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SAIMM

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VOLUME 118 NO. 8 AUGUST 2018

Let's get Clean

MIP Process Technologies (Pty) Ltd, a Sandton, Johannesburg, based process equipment supplier, has made huge in-roads in the South African gas cleaning market.

They do not only specialise in the supply of solid-liquid dewatering equipment, which includes, thickeners, flocculant plant, linear screens, slurry samplers, mixers and agitators, but have a range of gas cleaning equipment. This incorporates the supply of bag filters, scrubbers, stacks and associated equipment to provide a clean gas solution.

"We specialised in the more difficult gas cleaning operations", says Philip Hoff, Managing Director. "The more noxious the gasses to clean, the better we perform", he reiterates.

This philosophy has been proven in that they have supplied numerous Sulphur Dioxide (SO_2) cleaning plants for projects in the Democratic Republic of Congo and South Africa

They are currently executing a project for the supply of a gas cleaning plant which is one of the largest of its type in the world. The first section of the project deals with dust extraction and the second part the removal of SO_2.

MIP Process is arguably the only company that can provide all their own products when designing a plant such as the removal of Sulphur dioxide. Not only does a customer deal with one supplier, they will acquire superior equipment that has been proven.

Another niche gas extraction project is the removal of toxins that are generated during tap hole extraction.

This practise is common for smelters and MIP Process designs the hoods that capture and remove gasses which can be harmful to employees.

Although MIP's gas extraction business is doing well, the company's process equipment range continues to break new ground with a healthy growth programme.

MIP Process has added another product to enhance their highly dependable range, which will enhance its offering. Their new Peristaltic (hose) pumps are sourced from Europe and the company is the exclusive representative. "This product fits in with our supply of high quality products", says Hoff.

The pumps cover the full range of flow rates and applications, with hose designs suitable for food and beverage to mining applications.

MIP Process' philosophy and reputation is to deliver the highest quality equipment with the fastest turnaround time possible.

This service coupled with MIP's range of products, ensures customers are assured of a competitive offering.

Various projects have multiple thickeners and MIP Process can supply from as small as 1m diameter to 100m diameter.

Bag filters are generally the most cost effective and best solution for removal of dust particles.

Peristaltic pumps have a wide range of applications from polymer dosing to thickener underflows.

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SOCIETY OF MINING PROFESSORS 6TH REGIONAL CONFERENCE

PAPERS OF GENERAL INTEREST

SOMP Regional Conference

This edition of the *Journal* is dedicated to the Society of Mining Professors 6th Regional Conference 2018, which was held in Johannesburg, South Africa from 12 to 14 March. This Conference was hosted by Mining Engineering Education South Africa (MEESA) and the Southern African Institute of Mining and Metallurgy (SAIMM). MEESA is comprised of the School of Mining Engineering at the University of the Witwatersrand, the Department of Mining Engineering at the University of Pretoria, the Department of Mining Engineering at the University of Johannesburg, and the Department of Mining Engineering at the University of South Africa.

The theme of this regional conference was 'Overcoming Challenges in the Mining Industry through Sustainable Mining Practices', and the proceedings included papers on topics from research and development, sustainable development, and innovations in learning and teaching.

We hosted 46 delegates from 12 countries from the continents of Asia, Africa, Australia, and Europe. A total of 32 papers were presented, all of which were independently peer-reviewed by experts in the respective specialist fields in order to ensure the highest relevance and quality.

The conference had three keynote speakers – Mr Lucky Kgatle (Senior Vice President, Sasol Mining Pty Ltd), Ms Deshnee Naidoo (CEO, Vedanta Zinc International), and Professor Cuthbert Musingwini (Head of the School of Mining Engineering, University of the Witwatersrand), who between them covered the various aspects of the conference theme.

The conference also held a panel discussion on 'Mining Education of the Future'. The panelists were Professor Bruce Hebblewhite, Professor of Mining Engineering, School of Mining Engineering, UNSW Sydney and Secretary General for SOMP, Mr Gary Lane, Director, Vuuma Collaborations, and Dr Gordon Smith, Executive Head, Technical, Anglo American Platinum Ltd. This discussion was very well received by the delegates and formed the foundation for a similar discussion at the 29th SOMP Annual Meeting in Beijing, that was held from 3–6 July 2018 .

The conference was concluded with a technical visit to the South Deep mine operated by Gold Fields.

R. Mitra

President's Corner

Seeing the value of the SAIMM

Being the President of a professional organization like the SAIMM can be quite demanding. However, it's not always stressful because there are quite a few pleasures and privileges that accompany the job. One of the significant pleasures is the interaction with members at all the numerous events that are organized by the Institute. Here you get to talk and listen to the views of the members on a number of issues facing the minerals industry. You get to hear all the good, the not so good, and the bad about the industry that we serve. You are always kept up to date, simply through that interaction with other professionals in the field. This is of significant value, especially considering how quickly technology is changing the minerals industry landscape.

During my interaction with both members and non-members of the Institute, I have had to appraise them of the value of maintaining their SAIMM membership or becoming a member. The SAIMM is a well-recognized and valued professional organization that serves the needs and interest of its members, and by extension the mining profession. The objectives of the Institute are to disseminate scientific and technical knowledge to the benefit of the minerals industry and to promote continuing professional education across Southern Africa through a number of activities. The SAIMM brand is well established, well trusted, and widely recognized for organizing first-class events. Events such as conferences, schools, workshops, and colloquia are the foundation through which independent knowledge-sharing platforms are facilitated by the Institute to discuss topical issues and other developments in the minerals industry. These events provide industry professionals with an opportunity to learn about technological and (sometimes) societal issues in the minerals sector and to further network with local and international professionals in the field. Such opportunities are priceless when it comes to career development, as well as maintaining and enriching your engineering knowledge.

As part of fulfilling its objective on technical and scientific dissemination, the SAIMM is also part of OneMine.org (www.onemine.org), where members have free access to over 100 000 papers from around the world *via* this database. OneMine.org is coordinated by SME (www.smenet.org) in the USA. Papers published through the monthly SAIMM journal are also hosted on the database, thereby providing a wider distribution for authors and thus a wider network of readership and citation. The SAIMM also provides its members with free access to some services of the technical library that combines the previous holdings of the Chamber of Mines (now Minerals Council South Africa) and the Anglo American libraries. The technical library offers free access to the catalogue and a monthly bulletin. Up to 10 free articles per month can be sent to a member via e-mail.

The Institute is also part of the Global Mineral Professional Alliance (GMPA). The GMPA is a collaboration between leading professional organizations for the minerals industry around the world. It comprises the Australasian Institute of Mining and Metallurgy (AusIMM), the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), the Institute of Materials, Mining and Metallurgy (IOM^3), the Southern African Institute of Mining and Metallurgy (SAIMM), the Society for Mining, Metallurgy and Exploration Inc. (SME), and the Instituto de Ingenieros de Minas del Peru (IIMP). The participation of SAIMM in the GMPA ensures that members are able to enjoy reciprocal benefits from the other participating institutes and societies. The participation also enables SAIMM to obtain an understanding of how sister organizations in the minerals industry are operating, which helps benchmark the Institute's performance and activities for the benefit of its members.

President's Corner *(continued)*

The SAIMM is 124 years old, and for the organization to see another century, young members should be central to its activities. These young professionals are the future of the minerals industry and the Institute. The Young Professionals Council (YPC) is an initiative that was established by the SAIMM to serve the interests of its young members (35 years and younger). I was fortunate enough to witness the initial establishment of the YPC in 2013. The growth and development of the YPC since its inception has continued to surpass expectations. Some of the successful initiatives run under the YPC banner include the establishment of an e-magazine, known as 'Youth in Mining and Metallurgy', the mentorship programme, and the graduate employment database. The mentorship programme helps to link young members who have an interest in being mentored with mentors that are drawn from the older members of the SAIMM. The employment database is a portal that aims to link unemployed young members with potential employers. All these activities are making an impact on the lives and careers of the students and young professional members of the Institute. Recently, one official from the DST asked me why our Institute is not blowing its trumpet loud when it comes to the successes of the YPC. This demonstrates the positive impact the YPC has had in the industry in such a short space of time.

Another value for young members that has been created by the SAIMM is the Scholarship Trust Fund. The fund was established to provide assistance to a greater number of needy students in the mining and metallurgical fields, who depend on personal or family funds for their educational needs. The SAIMM gives an annual donation to universities who then disperse the funds accordingly. This is a testament to the value that the SAIMM places on the future leaders of the industry.

In the presidential article in last month's SAIMM *Journal*, I wrote about one new SAIMM initiative, the committee for Diversity and Inclusion in the Minerals Industry (DIMI). Recently there has been an outcry over an opinion piece that was featured in the South African Institute of Civil Engineering (SAICE) magazine, written by the CEO of SAICE, Manglin Pillay. The article, ironically published during the month when women are celebrated, clearly diminishes the value of women in the engineering fields. The SAIMM does not share such views but looks to promote diversity (in all its forms) and inclusion in the minerals industry. This includes advancement and promotion of the needs and interests of women who are making their career in the industry. Through the DIMI committee, the SAIMM seeks to bring awareness to the industry on the importance of valuing the contributions brought to it by all diverse individuals.

As I write my last article as the President of the Institute, I wish to say that the value of the SAIMM as a home for professionals in the minerals industry cannot be underrated. It has been both an honor and a privilege to lead such a dynamic and vibrant organization this past year. I have grown as an individual and I have learned a lot from the members of the Institute. I want to sign off by saluting the members of the Institute who selflessly participate at all levels, and dedicate enormous effort and time to ensure that the values and missions of the Institute are kept alive. I have no doubt that we will leave a legacy to our children, who together with their descendants, will still see another century of the Institute. So, as your outgoing President, I say thank you, *dankie*, ngiyabonga, *enkosi, kea leboha, ndo livhuwa*.

S. Ndlovu
President, SAIMM

http://dx.doi.org/10.17159/2411-9717/2018/v118n8a1

The costs of sampling errors and bias to the mining industry

by R.C.A. Minnitt

Synopsis

South Africa's mineral commodities generate approximately R420 billion per annum from export earnings. Of that amount coal (28.1%), gold (15.2%), iron ore (14.5%), and platinum (21.7%) account for 80%, and together with chrome and manganese account for 88% of the earnings. Payment for these products is based on the metal content, and in the case of coal, the energy content. Traders rely on the analytical results from samples of the products to obtain a fair price and true value of the sale. This paper covers three main issues. Firstly, the thrust of interest in sampling of particulate materials is shown to be primarily due to the financial implications of poor sampling and the vibrant trade in these mineral and metal products in the USA between the 1850s and 1940s. The importance of correct engineering for cutter operation and good maintenance of cutters in general in the sampling of bulk commodities is emphasised. Secondly, simulation of a low-grade iron ore deposit demonstrates that the principal offending factor in sampling events is the sampling bias, rather than the sampling error. Whereas sampling error may account for as little as 0.0016% error in the mean grade, sampling bias, which can be positive or negative, may affect the mean grade by as much as 10%. Thirdly, the contribution of individual particles of iron ore, particularly those in the larger fractions of the size distribution, is investigated. Relatively small changes in mean grade of about 0.106 %Fe can result in losses to the supplier of about US$11 600 per 100 000 t shipment of iron ore, a substantial amount of nearly seven million dollars per annum. Together the three aspects, principles of correct cutter operation, the effects of bias on the mean grade of samples, and the effect of size distribution on sample extraction error, contribute to potential financial losses in the bulk commodities trade.

Keywords

sampling, error, bias, uncertainty.

Introduction

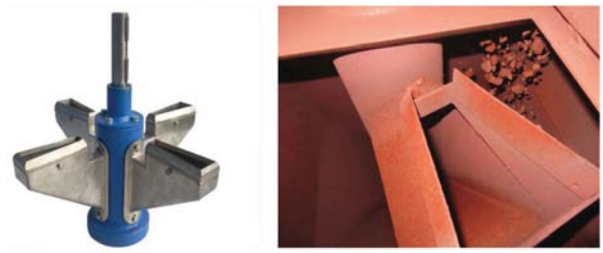
The theory and practice of particulate sampling grew out of the need for accurate assays of the grades of ores, concentrates, and coals that were traded in the mining areas of the USA and UK in the period between 1850 and 1930. The origins of sampling theory and practice are deeply embedded in the financial implications associated with sampling and follow-up trading of ores and concentrates. In fact, so prolific were the authors and writers about the issues involving the sampling of ores and concentrates that Sharwood and von Bernewitz (1922) from the US Bureau of Mines compiled a list of 906 pieces of literature on sampling up to July 1921. Some of the more famous sampling and blending authors of the time were Reed (1882), Brunton (1895), Hofman (1899), Warwick (1903), Rickard (1907), Richards (1908), Argall (1912), Woodbridge (1916), Pulsifer (1920), and Taggart (1927). Records indicate that the principal exponent of practical sampling was Henry Vezin, who apart from newspaper articles, wrote very little himself, but as early as 1850 had designed and published diagrams of his rotary sampler (Rawle, 2017). Vezin's outstanding design implies that he understood, even at this early stage, the principles of probabilistic sampling.. Although he never published it, Vezin's outstanding splitter design, examples of which are shown in Figure 1a and 1b, implies he understood, even at this early stage, the principles of probabilistic sampling, namely 'each and every fragment must have the same statistical opportunity as every other fragment of being in the sample'.

Table 2 in Volume 2 of Richards' 'Ore Dressing' (Richards, 1907, p. 850), extract shown below, provides a minimum mass calculation that he attributes to Vezin (1866), but which may have originated with Hofman (1899), a professor from Massachusetts Institute of Technology (MIT) (Rawle, 2017).

According to Rawle (2017), the superscript 14 next to Vezin's name refers to Hofman (1899), who in fact did not mention Vezin's name in the text.

The table quoted by Richards (1908) and attributed to Vezin provides data for sample mass and fragment size, which plots as a

* *School of Mining Engineering, University of the Witwatersrand, South Africa.*



a b

Figure 1—Correctly designed rotary Vezin splitters with radial lips for cutting vertical falling streams of (a) liquids, slurries, and fine-grained materials (Pitard, 2017), and (b) iron ore, just before interacting with the falling fragments (Holmes, 2015)

The simplest rule, adopted by Vezin[14] in 1866, is: first, to decide what weight (w) should be taken for assay or analysis after the ore has been ground to 100-mesh (approximately 0.125 mm. diameter); second, to compute the number (n) of maximum sized grains passing through a 100-mesh screen that would weigh (w); and third, to cut down to a weight after each crushing which will be equal to n of the maximum sized particles.

This rule may be said to use a constant number of particles whatever their size. The following figures show the weights of different sizes required by this rule on the basis of 0.1 assay ton (2.917 grams) of ore through a 100-mesh screen (0.125 mm.):

128	mm.	3,131	metric	tons.
64	"	391	"	"
32	"	48.9	"	"
16	"	6.12	"	"
8	"	764.6	kilos.	
4	"	95.57	"	
2	"	11.95	"	
1	"	1.493	"	
0.5	"	186.7	grams.	
0.25	"	23.33	"	
0.125	"	2.917	"	

The above rule demands finer crushing than practice indicates to be necessary, and it is, therefore, more expensive than is wise.

power curve with the equation as shown in Figure 2a, or alternatively, as a straight line on a log-log graph paper with an equation as shown in Figure 2b.

These data indicates that at an early stage (*ca.* 1866) Vezin had shown that the sample mass and the cube of the fragment size were related. Richards (1908) suggested the masses generated by Vezin's formula were too large, and arbitrarily changed the factor for fragment size in the equation in Figure 1a to d_N^2, although he provided no scientific grounds for doing so, a change that some have suggested set the science of sampling back 50 years. It was not until the 1950s that Pierre Gy set the record straight, restoring the factor to d_N^3. Unfortunately, Vezin never published much of his research and it was left to Warwick (1903) to pull his work together in an excellent volume, entitled '*Notes on Sampling*', after Vezin's death.

The following extract from Chapter 5, entitled '*Receiving, sampling, and purchasing*' on page 45 of Hofman's 1899 publication, refers specifically to the trade in gold- and silver-bearing ores (Rawle, 2017).

Reed* arrived by calculation at certain definite maximum sizes which ores of a varying tenor in silver and gold may have when they are reduced from a 100-ton lot down to the assay sample. His tabulated results, slightly condensed, are subjoined:

Quantity of Ore. Reducing	Value of Silver in Ounces Per Ton.			Size of Ore.
	Highest: 300. Average: 50.	Highest: 3000. Average: 75.	Highest: 10,000 Average: 500.	
100 tons to 10 tons	Cocoanut	Fist	Fist	Maximum permissible size of ore for given grade.
10 tons to 1 ton	Orange	Egg	Walnut	
1 ton to 200 pounds	Walnut	Chestnut	Chestnut	
200 pounds to 5 pounds	Pea	Wheat	Wheat	
5 pounds to bottle-sample	20-mesh	35-mesh	50-mesh	

An ore must run very much lower than 50 oz. silver per ton, if the seller will be satisfied with cocoanut-size in reducing 100

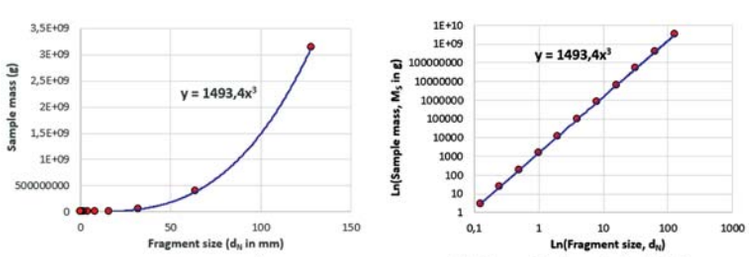
* *School of Mines Quarterly*, vi., p. 357.

This extract from Hofman's document clearly illustrates that early authors understood that accurate assay values depend on a relationship between the size of ore fragments and the mass of the sample. Vezin's donation of his notes, reports, and calculations recorded in the *Mining Reporter*, Denver, Colorado of October 1901, was written up in eighteen parts and reported in the local newspaper by Arthur Warwick (Rawle, 2017). A further extract from '*Notes on Sampling*', Warwick's (1903) compilation of Vezin's work, illustrates the scale of mining- and mineral-related financial transactions taking place in Colorado at the end of the 19th century.

them. The assays were made by two different assayers, each most reliable. The following are a few examples from the report referred to:

Example.	Assayer 1, ozs. gold	Assayer 2, ozs. gold.	Differences per ton.
1	10.08	5.36	$ 94.40
2	0.32	21.94	432.40
3	5.68	1.04	92.80
4	2.40	0.21	43.80
5	3.12	10.00	137.60
6	79.04	11.46	1,351.60

These are but a few of the extraordinary results obtained from two halves of the same pulp. This, be it remembered, of ore which it is customary to expect buyer and seller to agree within one dollar per ton. With greater differences the aid of an umpire is called in. Correct sampling is of more importance to-day than ever before, owing to the fact that miners are selling practically the whole of the output of their mines instead of milling or smelting their own ores. Thus in Colorado, with a mineral output aggregating more than $50,000,000 per annum, an error of two per cent. in the settlement would entail a loss of $1,000,000 to some one.



$M_S = 1493.4 \times d_N^3$ $\quad$ $lnM_S = 3lnd_N + ln1493.4$

a b

Figure 2—Vezin's data for sample mass (g) *versus* fragment size (mm) plotted (a) as a power curve and (b) as a straight line on a log-log plot. Equations for the data are shown below the curves

This also shows that the extensive research into sampling issues at the turn of the 19th century was motivated by the need to understand the scale of the losses that one or other partner would potentially face because of poor sampling practice. What these authors did not specify is the source of the errors. This paper aims to examine the effects of sampling errors and sampling bias on assays of traded commodities, and consequently the potentially incorrect payments to producers.

