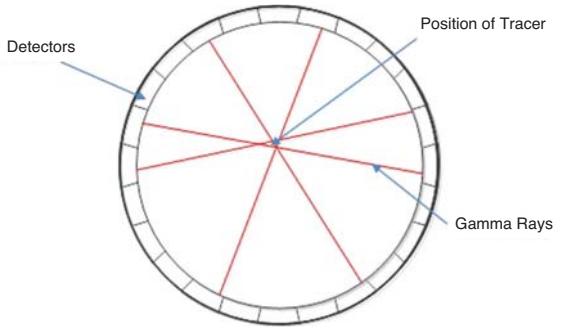

Figure 1—Cylindrical arrangement of detectors in a positron emission tomography camera

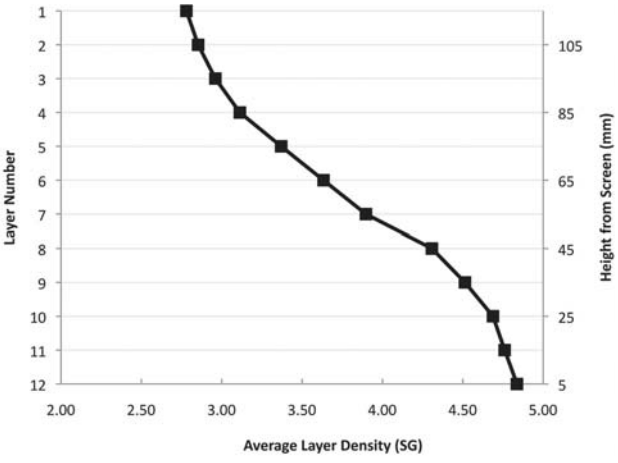

Figure 2—Density profile after stratification

Experimental

The iron ore sample was screened to 5–8 mm to minimize the effect of size. Initial test work was conducted to obtain information on the density distribution. The sample was jigged for 10 minutes, after which it was removed in layers and the density of each layer was analysed. The results are shown in Figure 2.

Tracer particles were selected from the sample. Their density and size was measured, and a small hole was drilled in each particle to accommodate the radio-isotope. Before each series of test runs commenced, the tracers were prepared by inserting Ga^{68} isotopes inside the iron ore tracer particle. The half-life of Ga^{68} allowed a six-hour window for test work on one tracer. The jig was filled to a bed height of 140 mm, which corresponded to approximately 8 kg of iron ore. The tracer was then placed in position and water was added to ensure that there was at least 50 mm of water above the jig bed during the entire jig cycle.

The only variables that were changed during these tests were the tracer particle shape, size, density, and starting position. Operating conditions of the jig are shown in Table I. The conditions chosen were based on preliminary test work and gave sufficient separation within the practical time frame. Four different tracers at various starting positions were tested. Table II shows the properties of the tracers.

Table I

Jig operating parameter

Pulse height	35 mm
Upward pulse velocity	300 mm/s
Top hold time	160 ms
Downward pulse velocity	100 mm/s
Bottom hold time	100 ms
Run time	10 min

Table II

Properties of tracers

Tracer	1	2	3	4
Density (SG)	5.01	2.92	4.11	3.99
Weight (g)	1.20	0.63	1.24	1.51
Size (mm)	6.53	5.39	4.15	6.65
Shape	Equant	Tabular	Bladed	Equant

Results and discussion

The results that follow represent typical results obtained during the tests.

General stratification

The top and side view of the jigging chamber are shown in Figure 3, with the trajectory of a tracer particle (density 5.01). To view the movement of the particle more clearly, the pulse movement of the particle is subtracted from its trajectory by using a background correction technique similar to that used during XRF data analysis. The particle started at the top of the chamber and moved down the vertical axis until it reached the bottom, where it started to move randomly in the horizontal plane.

From the three-dimensional data, the most important movement component for modelling purposes is the vertical component. Figure 4 shows the vertical movement of the tracer particle *versus* time. The first important feature of this

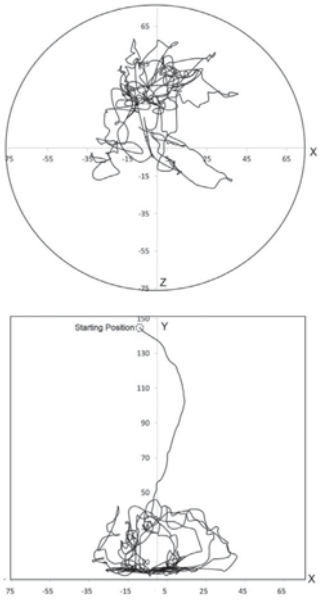

Figure 3—Movement of a particle (density 5.01) inside a batch jig

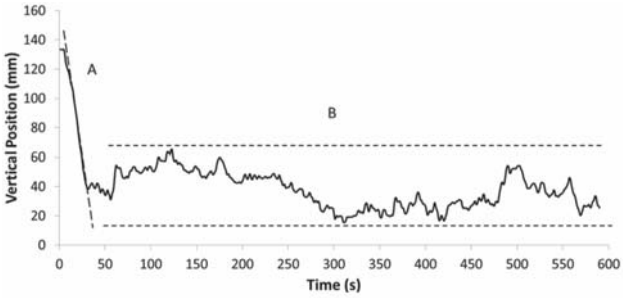

Figure 4—Vertical movement of a tracer particle (density 3.99)

curve is the initial movement of the tracer. The slope of this line gives an indication of the stratification rate of the tracer. Unfortunately, the data-set was too small to draw any conclusion on the effect of different variables on the stratification rate. The second important feature is the movement of the particle after it has reached its stratification position. The tracer will continue to move up and down in a band along the vertical axis (Figure 4, curve 'B'). A frequency plot of the tracer position in this region forms a normal distribution around a centre point, providing the statistical probability of where the tracer will end up.

Individual pulse

The resolution obtained from the PEPT camera is high enough to observe the tracer movement during a single pulse. The particle starts settling as soon as the upward pulse ends (Figure 5); about halfway down the particle seems to stop and remains stationary for about 100 ms. This is probably due to the 'kickback' that the particle experiences from the jig – the initial downward movement of the particle bed exerts a force on the piston that pushes it back slightly. The piston pushes back to produce an upward flow that causes enough drag on the bed to hold it stationary for a short time before the bed moves again. When the particle is at the bottom of the bed this effect is not evident (Figure 6); the particles at the bottom of the jig are not trapped within the bed and can settle even against the slight upward flow from the kickback.

Effect of starting position

There is a definite difference in the behaviour of tracer particles started at different positions in the jig bed. The following cases were considered: a high-density particle (density 5.01) with two starting positions, at the top centre and top side of the bed, and a low-density particle (density 2.92) with centre and side positions at the bottom of the bed. To investigate the effect of starting position on the stratification rate, the time to the final equilibrium position was noted (Table III). The settling rate for the heavy particle started at the sidewall is significantly lower than that of the particle started at the centre. There is no clear difference in settling rates seen for the lighter particle.

Another interesting phenomenon observed when comparing the movement of the tracers from different starting positions is illustrated in Figure 7. Heavy particles starting at the side of the jig chamber have a tendency to move to the centre as they settle, and light particles started at the centre on the bottom move to the side. This indicates that a secondary flow field is generated in the jig, as suggested by the results obtained by Williams *et al.* (1998), who

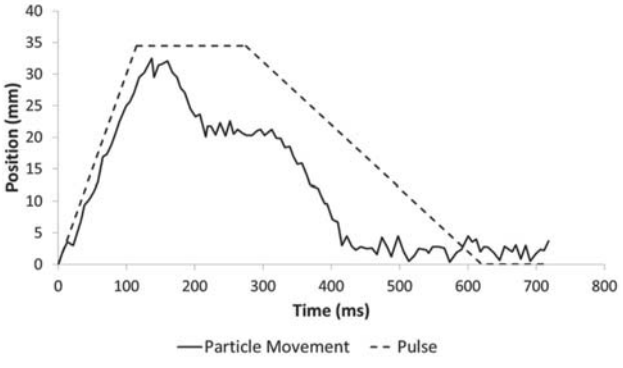

Figure 5—Jig pulse and particle movement with time

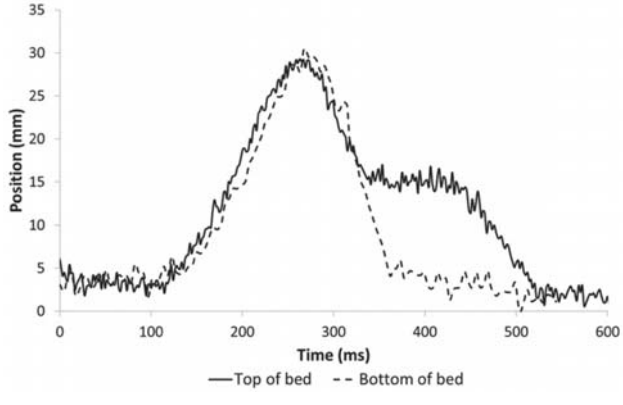

Figure 6—Particle movement with time at the top and bottom of the bed during a single pulse

Table III

Time to stratification at different starting positions

	Time to equilibrium position (sec)			
	Density 5.01 (top centre)	Density 5.01 (top side)	Density 2.92 (bottom centre)	Density 2.92 (bottom side)
Test 1	75	150	63	50
Test 2	63	144	50	50
Test 3	83	122	45	44
Test 4	50	110	25	44
Average	68	132	46	47
Standard deviation	14.5	18.7	15.8	3.5

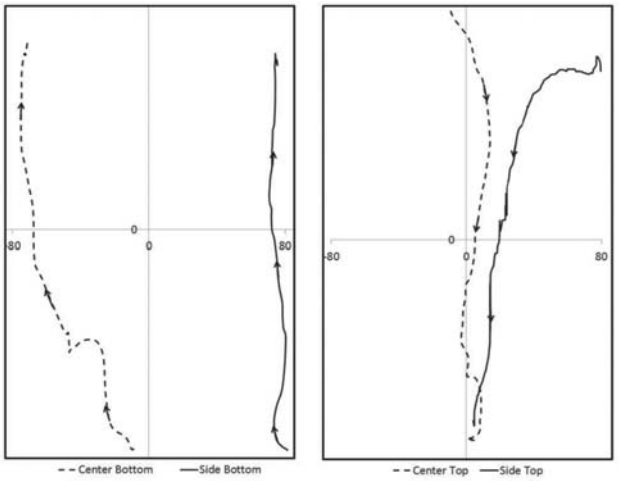

Figure 7—XY plane of the jig. (a) Movement of light particle, (b) movement of heavy particle

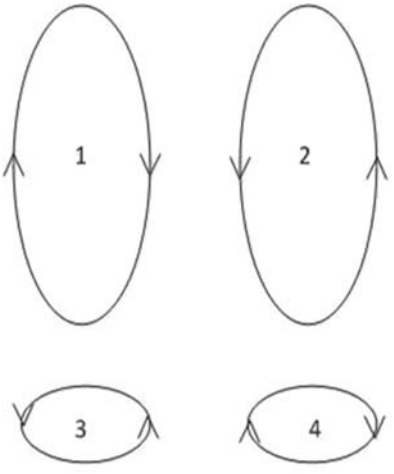

Figure 8—Diagram of flow patterns in the XY plane (Williams *et al.*, 1998)

discovered these flow patterns in a laboratory-scale jig using glass beads, with one of the beads containing the PEPT tracer. The flow fields discovered by Williams *et al.* (1998) shown in Figure 8 can explain some of the behaviour observed in this experiment. The heavy tracer at the side experiences a secondary upward flow that slows its stratification rate. Surprisingly, this effect is not observed to the same extent on the light tracer, which indicates that there are other factors involved.

Jigging is widely used in the mineral processing industry due to its low cost and simplicity of operation. However, jigging lacks the separation efficiency of some of the available technologies such as dense medium separation. For jigs to remain a viable option, their separation efficiencies have to be improved. This can be done either by intelligent operation based on the properties of the feed material or by optimizing the physical design of the jig. This study shows that PEPT should be able to provide data that can be used in generating models that will be useful for both design and operation of jigs.

Optimizing jig design can be an extensive exercise, since physical changes have to be made, which is typically done only when a serious problem arises; In the future, numerical modelling (discrete element modelling and computational fluid dynamics) will make jig design a much easier task, and the kinetic information (Figure 4) generated from PEPT is the ideal data to use when developing such models. PEPT might be able to aid in the development of material-specific models to predict retention time, and possibly efficiency, from data obtained on the feed material. These models can be very useful from an operational standpoint.

Conclusion

Test work on a laboratory jig using PEPT technology shows significant promise for improving understanding the jigging process. It provides new insights into a very ancient technique and presents the opportunity to re-evaluate old theories and develop new ones. The data generated by the PEPT technique can be used to validate theoretical models and to provide insight into specific industry problems, since a real ore can be tested. The resolution from the PEPT technique is such the movement of a particle can be tracked during an individual pulse of the jig. The particle trajectories suggest that there exist additional factors that strongly affect stratification, which require more consideration during further studies.

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The art and science of dense medium selection

by J. Bosman*

Synopsis

Medium suspensions play an integral part in the successful application of dense medium separation for both static and dynamic separators. Although models exist to predict particle movement as a function of medium density and viscosity, these models have been derived on the assumption of Newtonian rheology. Viscosity measurements of medium suspensions have shown that they are non-Newtonian, with a yield stress, and as such cannot be used in the existing models. As a result, medium selection remains an art based upon practical experience and indirect measurements of viscosity such as stability and cyclone differentials.

Keywords
DMS medium, rheology, dense medium drum, dense medium cyclone.

Introduction

Dense medium separation (DMS) is a process whereby particles are sorted primarily on the basis of their densities. Particles with a wide range of densities are introduced into a medium suspension of a given density. Particles that are lighter than the medium density rise. These are commonly referred to as floats. Particles that are heavier than the medium density sink, and these are commonly referred to as sinks (Gupta and Yan, 2006, p. 527).

The process of DMS provides a basis for the upgrading of a mineral or ore. Some examples are listed below:

- Diamonds – these have a particle density of 3.6 t/m^3 and are separated from either kimberlite, with a particle density of 2.6 t/m^3 or alluvial material with a density of 2.65 t/m^3
- Iron ore – in the form of haematite, pure iron ore has a particle density of 5.2 t/m^3 and it is separated from siliceous material with a particle density of 2.65 t/m^3.

The mixture of ore and medium is introduced into separating vessels within which the separation take place. These can either be static (where gravity is the driving force behind the separation) or dynamic (where centrifugal forces are the driving force and the particles experience multiple *g* forces).

Medium suspensions

Over the years, many different types of suspensions have been used for DMS. These can be broadly classed into homogeneous and heterogeneous suspensions.

Homogeneous suspensions consist of a single phase. Typical examples are organic liquids and salt solutions. Unfortunately, organic liquids are carcinogenic and nowadays are used only under controlled conditions in a laboratory environment. Salt solutions tend to be corrosive and are also harmful to exposed skin and eyes.

Heterogeneous suspensions normally consist of two phases, i.e. a liquid phase (normally water) and a solid phase which is suspended in the liquid phase. Some of the solid phases that have been used are sand, galena, baryte, magnetite, and ferrosilicon (Gupta and Yan , 2006, p. 530).

The most important properties of a suspension are its stability and viscosity.

Stability refers to the tendency of the solid phase in a heterogeneous suspension to settle out. This can be as a result of gravity alone (in static separators) or multiple *g* forces (in dynamic separators).

Stability is normally measured by allowing a suspension to settle under gravity and the mudline (interface between the solid and water phase) is tracked as a function of time. The more stable the suspension, the slower it will settle, and the more unstable a medium the faster it will settle (Grobler, Sandenbergh, and Pistorius, 2002, p. 84).

Medium stability can be used to compare the relative stabilities of suspensions with different solid properties and densities.

Viscosity can be interpreted as resistance

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to flow. The higher the viscosity of a suspension, the more difficult it is for a particle to move through it, and the lower the viscosity, the easier it is for a particle to move through it

Viscosity is measured by applying a force to a suspension and measuring the resultant shear rate. Viscosity is defined as the ratio of shear stress to shear rate. The various classifications of suspensions in viscosity terms are shown in Figure 1.

Ideal suspensions occur when the shear stress is directly proportional to the shear rate, and these are known as Newtonian fluids. Water is a Newtonian fluid with a viscosity of 1 centipoise (cP) or 0.001 Pa.s. The effect of Newtonian fluids is constant and predictable as the viscosity does not vary with shear rate.

A Bingham plastic requires a certain amount of force (stress) to be applied before it starts shearing. This is known as the yield stress. Toothpaste is a common example of such a plastic. A certain force must be applied to the tube before the paste begins to flow.

In shear thinning suspensions, the viscosity decreases with increasing shear rate, and in shear thickening suspensions the viscosity increases with shear rate.

Tomato sauce is an example of a shear thinning suspension. When squeezed out a bottle, it will thin out and flow, but once on the plate, it will retain its shape. Wet sand completely soaked with water (as on a beach) is an example of a shear thickening suspension, which is why when you walk on it, a dry area appears under your foot.

Heterogeneous suspensions such as those used for dense media tend to have a yield stress. The quantity of solid phase in the suspension is defined by the density of the suspension and is commonly referred to as the medium density. The flow curve for a given medium where only the medium density is varied is shown in Figure 2.

Apparent viscosity is defined as the viscosity at a given shear rate. The apparent viscosity curves corresponding to Figure 2 are shown in Figure 3.

From the apparent viscosity curves, increasing the medium density leads to an increase in the viscosity. Depending upon medium density, the viscosity behaviour can range from shear thickening to Newtonian to shear thinning.

Medium viscosity and stability are directly related. High stabilities correspond to high viscosities and low stabilities correspond to low viscosities.

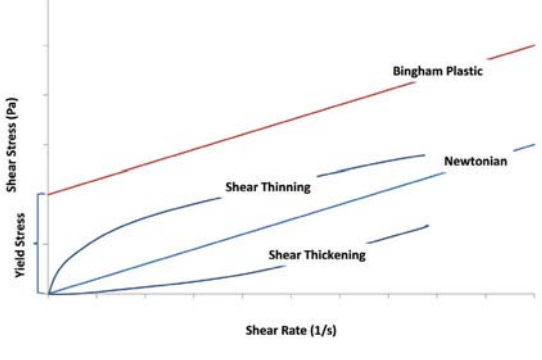

Figure 1—Classification of suspensions according to viscosity (after (Collins, Napier-Munn, and Sciarone, 1974, p. 105)

For high-density separations, i.e. greater than 2.6 t/m³, ferrosilicon is widely used. Ferrosilicon is a ferroalloy consisting of iron and silica in a very specific ratio. Although iron has a high particle density, it corrodes and in its pure form is not suitable as a solid phase in a suspension. Silica is added to improve the corrosion resistance of the iron. The only problem is that the more silica that is added, the lower the resultant density of the ferroalloy is. A silica content of 14–16% in the alloy has proven to provide the best corrosion protection without too great a reduction in the particle density (Collins, Napier-Munn, and Sciarone., 1974, p. 110).

Particle shape and particle size play an important role in the stability/viscosity of ferrosilicon suspensions (Collins, Napier-Munn, and Sciarone, 1976, p. 105).

Particle shape is a result of the production process. Milled ferrosilicon particles are angular in shape. Water atomization produces particle that are of a spherical nature. Gas atomization produces highly spherical particles. Example of the three particle shapes are shown in Figure 4.

For the same particle size and suspension density, viscosity and stability decrease as the particle sphericity increases (i.e. when changing from milled to water-atomized to gas-atomized particles).

For each particle shape, there are a range of particle sizes. The maximum particle size is typically 212 μm. From a practical point of view, the percentage passing 45 μm is often used to differentiate size distributions. This is due to the fact that dry screening at 45 μm is quick and simple. Commercially available size ranges associated with the three particle shapes are shown in Figure 5.

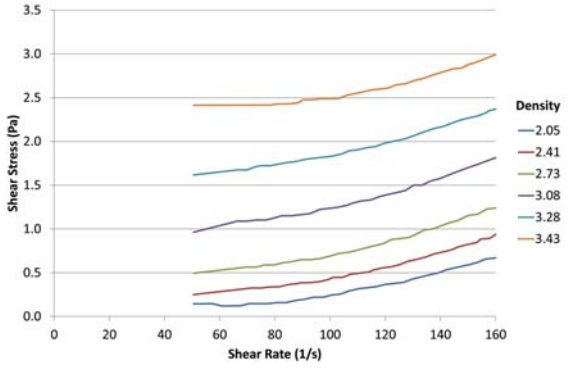

Figure 2—Medium suspension flow curve (after (Shi and Napier-Munn, 1996, p. 119)

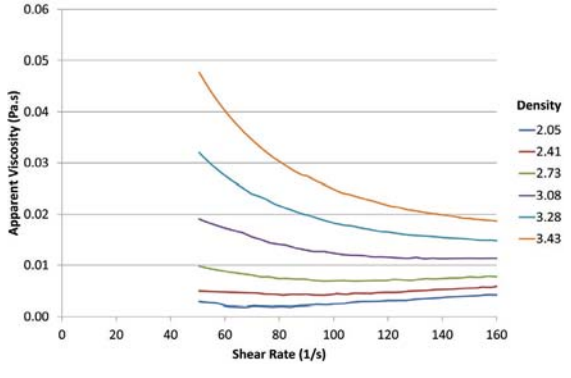

Figure 3—Apparent viscosity curves

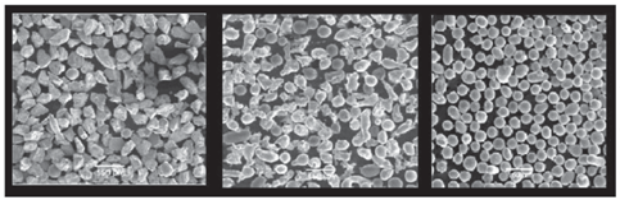

Figure 4—Milled, water-atomized, and gas-atomized ferrosilicon particle

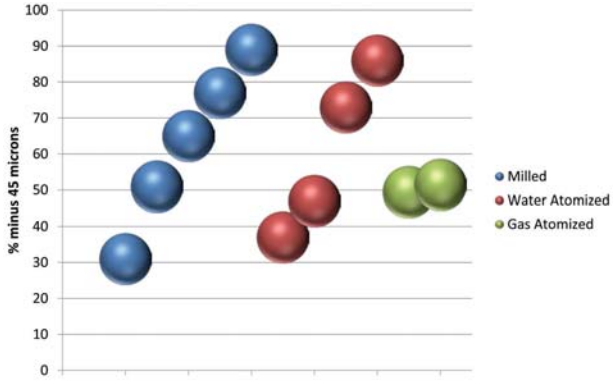

Figure 5—Ferrosilicon particle sizes

Milled and water-atomized ferrosilicon cover similar size ranges, whereas the gas-atomized ferrosilicon is found in the middle of the range.

Separators

Separators can be divided into two main classes, i.e. static separators, in which the separation process takes place under the force of gravity (1 *g*), and dynamic separators where the separation takes place under the influence of multiple *g* forces. These multiple *g* forces are generated by accelerating the mixture of ore and medium within the separating vessel (Wills and Napier-Munn, 2006, p. 248).

Static separators

Although a wide range of static separating vessels have been used over the years, one of the more widely used units is the WEMCO drum. WEMCO is an acronym for the Western Machinery Company, which patented the WEMCO drum in 1954 (Maust, 1954). Drawings from the patent are shown in Figures 6 and 7.

In the following description, the numbers in parentheses refer to the labels in Figures 6 and 7. In the WEMCO drum, ore and medium are introduced into the drum at the feed end (17). The drum shown in Figure 7 is rotating in a counter-clockwise direction. The drum is fitted with lifters along the length of the drum (11). The heavier particles settle out and the lighter particles float. As the heavier particles settle to the bottom of the drum, they are trapped in the lifters, which carry them to the top of the drum. At the top of the drum they fall under gravity into the sinks launder (20). Medium is added to the sinks launder to flush the heavy particles out of the drum. The lighter particles float to the top of the medium and are carried out of the drum with the medium flow through the outlet (19). The floating particles are kept away from the lifters by skirts (15).