From a detailed historical investigation of the origins of sampling, Rawle (2017) concluded that sampling theory arose from two papers published by Reed (1882, 1885) in the *Columbia School of Mines Quarterly*. Reed's 1882 publication, '*Ore sampling*' is a key document in which he explained that the minimum sample mass is proportional to the cube of the nominal fragment top-size. He apparently quotes David Brunton on this matter, but without reference to his publications. Although there is no record of published work earlier than 1895, Brunton (1895) later wrote about the theory and practical aspects of sampling. Brunton made the astute observation that '... *to be equally reliable two samples of a same material should be made of the same number of fragments, irrespective of the top particle size*.'

Perhaps the first person to expound an equation for the variances of what is now known as the Fundamental Sampling Error was Robert Richards (1908), who published the '*Ore Dressing*' volumes, which are regarded as the *de facto* mining publications. Vezin, Brunton, and Richards all knew one another and met on a number of occasions (Rawle, 2017). Vezin outlined a calculation very early on (maybe even in the 1860s) based on a simple statistical analysis of a minimum of 10 000 particles needed in the 99th percentile of the size distribution. In the 1880s, Reed published a similar analysis that was used by Hofman (1901), and subsequently Richards (1908).

Nature of sampling errors

The historical background tells firstly of the huge body of research that had taken place up to the early 1950s, when Pierre Gy dedicated himself to a study of the sampling of particulate materials, and secondly, that this research was driven by financial concerns. Accordingly, Gy (2004) honed the issues around sampling into what is called the Theory of Sampling (TOS) from the 1950s onward, and particularly in the 1970s when his work started to appear in English.

Gy (1953, 1967, 1976, 1979) was among the first researchers to name the different types of sampling errors and to identify the locations at which these errors were generated in the mining value chain. Gy's earliest taxonomy of sampling errors was simple, the source and nature of the sampling errors essential to an understanding of the financial implications of trusting a sample value being shown in Table I. The sources of sampling errors are categorized into four groups that deal firstly with the nature of the materials being sampled; secondly, the sampling equipment and materials handling; thirdly, the processes and procedures in the plant; and fourthly, with the analytical processes. This shows that sampling errors of the same or different types may occur at a number of different localities along the mining value chain. While some sampling errors arise predominantly in the early stages of the mining value chain, the same errors may re-occur later on at a different locality if the source and nature of such errors is not identified. For example, the Delimitation Error and Extraction Errors may occur during exploration, mine development, during grade control, and in the analytical laboratory (Pitard, 2006). Very often the sampling errors at different locations along the mining value chain vary in scale, rather than by type. Consequently, Delimitation and Extraction Errors at the exploration or mine development stages involve hundreds of tons of material, but these errors can also occur in the analytical laboratory where only a few tens of grams of material are being handled. Only the sampling errors associated with the materials themselves and those generated by materials handling are considered in this paper.

In a statistical sense, a sample is usually a small representative proportion of the lot, the total population. The information content of a single sample is small in that it provides a best, unbiased estimate of the mean value of the lot. In fact, the sample extraction process may have resulted in a biased sample, but this would never be known. Only when two or more, preferably many more, samples are assayed may it be possible to obtain a better estimate of the mean and begin to calculate the variability associated with these samples. Samples should bear the principal characteristic that they are representative of the lot from which they are drawn, and if representative, they are by definition unbiased. Furthermore, only correct samples, obeying the principle that every fragment in the lot should have an equal opportunity of being included in the sample,

Table I

The source and nature of the principal sampling errors identified in the Theory of Sampling

Sources of error	Name of error	Nature of error
Material characterization	*In situ* Nugget Effect (INE)	True error
	Fundamental Sampling Error (*FSE*)	True error
	Grouping and Segregation Error (GSE)	Bias
Sampling equipment and materials handling	Delineation Error (DE)	Bias
	Extraction Error (EE)	Bias
	Preparation Error (PE)	Bias
	Weighing Error (WE)	Bias
Plant process and procedures	Continuous Selection Error (CSE)	Error and bias
Analytical processes	Analytical Error (AE)	Error and bias

can be unbiased. Only samples that are collected according to an optimized sampling protocol will be representative. It is important not lose sight of the fact that samples are a first step in an investigation into the mean value for one or other characteristic of the population, the lot. Consideration of the spatial and directional reach of a sample value is important to sampling, but it is fundamental to the discipline of geostatistics. Sampling a stockpile by taking replicate samples over the surface of the pile will provide a more representative estimate if the footprint of the area within which samples are selected is bigger. Sampling a bigger area means the reach of the samples, the spatial influence of each sample, is extended. Samples extracted from flowing material streams, such as crushed ore on a conveyor belt, or leachate flowing in a launder of a hydrometallurgical process, are linearly separated in time and space. The space between samples and the size of the sample collected are crucial to obtaining an accurate estimate of the mean of the parent lot, the behaviour of the input material, and the ability of the plant to produce a product to customer specifications.

Sampling errors

Sampling errors arise because of both the constitutional and distributional heterogeneity of materials, broadly referred to as the materialization errors (Gy, 1967, 1979; Pitard, 1993). Three types of sampling errors arising from this heterogeneity have been defined, namely the *In situ* Nugget Effect (INE), the Fundamental Sampling Error (FSE), and the Grouping and Segregation Error (GSE).

In situ Nugget Effect

Pitard (2006) is the main proponent of the *In situ* Nugget Effect, the error arising from the presence or absence of gold nuggets in a piece of borehole core from a mineralized zone. Depending on the nature of the mineralization, nuggets may or may not be present. Solid gold nuggets may be up to several millimetres in diameter (in some cases much bigger). In some cases fine gold grains, about 50–70 μm in diameter, aggregate together to form nuggets up to 1000 μm in diameter. Such nuggets produce extreme variability in gold assay values from one drill-hole to the next. The INE can be reduced by increasing the diameter of the drill core.

Fundamental Sampling Error

The formula devised by Gy as early as 1952–1953 is given in Equation [1]. It relates the FSE to an ore-type-specific coefficient K, the diameter of the fragment size raised to the third power (d_N^3), and inversely to the mass of the sample (M_S).

$$\sigma_{FSE}^2 = \frac{K d_N^3}{M_S} \quad [1]$$

While it is clear that the FSE is inversely proportional to the mass (M_S) of the sample, and hence can be controlled by the mass of the sample or the number of samples, as indicated in Equation [2], it also depends on how the mass is constituted.

$$\sigma_{FSE}^2 \propto \frac{1}{M_S} \text{ or } \sigma_{FSE}^2 \propto \frac{1}{n} \quad [2]$$

M_S may represent a single, large sample, or it could represent a large number of small increments, n, composited to make up the sample. The FSE is an error variance in units of the analyte of interest.

The FSE arises from within-fragment and between-fragment variations in grain size, particle size, mineralogical composition or mineral constituents, density of the fragments, amount of gangue, and metal content from one fragment to the next. No matter how many samples are assayed, the results are always different from one another because of constitutional heterogeneity and the fact that only a portion of, and not the whole of, the population is sampled.

The FSE can be reduced to a minimum, but can never be eliminated. Oddly enough, it can also be calculated before the sample is taken, provided heterogeneity tests and essential material characterization have been carried out. Percentage precision, without any reference to units, is the coefficient of variation (standard deviation divided by the mean, multiplied by 100) used to compare one set of analyses with another. The parameters in the formula devised by Gy (1953, 1967) for the FSE indicate that larger sample masses, and in particular a reduction of fragment size, will improve the precision of the assays. The only way to ensure that the FSE is minimized is to ensure that the primary sample is collected according to the strict principles of the Theory of Sampling.

Grouping and Segregation Error

The Grouping and Segregation Error, another material-related source of error, arises from the tendency for grains or fragments of ore containing very dense target analytes to group under gravity and consequently segregate from the bulk of the ore. The behaviour of very dense fragments under gravity is much like that of a shoal of fish, which although not physically connected, segregates and moves under natural instinct as a group in water. A net thrown into a shoal will produce a catch – a strongly biased, non-representative distribution of fish in the ocean, as will a net that catches no fish. Sampling of strongly segregated particulate materials is typically clustered sampling in which groups of fragments (clusters) are extracted at random. The size of the clusters is critically important when sampling for genetically modified organisms (GMOs) in shipments of grain. If the clusters differ significantly in composition with respect to the target analyte due to the Grouping and Segregation Error, this can result in non-representative sampling. This so-called Grouping and Segregation Error (GSE) can be reduced during sampling processes, firstly by homogenizing the lot, and secondly by collecting and compositing numerous small increments rather than taking a single large sample. The principle of compositing numerous increments was well understood as early as 1882 by Reed (1882), who designed a shovel that could extract small increments from a lot.

While there are equations that relate the Grouping and Segregation Error to segregation and grouping coefficients, it is not possible to calculate the error, because the size of the error is transient and changes from one moment to another. It is, however, possible to identify, measure, and mitigate its overall contribution to particulate segregation and the distributional heterogeneity.

Sampling bias

Whereas sampling errors, namely the INE, FSE, and GSE, are due to variability in material composition or constitutional heterogeneity, sampling bias arises from materials handling, sampling equipment, and specifically interactions at the interface between the steel[1] of the sampling tools and the broken ores. Biased sampling occurs when certain particles within the lot, due to their size, shape, density, or moisture content, are consistently favoured over others during the sampling process; the contribution to sampling bias due to weighing and moisture determinations is not considered in this discussion. This means that not every fragment has the same statistical chance of being in the sample. Sampling bias can be engineered out of sampling equipment provided that two principles, namely the Principle of Symmetry and the Principle of the Centre of Gravity (CoG), are upheld (Pitard, 1993). The Principle of Symmetry requires that the interaction of the steel of sampling equipment with fragments is exactly the same as it enters and exits the flowing stream of material. Sampling tools with thick walls (Figure 3a) and angles that are not steep enough (Figure 3b) are likely to introduce a bias. Good sampling cutter designs have sharp edges and angles with $\alpha \geq 70°$ as shown in the good design of Figure 3b.

The Principle of the Centre of Gravity requires that fragments whose centre of gravity lies within the sampling tool must be included in the sample, whereas those whose centre of gravity lies outside the tool must not be part of the sample, as illustrated in Figure 4 (Pitard, 1992).

Sampling of bulk commodities such as iron ore, manganese, chromite, bauxite, limestone, and coal for commercial purposes or monitoring of processes that use these products, is standard practice in industry. A video documentary entitled '*Sampling of Bulk Commodities, Design and Operation of Sample Cutters*' compiled and produced by Robinson and Holmes (1990) at the Commonwealth Scientific and Industrial Research Organisation (CSIRO), Australia describes nine necessary conditions for sample cutters to provide unbiased samples. The principle for cutters to deliver unbiased samples is that 'all particles should have an equal chance of being sampled' (0:01:10). The documentary examines the performance of sample cutters for bulk commodities and provides advice regarding cutter designs based on the work of Gy (1979).

Robinson and Holmes (1990) examined many types of sampling equipment used to sample flowing streams of material on a conveyor belt. Cross-stream cutters located at the end of a conveyor belt, where the cutter intersects the falling stream of material, are generally considered the most efficient and unbiased equipment. Cross-belt cutters intersect the material on the conveyor at some point along the length of the belt. Continuous research into cutter shapes and the angle at which the cutter intersects the moving stream of material has led to considerable improvement in the performance of cross-belt cutters. However, they suffer from the limitation that the material closest to the conveyor belt is generally under-sampled, and generally they are not highly recommended (Robinson and Holmes, 1990).

Robinson and Holmes (1990) described nine necessary conditions for cutters to be unbiased, some of which relate to cutter design and others to cutter performance. The basic principle of correct sampling requires that when the cutter moves through the stream of particles, it must intersect the entire stream rather than just a portion of the stream. The cutter in Figure 5a is too far forward, and does not appear to cut the entire stream, meaning the back of the stream is inadequately sampled. Adequately powered motors or hydraulic mechanisms able to drive the cutter through the moving stream at a constant speed are essential. Biased samples arise from hand-operated cutters as like shown in

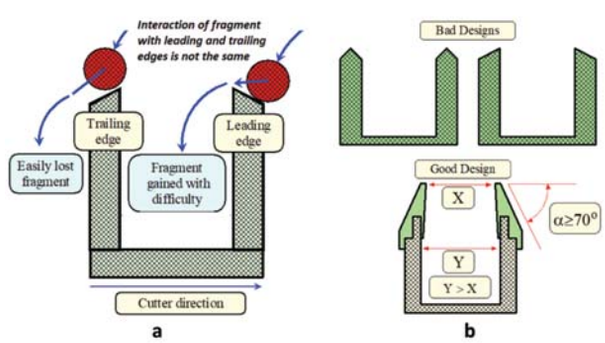


Figure 3—Principle of Symmetry requires that (a) the interaction between fragments and the sampling tool should be the same at the leading edge and trailing edge, and (b) the angle of the cutter blades should be steep and sharp (Pitard, 1992)

[1]One accepts that not all sampling devices are made of steel, but this is the preferred material, especially in highly abrasive mechanical environments where cross-stream or cross-belt cutters move through a stream of iron ore particles.

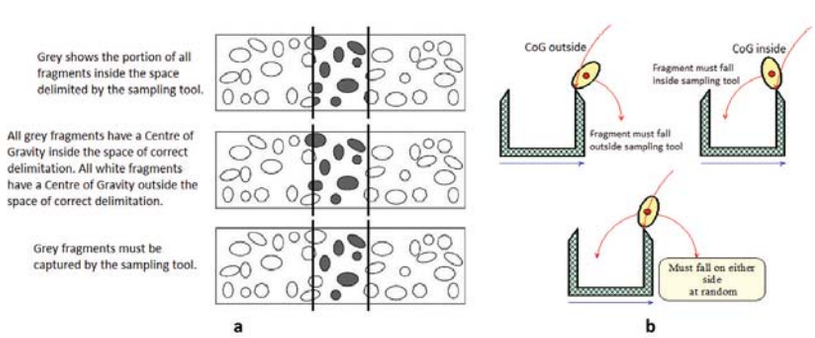


Figure 4—The Principle of the Centre of Gravity requires that (a) the CoG of fragments extracted by the sampling tool lies inside the space of correct delimitation, and (b) the final destination of fragments depend on where the CoG lies relative to the cutter blade (Pitard, 1992)

Figure 5b, because they cannot provide an even cut of the stream, nor does the cutter blade intersect the entire stream of particles.

Changes in the speed of the cutter as it moves through the stream during the sampling could introduce a bias. The highly abrasive environment in which the equipment operates means that cutter blades are subject to wear, so regular maintenance is required to ensure that the edges of the blades are sharp and straight. Wear and abrasion along the lips of the cutter could mean the size of the opening varies along the length of the cutter, being wider in the middle than it is at the ends. As a result, material along the centre of the stream is likely to be over-represented. An important aspect of cutter design is that the cutter must have sufficient capacity to hold all the material it extracts during a single traverse across the stream. The capacity of the cross-stream sampler shown in Figure 6a is too small, with the result that sample material falls out of the sampler onto the incoming stream once it has passed through the stream.

Loss of material from an overflowing cutter means the extracted sample cannot be correct or representative; it is biased. Losses of material from the sample, or extraneous additions of built-up material on the cutter blades, as shown in Figure 6b, mean samples will be either under-weight or contaminated. Cutter openings must be sufficiently wide that all particles have the opportunity of entering the sample cutter, and not as shown in Figures 5a and 6b. Gy (1979) suggested that cutter openings be at least three times larger than the nominal top size or the largest particle in the flowing stream. Where this is not the case, the largest particles interacting with one another at the cutter aperture could prevent other particles from entering the cutter. If the cutter aperture is too narrow, there is a possibility that larger particles could hit one cutter blade and bounce over the other blade, so that larger particles are under-represented. For cross-stream cutters it is essential that the cutter blades are at right angles to the trajectory of the falling stream. If the angle of the cutter blades to that of the incoming stream is small, large particles that should be included could hit the blades and easily bounce out of the cutter, resulting in a bias in which larger particles are under-represented. According to Van Niekerk (2017, personal communication) the +25 mm fraction of the lot tends to be missed during the sampling procedure. For Vezin samplers, such as those shown in Figure 1, the action is the same as a cross-stream sampler, but the cutter must move at constant speed, *i.e.* constant angular velocity, and the cutter blades must be radial to the centre of rotation, so that the falling stream is subjected to a constant cutting time irrespective of its position along the cutter. Swing-arm cutters and ramp cutters have openings that are at right angles to the stream and must move at constant velocity through the stream of particles. Significant research by the CSIRO into the size of cutter apertures, the speed of cutters, and the angle at which they intersect the stream, has provided the optimal conditions for cutter operation (Robinson and Holmes, 1990).

The principles for cutter operation in dry materials are essentially the same as for slurries and very moist streams of materials, except that there is no bouncing of the materials. For swing-arm and ramp cutters the bounces of particles are larger when the cutter enters the stream than when it exits. Particles that bounce sideways off cutter blades should not be able to bounce so far that they are not included in the sample (Robinson and Holmes 1990).

The study concluded that the simple aspects of cutter operation, such as routine maintenance, cutting the full stream, sufficiently wide cutter apertures and adequately powered motors are the most important (Robinson and

a b

Figure 5—Cross-stream cutters showing (a) a hand operated cutter which does not traverse the entire belt, and (b) a cutter whose aperture is too small for the material being sampled and wide, flat cutter blades that allow particles to wander around on top of the blades (Holmes 2015)

a b

Figure 6—Cross-stream cutters showing (a) insufficient capacity, resulting in massive reflux from a poorly designed primary cutter aperture at high flow rates, and (b) excessive sample build-up and partial blockage of a secondary cutter aperture (Holmes 2015)

Holmes, 1990). It is possible that uninformed operators could overlook deviations from standard sampling conditions and practices that may cause significant bias. Consequently, it is recommended here that an on-site competent person, who is fully aware of the principles of the Theory of Sampling, observes, audits, and verifies correct cutter operation at individual sampling locations on a regular, even weekly, basis to ensuring that no meaningful bias is introduced into samples.

Sampling uncertainty

Sampling uncertainty has to do with a model of statistical behaviour of sampling assays. Uncertainty can be established only when there is more than one measurement for which a variance can be calculated. Uncertainty defines an estimate of the statistical range within which a sample value could lie with a given level of confidence. The difference between an error and measurement uncertainty lies in the fact that an error is a mistake and implies responsibility, whereas measurement uncertainty implies no responsibility (Pitard, 2006). Uncertainty, an adjective describing an acceptance of a lack of knowledge in some measurements, can be aleatory or epistemic; aleatory uncertainty has to do with chance with no ability to predict the outcome, whereas epistemic uncertainty has to do with knowledge and an ability to predict the outcome. According to Gy (2004), sampling of particulate materials is always an aleatory operation.

Representation of sampling errors and bias

For illustrative purposes, this paper investigates the effect of sampling bias on the grade of iron ore and how such a bias affects the revenues derived from marketing of the product. A full description of the non-conditional Gaussian simulation product in a 1500×1500 m domain[2] using normal-score transformations of a parent distribution of the percentage Fe content in iron ores is described in Minnitt (2013, 2017). The iron ore simulations produced in this way constitute the daughter distributions kriged into a 150×150 domain used in this analysis. The 150×150 domain provides a common starting point from which to compare the effect of introducing error and bias to the sampling of the iron ore distribution.

Four sampling events were defined in this study. The first produced a control data-set on a 10×10 grid, where the actual percentage Fe is plotted against the estimated percentage Fe with no error and no bias, as shown in Figure 7a. A second sampling event of the actual percentage Fe on a 10×10 grid is plotted against the estimated percentage Fe including 10% error, but with no bias, as shown in Figure 7b. A third sampling event of the actual percentage Fe on a 10×10 grid included 10% error with a 0.9× bias (Figure 7c), and a fourth event of the actual percentage Fe on a 10×10 grid included 10% error with 1.1× bias, as shown in Figure 7d. Without the influence from poor sampling, the scattergram would be a straight line lying at a small angle to the 45-degree line of the perfect estimator. The effect of the strong negatively skewed the distribution of iron ore grades is evident in the scattergrams (Figures 7a to 7d), causing the tail to extend to the lower left corner of the scattergram.

[2]Domain refers to an area over an ore deposit in which the mean grade remains relatively constant, *i.e.* there is no trend.

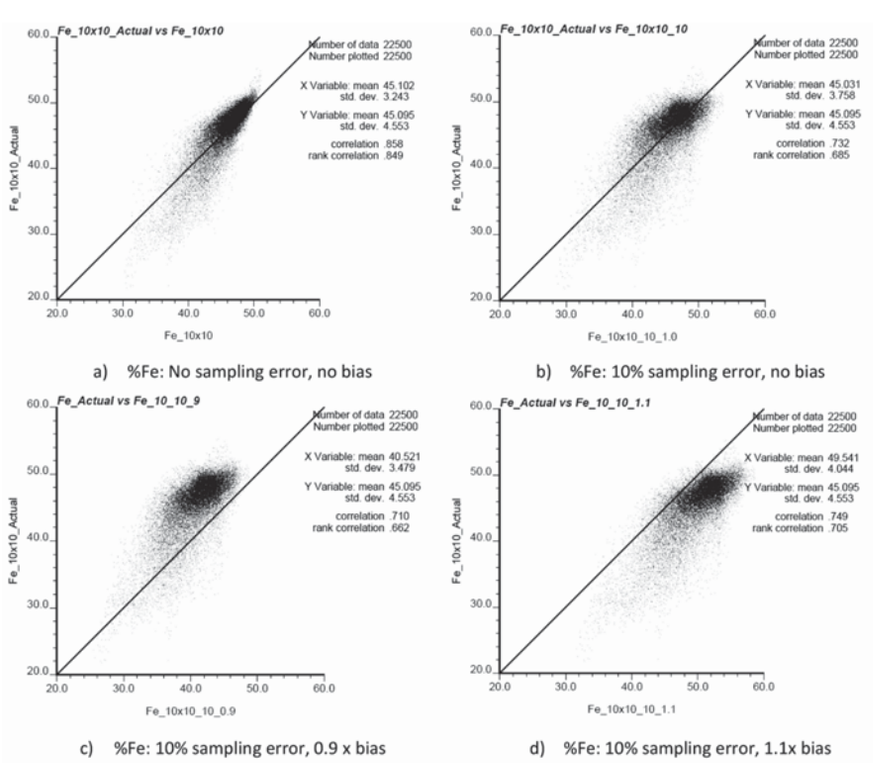


Figure 7—Comparison of actual %Fe values on a 10×10 grid with (a) kriged %Fe on a 10×10 grid with no sampling error and no bias, (b) kriged %Fe on a 10×10 grid with 10% sampling error only, (c) kriged %Fe on a 10×10 grid with 10% sampling error and 0.9× bias, and (d) kriged %Fe on a 10×10 grid with 10% sampling error and 1.1× bias

A comparison of grades of the actual and the kriged percentage Fe data on a 10×10 grid, shown in the scattergram in Figure 8, indicates that there is very good correspondence between the actual and kriged sets of data. Visually, the point cloud for the iron ore estimates is shifted upwards so that the bulk of the values lie slightly higher than the 45-degree line (Figure 8). The mean values differ by 0.007% Fe, while the standard deviation of the actual values tend to be somewhat larger than the kriged values. In the scatterplots in Figures 7a and 8, the actual 10×10 m block values on the y-axis are control data that is compared with the kriged 10×10 m blocks on the x-axis; there is no error or bias in this iron ore data.

As expected, the effect of adding 10% sampling error to the actual percentage Fe grades simply enlarges the point cloud of estimates, as shown in Figure 7b and Figure 9, indicating an increased variability of sample values around the perfect estimator and a negligible effect on the mean percentage Fe grade. Mean grade increases marginally (0.007%) from 45.095 to 45.102% Fe, but a 1.31% decrease in standard deviation from 4.553% Fe to 3.243% Fe is found, due to the smoothing effect commonly associated with ordinary kriging estimates (compare Figures 8 and 9).

Assume now that the starting points are the ordinary kriged iron ore values with a mean of 45.102% Fe and standard deviation of 3.243% Fe. Introducing a 10% error into the sampling (with no bias) results in a negligible

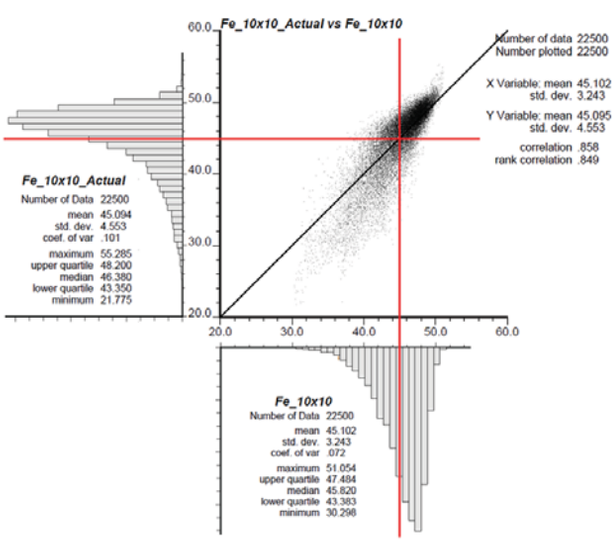


Figure 8—Scatterplot of the actual data (y-axis) against kriged values of %Fe on a 10×10 grid (x-axis) showing a 0.008% Fe difference in the mean values and a 1.31% Fe difference in standard deviations

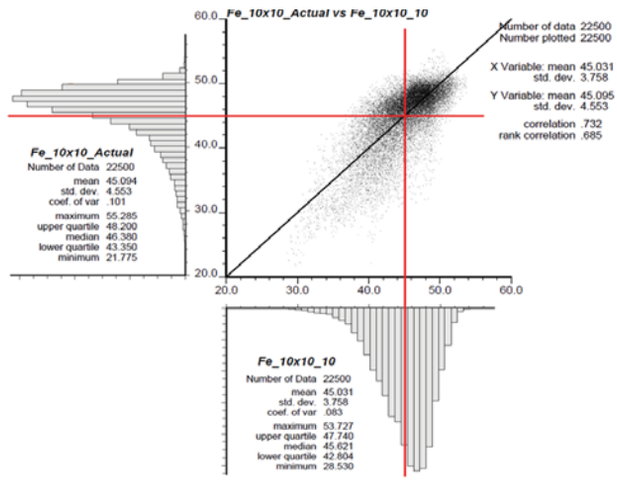


Figure 9—Actual %Fe data compared to kriged grades with a 10% sampling error and no bias; mean grade changes very little, by 0.064% Fe, but the standard deviation increases significantly by 13.7%, from 3.243 to 3.758% Fe (compare with Figure 7)

change from the actual to kriged mean value of about 0.0016% Fe. The standard deviation, however, increases from 3.243% to 3.758% Fe, a change of about 13.7% (Table II). This important result indicates that sampling errors arising from changes in constitutional heterogeneity simply expand the point cloud, resulting in greater variability of the grades, but negligible changes in the mean value.

However, when a bias is introduced, the scatterplots indicate that the effect of the bias on the mean value is much more severe than the effect of a 10% error, as is shown in Figures 7c and 7d.

In the case of 10% sampling error plus a 0.9× multiplicative bias (Figures 10 and 7c), the mean decreases by 10.16% from 45.102% Fe to 40.521% Fe, and the standard deviation increases by 6.78% from 3.243 (Figure 9) to 3.479% Fe (Figure 10).

For the sampling error of 10% with a 1.1× multiplicative bias (Figures 7d and 11) the mean increases 9.10% from 45.102% to 49.541% Fe, and the standard deviation increases 19.81% from 3.243% to 4.044% Fe.

With no sampling error and no sampling bias the actual (y-axis) and estimated (x-axis) percentage Fe grades are closely clustered around the 45-degree line – the unbiased estimator. The variation in iron ore grades tends to average out to approximately zero with repeated sampling over time. The change in the spread of the histograms in Figures 10 and 11, because of the introduction of 0.9× and 1.1×

Table II

Summary of observed changes in mean and standard deviation for 10% error, 0.9× bias, and 1.1× bias

	No error, no bias		10% error, no bias		10% error, 0.9× bias		10% error, 1.1× bias	
	Mean	Std dev	Mean	Std dev	Mean	Std dev	Mean	Std dev
Actual iron ore grades	45.095	4.553	45.102	3.243	45.102	3.243	45.102	3.243
Kriged iron ore grades	45.102	3.243	45.031	3.758	40.521	3.479	49.541	4.044
Difference	0.007	1.31	0.071	0.515	4.582	0.236	4.439	0.801
Percentage change	0.00016	28.77	0.0016	13.70	10.16	6.78	8.96	19.81

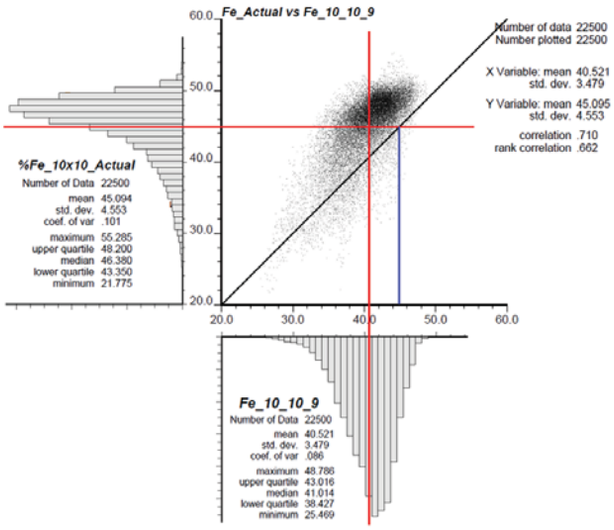


Figure 10—Actual kriged %Fe data compared with a 10% sampling error and 0.9× multiplicative bias results in a very large decrease of 4.574% Fe in the mean value from 45.095% to 40.521% Fe; the standard deviation changes relatively little from 4.553% to 3.479% Fe

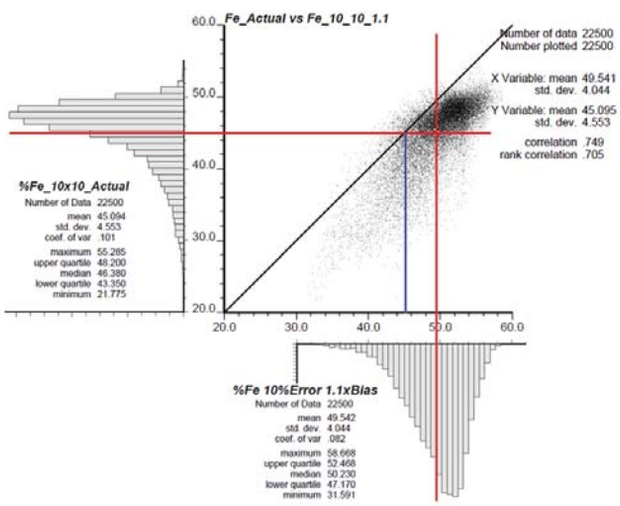


Figure 11—Actual kriged %Fe data compared to data with 10% error and 1.1× multiplicative bias results in a very large increase in the mean grade from 45.095% to 49.542% Fe; the standard deviation changes relatively little from 4.553% to 4.044% Fe

multiplicative bias, is large, changing the mean grade from 40.521% to 49.541% Fe, a difference of approximately 9.10% Fe. Negative sampling bias significantly decreases the mean grade by 10.16%, whereas a positive sampling bias significantly increases the mean grade by 8.96%. These biases are unrealistically high, but they provide an insight into the value-at-risk if biases were that high.

As already emphasized, generating biases this large in the iron ore industry is unlikely, but the example used here provides insight into how biases actually occur and how sampling bias, rather than sampling error, is the chief cause for concern in mineral and commodity trading. This is not to say that the FSE can be ignored, but rather to emphasise that sampling bias is a greater problem than sampling error because bias tends to be cumulative; it does not usually average out, even though it may change from time to time.

Financial impact of sampling error and sampling bias

Sales contracts between buyers and sellers of ores, concentrates, and metal products specify lower limits on grade and upper limits on deleterious elements for which penalties are payable. The financial consequences of systematic biased underestimation of the tailings grade are reported by Carrasco, Carrasco, and Jara (2004) for a tailings dump due for sale to another company. Approximately 96 000 t of tailings, supposedly containing 0.15% Cu but actually containing 0.20% Cu, reported to the dump every day. The bias of 0.05% Cu amounts to 48 t of copper per day, or 17 520 t of copper every year. At a copper price of US $6800 per metric ton, this represents an annual loss of approximately US$120 million (Lachance *et al.*, 2014). Statistical analysis of the sampling procedures and metal accounting systems are shown to provide control measures allowing allocations of gold contributions from different shafts to be established with confidence in the gold mining industry (Bartlett, Korff and Minnitt, 2014).

Iron ore contracts are quoted in US cents and the internationally agreed-upon unit of measure for iron ore pricing is the dry metric ton unit (dmtu). A dmtu consists of 1% of iron (Fe) contained in a ton of ore, excluding moisture. Iron ore is typically sold from South Africa at between 62% Fe and 66% Fe (Otto, 2018, personnel communication), so assuming a price of US$70 per ton for iron ore at a mean grade of 63.805% Fe would make the dmtu price US 109.7 cents per ton. The price per ton is calculated by multiplying the cents per dmtu price by the percentage of iron content, so a 100 000 t shipment of iron ore with a grade of 63.805% Fe should earn the producer or cost the customer about US$7.0190 million. For a 10% sampling error due to variations in the constitutional heterogeneity, the difference in %Fe on a lot with a grade of 63.805% Fe is 0.0016% (column 4, Table II), or 0.102% Fe, resulting in a US$11 230 difference, or 0.00116%, in the value of a 100 000 t shipment. Holmes (2013) indicates that the overall precision for iron ore grade specified in ISO 3082 varies depending on the size of the shipment, and could vary from 0.17 to 0.275% Fe for shipments varying from over 270 000 t to less than 15 000 tonnes, respectively.

For a swing arm (falling stream) cutter sampling iron ore at a nominal top-size of 32 mm with a flow rate of 10 000 t/h, equivalent to 2777 kg/s, at a belt speed of 4.3 m/s there is about 646 kg on each metre of the belt. Cutter speeds and cutter apertures on swing arm cutters vary from 200–800 mm/s and 30–100 mm respectively. These parameters mean that given that the mass of the average increment, given by Mass = $(C^*A)/(3.6^*Vc)$, where C is the flow rate of ore in tons per hour, A is the cutter aperture in metres, and Vc is the cutter speed in metres per second, the average primary cut could vary between 104 kg and 1388 kg, (Holmes, 2018, personal communication). The average increment would be about 463 kg every 10 000 t. A typical loading rate for iron ore in some South African facilities is 8 000 t/h with peaks up to 10 000 t/h, meaning a 100 000 t vessel could be loaded in approximately 10 hours (Steinhaus, 2017). When sampling lump ore according to ISO 3082 using minimum parameters, *i.e.*, a loading rate of say 10 000 t/h, a cutter aperture of 100 mm, and a cutter speed of 0.6 m/s the sample

would consist of 45 increments of 463 kg, giving a composite sample of about 21 t (Holmes, 2018, personal communication). So every 2222 t we take a 463 kg increment, therefore the sampling rate is 0.21 kg/t.

The grade of an iron ore assay is constituted from the contributions of individual fragments in a 21 t composite sample as shown in Table III (van Niekerk, 2017). Using a density of 5.15 g/cm^3 the diameter, radius, volume, and mass of a single particle in each individual size fraction is calculated. Using the percentage mass distribution of a typical lumpy ore, the mass of the size fraction in the composite sample can be calculated. Knowing the mass of one fragment, it is possible to calculate the number of particles in each size class. It is well known that the larger fragments of iron ore tend to have higher grades than smaller fragments, so the iron content of each size fraction can be calculated and is aggregated for each size faction (Table III). The cumulative grade for the different size fractions in the 21 t composite sample of ore with a standard particle size distribution as shown in column 7 of Table III, gives an average grade of 63.805% Fe.

According to the most recent (fifth) edition of ISO 3082 (2017) for sampling iron ore, the overall precision at a 95% confidence level varies from 0.34% to 0.55% Fe for shipments from more than 270 000 t to less than 15 000 t, respectively. For a 100 000 t lot, the overall precision is 0.38% Fe as given in the ISO 3082 (2017, Table 1) iron ore standard.

The question is: how easy is it to make an error like this? Assume a 21 t composite iron ore sample having a density of 5.15 g/cm^3, and size fractions from –6.3 mm to +31.5 mm, is extracted during loading of a 100 000 t vessel. The following example is hypothetical and one could adjust the figures, but the assumption made here is that poor engineering or lack of maintenance of sampling equipment results in 12% of high-grade fragments, ranging from 20 mm to +31.5 mm, being excluded from the 21 t sample. However, as a corollary, the volume of large particles not collected in the sample means that space is available for an equal volume of small, low-grade particles in the range –6.3 mm to –16 mm to be added to the sample. It is suggested here that the biased sampling equipment creates an exchange mechanism whereby small volumes of the larger high-grade particles sizes excluded from the sample are replaced by an equal (or much smaller) volume of the smaller low-grade particles. Assuming that small, low-grade fragments replace large, high-grade fragments, the overall Fe grade of the sample is reduced from 64.805% to 63.69 %Fe, as indicated in Table IV. That is an exchange of about 6.72% of the large for small fragment sizes in the original 21 t composite sample. The percentage sample mass loss from the 100 000 t shipment is not as important as the loss of mass from the 21 t composite sample itself resulting in a biased sample (Holmes, 2018, personal communication).

According to van Niekerk (2017, personal communication) the +25 mm fraction of the lot tends to be missed during the sampling procedure. Assume further that the percentage of fragments lost from the four largest size fractions (+31.5 mm to +20.0 mm) shown in Table IV is 12%. This results in an exchange of 1411 kg of the larger fragments for 1411 kg of the smaller fragments giving an overall reduction in grade of 0.1061% Fe, due to bias in the sampling equipment (Table IV). During the sampling process approximately 0.6 million large particles exchange places with 7.08 million small particles in the stream (Table IV), giving rise to a negative bias in the sample. The exchanges are never actually seen because the particles simply fall back onto the incoming stream and continue to the loading bay of the vessel.

The difference between the unbiased (63.805% Fe) and biased (63.699% Fe) grades, shown in Tables III and IV, due to the exchange between higher- and lower-grade fragments produces a bias of 0.106% Fe. What is noteworthy is that the bias remains the same (0.106% Fe) irrespective of the mass of the composite sample collected for assay. The 0.106% Fe bias in the grade for a 100 000 t load at a price of US$70 per ton and the lot grade of 63.805% Fe would amount to a loss of US$11 600, not much on a load worth US$7.0 million. However, South Africa exports 60 Mt of iron ore annually, and if 100 000 t shipments are used this amounts to 600 ships annually. Thus the cumulative losses in a year could be as much as US$6.96 million.