The distance from the bottom of the drum to the level of the medium in the drum (indicated by the symbol b) determines the maximum distance that the particles must travel to sink or float. Common practice is to feed the particles into the drum at half this distance, thereby ensuring that the distance that the heavier and the lighter particles must travel is the same.

The ferrosilicon particles in the medium suspension are subject to gravity and tend to settle out of the water suspension. The mixing action caused by the lifters as they slowly move through the medium helps to offset this to a degree, However, if the size distribution of the ferrosilicon is too coarse, settling will occur and an unstable medium with resultant efficiency losses will result. On the other hand, if the size distribution is too fine, the medium will become too viscous and efficiency will also be adversely affected.

Dynamic separators

A number of dynamic separators have also been used over the years, but the unit that is most widely used is the dense medium cyclone. It was first patented in 1942 by the Dutch

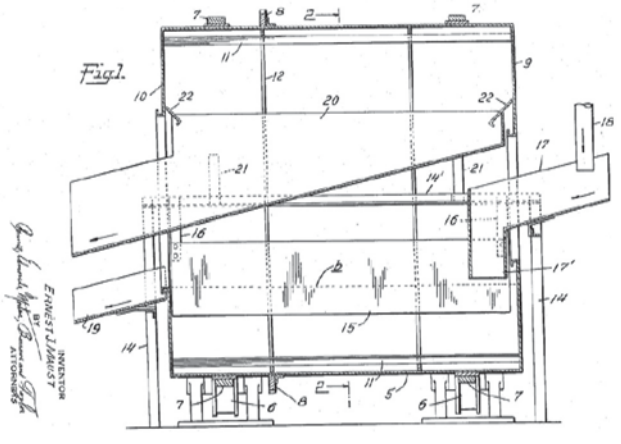

Figure 6—WEMCO drum separator – side view

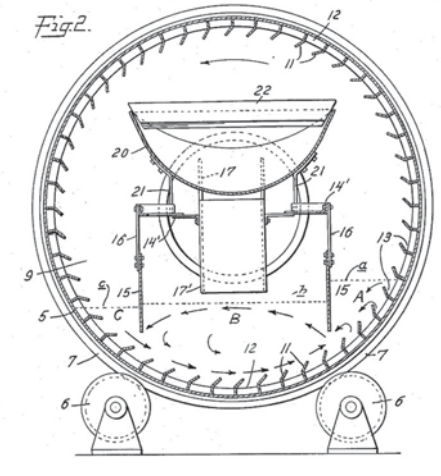

Figure 7—WEMCO drum separator – cross-section

State Mines and is often referred to as the DSM cyclone. A drawing from the original patent (Driessen, Krijgsman, and Leeman, 1951) is shown in Figure 8.

What is interesting to note is the lack of a vortex finder in the cyclone. This was introduced in 1948.

In the cyclone, a mixture of ore and medium is introduced into the feed inlet (15) under pressure. The pressure is supplied by either gravity or a pump feed. As the mixture enters the cyclone, the cylindrical inlet head (11) forces it to start rotating. As it rotates the ore is subject to a centrifugal force, which causes the larger and heavier particles to move to the outer wall of the cyclone, from where they are discharged through the spigot or apex of the cyclone (16). The finer and lighter particles are carried to the overflow (13) via the vortex finder.

An important point is that the ferrosilicon particles, which make up the solid phase of the medium suspension, are also subjected to the centrifugal force. This force classifies the ferrosilicon, with the coarser particles reporting to the cyclone underflow and the finer particles to the cyclone overflow. As a result, the cyclone overflow is of a lower density than the feed medium and the cyclone underflow is of a higher density. The differences in medium densities for the three cyclone streams are known as differentials. If the differentials become too large, the medium within the cyclone becomes unstable and efficiency is adversely affected. This occurs when the size distribution of the ferrosilicon is too coarse. On the other hand, if the size distribution is too fine, the medium will become too viscous and efficiency will also be adversely affected. In this case the differentials, especially between feed and overflow, reduce to zero i.e. the feed and overflow medium densities are the same.

Modelling separation

For both static and dynamic separators, separation is a result of particles moving through the medium. Modelling this process is different for static and dynamic separators due to the nature of the separators and the magnitude of the gravitational force that is applied to the separator.

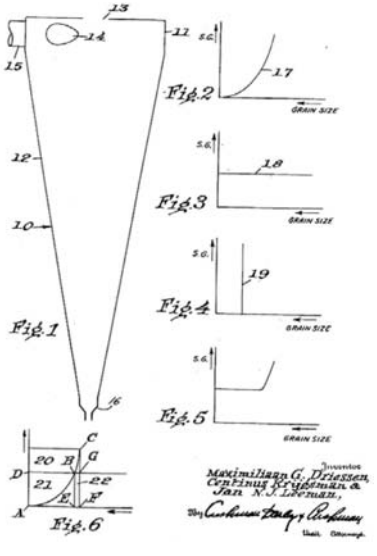

Figure 8—Original DSM cyclone patent

Static separators

In the Stokes regime, the terminal or free settling velocity of spherical particles in fluids of known density and viscosity can be described by the following equation (Fuerstenau and Han, 2003, p. 174):

$$v_t = \frac{\mu}{\rho_f d} 10^A$$

where
v_t – terminal or free settling velocity (cm/s)
μ – fluid viscosity (cP)
ρ_f – fluid density (g/cm3)
d – particle diameter (cm)
A – Archimedes number, which is given by:

$$A = 5\left[0.66 + 0.4\log\left(4/3Ga\right)\right]^{0.5} - 5.55$$

Ga – Galileo number, which is given by:

$$Ga = \frac{gd^3\rho_f\left(\rho_s - \rho^f\right)}{\mu^2}$$

ρ_s – particle density (g/cm3)
g – acceleration due to gravity (cm/s2)

Particles do not reach their free falling or terminal settling velocity instantaneously, but experience an initial period of acceleration until the terminal velocity is achieved.

The particle velocity as a function of time can be described by:

$$v = v_t\left[1 - \exp\left(-kt\right)\right]$$

with
v – velocity (cm/s)
k is given by:

$$k = \frac{9\mu}{2r^2\rho_s}$$

r – particle radius (cm)

The time required to achieve the terminal settling velocity is a strong function of the fluid density and viscosity as well as the particle size.

Free settling occurs when the volumetric concentration of particles in the fluid is less than 1%. If the concentration is higher, the particles interfere with one another and the settling rate is reduced. This is referred to as hindered settling. A correction factor C_F can be applied to the free settling velocity v_F to account for this reduction and calculate the hindered settling velocity v_H.

$$v_H = C_F \times v_F$$

The correction factor for hindered settling is a function of the volumetric concentration (γ) of the ore in the ore/medium mixture. The factor is a function of the volumetric fraction (γ):

For $\gamma < 0.3$:

$$C_F = \frac{(1-\gamma)^2}{10^{(1.82\gamma)}}$$

For $0.3 < \gamma < 0.7$

$$C_F = 0.123\frac{(1-\gamma)^3}{\gamma}$$

Dynamic separators

The simplest model for particle motion in a dynamic separator such as a cyclone is given by the equilibrium orbit theory (Wills and Napier-Munn, 2006, p. 216).

$$d = \sqrt{\frac{18\mu W r}{(\rho_s - \rho_l)V_t^2}}$$

where:
d – particle diameter (cm)
ρ_s – particle density (g/cm3)
ρ_l – liquid density (g/cm3)
V_t – tangential velocity of the particle (cm/s)
r – radius of the equilibrium orbit (cm)
μ - liquid viscosity (cP)
W – radial velocity (cm/s).

Unfortunately, this simple model is not of great value as the practical determination of the radial and tangential velocities and the radius of the equilibrium orbit for a given cyclone geometry and operating conditions is problematical.

Of all the classification cyclone models that have been formulated since the 1950s, the two that remain the most widely used are the Plitt model and the Nageswararao model. These models are semi-empirical in nature and are the result of applying regression analysis to measured data. The influence of viscosity has been studied and the results have incorporated into a modified Plitt model with the following result (Kawatra, Bakshi, and Rusesky, 1996, p. 46):

$$d_{50c} = \frac{K_2 D_c^{0.46} D_i^{0.6} D_o^{1.21} \varphi^{0.41} \mu^{0.35}}{D_u^{0.71} h^{0.38} Q^{0.45} (\rho_s - \rho_l)^{0.5}}$$

where
d_{50c} – particle size with an equal chance of reporting to either cyclone underflow or overflow (µm)
K_2 – calibration factor
D_c – cyclone diameter (cm)
D_o – vortex finder diameter (cm)
D_i – inlet diameter (cm)
Ψ - % volumetric solids in the cyclone feed
μ - liquid viscosity (cP)
D_u – underflow or spigot diameter (cm)
h – free vortex height (cm)
Q – cyclone volumetric flow rate (l/min)
ρ_s – particle density (g/cm3)
ρ_l – liquid density (g/cm3).

Summary and comparison of models

The models can be summarized and compared using the following factors.

Medium phase

For both models, this is described by the following parameters:
ρ_l – the medium density (g/cm3)
μ – the medium viscosity (cP)

Geometry

For the static (drum) model, the distance that a particle has to move to become either floats or sinks is the major variable. For modelling purposes, it is assumed that the feed is introduced halfway between the level of medium in the drum and the bottom of the drum. The distance that a float and a sink particle have to travel is the same and equal to half the distance between the level of medium and the bottom of the drum. The primary variable is therefore drum diameter.

The drum length, as it determines residence time, is the secondary variable.

For the dynamic separator (cyclone), the geometry is uniquely defined by the following parameters:

- Cyclone diameter
- Inlet diameter
- Vortex finder diameter
- Barrel section (defined by barrel length)
- Cone section (defined by internal cone angle)
- Spigot diameter.

In the absence of detailed cyclone drawings, the free vortex height (h) can be approximated by the distance from the bottom of the inlet head to the spigot.

Operational

For the drum, the main operational parameters are circulating medium density and residence time.

For the cyclone, the main operational variables are circulating medium density and inlet pressure or head.

Viscosity determination

The determination of the viscosity of settling slurries, regardless of whether the solid phase is an ore or ferrosilicon, remains a major problem.

Viscosity measurement

Although a number of different methods have been tried, the one that is most applicable to dense media in general and ferrosilicon in particular is that developed by Frank Shi while working at the Julius Kruttschnitt Mineral Research Centre in Australia (Shi and Napier-Munn,1996, p. 118).

The concept was originally developed by the De Beers Research Laboratory and makes use of a modified Stormer viscometer as shown in Figure 9.

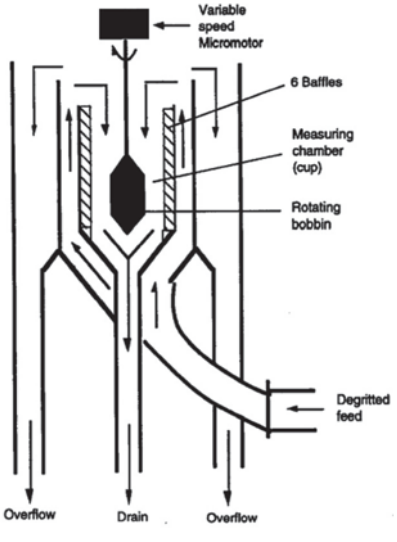

Figure 9—Modified Stormer viscometer

The concept was then taken further by Shi, who developed a model whereby the shear rate/shear stress relationship for a given medium can be modelled using the particle size distribution (laser-based *p*80 and *p*20), shape (milled or atomized), temperature, and medium density as inputs.

The model is summarized graphically in Figure 10.

Measured *versus* predicted data for 65D milled ferrosilicon is shown in Figure 11.

Stability measurement

The stability of medium suspensions is also used as an indication of viscosity.

The test is very simple to conduct and does not require expensive laboratory equipment. The procedure is as follows:

- In a 250 ml measuring cylinder, make up 250 ml of medium at the required density
- Seal the top of the measuring cylinder and invert it 10 times to ensure that the medium is well mixed
- Place the measuring cylinder right side up on a table
- Track the mudline (interface between the medium and water) and record the time as it passes each gradation on the cylinder (on a standard 250 ml measuring cylinder, each gradation is 1 mm, but it is worthwhile checking)
- Plot the results on a graph and determine the slope of the resultant line where it is linear. The slope is the stability value.

The experimental set-up is shown in Figure 12.

A typical result is shown in Figure 13.

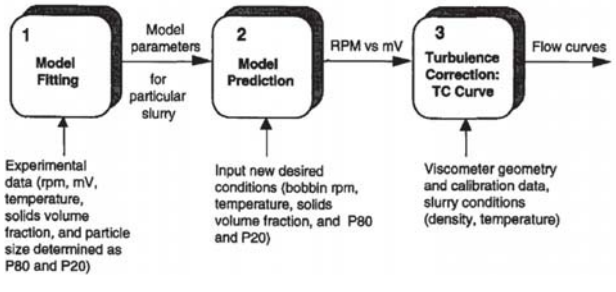

Figure 10—Slurry rheology model

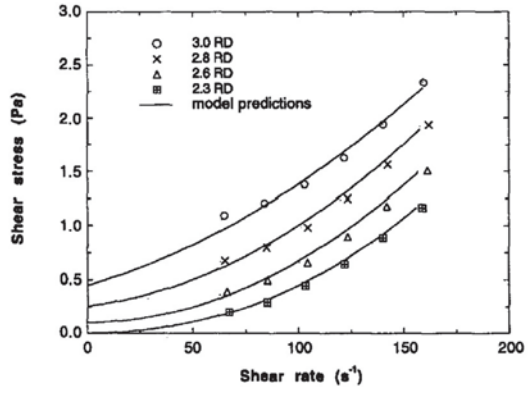

Figure 11—Measured versus predicted data for 65D milled ferrosilicon

Medium selection

Static (drum) separators

The science

The model that has been described includes a medium density and a viscosity term. Provided that the medium is relatively stable (i.e. does not settle too quickly), the density can be assumed to be constant throughout the separator.

In the derivation of the equations to model particle velocity, the viscosity term is based upon Newtonian slurry. Using Shi's slurry rheology model, the viscosity of a medium suspension can show Newtonian behaviour for a small range in density, but in general is either shear thinning or shear thickening depending upon the medium density. It also exhibits a yield stress.

If the medium in the static separator is exposed to a constant shear rate, then the apparent viscosity of the medium suspension for this shear rate can be legitimately used for the viscosity term in the model.

The assumption that a constant shear rate applies throughout the separator has yet to be proven, and appears to be unlikely. At the lifter/medium interface, the shear rate will certainly be different to that just below the surface of the medium.

Figure 12—Stability measurement experimental set-up

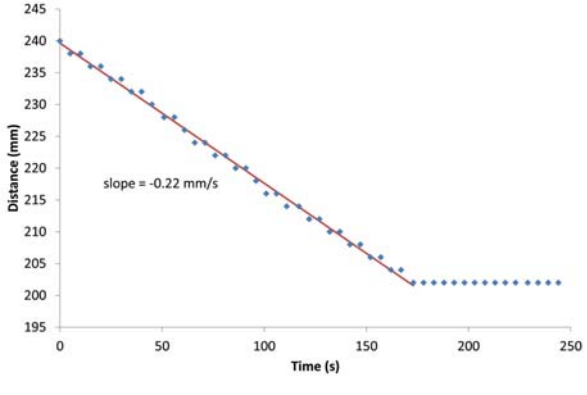

Figure 13—Stability test plot

It is thus not possible to directly use the Newtonian-based models for particle transport as a tool to predict the effect of different medium suspensions.

The art

From an operational point of view, it is fairly easy to determine if the medium suspension used in a static separator provides an acceptable separation. This is done by monitoring the quality of the product and waste streams and identifying misplaced material. Furthermore, by focusing on the performance of the finer particle sizes, the effect of the viscosity of the medium can be inferred. Increases in viscosity lead to greater increases in misplacement for finer particles compared to coarser particles, due to their lower particle mass.

Once the correct grade and density of medium suspension have been identified, a stability measurement can be performed and the settling rate determined. This settling rate can then be used as a parameter to indirectly control the viscosity of the medium.

Dynamic cyclone separators

The science

The viscosity-modified Plitt model was developed based upon the assumption that most mineral suspensions are Newtonian below 50% volumetric solids. The work done by Shi and his slurry rheology model have, however, proven this to be invalid for medium suspensions.

The shear rate within a cyclone can be described using the following equation (Kawatra, Bakshi, and Rusesky, 1996, p. 42):

$$S = \frac{dV_t}{d_r}$$

where

S – shear rate (s^{-1})

$\frac{dV_t}{d_r}$ – rate of change of tangential velocity with changing radius.

The maximum shear rate has been shown to occur close to the vortex finder, and the value decreases as the medium suspension moves closer to the wall of the cyclone. This variation in shear rate within the cyclone makes it impossible to use a single apparent viscosity value (corresponding to a given shear rate) in the modified Plitt equation to predict the cyclone cut size or d_{50}. However, if for a given medium grade and density the viscosity is close to Newtonian, this value can be used.

From the apparent viscosity curves generated by the Shi rheological model, there are a limited number of combinations of medium grade and density where this occurs.

The art

For a homogeneous medium, the feed, overflow, and underflow densities would be the same as there is no solid phase for the cyclone to classify. However, if this situation occurs with a medium suspension, it implies that the viscosity of the medium suspension is so high that the cyclone is unable to classify the solid phase of the medium. Operating practice has shown that if this is the case, then the efficiency of separation of an ore within a cyclone is also adversely affected.

Practical experience has shown that if the difference in the density of the cyclone feed and cyclone overflow is less than 3%, the medium is too viscous and separation efficiency is adversely affected.

If the medium suspension fed to the cyclone is too unstable, it classifies to an excessive degree within the cyclone. As a result, a range of densities occurs within the cyclone, with the lowest being close to the vortex finder and cyclone overflow where the maximum tangential velocity occurs, and the highest density towards the wall of the cyclone and the underflow. This also adversely affects the separation efficiency as particles build up within the cyclone due to the large density differences, and when the cyclone becomes too crowded they split randomly to the overflow and underflow.

Once again, practical experience has shown that if the difference in the density of the cyclone feed and cyclone overflow is more than 12%, the medium is too unstable and separation efficiency is adversely affected. The result is shown graphically in Figure 14.

Based upon the above, the cyclone can thus be used as a viscometer to ensure that the grade of the medium suspension that has been selected is correct. This is done by measuring the difference between the cyclone feed and overflow densities and calculating the percentage difference. As long as this difference is maintained between 3% and 12%, the viscosity of the medium suspension will not adversely affect the separation efficiency.

Conclusion

Although models for particle movement in both static and dynamic separators are available, their derivation is based upon Newtonian rheology.

Using the only definitive predictive rheology model that is currently available, the Shi model, the rheology of medium suspensions is non-Newtonian and also has a yield stress. The existing models thus cannot be used to predict the effect of different medium suspensions for static and dynamic separators.

For static separators, one must rely on the stability measurement of medium suspensions that have already been proven by efficiency analysis to be applicable as an indirect measure of medium viscosity.

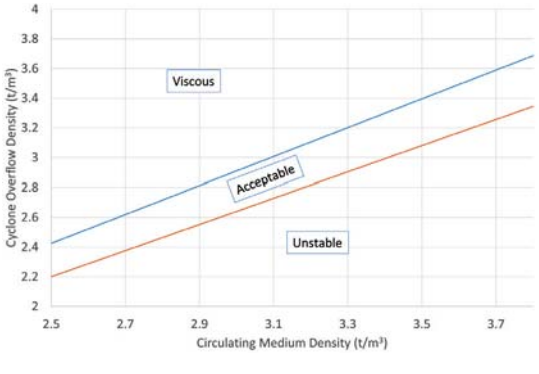

Figure 14—Cyclone feed – overflow differential plot

The art and science of dense medium selection

For dynamic separators, such as cyclones, the unit itself is used as a viscometer and the viscosity of the medium is inferred from the cyclone feed overflow differential.

Medium selection remains an art with a touch of science.

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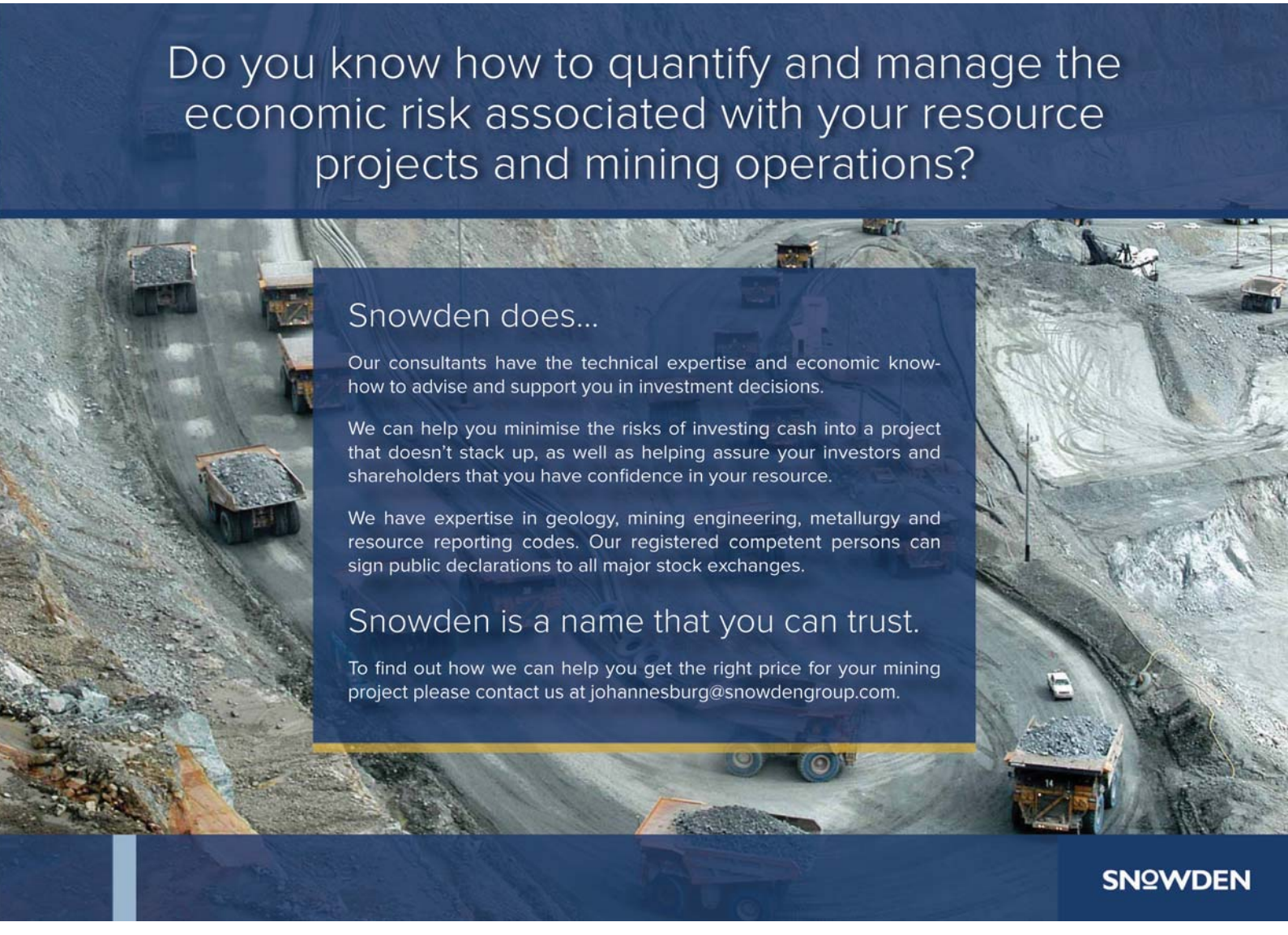

Current trends in the development of new or optimization of existing diamond processing plants, with focus on beneficiation

by P. van der Westhuyzen*, W. Bouwer*, and A. Jakins*

Synopsis

Gone are the days of stock-standard diamond beneficiation and final recovery circuits that could, with minor modifications, be adapted to treat any diamond-bearing ore source. A new era has dawned, offering the opportunity to streamline existing diamond processing operations or develop simpler, more efficient, and economical diamond processing plants, with the focus on more efficient comminution, beneficiation, and final recovery. This change in scene has been brought about by:

- The introduction, rapid development, and maturation of multiple comminution, sorting, and recovery technologies
- The need to adapt to a new standard of project approach *post* the commodity super-cycle phase, where optimizing existing operations and developing scalable, 'fit for purpose' new mines are fast becoming the norm in diamond processing plants in both primary and alluvial operations
- The quest for energy efficiency and lower labour costs
- More remote and inaccessible reserves (under lakes, tops of mountains etc.).