Table III

Particle size distribution, particle volume, particle mass, mass of size fraction in increment, mass of size fraction, and possible number of particles in a 21 t composite sample with an average grade of 63.805% Fe (van Niekerk, 2017, personal communication)

Size fractions	Largest particle diameter (mm)	Radius (mm)	Spherical particle volume (mm^3)	Mass of one particle (g)	Percentage mass distribution of a typical lump ore	Mass of size fraction in sample (kg)	Mass of size fraction (grams)	Possible particles in sample (n)	%Fe of size fraction	%Fe of total sample
+31.5 mm	31,5	15,75	0,00001636	84	1	210	210 000	2 493	66,0	
-31 +26.5 mm	26,5	13,25	0,00000974	50	7	1470	1 470 000	29 309	65,5	
-26.5 +25.0 mm	25	12,5	0,00000818	42	5	1050	1 050 000	24 934	65,0	
-25 +20.0 mm	20	10	0,00000419	22	21	4410	4 410 000	204 533	64,5	
-20 +16.0 mm	16	8	0,00000214	11	22	4620	4 620 000	418 501	64,0	
-16 +12.5 mm	12,5	6,25	0,00000102	5	18	3780	3 780 000	718 085	63,5	63,8050
-12.5 +10.0 mm	10	5	0,00000052	3	12	2520	2 520 000	935 007	63,0	
-10 +8.0 mm	8	4	0,00000027	1	7	1470	1 470 000	1 065 275	62,5	
-8 +6.3 mm	6,3	3,15	0,00000013	1	3	630	630 000	934 831	62,0	
-6.3 mm	5	2,5	0,00000007	0	4	840	840 000	2 493 352	61,5	
					100	21000	21 000 000	6 826 320		

Table IV

Effects of sampling bias on number, volume, and mass of particles lost, and particles gained (column 5) giving an average grade of 63.6989% Fe (van Niekerk, 2017, personal communication

Size fractions	No of +25 mm particles lost during sampling	Volume lost(+)/gained(-)	Mass (g) +25 mm particles lost during sampling	Remaining particles	Mass left over (kg)	%Fe of size fraction	Percentage left over	%Fe after large/small particle exchange
+31.5 mm	299	0,00489	25200	2194	185	66,0	0,8800	
-31 +26.5 mm	3517	0,03425	176400	25792	1294	65,5	6,1601	
-26.5 +25.0 mm	2992	0,02447	126000	21942	924	65,0	4,4001	
-25 +20.0 mm	24544	0,10276	529200	179989	3881	64,5	18,4804	
-20 +16.0 mm	50220	0,10765	554400	368281	4066	64,0	19,3604	
-16 +12.5 mm	-109637	-0,11206	-577130	827723	4357	63,5	20,7487	63,6989
-12.5 +10.0 mm	-142757	-0,07471	-384754	1077764	2905	63,0	13,8324	
-10 +8.0 mm	-162646	-0,04358	-224440	1227921	1694	62,5	8,0689	
-8 +6.3 mm	-142730	-0,01868	-96188	1077561	726	62,0	3,4581	
-6.3 mm	-380685	-0,02490	-128251	2874037	968	61,5	4,6108	
Fragments lost	81572	0,2740194	1411200	598196	21000		100	
Fragments gained	-938455	-0,2739346	-1410763	7085007				
Balance	-856883	0,0000848	437	7683203	0,44			(0,1061)

Conclusions

Sampling errors are those variations in the grade of the target analyte that occur because of constitutional heterogeneity, essentially differences in composition from one fragment of ore to the next. Sampling errors have to do with variability in assay values around the mean grade. Experimentation with a simulated iron ore deposit indicates that the effect of sampling error up to 10% has little or no impact on the mean values, and as a result the financial implications of random variability in iron ores is negligible. Sampling bias, by contrast, has an enormously significant effect on the mean grade of ores and is mainly responsible for financial losses in mineral trade.

Sampling bias arises from poorly designed and poorly installed sampling equipment that consistently favours certain material characteristics in the ores being sampled. A 0.9× to 1.1× multiplicative sampling bias can result in major changes in the means of the sample values and as a result, significant financial losses or gains can be incurred if sampling bias is undetected. These levels of bias are extreme, but they serve to illustrate that it is bias rather than sampling error that is responsible for potential financial losses. Careful attention to sampling equipment performance and maintenance is essential if unseen financial losses in the trade of commodities are to be avoided.

It is suggested here, based on the particle size distribution, that an equal or smaller volume of small, low-grade fragments replaces relatively small volumes of the large, high-grade iron ore fragments, excluded from the sample due to biased sampling equipment. Current indications are that biases in South Africa's iron ore industry are about 0.01% Fe, with probable losses of less than $638 per shipment or less than US$383 000 per annum on 600 shipments. If the status quo is not maintained, biased sampling equipment could decrease the average grade of a 100 000 t shipment by 0.106% Fe, which would result in losses of up to US$6.69 nillion in revenue for the South African iron ore industry.

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Prediction of the spontaneous combustion liability of coals and coal shales using statistical analysis

by M. Onifade and B. Genc

Synopsis

In this study we investigate the intrinsic factors influencing the propensity of coals and coal shales to undergo spontaneous combustion using statistical analysis. The intrinsic properties were determined by testing 14 *in situ* bituminous coals and 14 coal shales from the Witbank coalfield, South Africa. The relationships between these intrinsic properties (obtained from proximate and ultimate analysis) and spontaneous combustion liability indices (the Wits-Ehac Index and the Wits-CT Index) were established using linear and multiple regression analysis based on set criteria. The linear regression analyses indicate that moisture, volatile matter, ash, carbon, hydrogen, and nitrogen contents are the main factors affecting the spontaneous combustion liability of coals, while moisture, volatile matter, ash, carbon, hydrogen, nitrogen and total sulphur contents are the factors affecting the spontaneous combustion liability of coal shales. The regression analysis shows either a positive or a negative correlation coefficient between the intrinsic factors and the spontaneous combustion liability index. Multiple regression of the spontaneous combustion liability index on eight independent variables was used to develop acceptable and reliable predictive models as indicated by high R-squared values, high correlation coefficients, and low standard error of estimates. The use of the models derived from this study may enable the spontaneous combustion liability of coals and coal shales to be reliably predicted.

Keywords

spontaneous combustion, coal, coal shale, statistical analysis, Wits-Ehac Index, Wits-CT Index.

Introduction

Self-heating causes an increase in temperature without the contribution of heat from external sources. The reaction between coal and oxygen provides enough energy to support combustion without the influence of an external heat source (Onifade and Genc, 2018a). Spontaneous combustion in spoil heaps, waste dumps, highwalls, and coal shales is similar to coal oxidation. The self-heating of coal with a potential transition into endogenous fire constitutes a direct safety hazard in both underground and opencast mines, and unfavourably influences the mine environment. Most heat transfer may be by conduction, convection or radiation to the surrounding strata (Akande and Onifade, 2013; Akande, Onifade, and Aladejare, 2013). Rocks tend to be good insulators and retain heat within a coal seam or spoil heap. Self-heating as the major cause of coal shale and spoil heap fires is due to the accumulated influence of heat generating and heat dissipating mechanisms.

Coal and coal shale are sedimentary rocks that contain considerable amounts of organic and inorganic constituents (Dullien, 1979; Onifade and Genc, 2018b). This renders the rock permeable to water and air, and increases its surface area, thereby making the organic particles reactive to facilitate oxidation (Dullien, 1979). Extensive research has been conducted experimentally and computationally on the self-heating of coal in both surface and underground mines (Carras and Young, 1994; Genc and Cook, 2015; Kucuk, Kadioglu, and Gulaboglu, 2003; Stracher and Taylor, 2004). However, only limited studies have been conducted towards understanding the spontaneous combustion liability of coal shales exposed to atmospheric oxygen (Onifade, Genc, and Carpede, 2018; Onifade and Genc, 2018c; Onifade and Genc, 2018b).

Self-heating of coals and coal shales has been found to cause spontaneous combustion in selected bands of coal seams, highwalls, and spoil heaps (Onifade, Genc, and Carpede, 2018; Onifade and Genc, 2018c; Onifade and Genc, 2018b) (Figure 1). There is not sufficient information to evaluate and compare the spontaneous combustion liability of coal shales in relation to coals.

Different intrinsic and extrinsic factors affecting self-heating are the reason for the lack of better understanding of the mechanism of spontaneous combustion. These factors have been documented in various studies to predict the spontaneous combustion liability of coal (Banerjee, 1985; Beamish and Blazak,

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Figure 1—(a) Spoil heaps at Tweefontein Mine, (b) self-heating of coal shale away from the coal seam at Goedgevonden Colliery, Witbank, South Africa (Onifade and Genc, 2018b)

2005; Falcon, 2004; Guney, 1968; Kaymakci and Didari, 2002; Kim, 1977; Onifade and Genc, 2018b; Panigrahi and Sahu, 2004; Panigrahi and Sexana, 2001; Ren, Edwards, and Clarke, 1999). In the present work, selected intrinsic properties (moisture, ash, volatile, carbon, nitrogen, hydrogen, and, sulphur contents) of coals and coal shales were studied following standard procedures of the American Society for Testing and Materials (ASTM) and International Organization for Standardization (ISO). A statistical interpretation was carried out on coal and coal shale analysis data, and selected intrinsic factors affecting their liability toward spontaneous combustion examined. The combined effects of the selected intrinsic factors on the self-heating potential of these materials were evaluated for predictive purposes using multiple regression analysis. This will be useful for establishing significant relationships between coal and coal shale in terms of spontaneous combustion.

Materials and methods

Sample collection

Fourteen bituminous coals and fourteen coal shales were obtained from four coal mines in the Witbank coalfield using the ply sampling technique. Ply sampling provides a representation of the analysis of all the coal and mineral constituents in the seam as a whole. The samples were collected between selected bands of the coal seams (above and below) and highwalls, and sealed in airtight bags (made of aluminium-coated polyester) to avoid moisture loss and oxidation.

Sample preparation

Samples of coal and coal shale were reduced by crushing and ball milling to suitable sizes (<250 µm for geochemical analysis and <212 µm for spontaneous combustion tests) to obtain representative samples as required for each test. Volatile matter, ash content, moisture content, and fixed carbon were determined according to ASTM D-3175, D-3174, and D-3173. Carbon, hydrogen, nitrogen, and sulphur were determined using a LECO TruSpec CHNS analyser, after calibration with sulfamethazine according to ISO 12902:2001. The data processing was done by the software incorporated in the instrument. The results are given in percentages of carbon, hydrogen, and sulphur in the sample. The results for proximate, elemental analysis (percentage, air-dried basis), and spontaneous combustion tests (Wits-Ehac and Wits-CT Index) carried out on each sample are presented in Tables I and II.

Wits-Ehac tests

The Wits-Ehac Index was developed in the late 1980s to test the spontaneous combustion liability of coal (Eroglu, 1992; Genc, Onifade. and Cook, 2018; Genc and Cook, 2015; Onifade, Genc, and Carpede, 2018; Onifade and Genc, 2018b, 2018c, 2018d, 2018e; Onifade and Genc, 2018b, Uludag, Phillips, and Eroglu, 2001). Wade, Gouws, and Phillips, (1987) reported the details of the experimental procedure (Figure 2a). The index is calculated from the formula in Equation [1]. MS Excel is used to calculate the stages and generate the thermogram (Figure 2b).

$$\text{Wits-Ehac Index} = (\text{Stage II slope}/\ \text{XPT})*500 \qquad [1]$$

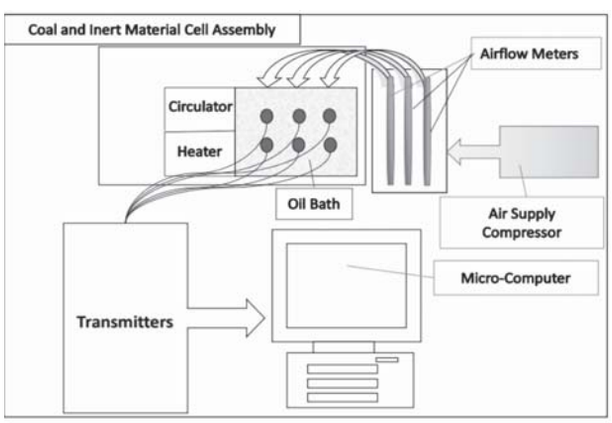

Figure 2a—Wits-Ehac apparatus set-up (Wade, Gouws, and Phillips, 1987)

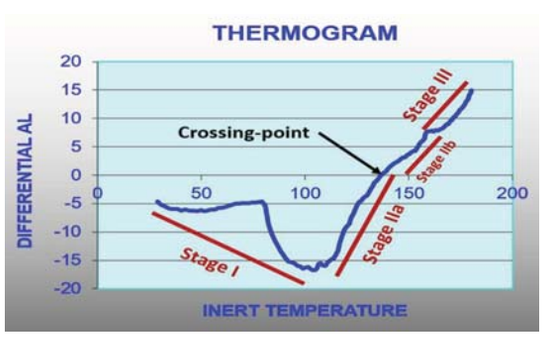

Figure 2b—A typical differential analysis thermogram of a coal sample produced by the Wits-Ehac Index

Wits-CT tests

The Wits-Ehac Index was developed to test the spontaneous combustion liability of coal. However, the Index failed to produce tangible results during the testing of some coal shales due to their low reactivity (Onifade, Genc, and Carpede, 2018). This is usually the case when the proportions of different organic (macerals) and inorganic matter (mainly crystalline) present in the samples vary. Consequently, a new method and apparatus were developed n the School of Mining Engineering, University of the Witwatersrand. This method is referred to as the Wits-CT test. The liability of various samples to spontaneous combustion was evaluated for 24 hours in ambient air. The details of the experimental procedure are documented by Onifade, Genc, and Carpede (2018). Figure 3 illustrates the experimental set-up. The index is calculated from the formula in Equation [2].

$$\text{Wits-CT Index} = (T_M/24 + T_R)^* \%C_{ad} \quad [2]$$

where T_M is the difference between the sum of the maximum temperatures of each thermocouple in the autoclave and room temperature (22°C), T_R is the difference between the peak temperature and initial temperature during oxidation in degrees Celsius, $\%C_{ad}$ is the air-dried percentage of carbon content of the sample, and 24 is the test duration in hours (constant).

Results and analysis

The results for proximate, elemental analysis, and spontaneous combustion tests conducted on coal and coal shale samples are shown in Tables I and II respectively.

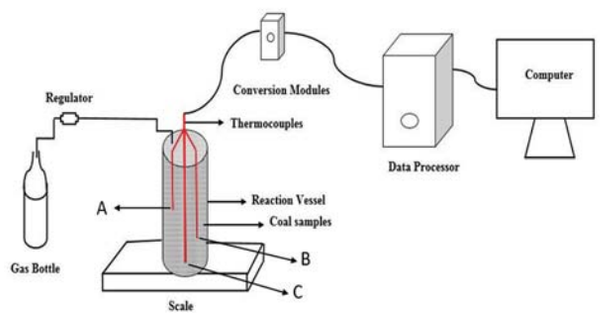

Figure 3—Schematic view of experimental set-up for Wits-CT tests (Onifade, Genc, and Carpede, 2018)

Table I

Proximate, elemental analysis (air-dried,%) and spontaneous combustion test results for the coals

Sample	Proximate analysis				Ultimate analysis					Liability indices	
	M	V	A	FC	C	H	N	S	Oc	WE	WC
CA	2.3	23.2	28.0	46.5	54.4	3.33	1.34	1.91	8.72	4.64	6.29
CB	2.3	21.0	20.0	56.7	61.4	3.36	1.48	1.11	10.4	4.64	6.96
CC	2.2	24.1	33.8	39.9	47.5	3.20	1.35	3.96	7.99	4.52	5.31
CD	2.3	25.5	20.5	51.7	61.4	3.78	1.53	0.86	9.63	4.60	6.80
CE	2.3	24.3	28.6	44.8	53.6	3.41	1.25	1.08	9.76	4.76	5.42
CF	2.5	23.6	46.9	27.0	35.9	3.01	0.89	3.42	7.38	4.49	3.97
CG	2.5	20.0	16.8	60.7	66	3.64	1.58	0.64	8.84	4.91	7.53
CH	2.4	26.9	18.8	51.9	65.2	4.21	1.55	2.19	5.65	4.69	7.51
CI	2.1	16.7	48.4	32.8	36.1	2.55	0.85	1.22	8.78	3.82	4.05
CJ	1.9	25.7	28.1	44.3	52.4	3.13	1.35	5.30	7.82	4.46	6.61
CK	1.6	22.1	13.7	62.6	69.7	4.02	1.60	0.76	8.62	4.44	9.10
CL	1.6	26.1	22.5	49.8	58.9	3.57	1.45	3.88	8.10	4.87	9.59
CM	1.6	22.0	17.0	59.4	66.7	3.77	1.57	0.59	8.78	4.76	7.27
CN	1.6	23.9	17.0	57.5	65.8	4.2	1.63	2.92	6.85	4.84	7.91

Table II

Proximate, elemental analysis, (air-dried,%), and spontaneous combustion test results for the coal shales

Sample	Proximate analysis				Ultimate analysis					Liability indices	
	M	V	A	FC	C	H	N	S	Oc	WE	WC
SA	1.4	11.2	78.5	8.9	11.5	1.34	0.34	0.54	6.39	3.09	1.33
SB	0.9	13.9	77.2	8.0	11.0	1.27	0.40	1.56	7.67	3.06	1.30
SC	1.1	12.7	74.6	11.6	13.8	1.60	0.42	0.35	8.13	-	0.91
SD	1.6	13.3	77.3	7.8	10.8	1.43	0.32	2.53	6.02	3.27	0.70
SE	1.7	15.9	68.4	14.0	15.8	1.78	0.41	6.90	5.01	3.73	1.60
SF	0.9	13.5	76.9	8.7	11.8	1.40	0.43	0.46	8.11	3.10	1.36
SG	0.8	10.7	84.3	4.2	6.02	1.04	0.29	0.73	6.83	-	0.67
SH	0.8	8.5	88.7	2.0	2.66	0.96	0.09	0.41	6.38	-	0.27
SI	1.0	11.9	79.6	7.5	9.12	1.41	0.26	0.22	8.39	-	0.95
SJ	0.9	11.9	86.9	0.3	3.42	0.75	0.08	0.75	7.19	-	0.42
SK	1.0	11.7	79.1	8.2	9.75	1.73	0.41	0.16	7.85	2.98	1.18
SL	1.0	16.0	74.0	9.0	10.5	2.14	0.39	0.12	11.85	2.99	1.34
SM	0.8	11.7	76.9	10.6	12.5	1.61	0.52	0.24	7.43	-	1.44
SN	1.5	16.6	51.5	30.4	33.7	2.87	0.96	0.31	9.16	3.77	3.99

M=moisture (%), V=volatile matter (%), A=ash (%), FC=fixed carbon determined by difference, C=carbon (%), H=hydrogen (%), N=nitrogen (%), S=sulphur (%), Oc =calculated oxygen (%), WE=Wits-Ehac Index, and WC=Wits-CT Index.

The moisture content varies between 1.6% and 2.5% for coal, and 0.8% and 1.7% for coal shales. The samples have a low moisture content and are thus more liable to spontaneous combustion. The low moisture content could be caused by the amount of water molecules absorbed on the external surface and internal open pore surface of the coals and coal shales. A low content of both physically and chemically absorbed water is one of the characteristics of higher rank coals. Similar studies are documented by Beamish and Hamilton (2005), McPherson (1993), and Onifade and Genc (2018d). The coal shales have low moisture contents similar to the tested coal samples. The moisture content of coals CG to CJ varies between 1.9% and 2.5%, between 2.2% and 2.3% for CA to CC, and between 2.3% to 2.5% for CD to CF. CK to CN have the lowest moisture content (all with 1.6%), while coals CF and CG have the highest moisture contents, followed by CH and then CA, CB, CD, and CE. It was found that the samples have approximately the same moisture contents and are more liable to spontaneous combustion, except for samples CF and CI, with lower liability indices.

Coal shales SE, SD, and SN have approximately the same moisture contents. Coal shale SN is more liable to spontaneous combustion than the other coal shales. This may be due to the presence of unidentified mineral matter that promotes the self-heating rate (Onifade and Genc, 2018d). The study shows that an increase in moisture content of coal and coal shales is enough to provide a high heat loss from evaporation, as the coal temperature increases during the oxidation reaction.

The volatile matter content varies between 16.7% and 26.9% for the coals and 6.5% and 16.6% for the coal shales. The volatile matter content for the coals is greater than 20%, except for sample CI (16.7%). Coals with a high volatile matter content are more liable to undergo spontaneous combustion, except for sample CF, which has a low Wits-CT Index as indicated in Table I. This is in line with the studies reported by Banerjee (2000) and Onifade and Genc (2018d). Among the coal shales, the highest volatile matter content is found in SN and SE, which are more liable to undergo spontaneous combustion compared to the other coal shales.

The ash content ranges between 13.7% and 48.4% for the coals, and 51.5% and 88.7% for the coal shales. It is known that the physical and chemical properties of coal changes during oxidation (Taylor *et al.*, 1998). The variation in ash content for samples from the same seam may be attributed to different proportions of mineral matter. The ash contents of coals CF and CI are high and these samples have low liability indices, while coals with low ash content have higher indices. The low and high liability indices of the coal samples could be due to the heat absorbing capacity of the minerals within the coal (Onifade and Genc, 2018d). Coal shales SN and SE have the lowest ash contents and the highest liability indices. Coal shales SH, SJ, and SG have high ash contents and indicated very low liability indices with the Wits-CT Index. Their liability indices could not be determined with the Wits-Ehac Index because of their low reactivity (Onifade *et al.*, 2018).

The carbon content varies between 35.9% and 69.7% for the coals, and 2.66% and 33.7% for the coal shales. Coals CF and CI have the lowest carbon contents and the lowest liability indices, whereas coals with high carbon contents have high liability indices (Onifade and Genc, 2018c). Coal shales SN and SE have the highest carbon content and the highest liability indices. It was found that coal shales with high and low carbon contents show similar characteristics towards spontaneous combustion as coals containing high and low carbon contents.

Coals CI and CF have the lowest hydrogen (2.55% and 3.01%) and the lowest liability indices, while coals CN and CH have medium-high hydrogen contents of 4.20% and high spontaneous combustion indices. Coal CN has the highest nitrogen content and a high liability index, while coals CI and CF have the lowest nitrogen contents of 0.85% and 0.89% and low liability indices. Coal shale SN has the highest hydrogen and nitrogen content of 2.87% and 0.96%, respectively and high liability indices compared to the other coal shales, while coal shale SJ, with the lowest hydrogen (0.75%) and nitrogen content (0.08%), has a low liability index. Coal CH has the lowest oxygen at 5.65%, and the highest oxygen content is found in coal CB (10.35%). Coal shale SE has the lowest oxygen content of 5.01% while coal shale SL has the highest oxygen content of 11.85%. This study found that oxygen content may have no influence on the spontaneous combustion liability index. Hence, the oxygen content of coal and coal shale does not seem to show a direct relationship to the readiness with which they absorb oxygen.

Statistical analysis

The initial evaluation of the intrinsic factors affecting spontaneous combustion of coals and coal shales was based on data obtained from spontaneous combustion liability tests and selected intrinsic factors obtained in the laboratory. The data was grouped into dependent and independent variables in order to facilitate the analyses. The statistical analysis was conducted by correlating coal and coal shale intrinsic factors as independent variables, with the values of the Wits-CT and Wits-Ehac indices as dependent variables. The R-squared values and the correlation coefficients were used to measure the trends and determine any significant relationships between the intrinsic properties and the Wits-Ehac Index and Wits-CT Index (Tables IV and V). The study interpreted the linear relationship between the intrinsic properties and spontaneous combustion liability index based on the criteria set by Onifade and Genc (2018b) as shown in Table III. Detailed descriptions of the statistical method and criteria are given by Onifade and Genc, (2018b).

Linear regression analysis

Data obtained from spontaneous combustion liability tests and intrinsic factors for the 28 coal and coal shale samples was analysed statistically. Tables IV and V presents the results of the linear regression analyses for both the coal and coal shale samples. Parentheses indicate a negative value.

The analysis of the independent and dependent variable pairs (moisture/Wits-Ehac Index; moisture/Wits-CT Index; volatile matter/Wits-Ehac Index; volatile matter/Wits-CT Index; ash/Wits-Ehac Index; ash/Wits-CT Index *etc.*) indicated similar trends in some cases. The spontaneous combustion liability index increases with increasing volatile matter, carbon, and hydrogen and decreases with increasing ash for the coals and coal shales. The proximate and ultimate analyses indicated that moisture, ash, carbon, hydrogen, and

Table III

Criteria for factors affecting spontaneous combustion liability of coals and coal shales (Onifade and Genc, 2018b)

Category	Criterion	Remarks
1	Correlation coefficient/R-squared value between 0.95 to 1 or -0.95 to -1	Perfect positive or negative linear relationship
2	Correlation coefficient/R-squared value between 0.51 to 0.94 or -0.51 to -0.94	Strong positive or negative linear relationship
3	Correlation coefficient/R-squared value between 0.25 to 0.50 or -0.25 to -0.50	Moderate positive or negative linear relationship
4	Correlation coefficient/R-squared value between 0.1 or 0.24 or -0.1 to -2.24	Weak positive or negative linear relationship
5	Correlation coefficient/R-squared value less than 0.1 but not zero	Very weak positive or negative linear relationship
6	Correlation coefficient of zero	No linear relationship

Table IV

Relationships between independent (wt.%, air-dried) and dependent variables obtained from linear redgression for coal samples

Independent variable	Dependent variables R-squared values		Dependent variables correlation coefficients	
	Wits-Ehac Index	Wits-CT Index	Wits-Ehac Index	Wits-CT Index
Moisture	0.0041	0.3418	(0.0637)	(0.5847)
Volatile matter	0.2666	0.1071	0.5164	0.3273
Ash	0.4739	0.7500	(0.6884)	(0.8660)
Carbon	0.4318	0.7336	0.6572	0.8565
Hydrogen	0.4377	0.5703	0.6616	0.7552
Nitrogen	0.4823	0.7026	0.6945	0.8382
Total sulphur	0.0047	0.0047	(0.0686)	(0.1016)
Oxygen	0.0001	0.0103	(0.0115)	(0.0687)

Table V

Relationships between independent (wt.%, air-dried) and dependent variables obtained from linear regression for coal shale samples

Independent variable	Dependent variables R-squared values		Dependent variables correlation coefficients	
	Wits-Ehac Index	Wits-CT Index	Wits-Ehac Index	Wits-CT Index
Moisture	0.5952	0.2031	0.7715	0.4507
Volatile matter	0.4082	0.4763	0.6389	0.6901
Ash	0.6975	0.8926	(0.8352)	(0.9448)
Carbon	0.6339	0.9273	0.7962	0.9630
Hydrogen	0.3358	0.7742	0.5795	0.8799
Nitrogen	0.4155	0.8953	0.6446	0.9462
Total sulphur	0.3353	0.0006	0.5791	0.0242
Oxygen	0.1031	0.1052	(0.3212)	(0.3243)

nitrogen are the factors influencing spontaneous combustion liability of coal based on the Wits-CT Index, while volatile matter, ash, carbon, hydrogen, and nitrogen are the main factors affecting spontaneous combustion liability with the Wits-Ehac Index. In addition, volatile matter, ash, carbon, hydrogen, and nitrogen are the factors influencing spontaneous combustion liability of coal shales, with stronger relationships using the Wits-CT Index, while moisture, volatile matter, ash, carbon, hydrogen, nitrogen, and total sulphur are the factors affecting spontaneous combustion liability of coal shales according to the Wits-Ehac Index. The linear regression indicates that the two spontaneous combustion liability indices show linear relationships with the intrinsic factors for both coal and coal shale samples based on the criteria used, and thus identify the major intrinsic factors affecting spontaneous combustion liability (Table IV and V). Similar findings are documented by Onifade and Genc (2018b). This study indicated that the intrinsic factors of coal and coal-shale may be used to measure spontaneous combustion liability. It was found that of the intrinsic properties considered, only the ash content has a negative effect on spontaneous combustion liability. Figures 4 and 5 illustrate the relationships between intrinsic factors and spontaneous combustion liability indices using linear regression analysis.

Influence of intrinsic factors on spontaneous combustion liability of coals and coal shales

Figures 4i and 5a illustrate the influence of moisture content on the spontaneous combustion liability of coals and coal shales. The results indicated a negative correlation for the

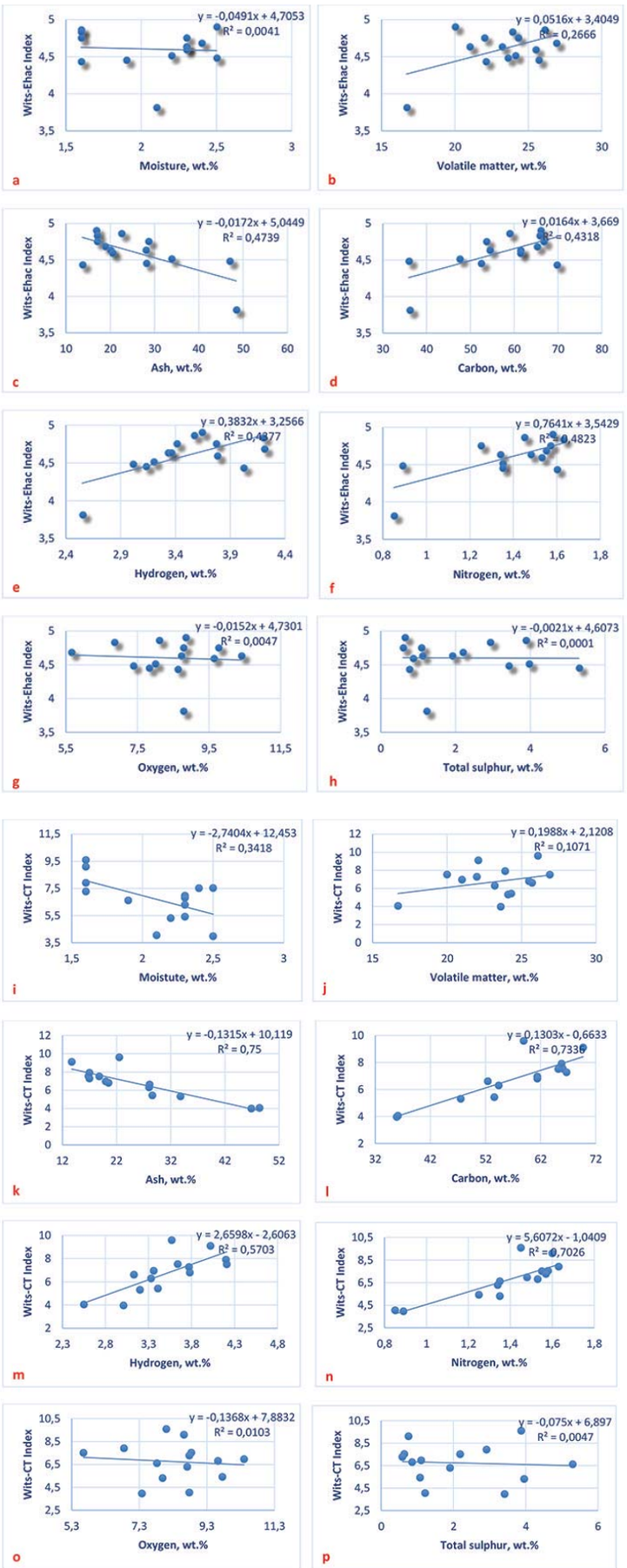

Figure 4—Linear relationships between liability indices (Wits-CT and Wits-Ehac) and intrinsic properties of coal

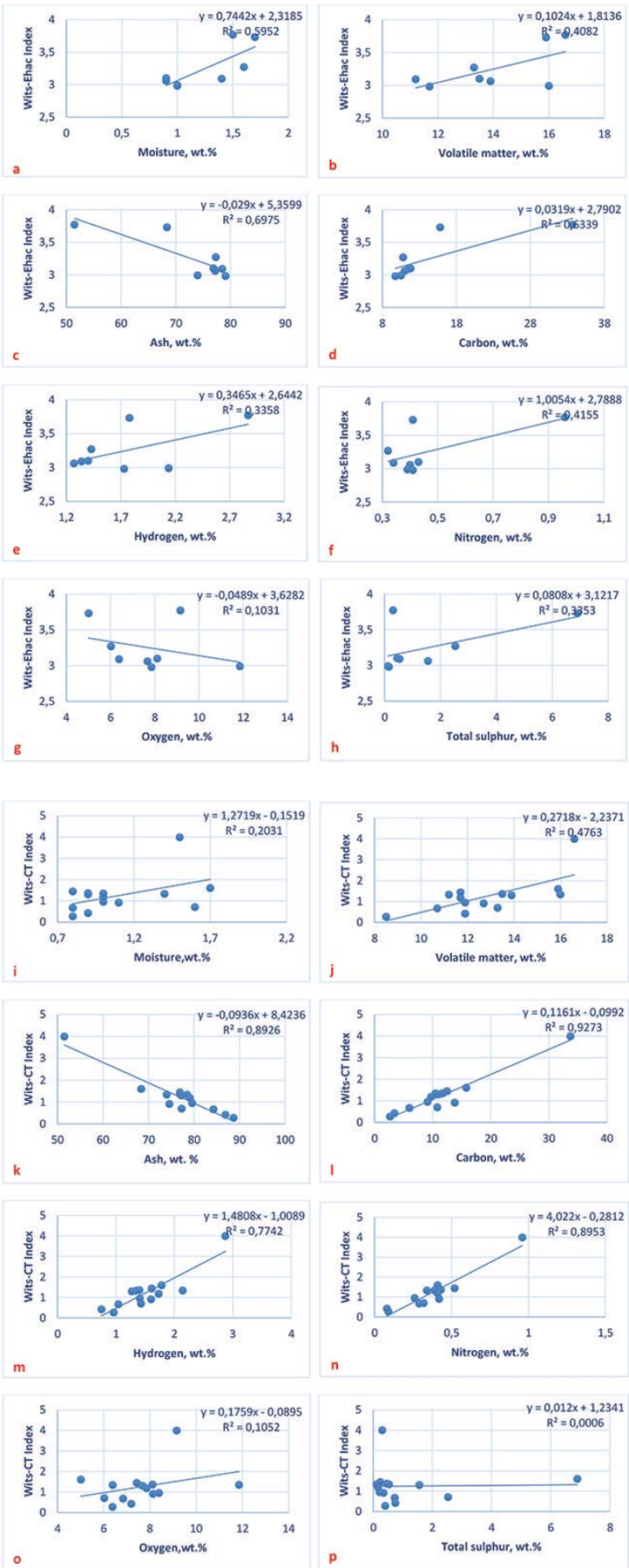

Figure 5—Linear relationships between liability indices (Wits-CT and Wits-Ehac) and intrinsic properties of coal shales

coals and a positive correlation for the coal shales. As the correlation displays an R-squared of 0.3418 for coals, it shows that the linear model has a moderate strength. It is different for the coal shales. Here the R-squared value is 0.5952, higher than for the coals. This indicates that the moisture content of the coal shales has a strong influence on the self-heating potential.

The R-squared value for volatile matter is 0.2666 for coals and 0.4763 for coal shales (Figures 4b and 5j). This is an indication that volatile matter content has a moderate influence on both coals and coal shales. The self-heating potential tends to increase for both cases but the increase seems to be more pronounced for coal shales than for coals. There is a positive relationship between the volatile matter and spontaneous combustion liability index for both coals and coal shales. Hence, as the volatile matter increases, the self-heating potential is likely to increase. Similar results are reported by Onifade and Genc, (2018b).

There is a negative relationship between the ash content and spontaneous combustion liability index for both coals and coal shales. A similar trend is reported by Onifade and Genc (2018b). The R-squared value is 0.75 for coals and 0.8926 for coal shales (Figures 4k and 5k). Therefore, the relationship between the ash content and spontaneous combustion liability index seems to be strong.

The R-squared value for carbon content is 0.7336 for coals and 0.9273 for coal shales (Figures 4l and 5l). The relationship between carbon content and spontaneous combustion liability seems to be strongly positive. Therefore, as the carbon content increases, the self-heating potential is likely to increase. Onifade and Genc, (2018b) reported similar results.

The R-squared value for the hydrogen content is 0.5703 for coals and 0.7742 for coal shales (Figures 4m and 5m). The relationships between the hydrogen content and spontaneous combustion liability index for coals and coal-shales are strongly positive. Hence, as the hydrogen content increases, the self-heating for both coals and coal shales may increase. Similar results were reported by Onifade and Genc, (2018b).

The R-squared value for the nitrogen content is 0.7026 for coals and 0.8953 for coal shales (Figures 4n and 5n). The relationships between the nitrogen content and spontaneous combustion liability index for coals and coal shales are both strong and positive. This indicates that as the nitrogen content increases, the self-heating potential is likely to increase, but this effect seems to be more pronounced for coal shales than for coals.

Total sulphur showed a negative correlation for the coals and a positive correlation for the coal shales (Figures 4p and 5h). The R-squared value of 0.0047 for coals (Figure 4p) indicates that the linear model does not fit well. This is not the same with the coal shales. Here, the R-squared value is slightly higher, 0.3353, than for the coals, thus indicating a moderate influence on self-heating potential, as opposed to the very weak effect for coals. Onifade and Genc, (2018b) reported similar results.

The calculated oxygen showed a negative correlation for the coals and a positive correlation for the coal shales. The correlation (Figure 4o) has an R-squared value of 0.0103 for coals, indicating that the linear model does not fit well. The R-squared for coal shales is slightly higher, 0.1052 (Figure 5o). Hence although the influence of this factor is very weak, it is more pronounced for coal-shales than for coals.

Multiple regression analysis

Multiple regression analysis was used to establish models for predicting the spontaneous combustion liability index of coals and coal shales. The models were developed by using eight independent variables (Table VI). A similar study by Onifade and Genc (2018b) indicated that the use of a single variable to predict the spontaneous combustion liability is unreliable. The study indicated that the influence of each intrinsic factor on spontaneous combustion liability varied, based on the linear regression. This motivated the need for further development of the models. The model that provides the highest correlation coefficient and lowest standard error of estimate was determined through multiple regression calculations.

The most reliable predictions of spontaneous combustion liability of the 14 coals and 14 coal shales are provided by the four models listed in Table VI. The developed models indicated a high correlation coefficient and low standard error. The study indicated that the Wits-Ehac Index and intrinsic properties show correlation coefficients of 0.815 and 0.998, and low standard errors of 0.254 and 0.011 for the coals and coal shales respectively, while the Wits-CT Index shows higher correlation coefficients of 0.937 and 0.991, and

Table VI

Models developed by multiple regression analysis

Formula no.	Models developed	R*	SEE†
All coals (A)	$WE=154.49–1.23M_{ad}–0.01V_{ad}–1.54A_{ad}–1.52C_{ad}–1.01H_{ad}–2.10N_{ad}–1.4S_{ad}–1.41O_c$	0.815	0.254
All coals (B)	$WC=968.39–10.37M_{ad}–0.04V_{ad}–9.70A_{ad}–9.43C_{ad}–9.62H_{ad}–15.30N_{ad}–9.13S_{ad}–9.50O_c$	0.937	0.931
All coal shales (C)	$WE=-2761+25.98M_{ad}+0.51V_{ad}+27.65A_{ad}+27.68C_{ad}+29.6H_{ad}+21.73N_{ad}+27.17S_{ad}+26.83O_c$	0.998	0.011
All coal shales (D)	$WC=-8396.61+79.12M_{ad}+1.16V_{ad}+83.99A_{ad}+84.17C_{ad}+88.71H_{ad}+70.40N_{ad}+82.88S_{ad}+82.18O_c$	0.991	0.193

* R = correlation coefficient, † SEE = standard error of estimate.

standard errors of 0.931 and 0.193. The models developed for the Wits-Ehac Index give lower standard errors of estimate and correlation coefficients than the Wits-CT Index for both coals and coal shales (Table VI). Therefore, the Wits-Ehac Index can yield more reliable results based on a low of standard error, while the Wits-CT Index can also be used to obtain suitable results based on high correlation coefficients. This indicates that the two liability indices can be used to predict the spontaneous combustion propensity of coals and coal shales. The models indicate that spontaneous combustion occurs due to the combined effect of various intrinsic factors. Onifade and Genc (2018b) reported similar results.