This paper serves to identify some technology advances and demonstrates how these could be considered as replacements for or in combination with conventional technologies to arrive at an optimum techno-economic solution. To name a few applications/technologies:

- Comminution: conventional cone crusher, modified/specialized cone crusher, and the high-pressure grinding roll (HPGR). Although significant advances have been made in recent years, this paper only briefly covers comminution within the beneficiation circuit design
- Waste sorting: NIR (near-infrared) sorting, optical (colour) sorting, XRF (X-ray fluorescence)
- Primary concentration by combining dense media separation (DMS) with either XRT (X-ray transmissive), pulsed X-ray, jigs, or pan plants depending on the application, scale, and economics
- Final recovery of diamonds with either conventional X-ray technology, pulsed X-ray technology, or XRT.

From recent studies, it can be concluded that there is no longer a standard solution, but rather the 'right' or appropriate solution. Through combining a sound knowledge of the ore source (ore dressing studies or on-mine data gathering) and leveraging off the advances in technologies, one is able, through trade-off studies, to arrive at the ultimate techno-economic configuration, the 'right' solution. Emphasis is placed on maximizing diamond recoveries through appropriate technology selection and minimizing the associated costs in an effort to de-bottleneck or improve efficiencies of existing diamond processing plants, or to arrive at the ideal new diamond process plant design..

Keywords
diamond recovery, process design, diamond concentration, optimization.

Introduction

The development of any diamond resource project inevitably requires the investigation of a vast range of issues across engineering disciplines (mining, metallurgical, civil, electrical, mechanical, environmental, geological etc.). No two resources, whether kimberlite or an alluvial deposit, are identical, hence no two process flow sheets are the same (Mackenzie and Cusworth, 2007).

Although technical viability is a strong focus and tends to dominate the assessment of the development potential, the principal purpose is to determine whether the development opportunity makes good business sense (Mackenzie and Cusworth, 2007).

There are basic principles and areas of importance within a diamond beneficiation plant that require focus and thorough understanding of the intended objective to ensure that optimum throughput and recovery are achieved by matching the appropriate interrelated unit processes.

The plant is designed on the basis of recovering diamonds from a source (kimberlite pipe, low-grade kimberlite stockpile, kimberlite tailings, alluvial deposit, or marine deposit) at a pre-determined rate (small scale, marginal, or large scale), considering:

- The ROM material (nature and variability of the feed, hardness, entrained waste, grade, and reserves)
- Diamond characterization (size frequency distribution [DSFD], revenue profile, type etc.)
- Retention strategy (chasing diamonds as opposed to concentrating diamonds from source material)
- Safety protocol

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- Product quality
- Footprint (time to construct and supporting infrastructure requirements)
- Overall cost (CAPEX and OPEX)
- Mining method
- Geographical location, availability of services (water and electricity/power), accessibility etc.

This paper does not focus on detailed or specific case studies. Any aspect of the general content of this paper would require a detailed, client-approved, and specialist study. The intention is merely to highlight the opportunities that new technology brings to the industry and to emphasize the importance of following an appropriate, impartial, and objective methodology in attempting to reach an appropriate design (be it for a brownfield optimization or greenfield project). The optimum solution is unlikely to reside in a single technology supplier, and will inevitably consist of the right combination of technologies leading to simple, flexible design, supported by an appropriate and sound techno-economic study that has considered all the technology options available.

Timeless principles of diamond plant design

Diamonds possess an enduring value, both visually and economically. As the value has endured the test of time, so have the principles that underpin the process design, which are discussed in the paragraphs to follow. The design process is reiterative in nature and requires constant evaluation of these principles during the project life cycle.

Business case

The first and foremost principle is quantifying and qualifying the business case, since the mineral industry is founded on the basic premise of business cases (Mackenzie and Cusworth, 2007). Investors, shareholders, and owners are in the business to realize profit commensurate with the risk.

The general factors that affect the profitability of an operation are typically cyclical commodity prices, declining grades, operating costs, and initiatives to upgrade and/or expand plants in order to maintain profit margins over the life cycle of the operation. Optimal unit process design requires delineating the flow sheet into logical business units, taking into account the following:

- Mass, water, and revenue balance
- Operating cost per unit time
- Efficiency parameters
- Performance indicators, i.e. recovery, grade, upgrade ratio etc.
- Realized revenue indicators (after subtracting operating costs).

Sight must never be lost of the business case during the process engineering design, and it should remain a key reference point during the project life cycle.

Diamond resource characterization

In the formulation of the business case, understanding (geology/mineralogy) and quantifying (grade and quantity) the source (kimberlite, alluvial deposit, tailings etc.) are key principles. For this purpose various drilling and/or sampling studies are undertaken.

Determination of the grade and revenue of the resource is central to the financial viability assessment. However, early characterization of the ore is vital to the assumptions around future treatment methods. As the project progresses through subsequent phases, the basic understanding of the resource is developed through more extensive drilling/sampling programmes, which ensure appropriate decision-making, translated into the most suitable techno-economic flow sheet.

Process design principles

Process design is founded on the availability and accuracy of information. A systematic approach is followed, and at concept level contains the following key elements:

- Ore dressing studies (ODS), which go hand-in-hand with the geological resource characterization and depend on ongoing drilling campaigns to secure ore samples. Samples are subjected to a myriad of metallurgical tests to develop an understanding of the ore. The aim of the characterization studies (on core or bulk representative samples) is not only to elucidate material character, but also to highlight potential problem areas in terms of the four stages in the diamond winning process, viz. comminution, concentration, recovery, and slimes handling. This information, although not definitive owing to the low sampling density at this stage of the project, can guide a more formal sampling programme during the subsequent phases of the project, geared towards developing a technically appropriate flow sheet to treat the material
- Development of the design criteria, which stipulate the basis of design and take into account the characteristics of the material to be processed and the desired product. In instances (early in the project development phase) where sparse information is available, assumptions are quantified
- Development of the flow sheet and associated mass balance, which typically begins with the selection of a beneficiation method. This process is driven largely by the business case and an understanding of the resource
- Various trade-off studies need to be conducted for the different unit processes and the combination of unit processes to determine the most appropriate flow sheet. It is imperative that the key findings of trade-off studies indicate the probability that material properties will vary in terms of important metallurgical indicators. Trade-off studies can therefore be indicative of upfront opportunities, which should ultimately decrease the operational risks associated with treating the resource. The trade-off studies will influence the design phase to reduce project risks and 'right-sizing' of the design, thus increasing the confidence in the approved detailed design solution.

The process design principles, whether for an alluvial deposit, hard rock (kimberlite), or tailings retreatment, remain unchanged in terms of the design criteria, and this, together with ore characterization test work on representative samples, continues to be the backbone of the design.

The process engineering value proposition is developed through the contextualization of recommendations in order to reduce the risks associated with realizing new opportunities,

and to maximize the potential revenue generated from these opportunities for the various flow sheets or unit processes within a flow sheet.

Why a new era in diamond process design?

If the principles are timeless, what has changed and why is this a new era? The answer to this lies largely in the fact that exceptional technologies have come to maturation over the last decade. Although this has made the process engineer's workspace exciting, it has brought more complexity due to the fact that more combinations and permutations are possible.

Plant operating knowledge across the unit operations has improved significantly in the past few years. Simulations that track revenue across streams and incorporate the variability in revenue efficiency of different unit processes have aided in this understanding. Sub-optimal operating conditions can more easily be identified and theory and practical knowledge are ever closer, ensuring that even small changes in operating conditions can result in revenue efficiency changes.

The sections to follow assume that the reader has a basic understanding of the four main process areas (feed preparation, concentration, final recovery, and slimes handling) within a typical diamond process plant.

The following tables depict some of the conventional and more recently developed technologies that are to be considered within the new-generation diamond process plant design arena. Limited focus is given to feed preparation (Table I), more focus on concentration (Table II) and final recovery (Table III), and some focus on waste sorting technologies (Table IV). The latter could be applied as part of either the feed preparation process or the concentration circuit. Slimes handling is omitted. The technologies listed are examples, and it is not necessarily intended to address all available technologies. Views or opinions are those of ADP Projects.

Thickening and disposal are not specifically covered in this paper. The latest technology for the optimal recovery and re-use of water is the ATA process, developed by Soane Mining and Soane Energy. This is a polymer-based technology capable of eliminating the requirement for tailings impoundment. Furthermore, significant improvements have been realized in fine screening (e.g. Bivitech, Derrick etc.).

These newly matured technologies (or the case of DebTech and Bourevestnik, only recently made commercially available to the broader diamond industry) will undoubtedly result in significant value being brought to bear on diamond projects in the future. The challenge is to appreciate the resultant complexity, opportunities, and associated risks (including the cost of poor decisions) and to perform the appropriate study work that precedes the projects (no matter their scale).

Tailored solutions

It is not a simple case of 'out with the old and in with the new'. There is a critical balance to be found between conventional and more recently developed technologies, especially given the varying degrees of maturity of these technologies in the diamond industry.

The authors have participated in many diamond process plant development studies/projects, and it is unquestionable that each project is unique and is deserving of a detailed and methodical application of knowledge, skills, understanding, and experience to arrive at the solution that will justify the expenditure and secure the revenue, taking into account human resources and skills available, as well as services (water and power availability and the cost of these).

Table I

Some technology choices for the feed preparation circuit (focus on dis-agglomeration/clay removal)

Category	Technology	Principles of operation[a]	Applications/advantages/disadvantages	Some technology suppliers
Disagglomeration for clay removal	Log washer	Abrasion-resistant paddles affixed to a horizontally mounted rotating shaft yields an aggressive washing action that breaks down clay	More applicable to tough plastic-type clays.	GreyStone, McLanahan, KPI-JCI, Trio (supplied by Diamond Equipment Group), QVM
	Hydro-Clean™	High-pressure washing technology, Material is cleaned by high-pressure streams of water from a washing rotor and spray nozzle combination	*Advantages:* – Weighs significantly less than equivalent technologies – Smaller footprint – Lower water consumption – Low residence time – Less CAPEX and OPEX	Haver and Tyler (subsidiary of the Haver Boecker group)
	Autogenous (AG) milling	A horizontal rotating mill. Larger rocks constitute the grinding medium, and are reduced by impact breakage with compressive grinding of finer particles	Extremely effective clay disagglomeration *Advantages:* – Single stage comminution circuit results in simpler plant design (smaller footprint) – Lower OPEX *Disadvantages:* – High power consumption – Initial CAPEX high	Outotec, Metso, Polysius FLSMidth, MechProTech
	Rotary scrubbing	A horizontal rotating cylindrical drum with internal lifters that continuously abrade material under controlled water-to -ore ratios	Not applicable for plastic-type clays, as these can pelletize through the tumbling action	McLanahan, Mechprotech, MT, FL Smidth, Bateman.

[a]Source: vendor specifications

Table II

Technology choices for the concentration circuit

Category	Technology	Principles of operation	Application/advantages/disadvantages	Some technology suppliers
Crushing (re-crush)[a]	Conventional cone crushing	Impact crushing	*Advantages:* – Lower capital expenditure *Disadvantages:* – Higher risk of diamond breakage – Requires choke-fed conditions	Sandvik, Metso, Osborn, IMS
	Modified cone crushing (specialized chamber design)	Impact crushing, but with modified chamber design. A degree of inter-particule crushing occurs	*Advantages:* – Significantly improved performance, capacity, and installed power, while retaining the reliability – Excellent product shape and high percentage product passing CSS after first pass, which is an industry benchmark with lower risk of diamond breakage *Disadvantages:* – Requires slightly more comprehensive test work to establish suitability of technology – More expensive	IMS (Kawasaki Cybas-i cone), Sandvik (Vibrocone – being explored for diamond processing)
	High pressure grinding roll (Daniel and Morley 2010: Olivier *et al.* 2010)	Inter-particle crushing	*Advantages:* – Various applications within flow sheet – High availability – Feed can be varied by varying the roller speed. Ability to produce finer PSDs – More diamond-friendly technology (Daniel and Morley, 2010) *Disadvantages:* – High CAPEX – Fine PSDs impose additional load on the slimes handling circuit – High power requirement, resulting in higher OPEX	Polysius, Weir Minerals (KHD), FL Smidth, Polysius (a ThyssenKrupp company), KHD (KHD Humboldt Wedag AG), and Köppern (Maschinenfabrik Köppern GmbH &Co KG)
Concentration	Jigging[b]	See footnote (b)	Typically employed at marginal high-throughput operations *Advantages:* – Low CAPEX – Semi-portable structure makes it appropriate for alluvial deposits, which are spread over large geographical areas *Disadvantages:* – Not as efficient as competitor technologies	Bateman, Kelsey, Gekko, PCF Engineering
	Rotary pan[b]	See footnote (b). Diamond-bearing material is mixed with water to create a slurry (puddle) which has a density of 1.3–1.5 g/cm^3. The mix is stirred in the pan by angled rotating 'teeth'. The concentrate settles and is pushed toward an extraction point, while lighter waste remains suspended and overflows from the centre of the pan as a separate stream	Typically employed at marginal high-throughput operations *Advantages:* – Lower initial CAPEX – Semi-portable structure makes it appropriate for alluvial deposits, which are spread over large geographical areas *Disadvantages:* – Not as efficient as competitor technologies – Requires operational skill	Various
	Dense medium separation (DMS)[b]	See footnote (b) Diamond concentration into a near-diamond density FeSi slurry medium through use of cyclones and peripheral equipment (JKMRC, n.d.)	Typically employed for higher LOM and affluent operations. *Advantages:* – Efficient diamond recovery – Lower initial CAPEX for diamond recovery from finer size fractions compared to bulk sorting technologies *Disadvantages:* – Higher initial CAPEX than jigging and rotary pan – Higher OPEX in comparison with all alternative technologie due to FeSi consumption – More labour intensive than bulk sorting technologies – High yields from sources with high heavy mineral content	EPCM designed and supplied i.e. ADP Projects, Bateman.

[a]Focus on re-crush function within concentration circuit only, other possibilities (i.e. VSI) not discussed as, for example, diamond damage aspects needs further clarification

[b]These systems are based on diamonds having a much higher specific gravity (density) of approx. 3.52 g/cm^3 compared to most of the gangue minerals

[c]The technologies listed are examples, and it is not necessarily intended to address all available technologies. Views or opinions are those of ADP Projects

Table II (continued)

Technology choices for the concentration circuit

Category	Technology	Principles of operation	Application/advantages/disadvantages	Some technology suppliers
	Bulk diamond sorting technologies			
	X-ray luminescence (XRL)	Based on the principle that diamonds fluoresce, and to some degree phosphoresce, when exposed to X-ray radiation (Raman and Jayaraman, 1950)	*Advantages:* – Lower CAPEX than XRT alternative – Lower CAPEX for diamond recovery from coarser size fractions than DMS – Efficiently recovers high-luminescing diamonds *Disadvantages:* – Risk of loss of low-luminescing diamonds – Risk of concentrate dilution by luminescing gangue – Higher CAPEX for diamond recovery from finer size fractions compared with equivalent DMS technology – Requires feed preparation to minimize the presence of coating on diamonds masking response to X-rays – Bottom size process restriction	DebTech Bourevestnik; Commodas; Flow Sort Stenert
	X-ray transmissive (XRT)	Diamonds are recovered on principle of atomic mass (Riedel and Dehler, 2010)	*Advantages:* – Efficiently recovers all diamond types – Potentially lower yields than DMS and XRL as yields are not influenced by gangue properties. Can detect diamonds in aggregates/conglomerates or embedded in shell – Lower CAPEX for diamond recovery from coarser size fractions than DMS, – Does not require extensive feed preparation as the technology does not rely on perfectly clean diamond surface *Disadvantages:* – Higher CAPEX than alternative bulk sorting technologies – Bottom size process restriction	Tomra (previously CommodasUltrasort), Steinert (supplied by IMS) DebTech (currently being tested).

[a]Focus on re-crush function within concentration circuit only, other possibilities (i.e. VSI) not discussed as, for example, diamond damage aspects needs further clarification
[b]These systems are based on diamonds having a much higher specific gravity (density) of approx. 3.52 g/cm^3 compared to most of the gangue minerals
[c]The technologies listed are examples, and it is not necessarily intended to address all available technologies. Views or opinions are those of ADP Projects

To illustrate this statement, some elements that relate to the project definition and associated technology selection considerations are discussed briefly (mining methods have been excluded from this discussion).

Project scale

Technology selection cannot be considered in isolation from the intended scale of operation, since the scale is inherently linked to the risk profile, or rather the adverseness to risk, which in turn is linked to the funding mechanism.

- Small-scale operations will require cost-effective 'off-the-shelf' capital-sensitive solutions, possibly to the detriment of efficiencies, to ensure that risk is minimized and steady income is generated. Ore dressing studies are generally not undertaken for this scale of operation
- Marginal-scale operations can bear more risk. These follow the middle path of higher capital outlay against increased returns, but still involve a minimization of risk to the detriment of efficiency. Ore dressing studies are limited to the bare necessities
- Large-scale operations tend to tolerate the least risk, since these are coupled to formal funding mechanisms requiring high initial capital investment. Investors require a secure understanding of the reserve and confidence in the diamond recovery process selection to assure the desired return on capital. These typically require extensive exploration, drilling, and ore dressing studies to facilitate the decision-making process.

Classification of the diamond resource

The classification of the diamond-bearing resource is mentioned here for consideration. This relates to the matrix of the material from which the diamonds are to be recovered:

- Alluvial/marine resources require little or no comminution, as the diamonds are generally liberated. The challenges are the possible higher levels of clay and lock-up of diamonds in shells or conglomerate. When selecting the appropriate concentration technique, the presence of shells potentially creates many challenges, one being increased FeSi consumption due to FeSi entrapment when DMS is utilized. If bulk sorting technologies are considered, the encapsulation of diamonds in shells could render them

Table III

Technology choices for final recovery (no focus on sort-house technologies)

Technology (Valbom and Dellas 2010)	Recovery principles[a]	Some technology suppliers
Magnetic sorting	This is a bulk reduction technique introduced prior to other sorting techniques to produce a lower volume upgraded diamond concentrate, and works on the basis of removing magnetic susceptible minerals	DebTech, Steinert (IMS) Eriez
Conventional X-ray luminescence	Diamond recovery on basis of luminescing properties	DebTech Modrup Technology, Impulelo Technologies, Flow Sort
Pulsed X-ray luminescence	Diamond recovery on basis of luminescing properties. Increased efficiencies compared with conventional X-ray luminescence	Bourevestnik
X-ray transmissive	Diamonds are recovered on the basis of atomic mass	Tomra (Commodas), Steinert (IMS) Debtech (under development)
Ultraviolet sorting	Can be used as scavenging technique and works on the principle of diamonds' response to UV light. *Advantages:* – Optimum diamond scavenging with minimum gangue material at high feed rates – Spillage free – Low concentrate mass – Operator and maintenance friendly – Complete operator safety – Low operating costs – Low CAPEX – Complementary technology to X-ray sorting	DebTech Osprey
Automated grease belt	Can be used as scavenging or primary recovery technique. Diamond recovery based on hydrophobic properties. Grease belts are automated, more secure, more efficient, and more expensive (but require a larger footprint) than grease tables	Mech Projects and Engineering (previously Oblique Engineering), Bateman Equipment Technologies (BET)
Vibrating grease tables	Can be used as scavenging or primary recovery technique. Diamond recovery based on hydrophobic properties. Grease tables are manually operated, less secure and less expensive (but require a smaller footprint) than grease belts	Bateman Equipment Technologies (BET), Vibramech.

[a]Source: Vendor specifications

Table IV

Some technology choices for waste sorting (could be applied in either feed preparation or concentration circuits)

Technology	Principles of operation	Application/advantages/ disadvantages	Some technology suppliers
X-Ray fluorescence (XRF)	Utilizes well established X-Ray Fluorescence technology. Simultaneously measures the concentration of up to four metals in the surface of each particle. A matrix of these elements (including actual quantities and/or ratios) is used to classify individual particle as discard or concentrate fractions	*Advantages:* – Robust and able to operate in extreme climatic conditions – Low maintenance cost – Mechanical or air ejections and very low electricity consumption	RADOS10, Redwave, Steinert (supplied by IMS)
Optical sorting by colour	Diamonds or diamond bearing/non-diamond bearing particles are recovered or rejected on the basis of surface colour and/or geometrical characteristics.	*Advantage:* – Simplified sorting algorithm when sorting only two facies *Disadvantage:* – Sorting efficiency declining with associated colour variation of known facies – Requires extensive feed preparation	Tomra (previously CommodasUltrasort), Steinert (supplied by IMS), Redwave, OptoSort
Optical sorting by NIR	Mineral recognition by individual absorption fingerprint in the near infra-red wavelength range.	*Advantage:* – Not subject to declining sorting efficiency associated with colour variations of known facies	Tomra (previously CommodasUltrasort), Steinert (supplied by IMS), Redwave, OptoSort.

invisible to X-ray technology. These considerations will require careful CAPEX and OPEX trade-offs against the life of the resource

- Hard rock (kimberlite) requires comminution to liberate diamonds. Comminution technology selection is critical to minimize diamond damage, but diamond size frequency distribution (DSFD), diamond revenue per sieve class, as well as CAPEX and OPEX trade-offs of the technology against the life of mine (LOM) also have to be considered. The comminution circuit and water treatment and slimes/slurry disposal go hand-in-hand
- Tailings reclamation has been receiving renewed emphasis in recent years and involves a unique cluster of conditions that are fundamentally different from treating hard-rock material: a finer feed distribution, potentially finer diamond distribution, and potentially

un-liberated diamond prevalence will require special consideration of alternative technologies to the primary processing techniques that were employed on the source material. Again, sight should not be lost to the CAPEX and OPEX implications against the estimated life of resource.

Diamond resource considerations

Grade and volume

Resource grade and volume relate directly to the LOM and require special emphasis on OPEX vs. CAPEX trade-offs when technology selections are made. For example:

- Short LOM prospects may not be able to bear the implementation of capital-intensive technologies, even though these might offer the benefit of increased diamond recoveries and lower OPEX. The net effect may not be justified over a short LOM as returns on investment may not be maximized
- Longer LOM prospects, however, present an entirely different picture as the increased revenue due to more efficient recoveries, improved liberation, and potentially lower OPEX (XRT and HPGR combination vs. DMS and cone crushing combination) against excessive initial CAPEX will make financial sense only over an extended LOM when the net profit will be maximized.

The LOM is, however, highly dependent on many other factors (mining method, nominal throughput, waste-to-ore ratio etc.) which are not covered in this paper. These are criteria that receive considerable emphasis during technology/flow sheet trade-off studies.

Waste-to-ore ratio

This section relates to kimberlitic ores and not alluvial deposits. Diamond recovery processes have traditionally struggled to treat feed sources with excessive waste content. The increase in waste material present in ROM feed normally causes increased internal recycle loads, for example:

- An increased secondary crushing recycle load due to harder waste material and the fact that the crushers cannot maintain the normal operating closed-side setting
- An increased recovery plant recycle load due to inflated DMS yields, placing unnecessary treatment constraints on the final diamond recovery process.

Although waste sorting techniques have been used in many other industries, waste sorting is an emerging technology in the minerals processing industry, receiving renewed attention in a variety of commodity applications. If applied appropriately, it offers the benefit of bulk reduction of the ROM ore by selectively removing the waste component. The objective of waste sorting is to upgrade low-grade kimberlite ROM sources and, in doing so, ensure the economically viable treatment of these sources by:

- Maximizing the revenue per hour by avoiding processing of a diluted ore of inferior grade
- Lowering the overall operating expenditure by minimizing the re-circulation and processing of high volumes of entrained waste
- Improving the overall process energy efficiency.

Diamond characteristics and occurrence

Diamond type has a direct impact on the applicability of certain concentration technologies e.g. a high prevalence of low-luminescing diamonds makes the selection of X-ray based technologies less attractive. Nevertheless, there are cases where the project cannot necessarily bear the excessive capital outlay associated with alternative transmissive technologies.