Validation of predicted model results

The results of the actual Wits-Ehac and Wits-CT indexes for coals and coal shales are presented in Tables VII and VIII.

The predicted spontaneous combustion liability indices (Wits-Ehac and Wits-CT) were validated with the actual indices, and the results are in line (Tables VII and VIII).

Table VII

Actual and predicted Wits-Ehac and Wits-CT indexes for coals

Sample	Wits-Ehac Index		Wits-CT Index	
	Actual	Predicted	Actual	Predicted
CA	4.64	4.47	6.29	6.20
CB	4.64	4.60	6.96	6.78
CC	4.52	4.41	5.31	5.31
CD	4.60	4.69	6.80	6.54
CE	4.76	4.56	5.42	6.17
CF	4.49	4.28	3.97	4.13
CG	4.91	4.67	7.53	7.29
CH	4.69	4.67	7.51	7.33
CI	3.82	3.88	4.05	3.94
CJ	4.46	4.53	6.61	7.49
CK	4.44	4.62	9.10	8.95
CL	4.87	4.58	9.59	9.15
CM	4.76	4.43	7.27	7.93
CN	4.84	4.68	7.91	8.35

Table VIII

Actual and predicted Wits-Ehac and Wits-CT indexes for coal shales

Samples	Wits-Ehac Index		Wits-CT Index	
	Actual	Predicted	Actual	Predicted
SA	3.09	3.10	1.33	1.01
SB	3.08	2.99	1.30	1.06
SC	-	2.85	0.91	0.99
SD	3.27	3.18	0.70	0.67
SE	3.73	3.37	1.60	1.50
SF	3.10	3.05	1.36	1.37
SG	-	2.94	0.67	0.62
SH	-	2.99	0.27	0.14
SI	-	2.90	0.95	0.66
SJ	-	2.72	0.42	0.19
SK	2.98	3.02	1.18	1.05
SL	2.99	2.89	1.34	1.19
SM	-	2.86	1.44	1.13
SN	3.77	3.63	3.99	3.98

Twenty-eight samples evaluated in this work have confirmed the consistency of the developed models. The Wits-Ehac indices for coal shales SC, SG, SH, SI, SJ, and SM, which were not obtained by the actual Wits-Ehac Index due to their low reactivities, were successfully predicted with the models. The models present a high level of confidence as they produced results in line with the actual liability indices.

Conclusion

The intrinsic properties of coal shales and coals could be used as a measure of the spontaneous combustion risk in coal mines. using linear regression. Moisture, volatile matter, ash, carbon, hydrogen, and nitrogen contents indicate better linear relationships with the spontaneous combustion liability index for coals, while volatile matter, ash, carbon, hydrogen, nitrogen, and total sulphur indicate better linear relationships with the spontaneous combustion liability index for coal shales. Multiple regression analysis shows that the Wits-Ehac Index and Wits-CT Index have high correlation coefficients with the intrinsic factors and a low standard error for both coals and coal shales. The spontaneous combustion liability of coals and coal shales can be predicted by a model consisting of various intrinsic factors. The predicted spontaneous combustion liability indices (Wits-Ehac and Wits-CT Index) were validated with the actual indices. Further research is under way towards establishing a generalized model involving the cumulative effect of other intrinsic properties such as petrographic composition and mineral matter.

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Geological mapping and modelling training in the University of the Witwatersrand Mine Tunnel, South Africa

by C. Birch

Synopsis

A simulated mine tunnel, stope, lamp room, and refuge chamber have been built in the basement of the School of Mining Engineering in the Faculty of Engineering and the Built Environment at Johannesburg's University of the Witwatersrand (Wits). Mock-ups are widely used in the mining industry to conduct training in a safe controlled environment.

This tunnel has been utilized for the past four years as part of the Advanced Mineral Resource Management Geological Modelling module. Students are required to map geological features depicted in the tunnel and link these to the information from a provided borehole database. This borehole database needs to be validated before the students create a 3D geological model that represents Bushveld Complex Critical Zone stratigraphy. Leapfrog Geo provides the software used for this assignment and they present two days of practical training in the creation of geological models to the students.

Students have been very positive about the learning experience and the majority of them have produced acceptable geological models. This combination of geological mapping in a simulated mine environment and the creation of a 3D geological model appears to be unique to Wits.

Keywords

geological modelling, mock-up, geological mapping, Leapfrog Geo, training, education.

Introduction

There are many people with vast experience in mining who never had the opportunity to attend full-time at a university or other tertiary institute. Their lack of formal academic qualifications and skills related to research and writing is hampering their promotional prospects. To address this gap, the Advanced Mineral Resource Management (MRM) programme was developed more than a decade ago in conjunction with a steering committee from the University of the Witwatersrand (Wits) School of Mining Engineering and the mining industry.

Eighteen modules are offered, all of which are run as short five-day courses that are assessed by means of examinations and assignments. The students, while completing the modules, develop presentation, writing, and research skills. Students find they are more confident in expressing their ideas verbally as well as in written form after completing the programme. The students on the Advanced MRM programme are predominantly non-graduate geological technicians, surveyors, mine planners, evaluation specialists, mining engineers, and plant metallurgists. Some of the students are university graduates and are doing the programme to supplement their mineral resource management skills. All the modules are considered to be at the South African National Qualifications Framework (NQF) Level 6.

Geological Modelling is one of the modules that form part of the Advanced MRM programme. For the past three years, the assignment exercise required for this module has been based on the Wits Mine Tunnel. Geological modelling training is primarily conducted in a classroom environment. Geological modelling courses are presented by institutes like the Geological Society of South Africa (Geological Society of South Africa, 2017) as well as the software providers (Leapfrog, 2017a) and consulting firms (SRK Consulting, 2017). Most of these courses target professionals working in the industry and are often focused on a specific software package or specialized task. None of the current courses found during an online search link basic geological mapping with creating a 3D geological model.

The Mine Tunnel

Mock-ups and simulators are well-established training tools in the mining skills development space. Many mines have mock-up stopes and tunnels to conduct basic mining training, including drilling, blasting, cleaning, and tramming activities. Mock-ups allow training to be conducted in a controlled, safe environment free from noise and other

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hazards found in the mining environment (Cirda Minerals Processing, 2007). Access to the mock-up is easy and not restricted by the constraints imposed by the production environment. Certain specialist contractors have built dedicated mock-ups specifically for their types of mining activities. An example of this is the shaft-sinking mock-up at Bentley Park (Carletonville) built by Murray & Roberts Cementation Training Academy (Murray & Roberts Cementation, 2007).

The Wits Mine Tunnel forms part of the Wits DigiMine Project (University of the Witwatersrand. 2015). Portions of the Chamber of Mines building on the West Campus have been converted into a 'mine'; with the roof of the building representing the 'surface', the stairwell representing the 'shaft', with the mock mine stope, lamp room, rescue chamber and tunnel located in the basement. The facilities cost approximately R15 million and were built with sponsorship from Gold Fields, New Concept Mining, and Sibanye (University of the Witwatersrand, 2015). The newly formed Wits Mining Institute is conducting extensive research in the mock-ups, including developing advanced software linking the sensors in the stope and tunnel to high-technology infrastructure, including a control room that monitors real-time ground movement, underground airflow modelling, and personnel access control and tracking (Engineering News, 2016).

The mock-up of the tunnel formed the first phase of the project (University of the Witwatersrand, 2015). This tunnel was established as a way to facilitate teaching and learning. The tunnel is 70 m long and construction took place in 2012/2013 (Figure 1).

As part of the teaching and learning functions of the tunnel, a decision was made to simulate geology onto the walls of the tunnel. This included moulding the walls to simulate rock and painting the walls to represent various lithologies. A dyke and a fault are also included in the simulation (Figures 2 and 3). The portrayal of the rock types and structures was conducted with assistance from the School of Geosciences at Wits. The bulk of the tunnel has sets installed and additional shotcrete is applied. Geological mapping in this portion is not possible. The sets were installed to investigate the propagation of radio waves through this type of tunnel support as part of the DigiMine project.

Figure 1—Construction of the Wits Mining Tunnel in 2013

The design portrays the rock types found in the Upper Critical Zone of the Bushveld Complex. These include dark grey/black to represent the melanocratic chromite-rich layers, light blue-grey to represent the leucocratic layers like anorthosite, and browns to represent the pyroxenite and norite layers. A fault is visible in two portions of the tunnel (the main portion as well as a spur). A dyke is also visible and is painted black.

There is no intersection visible between the fault and the dyke (it would occur in the portion of the tunnel where sets are installed). The age relationship between the dyke and the fault is thus open to interpretation and can really be determined only when the borehole information is included in the analysis. Students need to interpret the movement on the fault and determine if it is a normal or a reverse fault. Most students interpret the fault in the main portion of the tunnel correctly as a reverse fault and then get confused because in the spur it appears to be a normal fault. It is only after detailed inspection of the colours of the layer do they realize the one layer is darker than the other, and then they correctly interpret it as the same reverse fault as observed in the main portion of the tunnel.

Figure 2—Moulding the tunnel wall to simulate exposed rock face in the Wits Mine Tunnel (2013)

Figure 3—Painting the different rock types and structures in the Wits Mine Tunnel (2013)

Geological modelling

A geological model is a computerized representation of lithological, structural, geochemical, geophysical, and diamond drill-hole data on and below the Earth's surface (Fallara, Legault, and Rabeau, 2006). Geological models for sub-surface interpretation are generally based on limited data. They simplify complex environments and give scientists the ability to visualize interactively the interpretation of the subsurface geology. The application of geological models to mining enables the prediction of the presence, tenor, and spatial position of potentially economic mineralization. These predictions involve risk and require decision-making (Hodkiewicz, 2012).

An understanding of the potential, as well as limitations, of geological models is considered important in the Advanced Mineral Resource Management programme and thus the Geological Modelling module was created in 2012. Suitable software was required for the practical component of the module and Leapfrog Geo agreed to provide the software, as well as training for the delegates in the basic use of the program.

Traditional models (also referred to as explicit models) are created using wireframes based on the geological logging of boreholes. This type of modelling is time-consuming and a good understanding of the geology is needed for it to be effective. In this type of model, strings link the borehole intersections with the orebody. Implicit geological modelling, on the other hand, is a technique that uses a radial basis function to establish and update geological models relatively quickly and efficiently from borehole data, outcrop data, manually interpreted vertical or horizontal sections, and structural data. The radial basis function allows scattered 3D data-points to be described by a single mathematical function. Models can be isotropic, meaning without any trends or anisotropic, where there is a distinctive directional bias. Once anisotropy is identified from the data, this may be introduced into the modelling process. The implicit modelling process can also automatically split the model into fault blocks with the fault positions included in the data. Assays and any coded drill-hole data, such as lithology and alteration, can be interpolated (Hodkiewicz, 2012). Leapfrog Geo software is an example of this new approach to geological modelling. Implicit modelling is gaining acceptance in the industry, although it is important not to treat the software as a 'black box' and to constantly review and manually edit the computer-generated model (Birch, 2014).

Geological Modelling module

The Advanced MRM Geological Modelling (MRM 5) (Wits Enterprise, 2017) course commences with the students examining the geological information that is available for use in compiling the computer-generated geological models. Traditional methods of geological modelling are considered, and the advantages of using computer-generated geological models are debated.

The aim of this course is to:

- Provide a theoretical basis for ensuring that the geological orebody model used is representative of the orebody from which it was derived
- Ensure congruency between all types of geoscientific information used in the compilation of the geological orebody model
- Understand the geological modelling work flow
- Highlight and emphasize the importance of the orebody model and the role of geological input in maintaining the credibility of the model
- Link the geological model back to the South African Code for the Reporting of Exploration Results, Mineral Resources and Mineral Reserves (the SAMREC Code) and understand the importance of compliant reporting.

The assessment of the module is weighted 70% for the examinaton and 30% for the assignment. The assignment is conducted in two phases. The first phase is a practical geological mapping exercise conducted by the students in the Wits Mine Tunnel (Figure 4). The second phase is done by the students in their own time and requires the validation and correction of the supplied borehole database, selecting an appropriate modelling method using Leapfrog Geo (Leapfrog, 2017b), and linking the 3D model derived from the borehole data with the observations from the tunnel. The students are also required to link the simulated lithologies in the tunnel and borehole database with the types of lithologies that one would expect in the Upper Critical Zone of the Rustenburg Layered Suite in both the eastern and western limbs of the Bushveld Complex (Figure 5). The simulated stratigraphy presented in the borehole database and tunnel has been designated as the UG1 and UG2 layers. They are, however, a very loose representation of how these particular layers would appear realistically (for instance, the stratigraphic separation between the UG1 and UG2 is far greater than is depicted in the tunnel.

The Wits Mine Tunnel has been the venue for the mapping component for the 2014, 2016, and 2017 Geological Modelling module (the course was not run in 2015). Prior to 2014, a generic model representing Merensky Reef stratigraphy was utilized, where the students were supplied with a borehole database as well as a surface map. There was no practical mapping exercise.

Geological modelling assignment

Mapping exercise

Students are expected to record the relationships between the stratigraphic layering, dyke, and fault and gather sufficient

Figure 4—2016 class mapping the tunnel. The dyke can be clearly observed in the photograph, as well as the layers designated as UG2 H/W, UG2, and UG2 F/W

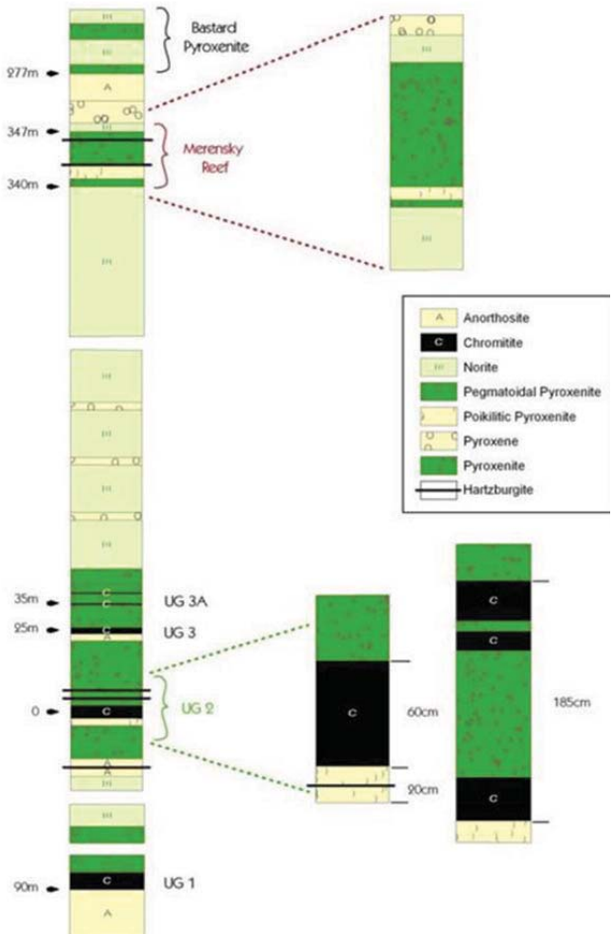

Figure 5—Stratigraphy of the Upper Critical Zone of the Rustenburg Layered Suite, Bushveld Complex and associated lithologies (Deloitte, 2009). The lithologies depicted in the tunnel have been designated as the UG1 and UG2 portions

information to ensure that the computer-generated model derived from the supplied borehole data is consistent with what can be observed in the tunnel. Sketches, as well as photographs of the various features, are required to be included in the assignment report. A 60 m soft tape is used as a reference and a clinorule is supplied to measure dip angles. For many of the students on the module, this is their first experience in recording geological information. Most of the students have experience of the underground environment and are familiar with the concepts of dip and strike. Discussions about concepts like true stratigraphic thickness and apparent dip take place during the mapping exercise.

Modelling exercise

The practical assignment is based on a borehole data-set of simulated Upper Critical Zone data supplied by Leapfrog South Africa. The database was created specifically for the module and includes 24 boreholes. The simulated stratigraphy includes:

- UG2 hangingwall
- UG2 A
- UG2 B
- UG2 footwall
- UG1 hangingwall
- UG1
- UG1 footwall.

Fault and dyke intersections are included in the borehole logs. Leapfrog South Africa also modified the data-set to include some obvious and some not so obvious data errors. The students are required to validate the data and produce a geological model. The Leapfrog Geo software has a built-in data validation tool which identifies missing information from the supplied borehole files (collars, survey, and geology). For example, one borehole does not have a collar file and thus cannot be located in 3D space. There are also inconsistencies between the maximum depths in the collar file and the geology file and overlaps between sections in the geology file. What the validation tool does not identify is that one of the collar files has an incorrect elevation and one survey file has an incorrect depth. A stratigraphic intersection thickness in one of the borehole files is very unrealistic compared to the same layer in the other boreholes. Effects like faults causing duplication of strata are covered by the course, and so the relationship between this abnormal thickness and known faulting is discussed. These types of errors can be identified only by visualizing the borehole traces in 3D space.

A large portion of the focus of the course is related to validating databases and ensuring only clean data is used for creating the geological models. Errors in data, as well as identifying poor quality sampling and reviewing the QA/QC processes, are extensively covered during the theoretical portion of the course. Thus the students are required to keep detailed notes during their data validation exercise, report on all the errors identified, and explain how these were rectified.

The model can be created using two broad modelling techniques – modelling the stratigraphic contact surfaces individually or modelling the stratigraphic units as a single set. Students are encouraged to experiment with these methods as well as to try different age relationships between the stratigraphic units, fault, and dyke. Ultimately, the aim is to create a model that was broadly consistent with the observations made in the tunnel (Figures 6 and 7).

Finally, students are required to make manual adjustments to the lithological contacts to concur with the observations made during the mapping exercise. The two obvious corrections that are required are shown in Figures 8 and 9.

The students are required to save their final models in the Leapfrog Visualizer format and send this together with their reports for assessment. The models are assessed for accuracy compared to the observations in the tunnel, as well to ensure that all the database errors were noted and corrected. The age relationship between the stratigraphy, dyke, and fault is also considered.

Observations

The use of the Wits Mine Tunnel allows the students to do their own geological mapping. Prior to 2014 there was no practical mapping component for this particular course. The tunnel has allowed students to gain a greater appreciation of how geologists do geological mapping and gather the

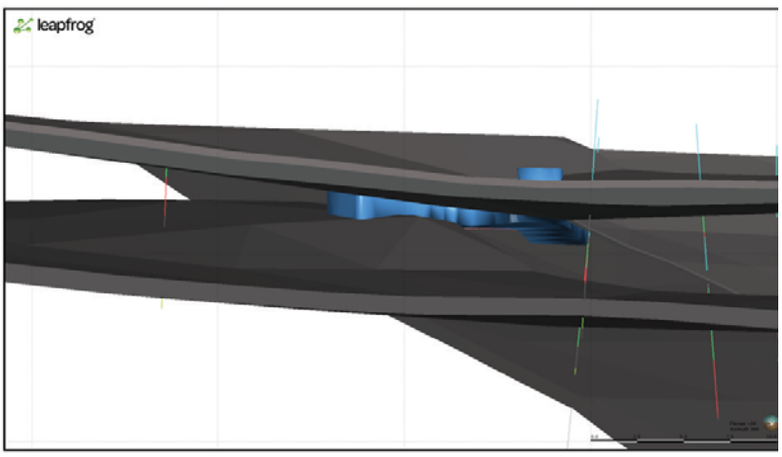

Figure 6—Leapfrog Geo model of the Wits Mine Tunnel showing UG1 and UG2 stratigraphy (2017)

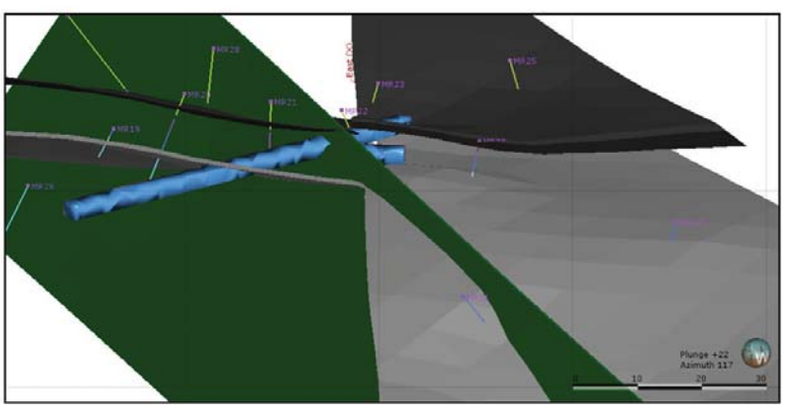

Figure 7—Leapfrog Geo model of the Wits Mine Tunnel showing UG1 and UG2 stratigraphy as well as the fault and dyke relationship (2017). The near-vertical plane shown in the figure is the dyke. For clarity, the fault plane is not shown; however, the position is clearly where the UG1 and UG2 stratigraphy is cut off

Figure 8—This figure shows the fault intersected in the tunnel spur. The dark coloured lithology on the left is the UG2, while the dark coloured lithology in the centre is the UG1. If the model is not manally corrected, the UG1 does not intersect the spur (as found in the tunnel) but is rendered deeper in the footwall (2017)

Figure 9—This figure shows the face of the main portion of the tunnel. The dark lithology observed is the UG2. If the model is not manually corrected, the UG2 is rendered above the face of the tunnel (2017)

information required to create valid geological models. The implicit modelling method utilized by Leapfrog Geo is fast and simple, but even a simple model like the one created by the students for this exercise does not represent the identified geology perfectly. There is thus a greater appreciation of the limitations of this type of modelling and the need for manual

Table I

Summary of assignment and overall performance for 2014, 2016, and 2017

Year	No. of students	No. who passed the assignment	Percentage who passed the assignment	No. who passed the course
2014	14	11	79%	12
2016	8	7	88%	6
2017	9	6	67%	5

interventions to force the model to honour the observations made in the tunnel.

Generally, the models received from the students are broadly correct (see Table I for a summary of performance). Some students failed to identify all the errors in the database, leading to some strange elements in their submitted models. In the 2017 class, all the students failed to adjust their models adequately to honour their mapping in the tunnel, while most of the students in the 2014 and 2016 classes did so. Changes to the Leapfrog Geo software have made this type of manipulation a multi-step process and this appears to have confused the students.

The feedback from the students based on their course review has indicated that the best part of the course was the practical modelling using the Leapfrog Geo software. Some students specifically commented on the tunnel mapping exercise as an enjoyable and valuable aspect of the course.

The pass rate for the Geological Modelling assignment ranges from 67% to 88% for the period that the tunnel has been the assignment topic. In 2014, fewer students passed the assignment than passed the course overall. This trend reversed in 2016 and 2017, where more students passed the assignment than the overall course.

Conclusions

The Wits Mine Tunnel was created in 2012/2013 to allow research and teaching to take place in a simulated underground environment in the School of Mining Engineering at the University of the Witwatersrand. The tunnel is part of a complex that includes a mine stope, rescue chamber, and lamp room and is is linked to a control room that forms part of the DigiMine Project (University of the Witwatersrand, 2015). This tunnel has been used for the Advanced MRM Geological Modelling (MRM 5) course in 2014, 2016, and 2017. The students map the tunnel and the mapping is combined with a borehole database for the students' assignment. They are required to use Leapfrog Geo software to create a 3D geological model and ensure that it is an accurate representation of the geology observed in the tunnel.

Mock-ups of the mining environment are not unique to the University of the Witwatersrand and are found in training centres around the mining industry. There are, however, no other examples noted where the geological information presented in the mock-up has been linked to digital information. This linking has given the students the opportunity to gather their own geological information to understand the full process of creating geological models. Students have been very positive about the learning experience and this type of learning is considered very beneficial to them. The mock-up is accessible without any special logistical requirements, meaning it can be easily accessed and simulated underground teaching takes place in a well-ventilated, cool, and safe environment.

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Integrated phytomining and ethanol production in the Zambian Copperbelt to minimize mine decontamination costs and environmental and social impacts: a review

by T. Sinkala

Synopsis

The mining industry in the Zambian Copperbelt has polluted the environment with heavy metals, the effects of which are a source of concern to host communities. It is globally known that remediation of polluted mine environments is expensive, and can be as high as US$48 000 or more per hectare, depending on the severity of contamination, using traditional physical and chemical approaches. These methods also often leave significant liabilities for host communities. This paper reviews available opportunities for mining companies in the Zambian Copperbelt to use integrated phytomining and production of ethanol, and its co-products, to minimize the costs for remediating polluted mine environments. The benefits of using this approach are manifold and include additional income streams from extracted metals and ethanol, creation of additional jobs for mine host communities, assured livelihoods for mine host communities even beyond mine closure, reclaimed land for food production and other activities, and improved corporate image for mining companies.

Keywords

phytoremediation, rehabilitation costs, ethanol production.

Introduction

Mining and mineral beneficiation activities in the Zambian Copperbelt have contributed considerable amounts of heavy metals to the environment (Křibek *et al.*, 2013; Swedish Geological AB *et al.*, 2005). These heavy metal pollutants have affected surrounding land, air, and water quality, which has long been a concern to host communities.

The metal mines in the Copperbelt generally contain variable amounts of sulphide minerals, either in the ore or in the host rocks. When these sulphide ores are mined and processed, heavy metals are leached from tailings under moist conditions owing to the decreased pH resulting from the oxidation of pyrite. This low pH increases the solubility of most heavy metals, which are then dispersed in the surrounding environment. Traditional measures to prevent acid drainage include physically stabilizing the waste by covering the acid-producing material with water, or covering the surface of dry tailings with soil and then revegetating the site to control pyrite oxidation (Renault, Sailerova, and Fedikow, 2000).

Dispersion of heavy metals from mining and milling operations also occurs by wind, where metals are contained in windblown dust. The major source of soil contamination in areas surrounding the mining operations is in fact pollutants spread by wind. Distances of dispersion have been recorded as far as 250 km for manganese and 60 km for copper (Renault, Sailerova, and Fedikow, 2000), making remediation efforts even more difficult and expensive such that mining companies are facing the choice of either paying for a statutory environmental assurance fund or nefariously avoiding paying for the funds (Chifungula, 2014; DMP, 2016; EJA, 2016; LHR, 2017). When traditional physical and chemical remediation approaches are applied, costs for rehabilitation works depend on the topography, the classification of the mine according to the mineral mined, the risk class of the mine, its proximity to built-up or urban areas (DWE, 2015), and the severity and extent of contamination. Following are examples of remediation costs incurred in a few selected countries.

In Zambia, of the projected US$226.65 million that 49 mining companies should have contributed to the environmental rehabilitation assurance fund for the period covering 2009 to 2012, the companies contributed only US$50.40 million (US$10.37 million in cash and the rest in bank guarantees) (Chifungula, 2014). In Western Australia, the total rehabilitation and closure costs for all mines operating under the Mining Act 1978 was estimated in 2010 to be between A$4 billion (US$3.01 billion) and A$6 billion (US$4.51

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billion) (DMP, 2016). As of financial year 2015–2016, compliance brought the total principal in the rehabilitation fund to approximately A$60 million or US$45.10 million (DMP, 2016). In Zimbabwe, a study conducted in 2011 by the Zimbabwe Environmental Management Agency (EMA) on four large decommissioned mines revealed a cumulative rehabilitation cost of US$32 million (ZAMI, 2017). In South Africa there are about 6 000 derelict and ownerless mines which have never been rehabilitated; previous owners have simply 'locked the gate' and walked away. As an example of the environmental liability cost, closure of the De Groote Boom mining area in South Africa, with a maximum 5 ha where activities included landscaping a free-draining topography, replacement of soil, revegetation, and general surface rehabilitation of the disturbed area, was estimated at R2.15 million (US$0.24 million), or about US$48 000 per hectare (DWE, 2015).

The above pollution scenario and associated rehabilitation costs indicate that it cannot be left to nature to restore the environment to an acceptable quality. Rather, aggressive efforts based on well-planned commercially viable restoration programmes are required.

The objective of this paper is to look at available opportunities for mining companies in the Zambian Copperbelt to apply phytoremediation to minimize expenses for rehabilitation of polluted mine environments, while also realizing new income streams when suitable hyperaccumulator plants are used to recover valuable metals and also to produce ethanol and associated co-products from the plants.

The discussion starts by explaining phytoremediation and the current international interest in this field, followed by examples of hyperaccumulator plants and their potential use as a source of feedstock for biofuels (biogas, bioethanol, and biodiesel) production. Issues to consider when identifying appropriate heavy-metal remediation plants that can also be used for ethanol production are highlighted, together with the benefits of using hyperaccumulator plants for both remediation and ethanol production.

Phytoremediation

Phytoremediation, the *in situ* use of plants to extract heavy metals from contaminated sites (Anderson *et al*., 2000), is a topic being increasingly investigated for its potential to cost-effectively decontaminate polluted areas (Hunt *et al*., 2014). The research efforts include the identification of metallophyte and pseudometallophyte plant species that can colonize areas that have been highly polluted with heavy metals and metalloids by mining and related industrial activities (Favas *et al*., 2014).

Favas *et al*. (2014) presented six different strategies for phytoremediation techniques that can be applied singly or severally, depending on the chemical nature and properties of the contaminant and the plant characteristics. These comprise:

- **Phytodegradation**/phytotransformation—using plants with specific enzymes to degrade/transform organic contaminants
- **Phytostabilization**/phytoimmobilization—using plants that incorporate the contaminants into the lignin of the cell wall of root cells or into humus
- **Phytovolatilization**—using plants that absorb and volatilize certain metals/metalloids
- **Phytoextraction**/phytoaccumulation/phytoabsorption/phytosequestration—using plants that absorb contaminants through the roots, followed by translocation and accumulation in the aerial parts
- **Phytofiltration**—using plants that absorb, concentrate, and/or precipitate contaminants, particularly heavy metals or radioactive elements, from an aqueous medium through their root system or other submerged organs
- **Rhizodegradation**/phytostimulation—using plants whose growing roots promote the proliferation of contaminant-degrading microorganisms in the rhizosphere that utilize the plant's exudates and metabolites as a source of carbon and energy.

Of interest in this paper are phytoextraction strategies for phytoremediation of metal- and metalloid-contaminated soils because of the potential to extract valuable minerals (Anderson *et al*., 1999; Favas *et al*., 2014; Krisnayanti and Anderson, 2014; van der Ent *et al*., 2015b). An extension of 'phytoextraction' (removal from the soil) is 'phytomining' (accumulating economic metal values in plant biomass) (Chaney and Mahoney, 2014; Chaney and Baklanov, 2017), defined as the use of hyperaccumulating plants to extract metal from soil with recovery of the metal from the biomass to return an economic profit (Lamb, Anderson, and Haverkamp, 2001). The same plants can similarly be used for agromining. According to van der Ent *et al.* (2015a), phytomining takes place on degraded or mined land as part of a rehabilitation strategy, while agromining takes place on low-productivity agricultural soils to generate economic returns to farmers. In this paper, both these activities are regarded as necessary to address environmental and socio-economic concerns in mining areas. There is therefore intensified research into plants which remove metals from soils in significant amounts, after which the valuable metals can be economically recovered from the plants. Over time, land is made available for other socio-economic uses once pollution in the soil has been reduced to acceptable levels.

Hyperaccumulator plants

Hyperaccumulators are plants that have the ability to store high concentrations of specific metals in their aerial parts. Hunt *et al.* (2014) cited van der Ent *et al.* (2013) on the current definition for a hyperaccumulator, which they summarized, for nickel, as a species that when growing in its natural environment accumulates at least 1 000 mg of nickel per kilogram (dry weight) within its leaves. They also report that hyperaccumulator species that concentrate other elements, including zinc (Zn), cadmium (Cd), lead (Pb), cobalt (Co), manganese (Mn), chromium (Cr), and selenium (Se), have been identified, and hyperaccumulation threshold limits have been established for each metal. They give examples of accumulation thresholds of 100 mg/kg for Cd, 1 000 mg/kg for Pb, and 10 000 mg/kg for Mn.

Examples of hyperaccumulators mentioned by Favas *et al*. (2014) include *Elsholtzia splendens, Alyssum bertolonii, Thlaspi caerulescens*, and *Pteris vittata*, which hyperaccumulate copper (Cu), nickel (Ni), Zn/Cd, and arsenic

(As), respectively. Hyperaccumulators include both terrestrial and aquatic plants. Examples of aquatic plants include *Centella asiatica* and *Eichhornia crassipes* (Mokhtar *et al.*, 2011). Aquatic plants are reported to be capable of bio-accumulating toxic metals and nutrients in large quantities in comparison to terrestrial plants (Wani *et al.*, 2017).

One of the concerns with phytomining is the low yield of metals per unit area in relation to the investment required to extract them (Hunt *et al.*, 2014; Mohanty, 2016). The low-grade biomass harvested from the plants may not necessarily be economically treatable by conventional mineral processing approaches.

Hunt *et al.* (2014) clearly stated that any application of phytoextraction requires plants with attributes that include fast growth rates, high biomass composition, deep roots, tolerance to metal uptake, metal specificity, and a high rate of metal transport from roots to shoots. They also stated that, when applied to metal recovery from wastes, phytoextraction can potentially result in environmental clean-up and the recovery of valuable metal products. To make the economics more attractive, however, it is considered that phytoremediation needs to be combined with other technologies, such as biofuel production (Mohanty, 2016).

Bioremediation plants as a source of feedstock for biofuels production

The biomass generated by remediation plants has potential to be a source of feedstock to produce biofuels (Hunt *et al.*, 2014; Mohanty, 2016; Warr, Kasonde, and Krishan, 2017), which can be gaseous (*e.g.* methane), liquid (*e.g.* ethanol and biodiesel), or solid (*e.g.* charcoal). For example, in research being conducted at Freiburg University in Germany, plants that accumulate germanium are harvested and fermented to produce biogas, after which the germanium is extracted (Kratochwill, 2015).

Hunt *et al.* (2014) have pointed out that biomass containing Cu (as a catalyst) has proven useful in the improvement of bio-oil quality produced through fast pyrolysis of biomass. The Cu in Cu-enriched biomass effectively catalyses the thermo-decomposition of the biomass and results in an improvement in the yield and heating value of the bio-oil compared with non-Cu containing biomass. Cu did not volatilize during treatment, which prevented metal contamination of the bio-oil. Hunt *et al.* (2014) also reported that research is being carried out into the use of Se-containing plants for applications such as fortified foods, biofuels, or potential bioherbicides and green fertilizers.

For the production of biofuels, care must be taken regarding the suitability of feedstock in relation to the metals being phytomined. In this respect, Gramss and Voigt (2016) studied the gradual accumulation of heavy metals in an industrial wheat crop grown on soil at a former uranium mine in East Germany, and the potential use of the herbage. They gave a caution regarding the use of grains, rather than of straw, with Cd and Cu concentrations above 3 and 12 mg/kg dry weight, respectively, as bioethanol feedstock. They found that Cd and Cu toxicities could lead to productivity losses in the fermentation of alcohol by *S. cerevisiae*, whereas a higher Mn, Ni, Pb, and Zn load could be tolerated by the yeast and be accepted according to forage hygiene guidelines. They observed that if As, Mn, Pb, and uranium (U) contents increased in the straw, the straw could still be used both as a roughage supplement for livestock and as a bioethanol feedstock.

Appropriate heavy-metal remediation plants for ethanol production

Ethanol is commonly produced from biomass feedstocks by a fermentation process. The feedstocks are sugar-based crops (such as sweet sorghum, sugarcane, and sugar beet) and starch-based feedstocks (such as cassava, sweet potatoes, and corn). Recently, technology has been developed to produce ethanol commercially also from cellulose (stringy fibre of a plant) derived from biomass, such as grasses, wood, and bagasse (Verardi *et al.*, 2012). Feedstocks for ethanol production via both common and advanced technologies are harvested from large areas and brought to a processing plant where ethanol is distilled from fermented material (or mash) at temperatures below 100°C (usually around 93 to 96°C). The material that remains after distillation is called vinasse. With reference to our subject, heavy metals would therefore be found in vinasse if the fermented material used to produce ethanol contained heavy metals from hyperaccumalator plants.

Suitability of traditional feedstocks for both remediation and ethanol production

The organic material that remains after ethanol disrillation, called vinasse, is where any metals derived from the harvested starch/sugar feedstocks can be found. However, as can be observed in Table I, the heavy metal content in vinasse derived from traditional ethanol feedstocks grown in non-mining environments is generally very low (Gamboa *et al.*, 2011; Rodrigues and Hu, 2017; Scull *et al.*, 2012).

Izah, Bassey, and Ohimain (2017) assessed the level of some selected heavy metals in cassava mill effluent-contaminated soil from the Ndemili community in the Niger Delta region of Nigeria. Results for both spatial and bimonthly distribution of heavy metals yielded for Cu 1.10–6.83 mg/kg, Zn 14.32–46.15 mg/kg, Mn 18.42–47.49 mg/kg, iron (Fe) 1303.29–4934.04 mg/kg, Pb 1.33–9.42 mg/kg, Cd < 0.001–0.24 mg/kg, Cr 0.19–3.41 mg/kg, Ni 1.57–3.76 mg/kg, and Co < 0.008–9.71 mg/kg. Levels for Fe appear relatively high, but unfortunately the authors did not report the metal elevations in the cassava tubers, which are the source of first-generation ethanol.

Wang *et al.* (2016) investigated the suitability of sugarcane as a crop for phytoremediation in a heavy-metal polluted farmland in Huanjiang County in the Huanjiang River Basin in Guangxi Province, southern China, where sugarcane is widely cultivated as one of the major economic crops. Mining activities in the area have resulted in heavy metals polluting farmland soils. Table II shows the heavy metal concentrations found in agricultural soils and the roots, upper stems, and leaves of sugarcane in the study area. It can be observed that most of the toxic heavy-metals intake by sugarcane accumulated in the roots, while only a small portion was transferred to the stems and leaves. Thus, if sugarcane were to be used for phytomining, harvesting would imply uprooting the entire plant. This indicates that

Table I

Levels of heavy metals in vinasse from the ethanol distillation process using traditional ethanol feedstocks grown in non-mining areas

Metal	Feedstock						
	Sugarcane feedstock[a] (mg/L)	Sugarcane feedstock[b]	Cane molasses[c] (mg/L)	Grapes – wine[c] (mg/L) (mg/L)	Agave – tequila[c] (mg/L)	Sweet sorghum[c] (mg/L)	Beet molasses[c] (mg/L)
Fe	44.9	0.07%	12.8–157.5	0.001–0.077	35.2–45	317	203–226
Mn	4.9						
Zn	1.2	0.04%					
Ba	0.54						
Cd	1.06		0.04–1.36	0.05–0.08	0.01–0.2		<1
Cr	0.15						
Ni	0.26						
Al	72.5						
Cu	0.06	6.60 ppm	0.27–1.71	0.2–3.26	0.36–4	37	2.1–5
Pb		12.90 ppm	0.02–0.48	0.55–1.34	0.065–0.5		<5
Co		6.49 ppm					

a - Rodrigues and Hu (2017)
b - Scull *et al.* (2012)
c - Gamboa *et al.* (2011)

Table II

Heavy metal concentrations in soils and sugarcane in Huanjiang County, China

Metal	Soil (mg/kg)	Sugarcane roots (mg/kg)	Sugarcane upper stems (mg/kg)	Sugarcane leaves (mg/kg)
Cu	33.054	9.3	3.22	3.13
Zn	707.95	197	87.61	42.30
Pb	929.25	161.1	2.95	6.89
Cd	1.15	0.6	0.05	0.029
As	60.89	10.3	0.05	0.12
Cr	47.43	94.5	2.68	4.53
Ni	16.64	4.8	0.44	2.90

the location of heavy metals in the part (roots, stems, leaves) of the plant, or in general the biomass distribution in the plant, determines how the plant will be managed and consequently the commercial viability of phytomining using such a plant.