Diamond size frequency distribution, revenue distribution, and extent of liberation impact on the:

- Top, middle, and bottom cut-off size selections. Some resources may contain a large percentage of small diamonds that may be of very low value, and others may contain a small percentage of very large diamonds that constitute the bulk of the value. This is a critical element that impacts directly on technology selection and on the project value proposition
- Balancing of the overall circuit with suitable technologies, respecting the cut-off size selections and technology constraints
- Appropriate selection and positioning of comminution technologies, specifically within regards to the re-crush function. Cognisance needs to be taken of diamond damage/breakage if the resource has a high prevalence of large diamonds.

Retention strategy

Investments in treatment processes are based on the financial model of a company, and it is imperative to characterize the 'net worth' and applicability of a particular unit process. Particular attention is given to the retention strategy of the mineral or valuable resource, both within a unit operation and within the overall plant (i.e. revenue in circulation). Every unit process can be characterized according to the value-add and should contribute positively to the overall financial model (Petersen, n.d.)

In the diamond treatment process the valuable stones should be recovered as soon as practically possible post-liberation, and not retained, in order to minimize revenue in circulation. In addition, the risk of damage or losses increases dramatically the longer the stones remain in circulation. Therefore, the technology selection within various processes, as well as the operability, must be taken into account, understood, and integrated to ensure the business case is sound.

Some technology evaluation scenarios

The following examples of technology trade-off scenarios are not client-specific, but have been generalized for the purpose of demonstrating how technology advances have influenced the diamond process design playing field.

Primary hard rock/kimberlite

New plant design

Consider a new treatment plant design for a primary kimberlite with large diamond incidence, coarse DSFD, and high waste-to-ore ratio. Table V outlines the technology selections for the conventional process design versus new-generation technology choices that could be considered in finding the optimal techno-economic solution.

Table V

Conventional *versus* new generation technology considerations

Process area	Conventional technology choices	New-generation technology considerations[a]
Feed preparation	• Primary jaw crusher followed by scrubbing and screening in a closed loop with conventional cone crushing for the secondary duty	• Waste sorting technologies (NIR, XRF or colour sorting) to remove waste prior to concentration • Alternative crushing technologies (HPGR or modified cone crushers) for secondary crusher duty for the protection of large diamonds
Concentration	• Treatment of the secondary crusher product by DMS (split or single-fraction approach would typically be considered, depending on the DSFD and revenue profile) into an upgraded concentrate • Inclusion of a conventional cone crusher in a re-crush duty for the further liberation of diamonds	• Bulk sorting technologies (pulsed X-ray or XRT) for the recovery of diamonds down to the MCOS, paired with DMS for the recovery of diamonds down to the BCOS • Alternative crushing technologies – HPGR (Daniel and Morley, 2010) or modified cone crushers in a re-crush duty for the protection of large diamonds
Final recovery	• Treatment of DMS concentrates through conventional X-ray technologies and/or grease recovery unit processes, to yield diamonds ready for export	• Magnetic separation (if high incidence of heavy minerals inflates DMS yields) • Pairing of bulk sorting and final recovery sorting technique is critical. This principle is to be adhered to in final recovery trade-off studies. • Inclusion of a scavenging technology (grease belt, table or UV sorter).

[a]Confidential client reports by ADP Projects

Trade-off/evaluation criteria that are to be considered in order to maximize the revenue per hour and minimize recirculation of revenue within unit processes (ADP Projects – confidential technical client reports):

- CAPEX (including equipment cost, support structure requirements, civil requirements, footprint etc.) and OPEX (labour requirements, reagent – e.g. FeSi in the case of DMS, power requirements of the technology selection)
- CAPEX outlay and potential revenue realization (a function of OPEX and diamond revenue) in relation to LOM
- Circuit balancing in terms of DSFD and revenue profile to achieve the optimal cut-off size selections, respecting technology restrictions while maximizing diamond revenue (consider diamond damage and/or liberation associated with comminution technologies and revenue losses when performing cut-off size trade-offs in terms of concentration technology limitations)
- Yields of the concentration technologies (DMS, XRL, XRT) in relation to ore characteristics, impacting on the final recovery throughput requirement
- Complexity
- Materials handling requirements
- Plant service requirements
- Operational challenges (including labour and skills availability)
- Flexibility to treat ore if resource characteristics change over LOM (e.g. clay content, breakage characteristics, waste–to-ore ratio, diamond grades, magnetic susceptibility profile etc.). Impact on downstream tailings handling requirements
- Diamond security challenges.

Existing plant optimization

Consider an existing plant treating an ore with finer DSFD and increasing incidence of waste with a typical flow sheet configuration consisting of primary crushing, scrubbing, screening, and closed-loop secondary crushing feeding multiple split fraction DMS modules (i.e. fines DMS modules treating the 1–12 mm size fraction and coarse DMS modules treating the 12–32 mm size fraction) (JKMRC, n.d.).

When tasked to find a solution to handle increased throughput requirements due to increasing waste dilution while minimizing the capital investment and maximizing recoveries, the following options could be considered and the capital investment phased to accommodate cash flow constraints (ADP Projects, confidential technical client reports).

- Inclusion of waste bulk sorting (NIR, XRF, or colour sorting) in the feed preparation as well as in the concentration re-crush/tertiary circuits to eliminate unnecessary load on the concentration and final recovery circuits. The net effect will be an overall increased ROM throughput capability, more efficient recovery of diamonds in downstream processes, and improved overall economics
- Converting all DMS circuits into fines DMS modules (with minor modifications), and bulk sorting techniques (XRL or XRT) to recover diamonds from the coarser size fraction
- Combining the abovementioned technologies with other changes to an existing plant can create additional head feed capacity, reduce unit costs, and optimize the operation as a whole.

Trade-off/evaluation criteria that are to be considered include (ADP Projects, confidential technical client reports).

- CAPEX (including equipment cost, support structure requirements, civil requirements, footprint, expansion requirements to downstream processes – slimes handling, final recovery etc.), and OPEX (labour requirements, reagent e.g. FeSi in the case of DMS, power requirements of the technology selection, downstream process expansions to accommodate additional throughput, slimes handling)
- Revenue realization in relation to LOM
- Circuit balancing, leveraging the benefits of technology selection while respecting limitations

- Yields of the concentration technologies (DMS, XRL, XRT) in relation to ore characteristics, impacting on the final recovery throughput constraints
- Complexity
- Materials handling requirements and incorporation of these into an existing plant
- Plant service requirements in relation to availability (power, water etc.)
- Operational challenges
- Flexibility to treat ore if resource characteristics change over the LOM (e.g. clay content, breakage characteristics, waste-to-ore ratio, magnetic susceptibility profile)
- Impact on downstream tailings handling constraints and final recovery constraints
- Expandability of existing plant infrastructure (e.g. final recovery building)
- Opportunity to incorporate bulk sorting technologies within final recovery building
- Impact on security philosophy
- Compatibility with downstream processing techniques (i.e. sorting technologies that are considered for the concentration circuit in relation to the existing final recovery process).

Alluvial examples

Consider the following alluvial type plants:

- Small-scale high-grade gravel (typically remote African operations):
  - The typical circuit would comprise a scrubbing and screening module followed by DMS and wet X-ray processing as the final recovery technique
  - The new-generation approach would consider combining DMS with bulk sorting technologies (XRT, XRL) or simply replacing the entire DMS module with bulk sorting.
- High-capacity low-grade gravel with low clay content (typically Orange River, South Africa)
  - The typical circuit will consider dry screening followed by rotary pans, feeding pan concentrates into DMS, followed by diamond recovery by wet XRL
  - The new-generation approach would consider bulk sorting (XRT/XRL) instead of, or combined with, DMS.
- Conglomerate, clay, and shell-rich gravel (typically South Africa West Coast):
  - The typical circuit configuration would include scrubbing/milling and screening, followed by a combination of jigs and DMS to produce concentrates that would be treated by wet XRL in the final recovery circuit
  - The new-generation approach would consider bulk sorting technologies (XRT or XRL) either exclusively or in combination with conventional techniques for the concentration circuit.

General trade-off/evaluation criteria that are to be considered:

- Requirement for mobile modular configuration
- CAPEX (including equipment cost, support structure requirements, civil requirements, footprint etc.) and OPEX (labour requirements, reagents e.g. FeSi in the case of DMS, power requirements of the technology selection)
- Cognisance to be taken of the fact that bulk sorting techniques are to be paired with final recovery techniques. If XRT is employed, diamonds that are locked up in conglomerates or shells have a fair chance of being detected and will report to concentrate. If the final recovery circuit makes use of XRL, these will report to tailings since the technology relies on a clean diamond surface to ensure detection
- Downstream tailings handling requirement
- Risk of diamond loss (technology limitations with regard to bottom size that can be treated).

Conclusion

Although new technologies have brought the opportunity for improvements in diamond recovery, they have also increased the probability of poor decision-making. The level of complexity has increased, requiring a systematic factual approach *versus* one that is riddled with traditions and inappropriate assumptions. New technologies, no matter how attractive in principle, are not always applicable or appropriate. This reality has been borne from experience, but must not deter the inexorable improvement to plant design that technology always can, and mostly does, bring.

The selection of the appropriate unit processes and/or the combination of these is reduced to the analysis of the trade-off between efficiency considerations and capital/operating costs. Various simple quantifications of the potential efficiency improvements over the life of mine *versus* the capital savings/increases must be included to aid in decision-making. In addition to the financial aspect, consideration must be given to practical operating, maintenance, and security aspects. These include the existing operating and maintenance culture. Unit processes should not be introduced to act as a safety net for inappropriate operating and maintenance culture, since culture within an operation is something that can be re-established or changed.

It is contingent on all process design engineers to apply themselves fully to understanding the technologies to the extent necessary to be able to make objective decisions on their potential applicability on projects. That, in turn, requires keeping an open mind, being impartial, and engaging meaningfully with and listening carefully to the technology suppliers at all times. Most importantly, it requires doing the 'hard yards' on the study work, never pre-judging the optimal flow sheet solution at the outset, and allowing the facts to speak for themselves whenever possible through thorough front-end engineering design work.

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Disclaimer

The SAIMM recognizes that the list of technologies and suppliers described within this paper are not comprehensive and as such there will be other technologies and suppliers available and it is incumbent on the reader to research their availability in their specific geographic location. The paper talks rather to principles that should be considered when designing Diamond processing plants. ◆

A simple framework for developing a concept beneficiation flow sheet

by J. Rabe*

Synopsis

Tasked with developing a flow sheet for a new resource, one typically turns the experts for help. The advice of an expert can be invaluable in reducing the time and costs associated with characterizing an ore (not to mention extracting maximum information from limited samples). While guiding and optimizing a metallurgical concept study requires a certain skill set, selling a project is something different.

The metallurgical specialist is usually not responsible for sharing the results of investigations with potential investors. However, the person responsible for selling the project can ensure that a metallurgical concept study is conducted thoroughly and responsibly without having the ability to do the work, while being sufficiently familiar with the metallurgical aspects to authoritatively sell the project.

The process of identifying suitable process routes through to constructing a concept flow sheet and mass balance is discussed as a basic example.

From basic mineralogical information and a thorough literature survey, the most likely beneficiation routes can be identified and targeted testing conducted. The final top size will depend on the required product properties (mainly grade), the selected beneficiation technology, and the ore's liberation properties.

Once the required top size has been identified, comminution options can be evaluated. Once again, literature surveys can be used to supplement concept-level comminution property information, and select possible equipment and circuit configurations.

Marketing, material handling, and logistics-related information can be used to determine required dewatering technologies. At this stage, enough information is available to develop a concept mass balance and conduct basic equipment sizing..

Keywords
comminution, ore beneficiation, characterization, titanomagnetite. Conceptual flowsheet

Introduction

The project manager responsible for delivery of a concept study on any resource does not always have an in-depth knowledge of the metallurgical processes required to generate a saleable product at a competitive cost.

In larger companies, the project manager will have access to a metallurgist, but in most exploration companies and junior mining houses, this will be an outsourced function – at least at this stage of the project. This means that the project manager has to trust an external expert to do the necessary work thoroughly enough to be able to submit the final study results to a competent reviewer without any come-backs. The primary marketer of the project also needs to gain enough insight into the process choices, opportunities, and challenges to be able to confidently sell the project to potential investors while being able to answer most of their technical as well as financial questions.

This may sound like a daunting challenge, but building on a few basic concepts enables one to understand the process choices and sell the project with confidence. For a typical concept study, a single technically and financially viable solution needs to be proven, with a typical accuracy of 30% for cost estimation – although this varies between companies. At the conclusion of the concept stage, targeted ore characterization and processing tests are typically identified for option generation and analysis in the project phases to follow. Depending on how well known the target ore is, as well as the expected ore properties and the acceptable amount of risk for the project, certain aspects of the ore processing may be based on literature reviews at this stage of the project.

Chemical characteristics, such as gold, iron, or copper grades, might give an indication of the potential value of a resource, but do not yield any information on the extractability of the target mineral, nor do they enable the final achievable product grades to be identified with regard to the target element as well as undesirable impurities. Knowledge of the mineral associations in the resource is essential to develop a suitable beneficiation strategy. With the use of information from literature reviews, basic processing options can be identified at this stage.

* *Pesco.*

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A simple framework for developing a concept beneficiation flow sheet

Separation based on particle size represents the simplest basis of beneficiation. In such a process, if either the product or gangue minerals can be broken down selectively, the ore can be upgraded by classification. Separation based on particle density is one of the most widely used beneficiation strategies, finding application in fields as diverse as gold panning and vegetable classification. Separations based on magnetic and surface properties have also been utilized for hundreds of years.

Along with each of the processing options being investigated, the top size that is required to produce the required product grade needs to be investigated. Liberation is a word with different meanings for different people, and even when the same definition is applied, there is still plenty of room for confusion.

At this stage preliminary beneficiation flow sheets can be drawn up, and relevant testing initiated. Typical tests will include liberation testing and confirmation of product grades and yields. Ore hardness and related comminution properties may be investigated at this stage of the project – although typically the lack of sufficient sample mass at coarse enough top sizes prohibits most such test work at this stage. Literature reviews and vendor experience are invaluable resources for conducting conceptual designs.

A schematic summary of the proposed methodology for developing a process flow diagram (PFD) is shown in Figure 1.

Although not discussed in detail in this overview, sorting of particles based on surface properties such as fluorescence or colour can also be considered. Traditionally this has been limited to coarse top sizes only, but as sorting technology has evolved the range of properties that can be used to discriminate between particles, as well as machine capacities, has improved.

Results from tests conducted on a titaniferous magnetite from the Bushveld Complex will be used as examples for discussion.

Mineralogy

For the purposes of this discussion, bulk mineralogy refers to techniques like X-ray diffraction (XRD), and automated mineralogy refers to the Mineral Liberation Analyser (MLA) or QEMSCAN (derived from the QEMSEM - Quantitative Evaluation of Minerals by Scanning Electron Microscopy). The term 'bulk' is not meant to refer to large sample sizes, but rather to the fact that the techniques give information regarding the average mineralogy of the entire sample – nothing can be said about mineral associations based on XRD only. QEMSCAN gives information on a micro-scale, meaning information about grain sizes and mineral associations at this level can be provided.

With the development of sophisticated mineralogical characterization tools such as the MLA and QEMSEM, obtaining copious amounts of mineralogical information early on in the project is becoming the norm rather than the exception. It is unfortunate that metallurgical interpretation of this rich mineralogical information is often lacking, and the potential of simpler techniques such as XRD is often forgotten.

One of the most important aspects of interpretation of mineralogical results is that certain mineral properties cannot be measured by or inferred from either micro-mineralogy such as the MLA or bulk mineralogy such as XRD with 100% accuracy – magnetic and flotation response are two typical examples of this.

Bulk mineralogy

XRD has been used to identify minerals based on their crystal structure since the 1920s. The commonly used term XRD refers to X-ray powder diffraction; a rapid non-destructive test to determine mineralogical make-up of a crystalline specimen. The basic principle involves impinging an X-ray beam onto a sample and measuring the intensity of the diffracted beam at a range of angles. Depending on the spacing and orientation of the crystal planes, higher intensity peaks will be observed at certain positions. Although it may not be possible to distinguish chemically between, for example, the titanium oxides anatase and rutile, their crystal structures are different, allowing each to be uniquely identified by XRD (Figure 2).

The patterns of all minerals in a sample are superimposed on each other in an XRD spectrum. The superimposed fingerprints of each mineral in the sample are then separated, allowing the relative quantity of each component to be estimated.

Automated mineralogy

Development of the QEMSEM started in the 1970s at CSIRO (FEI Natural Resources, n.d. (b)), with the first commercial sales from 1987. Following significant improvements to the system in the following decade, Windows™-based systems were developed in 1997, and called the QEM SCAN. FEI (originally Field Emission Inc.) bought all intellectual

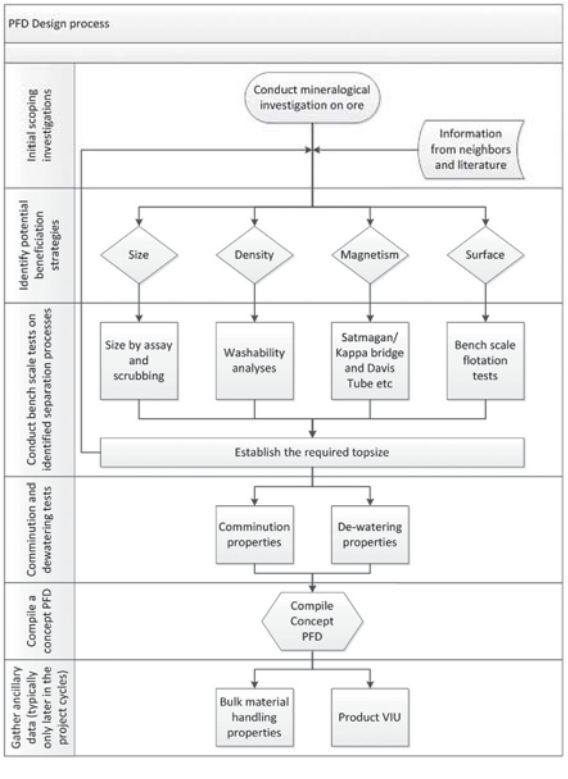

Figure 1—A schematic summary of a simple process to develop a PFD

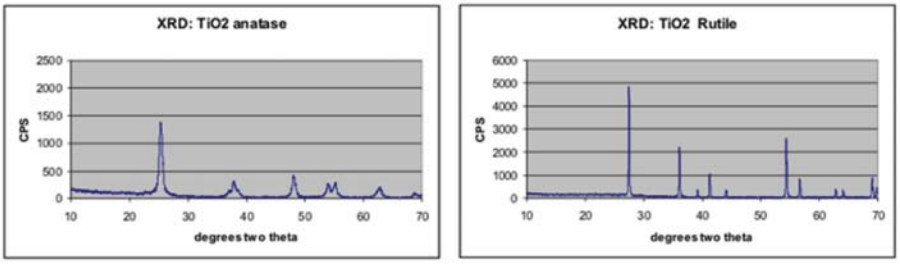

Figure 2 – X-ray diffraction patterns for two different forms of TiO_2 (University of Arizona, n.d.)

property as well as the software related to the QEMSCAN early in 2009. The MLA was developed by JKMRC in the late 1990s, and commercialized in 2000. In mid-2009 FEI bought the MLA business from JKTech.

The case for mineralogy can be illustrated by using an iron ore as example, where iron might be present in a number of different minerals, ranging from haematite and magnetite through hydrated oxides such as goethite to iron-bearing silicates.

The geological samples have been analysed, and high iron concentrations confirmed, but the target mineral has not been identified. Based on geological interpretations, magnetite has been targeted for extraction, and the relevant tests conducted to prove a business case.

An acceptable iron grade was produced in the concentrate, but recovery of iron was disappointing. XRD analysis of the ore would have indicated that a number of iron-bearing minerals are present, with some of them having weak magnetic responses. While this would raise concerns about using only magnetic separation to recover iron, the low recovery could also be due to weathering of the magnetite, in which case the magnetite could be intimately associated with haematite.

For the hypothetical ore depicted in Figure 3, only partial iron recovery would be expected by magnetic separation, while a high recovery would be obtained from the example in Figure 4.

Liberation

Defining liberation

The concept of liberation is a complex one, with a multitude of definitions. At one extreme, the liberation size of an ore is the size at which the smallest valuable mineral grain is liberated from its host matrix.

This definition is often used in automated mineralogy reports, and a common interpretation implies that a product with no gangue minerals is being aimed for – which is usually not the case. For example, a typical haematite product consists of around 63% iron with 10% gangue minerals. If a homogenous piece of ore with a grade of 63% iron and a liberation size of 100 μm is crushed to a top size of 31 mm, the resulting fragments will also consist of 63% Fe. A liberation investigation will indicate that significant further crushing is required, even though no further process is required to achieve the required product grade.

Top size

In many cases the ultimate mineralogical liberation is fairly simple to quantify, but the actual top size at which an ore will be sufficiently liberated to produce the required product grade

Figure 3—An iron ore sample containing three different iron-bearing minerals examined by MLA (FEI Natural Resources, n.d. (a))

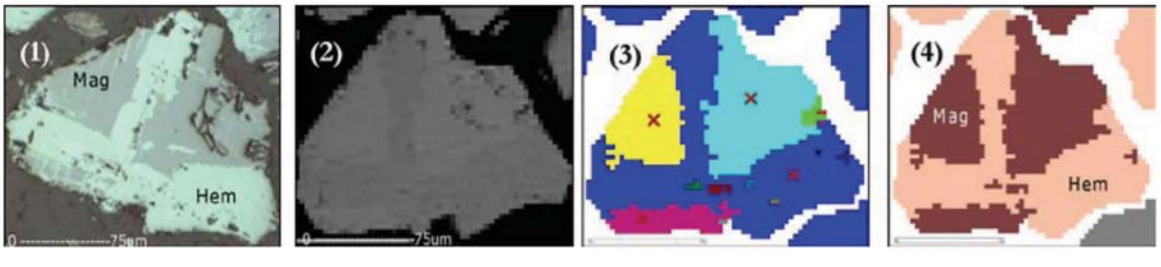

Figure 4—Automated mineralogy discriminating between magnetite and haematite grains, with magnetite and haematite intimately associated with each other (German, 2011)

is not so obvious. In the over-simplified example discussed previously, it may be obvious that the top size required for the production of a saleable product is dictated by factors other than liberation, which highlights the difference between top size and liberation size.

The goal is to determine the coarsest size at which an acceptable product grade can be produced. The optimum top size will typically be coarser than the liberation size, unless extremely pure products are required. Although there is a correlation between the mineralogical liberation size and optimum top size, the relationship is a function of factors such as:

- The grain size distribution of both valuable and gangue minerals
- The exact mineralogical chemistry for gangue and valuable minerals
- The relative abundance of the valuable mineral in the ore
- The required product grade
- Beneficiation process efficiency.

The best way to confirm the required top size for an ore is to crush representative samples to a range of top sizes and subject them to tests simulating the selected beneficiation option. The sizes at which testing is conducted will range around a size indicated by considering the list of factors mentioned above.

If more than one beneficiation technology will be applied (such as pre-concentration by DMS before milling and flotation), the final product grade can be estimated by testing an un-concentrated sample with the final process, unless feed grade has a significant effect on the achievable product grade (this is a typical challenge when flotation is involved).

In most cases the only factor that will require compensation is the recovery of the valuable mineral, with valuables discarded during pre-concentration typically lost to final waste.

Beneficiation tests

Based on the mineralogical results, one can identify the valuable mineral (or minerals) of interest, as well as the important gangue minerals. There are very few resources or types of resources that have not been investigated in the past, and information about successful beneficiation strategies can be obtained by conducting a thorough literature review, or by technical discussions with similar operations.

Once particles have been broken down to the required top size, the properties of the gangue and valuable minerals can be used to separate them to produce a saleable product.

The basis of separation for most ore types can be ascertained by surveying similar operations, and there are significant sources of mineral properties in the public domain. A few examples these are shown in Table I and Figure 5. Detailed information of most minerals can also be obtained through general literature surveys or from dedicated web sites like Webmineral (Mineralogy Database, 2012)and Mindat (Mindat, n.d.).

Certain separation techniques are applicable at various top sizes. Floating a 30 cm rock by attaching air bubbles to its surface is unlikely to succeed, as is optically sorting 100 μm particles. A good summary is provided by Kelly and Spottiswood (1982) (Figure 6). In the decades following publication of this textbook, some separation processes have been refined to allow a wider application, but these ranges remain a good rule of thumb in the absence of other information.

Beneficiation based on size

The typical run-of-mine ore is too large for easy handling during downstream processes and further crushing and screening, which also provides a convenient opportunity to separate minerals based on their relative hardness, is required.

Table I

Densities for selected common minerals[7] (The Engineering Toolbox, n.d.)

Mineral	Density (g/cm^3)
Magnetite	5.15
Braunite	4.76
Hausmanite	4.76
Jacobsite	4.75
Illemenite	4.72
Brannerite	4.5
Chromite	4.5
Fluorite	3.13
Musscovite	2.82
Talc	2.75
Calcite	2.71
Quartz	2.62

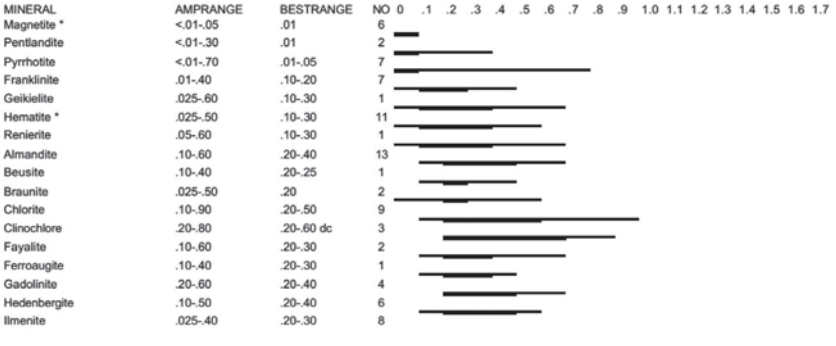

Figure 5—Extract of data summarizing the magnetic response for a number of minerals (Rosenblum and Brownfield, 2013)

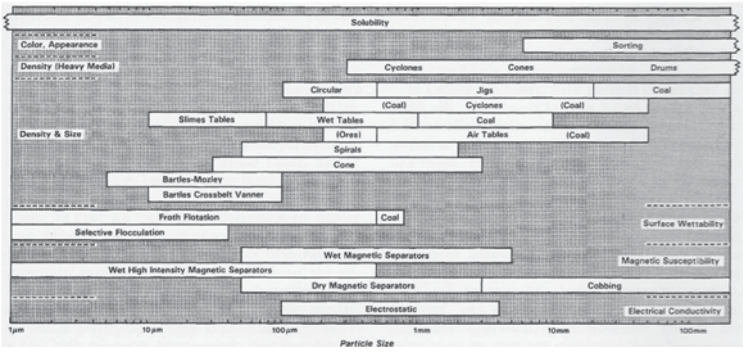

Figure 6—A graphical summary of separation processes applicable at various size ranges (Kelly and Spottiswood, 1982)

Typical examples of this are the use of Bradford breakers in coal processing (Figure 7), and scrubbing for upgrading of bauxites, manganese, diamond-bearing kimberlites, and other weathered materials.

Confirming the amenability of an ore to upgrading by classification can be done at a high level by wet screening of a representative sample, but the attritioning effect of particles being tumbled or mechanically agitated in a slurry is best simulated by conducting small-scale tumbling tests.

For smaller samples, these tests can be conducted by tumbling an ore in a vessel such as a cement mixer, followed by screening. If a few hundred kilograms of material are available, small scrubbing units can be used (Figure 8).

Beneficiation based on density

Separation of minerals based on differences in density dates back to the early Roman Empire, when alluvial gold deposits were mined using panning and sluices (Mining & Metallurgy, n.d.). Coal washing in South Africa dates back to the early 1900s (De Korte, 2010).

The upgrading of a weathered iron ore on a shaking table is shown in Figure 9, with the higher density particles in the dark band at the top of the picture, and the lighter gangue washed to the edge of the table at the bottom.

If mineralogical work indicates that gangue and valuable minerals can be separated based on density differences, testing needs to be done to show if this is will be feasible and at which top size optimal results are obtained. The most fundamental test of an ore's amenability to density-based separation is a sink-float test. In this procedure, a crushed ore sample is immersed in a liquid at a certain density. The lower density particles float, and are removed. The higher density 'sinks' are removed separately, and immersed in a fluid with higher density than the first bath. This process is repeated multiple times, fractionating the sample into density intervals.

The sink-float technique is typically limited to material coarser than 0.5–1 mm, but there are also service providers that conduct these tests down to 45 μm (ALS Global, n.d.) at densities up to 4.4 t/m^3. Using high-density liquids (Central Chemical Consulting, n.d.) with suspensions of tungsten carbide (WC), separation at densities as high as 5 t/m^3 is possible. MLA or QEMSCAN results can also be used to generate washability curves – typically useful for characterization of fine ore samples. These derived characteristics are very useful, but must be interpreted with care.

Figure 7—An illustration of a Bradford breaker, typically used in coal processing (TerraSource Global, n.d.)

Figure 8—An example of a pilot-scale scrubber

Figure 9—Using a shaking table to investigate rejection of the final remaining gangue particles in a fine iron product

There are a number of alternative techniques, such as the shaking table, the mineral density separator (MDS), and Viking/Reichert cone, for dividing ores into density fractions. All of these processes, however, have some element of process efficiency built into their results, and as such should be used with extreme caution. An MDS, for example, is quite suitable for investigating the jigability of an ore, but its outputs will not provide an accurate simulation of dense medium separation or an inline pressure jig.

Following the separation process, material in each density fraction is weighed and analysed to examine its chemical and/or mineralogical make-up. Based on the weight of material in each density fraction as well as its composition, a suitable separation density can be identified to pursue in further work.

The sink-float process characterizes the ore, allowing simulation of a separation process by using sigmoidal equations, typically the Whiten equation:

$$P_i = \frac{1}{1 + e^{\frac{1.099(\rho_{50} - \rho_i)}{Ep}}}$$

with

$$Ep = \frac{\rho_{75} - \rho_{25}}{\rho_{50}}$$

Ep is the slope of the partition curve, and can be calculated by dividing the difference between the densities at which 75% and 25% of the material in a density fraction report to sinks by the cut density of the operation. A lower number therefore indicates better separation. If a reasonable assumption can be made for the *Ep*, a simple weighted average product grade can be used to identify the required separation density.

The Whiten equation assumes symmetrical behaviour, with equal misplacement of both low- and high-density material, and it ignores the effect of particle size. The assumption around particle size is typically acceptable for dense-medium-based processes – as is the assumption regarding symmetrical misplacement.

These assumptions are typically not valid for water-based processes, for which the Erasmus curve (Erasmus, 1973) would be more applicable:

$$P_i = \frac{t_{e2} - \arctan\left(k(\rho_i - c)\right)}{t_{e2} - t_{e1}} \times 100$$

For symmetrical curves t_{e2}=-t_{e1}

Typically, 0.02+ρ_{50} < C < ρ_{50}-0.02

The Erasmus curve can only be used to evaluate partition curves for recoveries less than 99% and more than 1%, as is graphically shown in Figure 10.

While dense medium processes can be simulated with acceptable accuracy by using typical process efficiencies, water-based separation processes will require further test work in order to determine relevant process performance parameters. A few examples of such equipment include jigs (conventional and in-line pressure), spirals, and hindered settlers.

Beneficiation based on magnetic separation

If the gangue and valuable minerals have different magnetic susceptibilities, it is possible to separate them based on this property. Typically this is achieved by moving the particles through a magnetic field, with magnetic particles attracted by the field and the non-magnetic particles being rejected.

Depending on the size of the particles and their magnetic susceptibilities, a range of possible separators can be used – a few examples are shown in Figures 11–13.

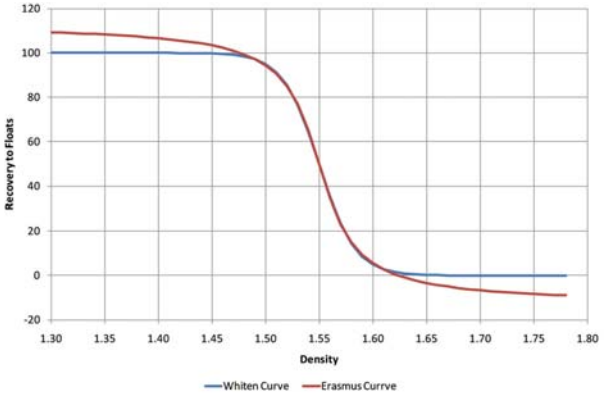

Figure 10—An illustration of the partition curves generated by the Whiten and the Erasmus equations

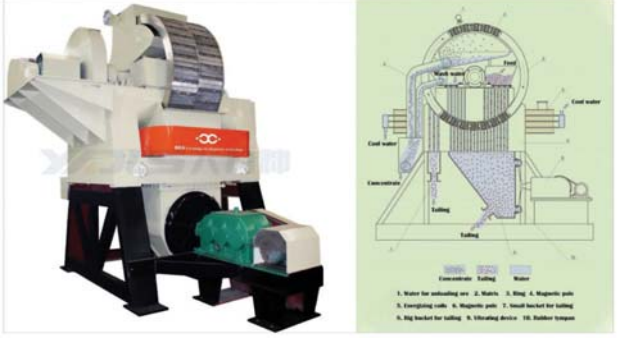

Figure 11—A high-intensity pulsed vertical magnetic separator (YDLS, 2013)

Figure 12—A fairly typical multistage rare-earth roll magnetic separator (YDLS, 2013)

Wet high-intensity separation is typically used for fine to ultrafine materials with low magnetic susceptibility, wet low-intensity separation for fine materials with high magnetic susceptibilities, and rare-earth rolls for medium to fine material with intermediate to low magnetic susceptibilities.

How does one decide on the correct equipment for a given application? While sources like the USGS (Figure 5) and original work by E.W. Davis are good places to begin, many minerals do not have a single value for their magnetic susceptibility (Rosenblum and Brownfield, 2013), with mineral chemistry influencing magnetic response. There are various ways to examine the magnetic response of an ore sample.

Magnetic susceptibility is commonly measured by (Marcon and Ostanina, 2012):

- Satmagan (Rapidscan systems, n.d.).
- Kappabridge (ASC scientific, n.d.), (AGICO, n.d.)

The magnetic susceptibility of pure minerals is of limited value when the requirement is to determine the amount of material that can be extracted, as well as the grade of product, as a function of variables such as top size, feed grades, and separation intensities. The Davis tube (Figure 15) is a standard tool used for high-susceptibility minerals such as magnetite. For lower susceptibility minerals such as the rare-earth minerals, various laboratory-scale versions of rare-earth or induced roll magnetic separators are available.

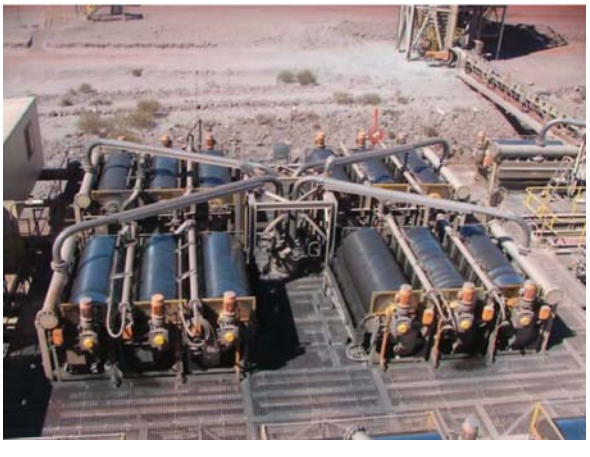

Figure 13—A typical magnetite concentration circuit utilizing low-intensity magnetic separation

Flotation

In the process of froth flotation, particles are separated based on differences in their surface properties. An over-simplified explanation is that particles are selectively attached to air bubbles in a frothy slurry, with the air bubbles rising and carrying attached particles with them. The particle-laden froth is then removed at the top of the flotation cell, with the balance of the material removed at the bottom of the cell.

Figure 15—A Davis Tube

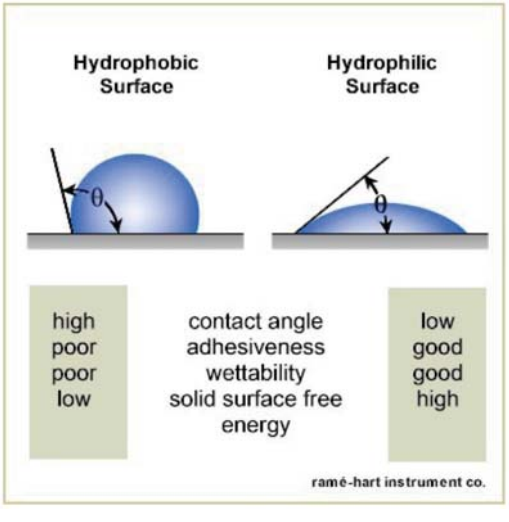

Figure 16—I Illustration of two different particles subjected to a contact angle test (Ramé-hart Instruments, n.d.)

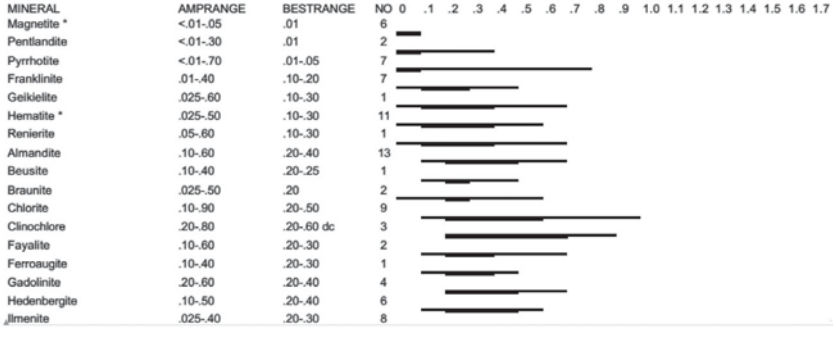

Figure 14—Summary of magnetic response for a number of minerals (Rosenblum and Brownfield, 2013)

In theory one could examine the surface properties of a particle and predict its flotation behaviour. There are a number of tests to determine the surface characteristics of materials, a few of which are discussed below.

In principle, a particle can be floated only if it is not wetted by water (hydrophobic). One way of determining this, and to investigate the effect of various flotation reagents, is to measure the angle between a drop of water and the surface of the material in question (Figure 15).

Zeta potential

The electrical potential of a particle surface can also be used as an indication of separation selectivity compared to other particles (Azonano, 2013; Fuerstenau, 2005).

Similar to the case with magnetic susceptibility, application of both the methods listed above in predicting flotation performance is problematic, with only limited correlation between contact angle, measurements of zeta potential, and flotation performance in the absence of flotation tests.

Various small-scale experimental set-ups exist for flotation performance prediction, with a conventional laboratory flotation cell the most commonly used. An example of such a cell is shown in Figure 17, with a test illustrated in Figure 18.

Figure 17—A laboratory flotation machine with a transparent 1 litre cell (Sepor, n.d.)

Figure 18—Froth being scraped off (from the flotation cell at the back into a bin in front) during a laboratory-scale flotation test

Comminution properties

A number of tests can be conducted to determine comminution properties for an ore. A few of the most common tests include:

- Crushing Work Index (CWI)
- Bond Ball mill Work Index (BBWI/BWI)
- Bond Rod mill Work Index (BRWI).

In addition, point load index, drop-shatter tests, small high-pressure grinding roll tests, abrasion tests, and other specialized tests can be conducted.

Crushing Work Index

A number of individual lumps of the ore between 50 mm and 75 mm in size are used in this test. Twenty particles are typically required. The particles are subjected to impact from a falling pendulum (Figure 19), and the amount of energy consumed in breaking each particle is recorded.

Bond Work Index

This procedure was developed to assess the energy required to mill particles from a top size of 3.35 mm to the required top size, and includes the effect of the circulating load. The standard Bond test consists of a detailed procedure (JKtech, n.d.), executed under strictly controlled conditions using well-defined equipment (Figure 20). Although there may be debate about the validity of the Bond mechanical equation under some conditions, the Bond Work Index remains one of the most reproducible methods of estimating the power required to grind an ore.

Figure 19—An example of a Bond Crushing Work Index set-up (Laarmann, n.d.)

Figure 20—A standard Bond mill

Example of PFD development

In the case of a titanomagnetite deposit, the base-case assumption was that the final product would be produced by magnetic separation. This decision was made based on previous experience when working with these ores as well as information in the literature.

The deposit consists of two ore types; one a finely disseminated and the other a massive titaniferous magnetite, as illustrated in Figure 21. Since the deposit outcrops, both of these ore types are also present in weathered and un-weathered conditions.

Some of the questions that had to be answered during the concept study were:

- Can a coarse product be produced – at least from the massive ore?
- Is scrubbing a viable way of silica rejection?
- What is the optimum grind for product grade and yield maximization?

Evaluation of scrubbing

Part of the resource has been weathered, possibly breaking down the silicates sufficiently to allow upgrading of the sample through scrubbing. This process step aims to break down the friable weathering products during a wet tumbling process, followed by screening in order to remove the fine (low-grade) fraction.

The results of one test are summarized in Figure 22, where increasing mass yield to the product corresponds to reducing the size below which material is rejected to waste. A mass yield of 100% implies no screening, and reflects the head grade of the sample.

Based on these results it is possible to increase the total iron content from 43% to 46% by rejecting 16% of the material, with only a limited further increase in the grade possible. In this case the upgrade was judged insufficient to justify the installation of a scrubbing step.

Figure 21—Examples of a massive and disseminated titaniferous magnetite

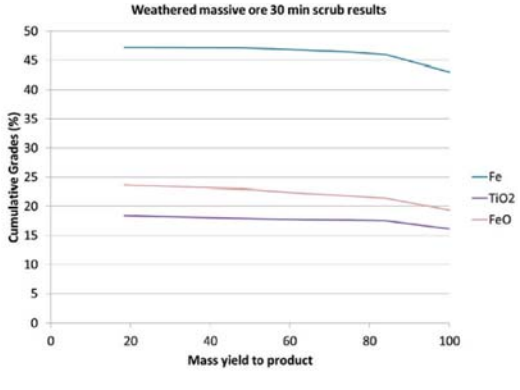

Figure 22—Cumulative grades as a function of yield (i.e. decreasing screen size) for the major components in a scrubbed sample

Dense medium separation

Sink-float tests were conducted to establish the viability of a density-based method to separate ore and gangue minerals.

These tests were conducted at two top sizes, 6 mm and 12 mm, which were selected based on visual interpretation of a number of drill core samples by the project's geological team. The purpose of these tests was to ascertain whether a final product could be obtained, or whether waste could be rejected early in the flow sheet. Two density points were therefore identified for testing – 2.96 t/m^3, and 3.6 t/m^3. The higher density was targeted to give an indication of a high-grade product's quality, and the lower density to establish the potential for silicate rejection.

Figure 23 illustrates the cumulative recovery of a number of species as a function of decreasing separation density at a top size of 6 mm. The dotted red line indicates no up- or down-grading, with valuable and waste materials both simply splitting in the same ratio.

It was observed that silicates preferentially reported to the low-density float fractions, with valuable iron and titanium reporting to the higher density fractions. As the separation density is increased, titanium and iron are selectively recovered, with gangue minerals being selectively rejected.

With 86% of the iron in this sample recovered into 73% of the total mass and 72% of the silica in the sample rejected, density-based separation was identified as a feasible processing step – even if only to reject gangue minerals before further milling to produce a high-grade concentrate.

Establishing optimum top size

Based on previous knowledge, magnetic separation at low intensities was identified as a suitable base-case process option for this ore. Test work aimed at ascertaining the best product grade, as well as the grind-grade relationship was therefore conducted using a Davis tube.

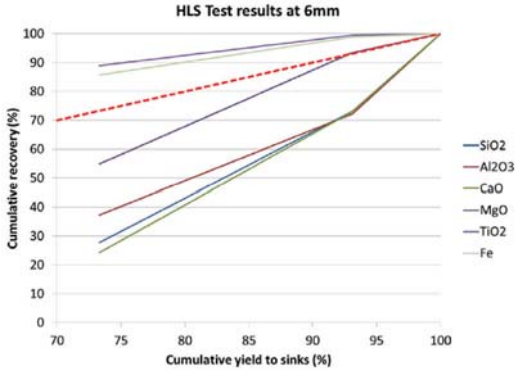

Figure 23—A summary of an HLS test on a titanomagnetite, with elemental recovery on the y-axis and mass recovered to sinks on the x-axis

As the optimum product grade for the project does not necessarily correspond to the highest grade (and the finest grind), a series of top sizes was investigated, from 80% finer than 500 μm to 80% finer than 38 μm. Figure 24 and Figure 25 summarize the results of these tests.

Based on this evidence there is no reason to grind finer than 500 μm, but because no decrease in product grade is observed at the upper end of the test range, it is possible that a coarser top size could in fact work equally well.

Further work

Although not discussed above, more detailed work regarding the optimum top size and test conditions is recommended. The lack of liberation at finer sizes is not typical, and would need to be confirmed during the concept phase.

No comminution work has been done yet, and typical numbers were used for plant design purposes. Literature reviews as well as vendor opinions were used for design of the comminution circuit. This was acceptable at the current stage of the project, but these assumptions will have to be confirmed by test work in subsequent study phases.

Thickeners were sized using a target rise rate, and this sizing confirmed by vendors. Settling behaviour and filtration rates will also have to be confirmed by testing.

Conclusion

Based on a geological interpretation of an orebody, followed by some bulk and/or micro-mineralogy, sufficient information can be obtained about an ore to identify potential beneficiation routes.

One of the easiest ways to beneficiate an ore is by methods based on particle size. If the gangue or the valuable component can be selectively broken down, a simple classification step can be used to upgrade the ore. If siliceous gangue minerals have been weathered by prolonged exposure to water and oxidation, there is a good probability that a wet tumbling process can be used to break down the gangue minerals and separate them from the valuable components. Following crushing and/or scrubbing, the sample can be screened into various size fractions and analysed to determine if sufficient upgrading has occurred.

Density, magnetic susceptibility, or surface properties can also be used to effect separation of valuable and gangue minerals. Density is the simplest of these properties to exploit. Once a sample has been reduced to the required top size, the material can be subjected to density characterization by sink-float analysis. Select laboratories can conduct sink-float testing of fine particles, and micro-mineralogy (MLA etc.) can be used to determine the washability of an ore. This washability information can then be used to simulate density-based separation by using information about the separation process from the literature and vendors.

Magnetic separation and flotation represent two less well-understood separation techniques. Fundamental characterizations such as magnetic susceptibility and contact angle measurement are available for each of the processes respectively. These characteristics cannot however, be used to predict product grades and yields. Empirical characterization, such as Davis Tube and bench-scale flotation, can be used to predict an ore's behaviour during magnetic separation or flotation.

Although there are many definitions for liberation, this need not be a significant concern during the early phases of a study. The coarsest top size required to yield a saleable product grade is typically determined by tests using the methods described previously.

Depending on the project, comminution and de-watering tests are typically conducted only during later stages of the project (pre-feasibility or feasibility). This is often due to lack of suitable material for testing. Comminution tests, for example, typically require significant masses of material representing all ore types in a deposit and at top sizes significantly coarser than that obtained by conventional core drilling. Coal is one commodity for which a procedure has been developed to characterize large diameter drill core material to predict the particle size distribution (PSD) that will be observed in a plant, although even this procedure involves certain problems.

Dewatering tests (thickening and filtration) require a significant volume (tens of litres) of representative material. For example, obtaining samples that accurately represent flotation plant tailings at this stage of a project would be a significant problem.

In the absence of test data, comminution and dewatering processes are typically designed based on literature reviews, historical information from similar operations, and the experience of equipment vendors.

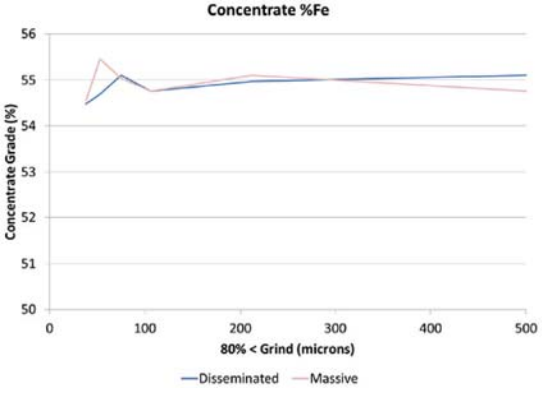

Figure 24—Product Fe grade as a function of grind for two titaniferous magnetite samples

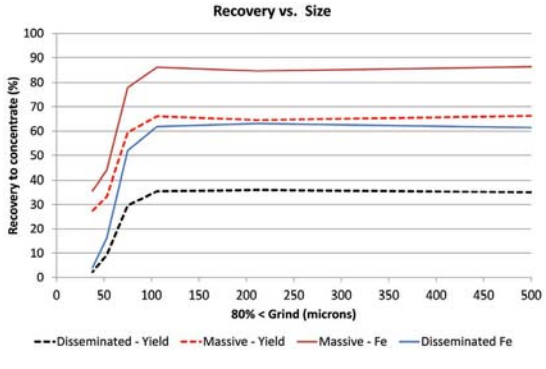

Figure 25—Recovery of iron and mass yield to concentrate for two titaniferous magnetite samples

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Maximizing haematite recovery within a fine and wide particle-size distribution using wet high-intensity magnetic separation

by M. Dworzanowski*

Synopsis

The physical beneficiation of iron ore that has a wide particle-size distribution is problematic, regardless of the process applied, whether dense medium separation, gravity concentration, magnetic separation, or flotation. The problem of particle size is further compounded when there is a significant -10 μm fraction. Generally the approach to a wide particle-size distribution is to split into narrower size ranges and treat each separately. More often than not the -10 μm fraction is not treated but discarded. This approach results in a more complicated and expensive flow sheet and the loss of any potential value in the -10 μm fraction.

Wet high-intensity magnetic separation (WHIMS) bench-scale test work was conducted on a haematite-rich material with a particle size of -200 μm. What made this material different was that it contained a 60% -10 μm fraction, hence discarding the -10 μm material was not an option. The objective of the test work was to determine how to maximize the recovery of the haematite across the full particle size range. Given the unusual particle size distribution, it was concluded that WHIMS would be the only practical beneficiation route. The -200 +10 μm and -10 μm fractions were treated separately and together under varying WHIMS conditions. For a given concentrate grade, the mass yield obtained was greater when the total particle-size distribution was treated. The inferred optimum conditions, using the same material, were tested on a pilot-scale WHIMS and similar results were obtained.

Keywords

WHIMS, haematite recovery, particle size distribution, -10 μm fraction.

Introduction

The beneficiation of iron ore in the size range of -32 +0.2 mm is well established globally. Dense medium separation (DMS) and gravity concentration are the beneficiation processes used. Iron ore production in this size range is almost exclusively based on haematite. Table I provides a summary of the processes commonly used for various size fractions.

The beneficiation of -200 μm iron ore will become more important as lower grade resources will require liberation of the contained iron ore at finer sizes. Currently, the treatment of -200 μm haematite-based iron ore is predominantly via flotation, specifically in Brazil and the USA. For -200 μm magnetite-based iron ore, treatment is predominantly via wet low-intensity magnetic separation (WLIMS).

The physical beneficiation of iron ore that has a wide particle-size distribution is problematic, regardless of the process applied, whether DMS, gravity concentration, magnetic separation, or flotation. The problem of particle size is further compounded when there is a significant -10 μm fraction. Generally the approach to a wide particle-size distribution is to split into narrower size ranges and treat each separately. More often than not the -10 μm fraction is not treated but discarded. This approach results in a more complicated and expensive flow sheet and the loss of any potential value in the -10 μm fraction.

Kumba Iron Ore's Sishen Mine currently produces over 3 Mt/a of -200 μm iron ore, which is sent to final tailings. This material contains 50–80% by mass of largely liberated haematite, representing a mineral resource from which no value is recovered. The viable beneficiation of this material would result in extra iron ore production as well as reducing the mine's tailings footprint and water consumption.

The beneficiation of -200 μm haematite-based iron ore can be accomplished in principle using gravity concentration, flotation, or magnetic separation. However, in this case 60% of the material is finer than 10 μm, which makes magnetic separation the only viable option.

The difficulty in recovering -10 μm haematite

The recovery of -10 μm mineral particles is extremely difficult in most cases and haematite is no exception. There are only two practical options for recovering -10 μm haematite, namely magnetic separation and flotation.

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Table I

Iron ore beneficiation processes

Size range	Processes
-32 +6.3 mm	DMS or jigs
-8 +1 mm	DMS or jigs
-1 +0.2 mm	Spirals or hindered settling separators

Iron ore flotation is practised extensively in Brazil and test work there has shown (Ma, Marques, and Gontijo, 2011) that regardless which flotation option is considered, the inclusion of -10 μm material in the flotation feed is not recommended due to the substantial increase in reagent consumption, which makes this option uneconomic. This is why most iron ore flotation plants deslime their feed at 10 μm.

Because of the low magnetic susceptibility of haematite, wet high-intensity magnetic separation (WHIMS) is applied to the magnetic recovery of -1 mm haematite. In 1982 a study was conducted (Forssberg and Kostkevicius, 1982) in which a number of different WHIMS machines were compared in terms of haematite recovery with decreasing particle size. Down to 20 μm all the machines managed good recoveries. Below 20 μm there was a noticeable drop in recovery, and below 10 μm there was a substantial drop in recovery. In 1989 a WHIMS circuit was installed at the Goldsworthy iron ore beneficiation plant in Australia (Miller, James, and Turner (1983) and the feed was deslimed at 10 μm, since test work had shown that treatment of this fraction by WHIMS would not be economical due to low recovery.

The issue of competing forces

The recovery of -10 μm haematite using magnetic separation, specifically WHIMS, is dependent on the probability of the haematite particles coming into contact with the magnetic matrix and the probability of the captured particles adhering to the magnetic matrix. The current design of WHIMS magnetic matrices ensures that the probability of the haematite particles coming into contact with the magnetic matrix is high, but adhesion to the matrix is a problem.

Paramagnetic particles, like haematite, adhere to a magnetic matrix by virtue of the magnetic force on the particles. Svoboda and Fujita (2003) represented the magnetic force by the following relationship:

$$F_m = k/\mu_0(VB\blacktriangledown B) \quad [1]$$

where k is the volumetric magnetic susceptibility of the particle, μ_0 is the magnetic permeability of vacuum, V is the volume of the particle, B is the external magnetic induction, and $\blacktriangledown B$ is the gradient of the magnetic induction. Magnetic force is thus proportional to the product of the external magnetic field and the field gradient. Modern WHIMS designs are able to generate a substantial magnetic force. However, the magnetic force is also proportional to particle volume and magnetic susceptibility. Therefore, for a -10 μm haematite particle with low magnetic susceptibility the magnetic force will be substantially reduced.

In WHIMS the magnetic force holding the -10 μm haematite particle will be competing with other forces, namely force of gravity, inertial force, hydrodynamic drag, and interparticle forces. In this particular case, hydrodynamic drag will have by far the greatest impact. The hydrodynamic drag is given by Svoboda and Fujita (2003):

$$F_d = 6\Pi\eta b v_p \quad [2]$$

where η is the dynamic viscosity of the fluid, b is the particle radius and, v_p is the relative velocity of the particle with respect to the fluid. Hydrodynamic drag is thus mainly proportional to the viscosity and velocity of the slurry entering the WHIMS machine.

Therefore, to increase the probability of the -10 μm haematite particles adhering to the magnetic matrix, one must increase the magnetic force and reduce hydrodynamic drag.

Promoting a favourable balance of competing forces

Magnetic force on a particle can be increased in two ways. Firstly, by increasing magnetic field strength and gradient, which are functions of WHIMS operation. Secondly, by increasing particle size. The size of very fine particles can be increased by chemical flocculation. Although it is possible in principle to selectively flocculate haematite particles using sodium oleate (Roy, 2012), this method is not practical because the presence of very fine gangue particles will result in entrainment, thus lowering potential product Fe grade. This process also requires the addition of sodium silicate (dispersant) and sodium hydroxide (pH modifier), thus making the operating costs very high. The only other way of 'increasing' particle size is to ensure that there is a reasonable quantity of +10 μm haematite particles in the solids feed to the WHIMS. The coarser haematite particles will have a higher probability of adhering to the magnetic matrix and will tend to act as 'shields' for the finer haematite particles, thus increasing their probability of adhesion.

To decrease hydrodynamic drag in a WHIMS would require two actions. Firstly, decreasing the viscosity of the WHIMS slurry feed would require a reduction in the feed solids concentration. As a guideline, the WHIMS slurry feed density should be no more than 30% solids by mass. Secondly, to decrease the velocity of the WHIMS slurry feed it is essential that the WHIMS magnetic matrix comes into contact with the slurry in a relatively 'quiescent' manner. In other words, the feed slurry stream directly impacting the matrix will be detrimental. This is where vertical carousel WHIMS machines have a distinct advantage over horizontal carousel WHIMS machines.

Methodology

Sample characterization

A composite sample of -200 μm iron ore was taken from the slimes thickener underflow at Sishen's DMS plant. The particle size distribution (PSD) of this material as well as chemical analysis by size fraction is shown in Table II. The PSD was determined by using sieves down to 25 μm, and the 10 μm screening was done using screen cloth in an ultrasonic bath. The chemical analysis was by X-ray fluorescence (XRF).

Table II

Material PSD and chemical analysis by size fraction

	PSD%	Fe%	SiO_2%	Al_2O_3%	K_2O%	P%	Mn%	CaO%	MgO%	S%
+212 μm	1.11	58.60	9.62	3.00	0.60	0.12	0.12	0.39	0.09	0.04
-212+45 μm	19.01	54.30	14.20	4.08	0.83	0.12	0.15	0.39	0.12	0.04
-45+25 μm	6.84	56.73	11.06	3.62	0.72	0.13	0.10	0.46	0.14	0.06
-25+10 μm	11.67	56.00	11.20	4.10	0.81	0.14	0.10	0.46	0.14	0.04
-10 μm	61.38	55.00	10.50	6.05	1.20	0.16	0.09	0.39	0.13	0.04
Unsized	100.00	55.00	11.60	5.44	1.10	0.15	0.11	0.38	0.14	0.04

The bulk modal mineralogy by size fraction, as determined by X-ray diffraction (XRD) analysis using the Rietveld method, is shown in Figure 1. The liberation of haematite by size fraction is shown in Table III. In this case, liberation is defined as the percentage of total haematite that occurs as particles with 95% or greater haematite content. The liberation was determined using QEMSCAN. QEMSCAN uses energy-dispersive spectra (EDS) from X-rays emitted by the sample when it interacts with an electron beam. These X-rays give the chemical composition of the mineral phases and the spectra are used to build up mineral maps, from which the liberation and mineral associations can be determined.

As part of the mineralogical investigation, particle shape was examined to determine if there was any significant difference in haematite particle shape between the size fractions. Particle shape was assessed by determining particle aspect ratio (AR), which is the ratio of the longest axis of the particle to the shortest axis. The distribution of particle shapes between the various size fractions is shown in Table IV.

The material characteristics of this -200 μm iron ore stream can be summarized as follows:

- There is no significant difference in chemical composition between the size fractions, which means that the recovery of all the size fractions will contribute to product quality
- There is no significant difference in bulk modal mineralogy between the size fractions. The minerals that are not iron oxides are non-magnetic. Therefore, there are no mineralogical obstacles to the application of magnetic separation to this material
- Haematite liberation is extensive across all the size fractions. The liberation in the +45 μm fraction is almost 10% lower than the other fractions, but in terms of magnetic recovery this will be compensated by the coarser particle size
- There is no significant difference in haematite particle shape by size fraction. Therefore, particle shape will not influence magnetic recovery.

Test work to demonstrate the effect of magnetic force on the recovery of -10 μm haematite

The test work was conducted using a SLON 100 bench-scale WHIMS (Figure 2).

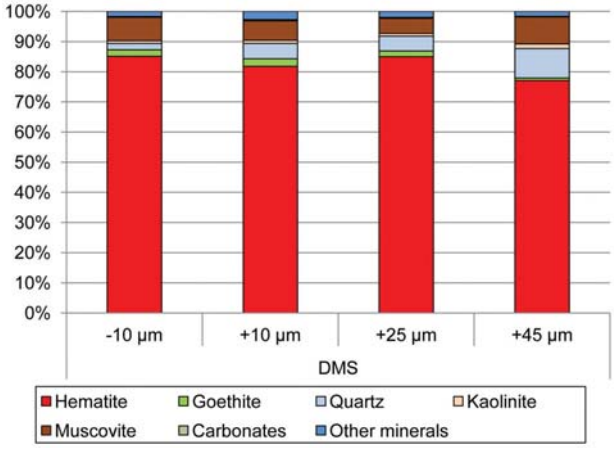

Figure 1—Bulk modal mineralogy by size fraction

Table III

Haematite liberation by size fraction

Size fraction	% haematite liberation
+45 μm	89.2
-45 +25 μm	97.7
-25 +10 μm	98.4
-10 μm	98.6

Table IV

Haematite particle shape distribution by size fraction

Size, μm	AR <1	AR <2	AR <3	AR <4	AR <5	AR <6	AR <7	AR <8	AR <9	AR < 10
+45	0.01	81.74	15.08	2.23	0.47	0.23	0.13	0.05	0.01	0.02
-45 +25	0.03	79.78	15.64	3.11	0.92	0.34	0.12	0.06	0.00	0.01
-25 +10	0.00	80.57	15.25	2.83	0.82	0.34	0.06	0.07	0.03	0.01
-10	0.13	80.43	17.09	1.97	0.22	0.09	0.06	0.00	0.00	0.00

The SLON 100 allows magnetic field strength to be varied by varying the electrical current to the electromagnet. Magnetic field strengths of up to 13 000 G (1.3 T) were used.

Figure 2—SLON 100 bench-scale WHIMS

The matrix is in the form of stainless steel rods with differing diameters, namely 1 mm, 1.5 mm, 2 mm, and 3 mm. A decrease in rod diameter results in an increase in magnetic gradient. Pulsation of the sample is also used to promote rejection of non-magnetic particles. For the purpose of this test work, pulsation was kept constant at a relatively low setting of 95 r/min for all the tests, since pulsation has no direct influence on the impact of magnetic force.

A matrix block with a given rod diameter was inserted into the SLON 100 feed chamber. The required magnetic field strength was set together with the fixed pulsation rate. For each test a 100 g sample was slurried to 25% solids by mass and placed in the feed chamber. The SLON 100 can accommodate a sample size of up to 300 g, therefore with a 100 g sample there is little chance of saturating the matrix. A total of 80 tests were conducted under the conditions shown in Table V. The unsized material and the four size fractions (-212 +45 μm, -45 +25 μm, -25 +10 μm, and -10 μm) were each tested under 16 different magnetic force settings.

Results

The results of the SLON 100 test work on various size fractions are presented in.Tables VI to X

Table V

SLON 100 test work conditions

Matrix	Field strength T	Sample				
		Unsized	-212+45 μm	-45+25 μm	-25+10 μm	-10 μm
1.0 mm	0.6	x	x	x	x	x
	0.8	x	x	x	x	x
	1	x	x	x	x	x
	1.3	x	x	x	x	x
1.5 mm	0.6	x	x	x	x	x
	0.8	x	x	x	x	x
	1	x	x	x	x	x
	1.3	x	x	x	x	x
2.0 mm	0.6	x	x	x	x	x
	0.8	x	x	x	x	x
	1	x	x	x	x	x
	1.3	x	x	x	x	x
3.0 mm	0.6	x	x	x	x	x
	0.8	x	x	x	x	x
	1	x	x	x	x	x
	1.3	x	x	x	x	x

Table VI

Effect of magnetic force on mass yield and grade, unsized sample

		Unsized samples							
Matrix	Pulsation r/min	0.6 T		0.8 T		1 T		1.3 T	
		Mass yield %	Fe grade %	Mass yield %	Fe grade %	Mass yield %	Fe grade %	Mass yield %	Fe grade %
1 mm	95	31.4	67.5	47.5	67.0	48.9	67.2	57.4	66.9
1.5 mm	95	20.3	67.4	34.7	67.5	41.0	67.0	47.3	66.8
2 mm	95	25.4	65.5	28.1	66.2	35.8	65.8	32.2	67.1
3 mm	95	12.6	66.6	22.5	64.1	20.4	66.8	22.1	65.3

Table VII

Effect of magnetic force on mass yield and grade, 212 +45 μm fraction

-212 +45 μm samples									
Matrix	Pulsation r/min	0.6 T		0.8 T		1 T		1.3 T	
		Mass yield %	Fe grade %	Mass yield %	Fe grade %	Mass yield %	Fe grade %	Mass yield %	Fe grade %
1 mm	95	42.5	65.5	55.9	64.4	62.9	64.9	65.9	63.4
1.5 mm	95	28.4	66.3	45.5	66.1	50.4	66.3	55.9	65.1
2 mm	95	25.1	65.0	39.9	65.2	40.8	65.0	42.1	65.2
3 mm	95	13.6	65.3	19.9	63.3	25.8	65.2	29.6	64.9

Table VIII

Effect of magnetic force on mass yield and grade, -45 +25 μm fraction

-45 +25 μm samples									
Matrix	Pulsation r/min	0.6 T		0.8 T		1 T		1.3 T	
		Mass yield %	Fe grade %	Mass yield %	Fe grade %	Mass yield %	Fe grade %	Mass yield %	Fe grade %
1 mm	95	41.0	67.0	51.3	66.4	54.7	66.4	52.9	65.3
1.5 mm	95	35.8	67.1	44.5	66.9	48.5	67.1	49.5	66.9
2 mm	95	27.0	66.8	37.5	67.0	38.1	66.3	40.4	66.6
3 mm	95	18.2	66.7	26.3	67.1	28.9	66.7	29.1	66.8

Table IX

Effect of magnetic force on mass yield and grade, -25 +10 μm size fraction

-25 +10 μm samples									
Matrix	Pulsation r/min	0.6 T		0.8 T		1 T		1.3 T	
		Mass yield %	Fe grade %	Mass yield %	Fe grade %	Mass yield %	Fe grade %	Mass yield %	Fe grade %
1 mm	95	46.0	67.3	55.5	66.8	56.2	67.0	62.3	67.2
1.5 mm	95	33.8	68.7	50.5	68.2	54.0	67.2	49.0	67.4
2 mm	95 rp	29.5	68.5	42.8	67.9	48.0	68.0	47.9	68.3
3 mm	95	22.4	68.1	31.2	67.4	32.7	67.4	37.7	68.0

Table X

Effect of magnetic force on mass yield and grade, -10 μm fraction

-10 μm samples									
Matrix	Pulsation r/min	0.6 T		0.8 T		1 T		1.3 T	
		Mass yield %	Fe grade %	Mass yield %	Fe grade %	Mass yield %	Fe grade %	Mass yield %	Fe grade %
1 mm	95	23.4	66.1	28.3	68.3	31.3	67.3	37.6	67.6
1.5 mm	95	13.2	66.7	21.6	65.9	23.7	66.6	23.6	65.9
2 mm	95	11.9	67.3	13.6	66.6	20.4	66.6	17.9	66.8
3 mm	95	9.4	67.3	15.3	67.1	17.1	67.3	18.7	65.8

Figures 3 to 6 show the effect of increasing magnetic field strength on mass yield for a given matrix rod diameter for the unsized material and the four size fractions.

Figures 7 to 11 show the effect of matrix rod diameter on mass yield and Fe product grade for a given magnetic field strength for the unsized material and the four size fractions.

Discussion

In magnetic separation, magnetic field strength is increased with decreasing magnetic susceptibility of the target mineral, and magnetic gradient is increased for decreasing particle size. In this investigation the test work results show that fine haematite recovery is clearly dependent on maximizing the magnetic force. The results also clearly demonstrate that treating the -10 μm haematite together with the coarser haematite significantly improves the recovery of the -10 μm haematite when compared to treating only the -10 μm fraction. Although the results suggest that operating at 1.3 T and using a 1 mm matrix would maximize mass yield, there are practical problems with this operating regime.

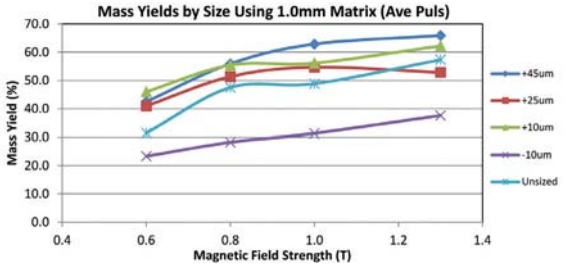

Figure 3—Effect of magnetic field strength, 1 mm matrix

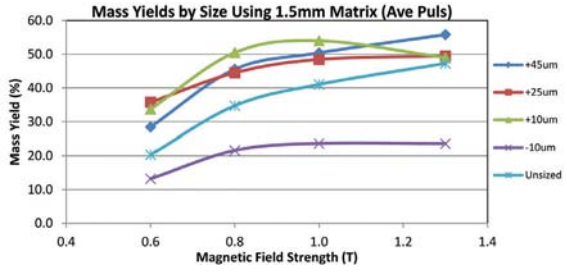

Figure 4—Effect of magnetic field strength, 1.5 mm matrix

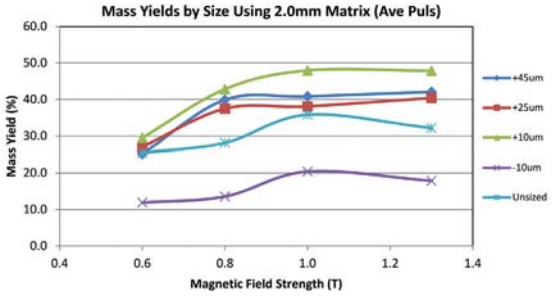

Figure 5—Effect of magnetic field strength, 2 mm matrix

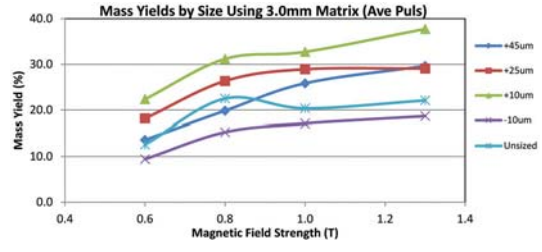

Figure 6—Effect of magnetic field strength, 3 mm matrix

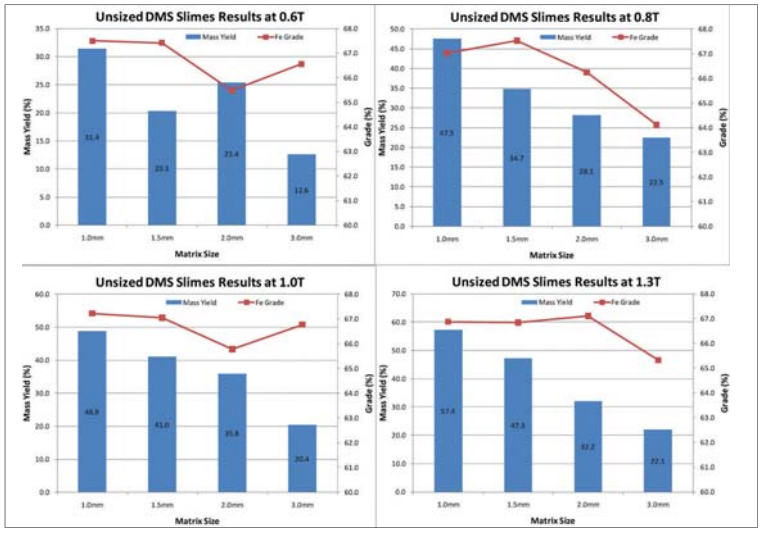

Figure 7—Effect of matrix rod diameter, unsized sample

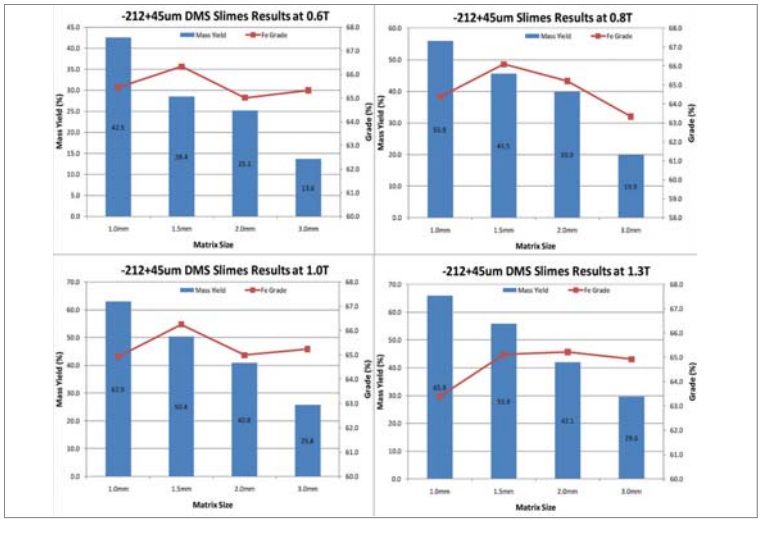

Figure 8—Effect of matrix rod diameter, -212 +45 μm fraction

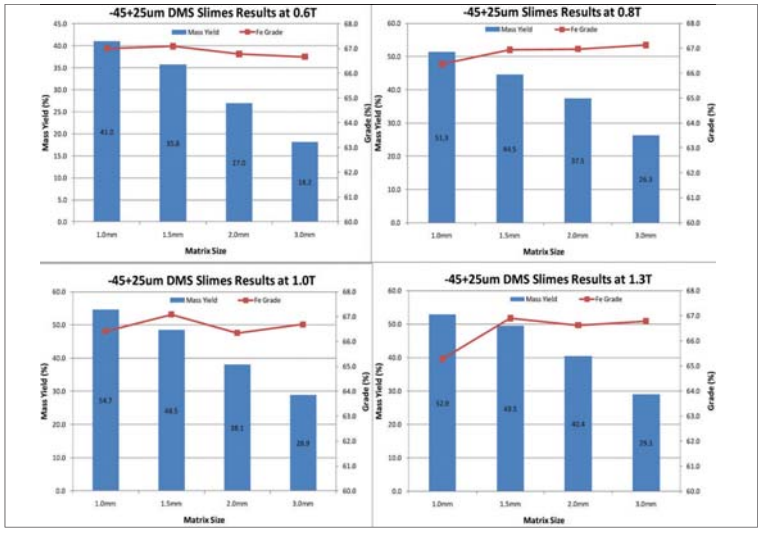

Figure 9—Effect of matrix rod diameter, -45 +25 μm fraction

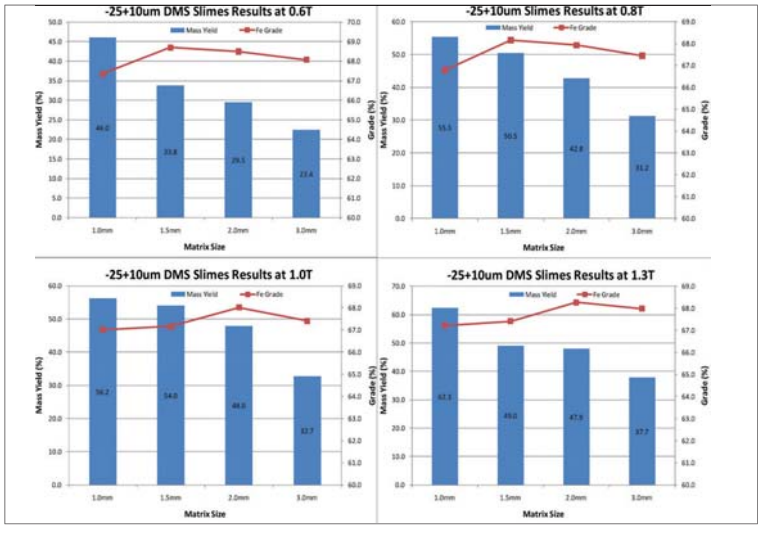

Figure 10—Effect of matrix rod diameter, -25 +10 μm fraction

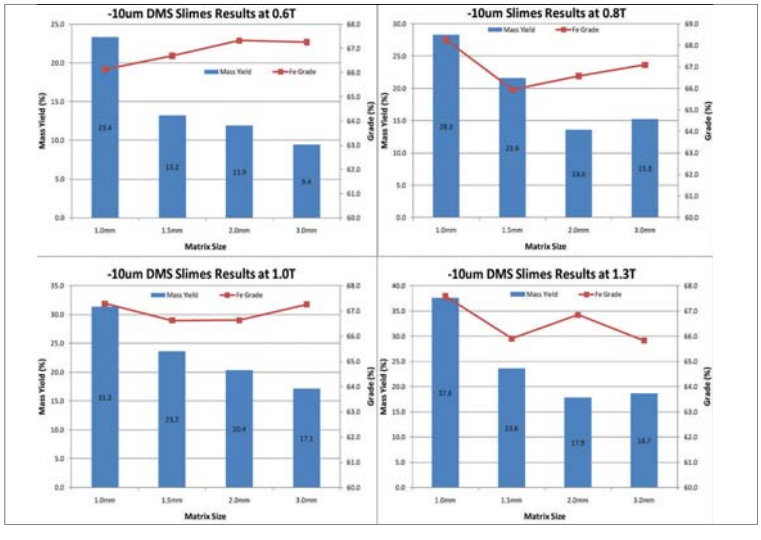

Figure 11—Effect of matrix rod diameter, -10 μm fraction

Most commercial WHIMS operate at a maximum of 1 T. Although some manufacturers can offer WHIMS with a 1.3 T capability, the extra capital and operating costs will be significant and the metallurgical benefit at a larger scale is unlikely to be achieved. This is because the recovery of paramagnetic minerals by WHIMS is subject to a recovery peak. As the magnetic field strength increases, the matrix loading increases to the point where further paramagnetic mineral recovery is not possible (Svoboda, 2004). The recovery tends to peak at around 1 T, which is corroborated by the results from this investigation (Figures 4 and 5).

Figures 3 to 6 illustrate what appear to be anomalies, in that higher yields are obtained from the +10 μm fraction than from the +25 μm and +45 μm fractions. This can be explained by differences in haematite liberation. With the same magnetic force conditions, a clear increase in yield with increasing size fraction will result only if the haematite liberation in each size fraction is similar. With Sishen slimes this is definitely not the case, because the material is not derived from a controlled comminution process.

The commercial application of the rod matrix in vertical carousel WHIMS is well proven down to a diameter of 1.5 mm. Although it is possible to install a 1 mm diameter rod matrix, the manufacturers at present are not prepared to guarantee performance because the mechanical stability of the matrix is poor. The wear rate of the matrix is very high and the rods become deformed at higher magnetic field strengths.

Therefore, taking into account practical considerations, the test work results point to a magnetic force requirement of 1 T with a 1.5 mm matrix. Under these conditions, the extra recovery of haematite obtained when treating the full PSD as opposed to individual size fractions is shown in Table XI. The results show that treating the full PSD will increase the mass yield to the concentrate by almost 20% at for the same concentrate Fe grade.

The prevailing conditions in a SLON 100 bench-scale WHIMS are very different to those of a full-scale vertical carousel WHIMS. The recovery of -10 μm haematite would therefore need to be studied on a pilot scale to determine the true magnetic recovery. A three-stage pilot plant circuit has been operated at Sishen using a pilot-scale SLON WHIMS with a 750 mm diameter vertical carousel to treat DMS slimes with similar material characteristics to the feed tested with the bench-scale SLON 100.

The scavenger WHIMS in this pilot-plant circuit was operated under the same optimum magnetic force conditions inferred from the SLON 100 bench-scale test-work, 1 T magnetic field strength and a 1.5 mm matrix. The scavenger feed slurry density was 15% solids by mass, thus slurry viscosity would not be an issue. Owing to the vertical carousel design of the SLON WHIMS, slurry velocity would also not be an issue. Figure 12 illustrates the response of the -10 μm fraction. This figure shows that there is consistent recovery of -10 μm haematite, since the average proportion of -10 μm in the concentrate is very similar to that of the feed. The relatively small and consistent difference between the -10 μm content of the feed and concentrate implies that because the feed comprises the full PSD the recovery of -10 μm is not critical, otherwise the difference between feed and concentrate would be considerably wider. The pilot plant data and bench-scale data cannot be compared directly because the operating environments are too dissimilar. However, the trends generated from the bench-scale and pilot scale testwork both confirm that maximizing the recovery of -10 μm haematite is dependent on treating a full PSD feed and applying a relatively high magnetic force.

Table XI

Yield / grade comparison at 1 T with a 1.5 mm matrix

Sample	Mass yield, %	Mags, % Fe
Full PSD	41.0	67.0
Sum of size fractions	34.3	66.6

Conclusions

The following conclusions can be drawn from the test work conducted:

- Although the beneficiation of material with a wide particle size distribution (PSD) is generally avoided, the same is not applicable to wet high-intensity magnetic separation (WHIMS). The bench-scale WHIMS test work conducted on -200 μm haematite-rich material demonstrated that separate treatment of different size fractions gave a concentrate mass yield of almost 20% less than the treatment of the full PSD
- The recovery of -10 μm haematite is considered problematic and generally is not practised. However, under the right conditions of hydrodynamic drag and magnetic force, reasonable recovery is possible
- Hydrodynamic drag can be reduced by decreasing slurry viscosity through the use of lower slurry feed densities (less than 30% solids by mass) and by reducing the velocity of the slurry impacting the WHIMS matrix, which is most effectively achieved by using a vertical carousel WHIMS instead of a horizontal carousel WHIMS
- The test work showed that a practical maximum magnetic force would be equivalent to 1 T magnetic field strength and the use of a 1.5 mm diameter rod matrix
- Repeating the bench-scale test work on a pilot scale with the same magnetic force and minimized hydrodynamic drag resulted in the same basic trend, with reasonable haematite recovery across the full PSD.

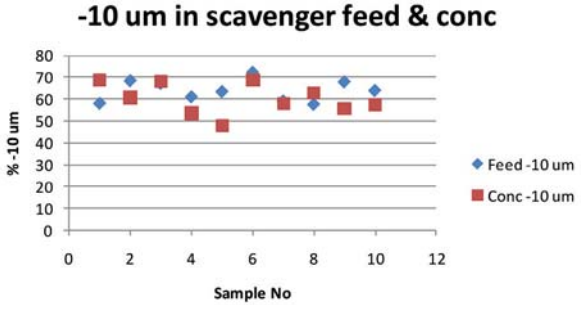

Figure 12—Scavenger WHIMS, -10 μm response

As a final summary:

- Generally, a feed particle top size of 1 mm is applied to WHIMS, whether horizontal or vertical carousel. With a feed top size of 1 mm or finer, classification of the feed ahead of WHIMS is to be avoided
- Apply a practical magnetic force – every material has a recovery peak
- The finer the feed PSD, the lower the feed slurry density should be
- A vertical carousel WHIMS is likely to produce better results than a horizontal carousel WHIMS.

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Operation and performance of the Sishen jig plant

by H.A. Myburgh* and A. Nortje†

Synopsis

Sishen Iron Ore Mine previously used only A-grade material (>60% Fe *in situ* value) from the pit for beneficiating in the DMS plant to a final product grade of 66% Fe in lump and 65% Fe fine ore. The B-grade (between 50% and 60% Fe) and C-grade material (between 35% and 50% Fe) were stockpiled separately, owing to the inability of the existing DMS plant to beneficiate material at densities higher than 3600 kg/m³. The ability to beneficiate the B-grade material at densities higher than 3600 kg/m³ was evaluated, and air-pulsed jigs were found to be techno-economically feasible and value maximizing.

The beneficiation of B-grade material would add to the existing DMS production of 28 Mt/a, with no additional mining cost and only limited costs for the handling of waste and B-material. The objective of the Sishen Expansion Project (SEP), i.e. the jig plant, was to produce 10 Mt/a of saleable product with six modules to the set physical and chemical specifications by 2009. During the start of construction, it was decided to add another two jig modules to the plant to increase production to 13 Mt/a.

During commissioning and ramp-up the shortcomings and advantages of the jigs were fully experienced and understood, resulting in many changes to optimize jigging performance..

Keywords

jigging, jig efficiency, jig control, physical separation, process optimization.

Introduction

Kumba Iron Ore's Sishen mine in South Africa produces 41 Mt of iron ore per annum from its beneficiation plants. The orebody consists mainly of laminated and massive haematite ore, of which 34 M/a is crushed to –90 mm before beneficiation by means of a combination of Wemco drums and dense medium cyclones. In addition, 22 Mt/a is crushed to -25 mm for beneficiation in the jig plant, which consists of eight modules of three jigs per module. The feed to each module is screened into three fractions that feed three different jigs. The Coarse Jig receives -25 mm +8 mm material, the Medium Jig is fed with -8 mm +3 mm material, and the Fine Jig receives -3 mm +1 mm material.

The jig plant with its 24 jigs was in full production towards the end of 2009, providing an equivalent of 10 Mt product per annum and above 13 Mt in the following two years. The jig plant was designed to receive lower grade material that needs a cut density of 4100 kg/m³ and higher to be able to produce the product to the specification at a design yield of 57.8% from an average Fe feed grade of 56%.

During hot commissioning and for two years after commissioning, many challenges were encountered and rectified, resulting in improved jig plant performance.

Plant layout

The feed material to the jig plant at Sishen Iron Ore Mine is reduced to a -25 mm top size in a three-stage crushing circuit and longitudinally stacked on two pre-beneficiation feed beds (Figure 1). The pre-beneficiation feed bed material is reclaimed by a drum reclaimer and conveyed to eight feed bunkers. Each of the eight conveyers feeds one jig module and belt scales on each conveyer control the feed rate.

The jig plant consists of eight modules with three jigs each: the Coarse Jig (-25 mm +8 mm), Medium Jig (-8 mm +3 mm), and Fine Jig (-3 mm +1 mm). The feed to the plant is spread evenly over the width of the primary double-deck screen with a shuttle conveyer. The top deck overflow feeds the Coarse Jig. The overflow of the bottom deck of the primary screen flows to a shuttle conveyer that spread the material evenly over the width of the chute that feeds the Medium Jig. The underflow of the primary deck flows to a single-deck secondary screen where the -1 mm material is screened out before the oversize material flows into the Fine Jig. The waste from all the jigs is conveyed on one conveyer system to the waste dump.

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Operation and performance of the Sishen jig plant

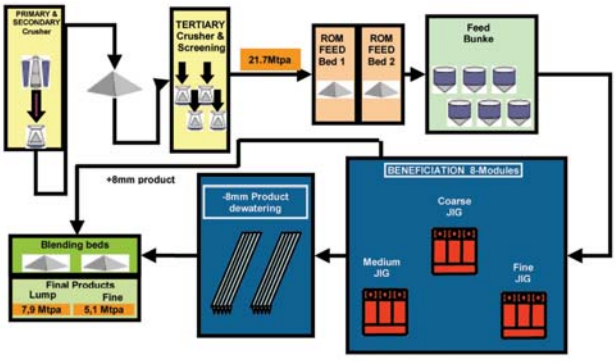

Figure 1—Overall jig plant layout

After beneficiation, the lumpy ore product (-25 mm +8 mm) is conveyed and stacked on the product blending beds, while the fine material (-8 mm +1 mm) is conveyed to the dewatering bunkers, where it remains for about four hours before being stacked on the fine product blending beds.

Jig construction and control

The jigs at Sishen Iron Ore Mine are under-bed pulsating jigs that are fully PLC-controlled. All the jigs are 3 m long, and the width varies from 4 m for the Coarse Jig to 3.5 m for the Medium and 2.2 m for the Fine Jig. The different widths for the jigs were selected to cater for the expected product size distribution (PSD) from the crusher circuit, taking in account the required specification for the final product.

The jig has a screen deck to support the jig bed and allow the water pulse generated in the air chambers to lift the beds to an acceptable height for the specific material and the hutch water. The hutch water flows through the bed at a constant rate to assist with the separation efficiency and keep the bed fluidized for longer. At the end of the jig, a float measures the stroke of the bed and indicates the product bed height. The height of the product bed is an indication to the PLC to open or close the product gates in small increments to control the product bed height in a narrow band around the set value. The waste flows over the weir at the end of the jig while the product is collected in the hopper underneath the jig. High- and low-level probes start and stop the feeder to control the extraction of the product (Figures 2 and 3).

The pulse is created by air that enters and exits the air chambers situated underneath the screen deck. The air forces the water in the air chamber down, creating the pulse on the ore bed, and lets the air out to allow the ore bed to settle on the jig screen deck before the next pulse begins. The air is generated by a blower and stored in the working air vessel. Poppet valves control the air that enters and exits the air chambers. The air/water interface level in the air chambers is measured by level probes, which control the poppet valve timing to keep the stroke in the air chamber constant.

Plant ramp-up

The throughput of the jig plant, as in most plants, is based on three primary drivers: run hours, feed tempo, and yield. In order to achieve the plant's nameplate capacity of 13 Mt/a, each of these had to be addressed and understood as each affects the others. The plant ramp-up was planned over a period of one year after the last module was hot-commissioned, although reaching full capacity took three years due to a number of reasons. However, the yield expectation changed significantly, along with a change in feed and product quality, in the years that followed.

The initial ramp-up was aimed at understanding the basics of jigging control parameters and their effect on feed tempo and yield. During this period a lot of focus was placed on jig control to achieve the specified qualities of 64% Fe lump and 63.5% Fe fine product at the highest yield possible. It was soon realized that running a jig plant is not only a matter of correctly controlling the jig, but that each of the drivers would have to be addressed in order achieve nameplate capacity at the correct product quality.

After the basics and the control of the jig were understood, a lot of focus was placed on uptime/runtime as this is the basis for throughput. The major issues were components continually failing, control instruments not being fit for purpose, and long times required to repair and/or

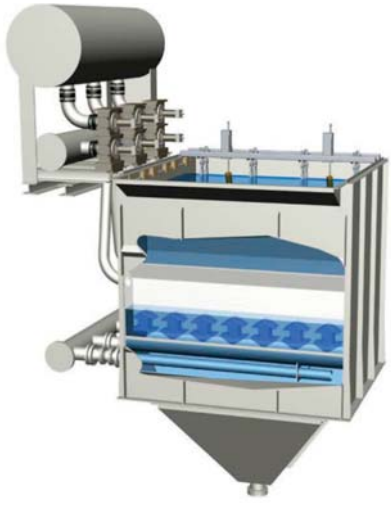

Figure 2—Jig layout with horizontal poppets

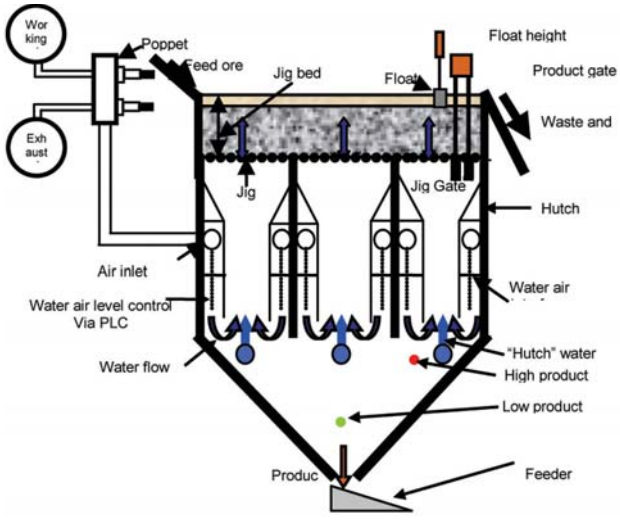

Figure 3—Schematic of jig construction

replace components. It also became apparent quite early that the wear of the jig components was much higher than expected, which also caused the loss of run-hours.

Through good understanding of the process and good collaboration between metallurgy, maintenance, and operations, various innovative in-house solutions were created. The same approach was followed for finding external solutions, in which the suppliers played an active part. This ensured a steady ramp-up in run-hours, as shown in Figure 4, and made a large contribution to the overall ramp-up in throughput. It should be noted that the 2012 run-hours were lower than 2011 due to the lack of feed available from mining as well as a planned hutch replacement, which took out three modules for a whole week at a time.

The correct feed tempo to the jig is important in order to achieve sufficient stratification. The impact of higher tempos needs to be understood as it influences the retention time of particles in the jig, allowing less stratification time.

The yield (mass recovery) is the single most important driver and also the most difficult to understand as there are many factors that affect quality, which in turn affects yield. A lot of dedicated effort was put into understanding these factors. They include feed quality and blend, mechanical condition of the jig components, as well as metallurgical issues affecting the separation efficiency, e.g. bed lift, near-density material, etc. However, understanding is only a part of the solution and effort went into collaborating with the mining as well as plant maintenance and operations teams to improve the overall yield.

The effect of this can be seen in the actual yield figures as shown in Figure 5:

- The 2008 yield, at 45.0%, was low as the plant was newly commissioned and little was known about the control philosophy of jigging that influence separation efficiency, with a consequent negative effect on the yield
- The 2009 yield, at 55.4%, was a major improvement, brought about through the increased understanding of the jig control parameters. A lot of focus was also placed on feed quality and plant metallurgical acceptability
- The 2010 yield, at 64.9%, was a further big improvement on the previous year's yield. This was the direct result of a metallurgical focus on the jig as well as high-grading (adding more A-grade material) in order to achieve the target chemical qualities at yield. There was also a shift in mining focus from considering not just feed quality but also beneficiation qualities
- The 2011 yield, at 62.3%, was a great achievement on the back of much lower feed qualities (1.6% Fe less than in 2010). The focus was on optimizing the use of A-grade material, which meant that less A-grade material was sent to the jig plant. However, the chief contribution to the good yield was the production of lower quality products (63.2% Fe lump and 62.2% Fe fines) in the second half of 2011
- The 2012 yield, at 58.6%, dropped significantly on the back of even lower feed qualities than the previous year (1.9% Fe less compared to 2011). However, the yield was very good considering the feed quality. Increasing focus was being placed on better efficiencies through improved monitoring and understanding through quantification, as well as improving separation efficiency factors. Predictions on product yield improved significantly, approaching the original estimated grade and yield relationship.

The most important learning outcome during the first three years of the plant was an understanding of the basic factors that affect jig separation efficiency. If these factors had been known and understood during ramp-up, nameplate capacity would have been reached sooner.

Factors affecting jig separation efficiency

Basic factors of jigging

Jigs have been used in mineral beneficiation for more than 100 years, mainly because of their simplicity and low cost.

The principle of operation is that the terminal velocity of a particle falling through water is determined by the drag of the water on the particle, creating an upward force, and the downward force of gravity. The drag is a very strong function of the surface area, which is in turn determined by two factors, namely the size of the particle and its shape. Terminal velocity is thus a function of three factors – particle size, shape, and density. A spherical particle has a higher terminal velocity because it experiences less drag than a flat particle with the same mass. Similarly, a lighter particle will have a smaller terminal velocity than a heavier particle of the same shape and size. Finally, small particles will have a lower terminal velocity than large particles of the same density and shape.

Separation in a jig takes place in a bed of particles about 250 to 500 mm thick, suspended on a screen deck and fully submerged in water. The aperture size of the screen is

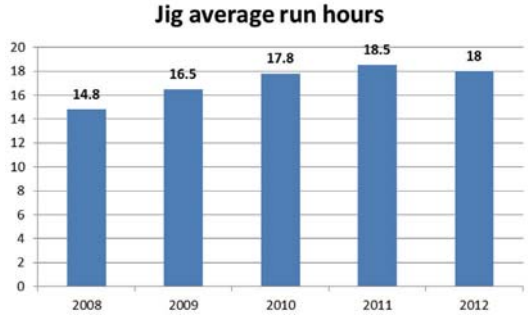

Figure 4—Improvement in run-hours

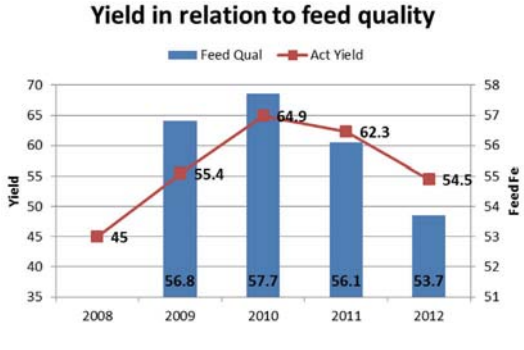

Figure 5—Relationship between yield and feed quality

smaller than the smallest particle. A water pulse is applied for a fraction of a second from underneath the screen through the bed of particles, at a velocity higher than the maximum terminal velocity of the particles. A continuous stream of hutch water with a continuous positive upwards flow improves the separation during the settling phase of the bed. All the particles are thus forced upwards, but those with a higher drag and lower mass will be displaced to a higher level under the upward water pulse. This upward force is then removed and the particles are allowed to settle. With continual repetition of the cycle, the low-drag high-density particles migrate to near the screen deck while the high-drag low-density particles migrate to the top of the bed. The product is removed through the product gate at the end of the screen and the lighter particles on top of the jig bed flow over the discard weir.

To summarize, the high-density particles, the large particles, and those with a spherical shape collect at the bottom of the bed, while the low-density particles, the small ones, and the flaky particles end up on top. This also exposes the weakness of the jig. The separation will be poor if the desired product is flaky and the discard more equidimensional and/or has a small density difference. The same applies if particles of all densities occur in the feed, in which case a layer is formed that contains particles of intermediate density, size, and shape, called the middling or near-density fraction. If there are too many near-density particles, the product layer tends to be thin, resulting in poor separation. A jig performs the best where proper liberation of the ore has taken place and where large differences in density occur, as in the case of metal separation from slag or coal from shale.

External factors that affect the separation efficiency of the jig

The external factors that have the biggest influence on jig efficiency are:

- Feed rate (retention time in jig)
- Chemical composition of feed material
- Percentage of near-density material in the feed
- Particle size and mass distribution into the jig
- Screen efficiency
- Particle size distribution (PSD) of feed material.

Pre-beneficiation beds were included before the beneficiation process to homogenize the feed to the jig plant, due to the following reasons:

- The varying chemical composition in the feed from the mine, which can also result in a varying percentage of near-density particles, depending on the type of ore mixture available in the pit at that time
- The varying PSD in the ROM feed, which will affect the split between the three jigs in a module, thus the feed rate (retention time)
- The ability to set up the jig operational parameters optionally for each jig, thereby optimizing product grade-yield interdependency
- Less selective mining affects the feed to the jig plant, as is also the case with feeding the DMS plant.

Because of the size of the open pit (12 km × 2.5 km), the feed to the plant is trucked from different loading points in the mine with ore types that differ mineralogically, resulting in variation in the chemical composition of the feed. Different kinds of ore also have different crushing characteristics, thus resulting in a variable PSD in the ROM feed.

Varying chemical compositions and amounts of near-density material in the feed to a jig would require continuous changing of the jig settings for optimum beneficiation efficiency. Since laboratory analysis is undertaken on composite product samples taken at two-hourly intervals, it would be difficult to maintain the optimum efficiency in the plant and some product would be misplaced due to the requirements of maintaining a constant chemical specification on the products beds.

Owing to the PSD variation in the feed, the feed rate to the different jigs varies continuously, with the possibility of exceeding the maximum feed rate to a certain jig and thus lowering the beneficiation efficiency of that specific jig.

With the pre-beneficiation bed capacity varying from a minimum of 80 000 t to 160 000 t per bed, the plant can run for two days or more with minimum changes to the jig settings, with slight adjustments to maintain chemical specification on the product beds, depending on the product bed chemical composition. This makes it possible to extract the optimum yield and product grade from the feed material if the pulse characteristics are optimized and maintained for each feed bed.

The shuttle conveyers used in the plant ensure that feed material at a constant PSD is evenly introduced into the jigs. Uneven distribution of feed over the width of the jig results in uneven stratification of particles and a large variance in particle retention time in the jig, causing extensive product and discard misplacement. It is also important that the PSD spread over the jig width is homogeneous to maximize the effectiveness of the separation process.

The negative impact of the percentage near-density material in the feed can be controlled to a certain degree by reducing the feed rate and/or increasing the weight on the floats. By increasing the weights the cut density is increased and more near-density material is removed to waste, decreasing the thickness of near-density particle layer underneath the float and thus reducing the misplacement of waste in product. The effect of varying feed chemical composition will have an effect on the product quality, and can be controlled by increasing or decreasing the height of the product bed that forms underneath the float, forcing more or less near-density particles to the product.

During the study phase of the jig plant project, different types of material were treated in a side-pulse test jig (3 m length x 0.620 m width). After the coarse fraction (-25 mm +8 mm) was successfully tested, the fine fraction (-8 mm +1 mm) was treated. After a few tests on the fine fraction the material was split into two fractions (-8 mm +3 mm and -3 mm +1 mm). Beneficiating the two fractions separately resulted in a 12% improvement in product yield compared to beneficiating the single fraction (-8 mm +1 mm). This is due to the fact that the finer -2 mm material is lost to the waste because of its particle size and the control parameters needed to successfully separate the top size 8 mm particles in the jig bed. This also indicated that the screening of the feed into the three size fractions to the different jigs must be within acceptable tolerances, otherwise the misplaced fine material to the wrong jig will result in the loss of product to waste.

Internal factors that affect the separation efficiency of the jig

The efficiency of the stratification of iron ore in jigs depends mainly on correct pulse characteristics, i.e. jig pulse shape, amplitude, and frequency. The mechanical condition of the various components used to create the pulse also has a direct influence. It is most important that the whole bed lifts up simultaneously, and no 'dead' areas must be present on the screen deck. No bed lift will occur in dead areas, thus no separation will take place and the forward movement of the rest of the ore bed will cause a downwards suction effect that will draw the waste in this area down to the bottom layer, undoing the stratification effort of the jig bed. After the material moves out of the dead area into the live area, separation will start again. If the dead area is near the product gate the mixed bottom layers will be drawn into the gate and a lower quality product will be obtained. The height of the lift should be at least three times the top size for coarse particles (> 75 mm lift for -25 mm +8 mm ore) and up to six times the top size for fine particles (> 18 mm lift for -3 mm +1 mm ore).

Provided that the pulse characteristics are correct, the beneficiation efficiency will be maximized. Some of the internal factors that can affect the separation efficiency by affecting/changing the pulse characteristics are:

- *Blocking of the punch plate at gate and screen deck*
  Any dead area in a jig bed, as described, will have an effect on separation efficiency. These problems were rectified where possible
- *Weight of floats*
  The weight of floats can vary by up to 500 g and needs to be calibrated. Weights added on the floats for bed chemical quality control should be only 100 g at a time, and float weights that differ by more than 50 g will make product chemical quality control difficult
- *Poppets*
  - Poppet timing (travel speed)
  - Poppet travel distance.

Any defect affecting the speed of the poppet and travel distance will affect the pulse control and thus the separation efficiency.

- *Poppet arrangement*—The first poppets were functioning horizontally, increasing the wear on the moving parts. The poppets were changed so that they functioned vertically, increasing the life of the moving parts from a few months to a few years
- Accurate calibration of all instruments is essential for consistent product quality and yield optimization on 24 jigs
- *Wave breakers inside air chambers*—With jigs up to 4 m wide, it is difficult to control the water-air interface in the air chambers and waves start to form, affecting the pulse and lift on the bed, which in turn adversely affect the separation efficiency. Wave breakers were installed in the air chambers to overcome this problem
- *Rubber seals on jig product gates*—Rubbers were installed on the gates of the fine and medium jigs to ensure that the product does not leak through continuously, thus improving control on the product bed and the extraction of product from the jig
- *Air chambers level sensors*—Changing the reading speed from 200 ms to 100 ms helped with the control of the bed lift. A faster sensor could improve the control even further.

If all the above-mentioned factors are 100% correct but the product gate design is wrong, a low-quality product will result. The product gate must be designed in such a way that the ore bed in the jig above the gate is still alive (moving upwards with the pulse) and will lift to the desired height, depending on the top size. The extraction of product via the product gate must be controlled in such a way that the height of the product bed (underneath the float) in the jig will stay within a narrow margin. If the product bed drops too low, the possibility of extracting waste and undesired near-density material increases, resulting in reduced product quality and product yield through compensating for quality.

Pulse timing and travel distance

A pulse cycle starts when the working air poppet receives a signal to open and allows air to enter the air chambers. The air will push the water/air interface in the air chamber downwards, carrying the energy over to the ore bed and lifting the jig bed with the desired stroke upwards. The cycle ends after the water/air interface in the air chamber has moved back to its original position, implying that the ore bed has settled back to its original position. This is just before the air enters the air chamber to start the next cycle.

Two poppets per chamber control the inlet and outlet of the air in the air chambers and the timing is controlled by a PLC. During the pulse cycle the working air poppet must open and close before the exhaust poppet opens and closes. The time to open and the distance the poppet must travel are thus important for controlling the pulse and lift (stroke) on the bed in order that proper segregation takes place.

The pulse frequency of the jigs is currently set at:

- Coarse Jig – 60 pulses per minute
- Medium Jig – 70 pulses per minute
- Fine Jig – 90 pulses per minute.

The time for each pulse cycle to be completed is as follow:

- Coarse Jig – 1000 ms
- Medium Jig – 857 ms
- Fine Jig – 667 ms.

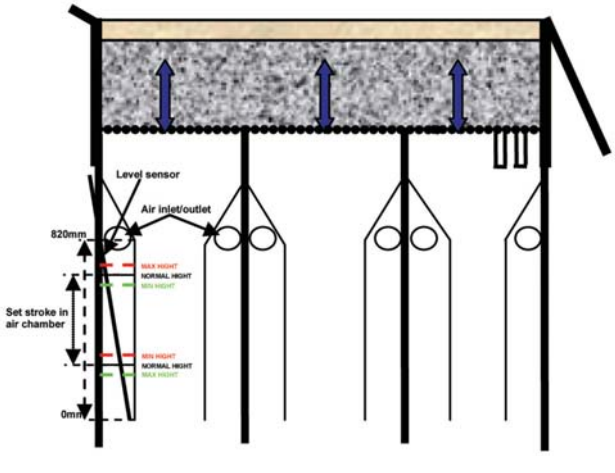

Figure 6—Air chamber detail showing water/air interface

The Fine Jig poppets have the shortest cycle period. The poppet construction and functioning for all three jigs, is the same, therefore it can be assumed that if the poppet travel speed and distance is correct for the fine, medium, and coarse jigs, then the control of the pulse forming and the lift (stroke) of the ore bed will be optimized.

The distance set for the poppets to travel for all the jigs is 55 mm and can be readjusted, if necessary. By keeping this value constant for all the jigs the rest of the settings can then be adjusted for optimum pulse control, keeping in mind that the thickness of the poppet rubbers sealing the working air from the jig's air chambers will start to decrease, increasing the poppet travel distance.

Figure 7 illustrates the poppet timing (black dotted line) and the reaction of the water/air interface (blue line) in the air chambers. It takes a new poppet 75 ms to travel the 55 mm set distance. At time T_T1W the working air receives a signal to open and will be fully opened at time T_T2W, well before the order to close reaches the poppet at time T_T3W. At time T_T4W the poppet is closed. At this stage the water/air interface reaches the set value after time T_P2 in the air chamber and the bed reaches the maximum lift required for optimum separation. The ore bed movement will differ from the blue graph, especially at the beginning of the pulse, because of the inertia of the ore bed at the beginning of the cycle. After the working air poppet has closed and before the exhaust poppet opens at time T_T1E the bed will be in suspension and will start to settle. During this period separation takes place as the heavier ore particles settle faster than the lighter waste particle.

At time T_T1E the exhaust poppet will receive the signal to open. It will be fully opened at time T_T2E, well in advance of the signal to close at time T_T3E, and it will be fully closed at time T_T4E. During this period the water/air interface in the air chamber will move back to its original set position, forced only by the water in the jig and the ore on the jig screen. After a few milliseconds the cycle will start again to create the next pulse.

Times T_T1W and T_T1E are fixed time settings and can be changed only on the PLC. Times T_T3W and T_T3E have three values, a normal, maximum, and minimum value. In the air chamber a certain stroke is needed to obtain the correct stroke on the ore bed in the jig. If the water/air interface has drifted off the normal of the two set values (height values) in the air chamber to either the maximum or minimum side, the PLC will advance or retard the poppet's timing by a few milliseconds to try and keep the two set values at the normal set value and thus maintain the stroke in the air chamber and the pulse characteristic on the ore bed in the jig.

Any mechanical defect in the poppets affecting the time/speed of the poppet that cannot be rectified by the PLC will have a negative effect on the pulse characteristics and will affect the jig separation efficiency (Figure 8).

Conclusion

Attaining the ramp-up targets of the Sishen jig plant as planned was no mean feat, and was achieved through a collaborative effort between metallurgy, maintenance, operations, and suppliers, and by not accepting any performance as the norm but adopting an approach of continuous improvement and teamwork. The product target of 13 Mt/a was achieved, assisted by a higher yield than planned and by the increase in working knowledge obtained during this period.

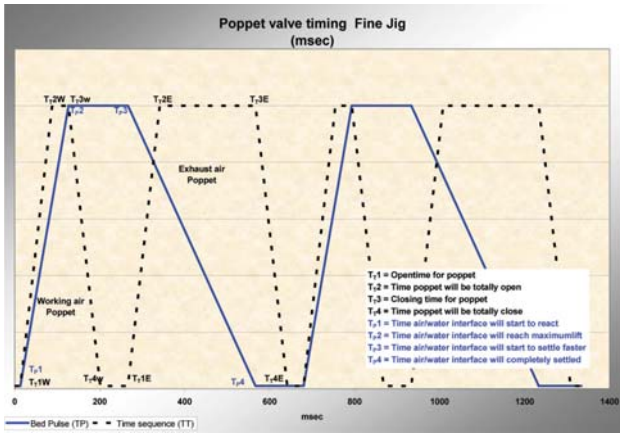

Figure 7—Effect of poppet valve timing on water/air interface reaction

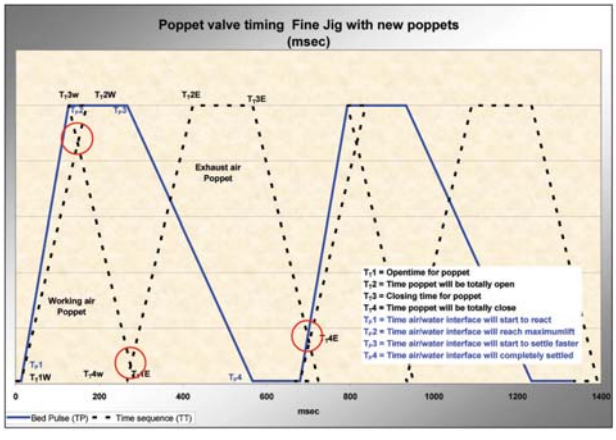

Figure 8—Effect of poppet valve timing on pulse separation efficiency

The most important basic factors to take into consideration and on which the efficiency of the stratification of iron ore in jigs depends are the correct pulse characteristics such as pulse shape, amplitude, and frequency. However, in order to control the pulse characteristics, the mechanical equipment must function correctly.

The stroke needed on the jig ore bed must be taken into consideration while setting the pulse. The pulse must be set so that the stroke on the ore bed is at least three times the top size of the ore feed to the jig for coarse material and up to six times for fine material. Most importantly, the whole bed must lift up simultaneously and no dead areas must be present on the screen deck.

By regularly monitoring the condition of the equipment responsible for the pulse characteristics, optimum separation efficiency will prevail.

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NORTJE, A. Internal report and data. Kumba Iron Ore. ◆

INTERNATIONAL ACTIVITIES

2014

15–16 July 2014 — Mine Planning School
Mine Design Lab, Chamber of Mines Building,
The University of the Witwatersrand
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4–5 August 2014 — Pyrometallurgical Modelling Principles and Practices
Emperors Palace Hotel Casino Convention Resort, Johannesburg
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6–8 August 2014 — MinPROC 2014
Lord Charles Hotel, Somerset West, Cape Town
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20–22 August 2014 — MineSafe Conference 2014 Technical Conference and Industry day
Emperors Palace, Hotel Casino Convention Resort, Johannesburg
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1–2 September 2014 — Drilling and Blasting
Swakopmund Hotel & Entertainment Centre,
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9–11 September 2014 — 3rd Mineral Project Valuation School
Mine Design Lab, Chamber of Mines Building,
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16–17 September 2014 — Surface Mining 2014
The Black Eagle Room, Nasrec Expo Centre
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20–24 October 2014 — 6th International Platinum Conference
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12 November 2014 — 12th Annual Southern African Student Colloquium
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18–19 November 2014 — Third International Engineering Materials and Metallurgy Conference and Exhibition (iMat 2014)
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2015

8–10 April 2015 — Sulphur and Sulphuric Acid 2015 Conference
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14–17 June 2015 — European Metallurgical Conference
Dusseldorf, Germany, Website: http://www.emc.gdmb.de

14–17 June 2015 — Lead Zinc Symposium 2015
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6–8 July 2015 — Copper Cobalt Africa Incorporating The 8th Southern African Base Metals Conference
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28 September-2 October 2015 — WorldGold Conference 2015
Misty Hills Country Hotel and Conference Centre,
Cradle of Humankind
Gauteng, South Africa
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12–14 October 2015 — Slope Stability 2015: International Symposium on slope stability in open pit mining and civil engineering
Cape Town Convention Centre, Cape Town
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8–13 November 2015 — MPES 2015: Twenty Third International Symposium on Mine Planning & Equipment Selection
Sandton Convention Centre, Johannesburg, South Africa
Contact: Raj Singhal, E-mail: singhal@shaw.ca or
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Company Affiliates

The following organizations have been admitted to the Institute as Company Affiliates

AECOM SA (Pty) Ltd
AEL Mining Services Limited
Air Liquide (Pty) Ltd
AMEC GRD SA
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Aurecon South Africa (Pty) Ltd
Aveng Mining Shafts and Underground
Aveng Moolmans (Pty) Ltd
Axis House (Pty) Ltd
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Barloworld Equipment -Mining
BASF Holdings SA (Pty) Ltd
Bateman Minerals and Metals (Pty) Ltd
BCL Limited (BCL001)
Becker Mining (Pty) Ltd
BedRock Mining Support Pty Ltd
Bell Equipment Limited
BHP Billiton Energy Coal SA Ltd
Blue Cube Systems (Pty) Ltd
Bluhm Burton Engineering Pty Ltd
Blyvooruitzicht Gold Mining Company Ltd
BSC Resources Ltd
CAE Mining (Pty) Limited
Caledonia Mining Corporation
CDM Group
CGG Services SA
Chamber of Mines
Concor Mining
Concor Technicrete
Council for Geoscience
CSIR Natural Resources and the Environment
Department of Water Affairs and Forestry
Deutsche Securities (Pty) Ltd
Digby Wells and Associates
Downer EDI Mining
DRA Mineral Projects (Pty) Ltd
Duraset
E+PC Engineering and Projects Company Ltd
Elbroc Mining Products (Pty) Ltd
eThekwini Municipality
Evraz Highveld Steel and Vanadium Limited
Exxaro Coal (Pty) Ltd
Exxaro Resources Limited
Fasken Martineau
FLSmidth Minerals (Pty) Ltd (FFE001)
Fluor Daniel SA (Pty) Ltd
Franki Africa (Pty) Ltd-JHB
Fraser Alexander Group
Goba (Pty) Ltd
Hall Core Drilling (Pty) Ltd
Hatch (Pty) Ltd
Herrenknecht AG
HPE Hydro Power Equipment (Pty) Ltd
Impala Platinum Holdings Limited
IMS Engineering (Pty) Ltd
JENNMAR South Africa
Joy Global Inc.(Africa)
Leco Africa (Pty) Limited
Longyear South Africa (Pty) Ltd
Lonmin Plc
Ludowici Africa (Pty) Ltd
Wekaba Engineering (Pty) Ltd
Magnetech (Pty) Ltd
MAGOTTEAUX (PTY) LTD
MBE Minerals SA Pty Ltd
MCC Contracts (Pty) Ltd
MDM Technical Africa (Pty) Ltd
Metalock Industrial Services Africa (Pty)Ltd
Metorex Limited
Metso Minerals (South Africa) (Pty) Ltd
Minerals Operations Executive (Pty) Ltd
MineRP
Mintek
Modular Mining Systems Africa (Pty) Ltd
MSA Group (Pty) Ltd
Multotec (Pty) Ltd
Murray and Roberts Cementation
Nalco Africa (Pty) Ltd
Namakwa Sands (Pty) Ltd
New Concept Mining (Pty) Limited
Northam Platinum Ltd - Zondereinde
Osborn Engineered Products SA (Pty) Ltd
Outotec (RSA) (Proprietary) Limited
PANalytical (Pty) Ltd
Paterson and Cooke Consulting Engineers
Paul Wurth International SA
Polysius A Division Of Thyssenkrupp Engineering
Precious Metals Refiners
Rand Refinery Limited
Redpath Mining South Africa (Pty) Ltd
Rosond (Pty) Ltd
Royal Bafokeng Platinum
Roymec Technologies (Pty) Ltd
RSV Misym Engineering Service (Pty) Ltd
RungePincockMinarco Limited
Rustenburg Platinum Mines Limited
SAIEG
Salene Mining (Pty) Ltd
Sandvik Mining and Construction Delmas (Pty) Ltd
Sandvik Mining and Construction RSA(Pty) Ltd
SANIRE
Sasol Mining (Pty) Ltd
Scanmin Africa (Pty) Ltd
Sebilo Resources (Pty) Ltd
SENET (Pty) Ltd
Senmin International (Pty) Ltd
Shaft Sinkers (Pty) Limited
Sibanye Gold Limited
Smec SA
SMS Siemag
SNC Lavalin (Pty) Ltd
Sound Mining Solution (Pty) Ltd
SRK Consulting SA (Pty) Ltd
Time Mining and Processing (Pty) Ltd
Tomra Sorting Solutions Mining (Pty) Ltd
TWP Projects (Pty) Ltd
Ukwazi Mining Solutions (Pty) Ltd
Umgeni Water
VBKOM Consulting Engineers
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Weir Minerals Africa (Pty) Ltd
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Forthcoming SAIMM events...

EXHIBITS/SPONSORSHIP
Companies wishing to sponsor and/or exhibit at any of these events should contact the conference co-ordinator as soon as possible

For the past 120 years, the Southern African Institute of Mining and Metallurgy, has promoted technical excellence in the minerals industry. We strive to continuously stay at the cutting edge of new developments in the mining and metallurgy industry. The SAIMM acts as the corporate voice for the mining and metallurgy industry in the South African economy. We actively encourage contact and networking between members and the strengthening of ties. The SAIMM offers a variety of conferences that are designed to bring you technical knowledge and information of interest for the good of the industry. Here is a glimpse of the events we have lined up for 2014. Visit our website for more information.

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SAIMM DIARY

2014

- **SCHOOL**
  Mine Planning School
  15–16 July 2014, Mine Design Lab, Chamber of Mines Building, The University of the Witwatersrand
- **CONFERENCE**
  Pyrometallurgical Modelling Principles and Practices
  4–5 August 2014, Emperors Palace, Hotel Casino Convention Resort, Johannesburg
- **CONFERENCE**
  MinProc 2014
  6–8 August 2014, Lord Charles Hotel, Somerset West, Cape Town
- **CONFERENCE**
  MineSafe Conference 2014
  Technical Conference and Industry day
  20–21 August 2014, Conference
  22 August 2014, Industry day
  Emperors Palace, Hotel Casino Convention Resort, Johannesburg
- **SCHOOL**
  Drilling and Blasting
  1–2 September 2014, Swakopmund Hotel & Entertainment Centre, Swakopmund, Namibia
- **SCHOOL**
  3rd Mineral Project Valuation School
  9–11 September 2014, Mine Design Lab, Chamber of Mines Building, The University of the Witwatersrand
- **CONFERENCE**
  Surface Mining 2014
  16–17 September 2014, The Black Eagle Room, Nasrec Expo Centre
- **CONFERENCE**
  6th International Platinum Conference
  20–24 October 2014, Sun City, South Africa
- **COLLOQUIUM**
  12th Annual Southern African Student Colloquium
  12 November 2014, Mintek, Randburg
- **CONFERENCE**
  Accessing Africa's Mineral Wealth: Mining Transport Logistics
  19–20 November 2014, Emperors Palace, Hotel Casino Convention Resort, Johannesburg

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