There is therefore a need to look at plants that can extract and accumulate significant amounts of valuable metals to make phytomining economically viable. The use of phytoaccumator plants that can also be applied to produce biofuels will create another income stream which can contribute to the reduction of remediation costs, because of the revenues from biofuels, even if the recoverable valuable metals from the plants are not in economically viable amounts. Furthermore, land will be made available for general agricultural use as the heavy metals in the soils are progressively reduced to acceptable levels (van der Ent. *et al*, 2015a).

Growing periods for sugar- and starch-based crops differ. For example, the period is 9–18 months for sugarcane, 4–4.3 months for sweet sorghum, 10–16 months for cassava, and about 6 months for sweet potatoes (Sinkala, Timilsina and Ekanayake, 2013), during which time the crops accumulate metals according to their respective abilities. Sugarcane and sweet sorghum, following the first harvest, are thereafter harvested from ratoons (Sinkala, Timilsina, and Ekanayake, 2013; Verheye, 2017). Harvesting from ratoons can be after 7–9 months from the virgin crop harvest (Verheye, 2017). Furthermore, the cost range for replanting sugarcane is about US$1 135–US$2 530 per hectare, whereas for managing a ratoon (or stubble) crop it is much cheaper at US$321–US$633 per hectare (PECEGE and CNA, 2016; Deliberto and Salassi, 2015). Thus, uprooting a sugarcane crop would be a relatively expensive operation as sugarcane would have to be replanted every year or after every harvest, as opposed to harvesting from cheaper ratoon/stubble crop.

Hyperaccumulator terrestrial plants

A study by van der Ent *et al.* (2015c) revealed that there are more than 30 Cu-Co hyperaccumulator plants (see examples in Figure 1) in the copper-cobalt belt of the Democratic Republic of Congo and Zambia that accumulate extraordinarily high concentrations of Cu and Co metal in their living tissues. They pointed out that such plants can be grown and harvested to remove Cu-Co from (polluted) soils, thus serving to remediate contaminated soils, for example around smelters (phytoextraction), or to create a 'metal-

Figure 1—Some Cu-Co hyperaccumulator plants found in the copper-cobalt belt of the Democratic Republic of Congo and Zambia

Table III

Highest detected metal contents from preliminary XRF analysis of plant material from the Zambian Copperbelt, (mg/kg)

Family	Name	Se	As	Zn	Cu	Co
Cyperaceae	Cyperus dives	128				
Pteridaceae	Pteris vittata L.		259			
Orobanchaceae	Alectra sessiliflora (Vahl) Kuntze			918		
Apiceae	cf. Diplolophium sp.				3 038	
Amaranthaceae	Celosia trigyna L.					1 060

Compiled from van der Ent *et al.*, 2015c

Table IV

Highest detected metal contents from preliminary soil analysis by DTPA extraction (indicative of plant-available metal concentrations), mg/kg

Type	Location/species	Co	Cu	Fe	Mn	Ni	Zn
Rhizosphere	Persicaria capitata	307	958	30		3.6	
Topsoil	Tailings (at Luanshya)				14		
Rhizosphere	Persicaria puncata						18

Compiled from van der Ent *et al.*, 2015c

enriched crop' (phytomining). The project aimed to elucidate metal speciation and elemental distribution in selected Cu-Co hyperaccumulators with high potential for phytoextraction, for which 200 plant specimens, 25 soil samples, and 10 mineral samples were collected for chemical analysis.

Table III indicates the highest detected metal contents from preliminary X-ray fluorescence (XRF) analysis of plant material from the Zambian Copperbelt, while Table IV indicates the highest detected metal contents from soils collected in the same area and extracted with diethylene triamine pentaacetic acid (DTPA). It can be observed that the levels of metal in both plants and soil are significantly higher than those reported in Table II for the mining area in China. However, using these plants to produce ethanol may require advanced biofuels technologies.

Kříbek *et al.* (2013) studied the content of metals and As in the leaves and tubers of cassava (*Manihot esculenta*) and sweet potatoes (*Ipomoea batatas*) growing on uncontaminated and contaminated soils of the Zambian Copperbelt. They found that the order in which metal concentration increased in different plant organs was tuber, root, stem, leaf stalk, and leaf. The contents of Cu in cassava and sweet potato leaves growing on contaminated soils were as high as 612 mg/kg total dry weight (dw) and 377 mg/kg

dw, respectively. The contents of Cu in leaves of both plants growing on uncontaminated soils were much lower (up to 252 and 198 mg/kg dw, respectively). The contents of Co, As, and Zn in leaves of cassava and sweet potatoes growing on contaminated soils were higher than in uncontaminated areas, while the contents of Pb did not differ significantly.

The highest detected copper content of 612 mg/kg in cassava leaves growing in Copperbelt soils (Kříbek *et al.* 2013) translates to 0.0612% grade (or 0.612 kg/t), while the highest copper content of 958 mg/kg detected in Persicaria capitata (van der Ent *et al.* 2015b) translates to 0.0958% grade (or 0.958 kg/t). The highest Cu content of 3 038 mg/kg detected by van der Ent *et al.* (2015b) in Apiceae cf. *Diplolophium* sp, and the 1 060 mg/kg Co in Amaranthaceae *Celosia trigyna L.*, translate to 0.3058% grade (or 3.058 kg/t) Cu and 0.1068% grade (or 1.068 kg/t) Co, respectively. Compare these to prevailing prices of US$7 100 per ton for Cu and US$75 500 per ton for Co (LME, 2018), and to published copper ore grades for surface mines in Zambia of 0.46% for Barrick's Lumwana mine (Barrick, 2014) and 0.5% for First Quantum's Kansanshi mine (FQML, 2017). Furthermore, for Zambia there are reasonable margins between production costs of ethanol and the pump price of petrol (gasoline). Ethanol is the equivalent of petrol and can be used either 100% or in any ethanol/petrol proportion in a flexible fuel engine, or can be blended with petrol in appropriate petrol engines (Barros, 2010). Production costs of ethanol range from about US$0.4 to US$0.7 per litre (PECEGE and CNA, 2016; Sinkala, Timilsina and Ekanayake, 2013), depending on the feedstock and refinery technology used, while the petrol price at the pump in Zambia is currently US$1.375 per litre. Clearly, the above metal grades concentrated by plants, and the ethanol production margins, appear to be sufficiently attractive to warrant feasibility studies on the possible implementation of integrated agro/phytomining with bioethanol production in viable areas.

Hyperaccumulator aquatic plants

There is also interest in appropriate aquatic hyperaccumulator plants that are technically and economically viable for both phytoremediation and ethanol production. Among the plants of interest that can be used to accumulate heavy metals and also to produce first-generation ethanol are cattails, or *Typha latifolia* L (see Figure 2). Cattails are herbaceous, rhizomatous perennial plants with long, slender green stalks topped with brown, fluffy, sausage-shaped flowering heads that are often found in marshlands where they extract dissolved nutrients out of water, leaving the marsh relatively clean (Acres USA, 2008). The plants are about 1.5–3.0 m tall, and are fairly high in starch content, usually about 30–46% (USDA, 2006). Cattails grown under marsh conditions using sewage yield 7 500 gallons of alcohol per acre (70 155 litres of alcohol per hectare) (Acres USA, 2008). The average yield of cattails from constructed wetlands is reported to be 16.1 t/ha, with a maximum of 42.7 t/ha (Suda, Shahbazi and Li, 2007).

The cattail core can be ground into flour (see Figure 2c), with one acre of the plants yielding about 6475 pounds (16 t) of flour per hectare (USDA, 2006). Projectgaia (2015) has quoted ethanol yields of 10 051 L/ha for wild cattail, 23 375 L/ha when produced from cattail starch only, and 93 500 L/ha for cellulose from cattails grown in sewage. Indicative levels of heavy metals accumulating in this plant can be seen in Table V.

(a) Cattail (Typha latifolia L.) aquatic plant (www.publicdomainpictures.net). *(b) Cattail (Typha latifolia L.) roots (https://i.pinimg.com).* *(c) Cattail (Typha latifolia L.) flour and fibre (https://www.tapatalk.com).*

Figure 2—Examples of aquatic hyperaccumulator plants

Table V

Heavy metal accumulation in various parts of cattail (*Typha latifolia L*) (mg/kg)

Data source and plant location	Cu	Co	Fe	Zn	Ni	Cr	Pb	Sn	As	Mn
Substrate[a]	1 156.7			1 231.7	296.7					
Root[a]	93.3			391.7	55.0					
Stem/leaves[a]	15.0			60.8	27.5					
Tissue[b]	48.0	7.7	341.0	1 412.0	524.0	15.0	27.0	138.0	157.0	3 161.0
Sediment[c]	3 738.0			30 09.0	9 372.0	92.1	5 686.0			
Root[c]	50.0			946.0	55.0	44.0	1 108.0			
Leaf[c]	30.0			215.0	40.0	21.0	40.0			

a – Manios *et al.*. (2003)
b - Hussain *et al.*. (2014)
c – Beisner *et al.*. (2014)

For every ton of combined roots and shoots (stems and leaves) of cattails harvested in ratios of about 86.15:14.85 respectively, the 108.3 (93.3 + 15) mg/kg of Cu obtained by Manios *et al.* (2003) would yield 0.1083 kg of Cu. At the time of writing this paper, literature quoting heavy metal tolerance levels for cattail could not be found. Such levels would have helped to determine the maximum amount of metals that can be extracted from this plant.

Benefits of using hyperaccumulator plants for both remediation and ethanol production

The above review of the literature shows that there are plants that can be used both to remediate mine environments polluted with heavy metals and to produce biofuels. Terrestrial plants such as Apiceae cf. *Diplolophium* sp identified in the Zambian Copperbelt and aquatic plants such as *Typha latifolia* L. (or cattail) that are found in the country (Catarino and Martins, 2010) are very promising. Below are examples of conceptual major socio-economic benefits that can be realized from a combined phytomining and biofuels/ethanol production approach:

- Unlike other methods, where money is spent to remediate contaminated mining areas, the use of appropriate plants for phytoremediation would instead generate income streams through reclaimed valuable heavy metals and production of ethanol and associated co-products from the harvested plants.
- Mining communities can be engaged in the agromining and ethanol production value chains, thus creating jobs for the communities.
- After mine closure and remediation of mine areas, ethanol production and its value chain activities can continue to provide livelihoods for host communities, thus diverting the communities from engaging in resource-degrading activities for livelihoods.
- The remediated land can be used for the production of food crops and other socio-economic activities, in addition to continued ethanol production.
- Mining companies would overall improve their corporate social responsibility image.

Countries engage in development of biofuels industries for a variety of reasons. Reasons often reported include energy security, job creation, rural industrialization, retention of wealth created from national resources, enhancing food security, and minimizing impacts of climate change (Filho, 2017; Hodur and Leistritz, 2009; Silalertruksa *et al.* 2012; Sinkala, Timilsina, and Ekanayake, 2013; IRENA, 2018).

Below are a some examples from a few countries, illustrating the benefits that would accrue to the Zambian Copperbelt due to the biofuels industry alone, if successfully implemented:

Economic contribution

In **Brazil**, the sugarcane sector contributes about US$43.8 billion to country's gross domestic product (GDP) – equivalent to almost 2% of the entire Brazilian economy. When various suppliers and stakeholders who depend on Brazil's sugarcane industry are added, the entire sugarcane agro-industrial system generates gross revenues totaling more than US$87 billion annually, to which ethanol contributes about US$12.42 billion (Filho, 2017; Neves *et al.*, 2008).

In a projection to the year 2022, in **Thailand**, the biofuels sector's contribution to the GDP is around US$150 million, with savings in imported goods worth US$2.547 billion compared to petroleum fuels (Silalertruksa *et al.*, 2012).

Sweden, which currently produces 55 ML of ethanol and 403 ML of biodiesel per annum (Ekbom, 2018), has made foreign exchange savings of more than US$600 million. Of the biodiesel, 100 ML per year are produced from wood pulp.

In the **USA**, a study by Hodur and Leistritz (2009) on the economic impacts of biofuel development revealed that an investment of US$176.5 million into the production of 190 ML of ethanol per year in North Dakota resulted in an annual operating expenditure of US$74.6 million. Of this, US$53 million constituted payments to North Dakota entities, of which the largest expenditure item by far (US$36million, or 68%) went to feedstock purchases and related supply logistics. In the Zambian context, feedstocks are expected to be produced largely by smallholder farmers, who would thus be among the major beneficiaries.

In 2017, the ethanol industry contributed more than US$24 billion to the US economy. The most significant impact of the ethanol industry is increased income to farmers who benefit from the demand for feedstock, which leads to both increased production and increased prices as well as earnings from locally-owned ethanol plants (Urbanchuk, 2018). This, again, illustrates the likely benefits that would accrue to smallholder farmers engaged in feedstock production.

In 2013, production of 6.4 billion litres of biodiesel contributed US$16.8 billion to the US economy (NBB, 2018). Use of 4.3 billion litres of biodiesel was estimated to lower greenhouse gas emissions by nearly 10 million metric tons of CO_2 equivalent.

Job creation

In **Brazil**, the sugarcane industry employs 1.09 million workers, according to 2011 data from the Ministry of Labor and Employment's Annual Report of Social Information (Filho, 2017). Salaries for sugarcane industry workers are among the highest in Brazil's agricultural sector, second only to wages in the soybean industry. For example, in 2008, sugarcane workers employed in Brazil's South-Central region (the country's main cane-producing zone) earned an average monthly income of R$1 062.55 (US$487.41), while in the North-Northeast region the average was R$666.20 (US$305.60). For context, the national average monthly salary amounted to R$942.02 (US$432.12) that year, and the minimum was R$415.00 (US$190.37). This is indicative of the reasonable incomes for the labour force that would be engaged in the biofuels sector in the Copperbelt.

In **Thailand**, a study by Silalertruksa *et al.* (2012) showed that producing ethanol and biodiesel requires about 17–20 more workers than gasoline, and 10 times the number for diesel, as per equivalent energy content, and that direct employment in agriculture contributes to more than 90% of total employment. In a projection to the year 2022, the estimated employment generation was found to be between around 238 700 and 382 400 person-years. Thus, the

Zambian Copperbelt Province, which has the largest percentage (29%) of youth unemployment, would benefit from the large labour requirements in the biofuels industry (CSO, 2016).

In **Sweden**, a study in 1998 found that each extra TWh of bioenergy use resulted in 300 extra jobs in the whole supply chain. The study did not include employment in construction, *e.g.* of heat plants. It was also found that one big advantage with bioenergy was that the employment was spread over the whole country, which is of great significance to the smaller communities and to rural development (Anderson, 2015). This scenario is in support of expectations from the success of the concept presented in this paper.

In the **USA**, when the direct, indirect, and induced jobs supported by ethanol production, construction activity, agriculture, exports, and R&D are included, the ethanol industry contributed nearly 360 000 jobs in 2017 (Urbanchuk, 2018). In 2012, the total employment in the biodiesel industry in the USA stood at nearly 47 000 jobs, with more than US$2.6 billion in wages paid (LMC International, 2013). Employment has since risen to 64 000 jobs in 2018 (NBB, 2018).

Clearly, there are demonstrable benefits of a biofuels industry if well implemented. Phytomining, on the other hand, is yet to be commercially implemented. To deploy the concept presented here, van der Ent *et al.* (2015a) proposed 'agromining' (a variant of phytomining). This is a type of agriculture to be carried out on degraded lands, where farming would not be for food crops but for valuable metals. They also presented a demonstration of how this can be carried out, as shown in Figure 3. The production sequence proposed in this paper is that processing of metals would be done after bioethanol has been produced from fermented material (Peters and Stojcheva, 2017; Verardi *et al.*, 2012).

Conclusions

It is globally recognized that traditional physical and chemical approaches to remediation of polluted mine environments are expensive, and often leave significant liabilities for host communities. These include permanently altered soil properties, destruction of host soil and microflora, and the creation of additional pollution problems, such as the generation of large volumes of chemical waste (Ayangbenro and Babalola, 2017). The abandoned, degraded land becomes a loss for future generations. Where communities continue to use the polluted land, their health is adversely affected, and so is that of wildlife.

Agro/phytomining, which involve planting and subsequent harvesting of appropriate vegetation that selectively concentrates specific metals from the environment into the plant tissues and recovery of valuable metals from the plants, offer hope of finding win-win methods of remediating contaminated mine environments. This is because in addition to cheaply remediating the environment, the valuable accumulated heavy metals in plants can be commercially exploited to add to normal mining revenues.

Some hyperaccumulator plants can also be used to produce biofuels, such as ethanol, on a commercial basis, which would also add to income streams for mining companies. The participatory nature of biofuel production would significantly benefit mining host communities, both during and beyond mine life.

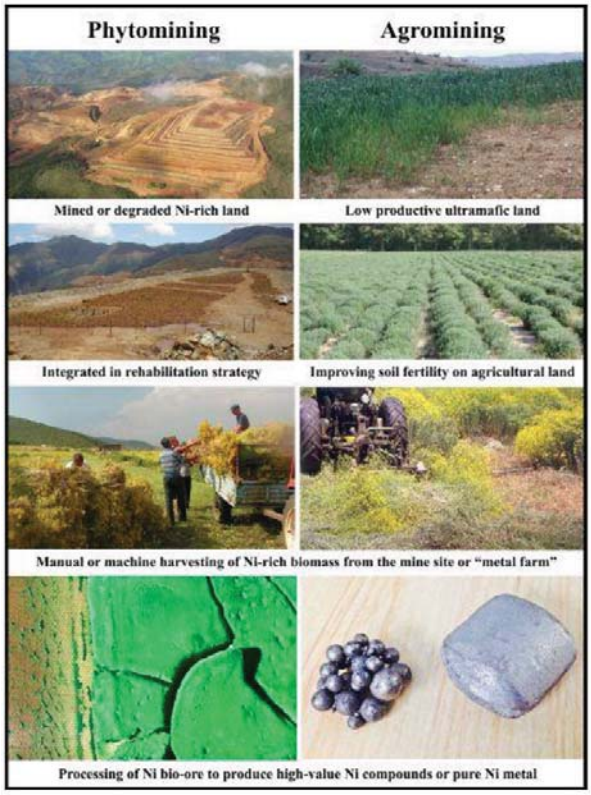

Figure 3—Phytomining/agromining operations with harvesting of biomass and processing of bio-ore (van der Ent *et al.*, 2015a)

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Mining and metallurgical wastes: a review of recycling and re-use practices

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Synopsis

Mining and metallurgical processes produce significant amounts of waste. In South Africa, mining and metallurgical wastes constitute one of the biggest challenges to the environment. If not managed properly, these types of wastes can result in irreversible damage to the environment and constitute a hazard to humans. Despite the environmental challenges associated with mining and metallurgical wastes, the mining and metal extraction industries can be integrated to form a circular economy model that promotes zero waste through the recycling and re-use of these waste materials. In other words, the different waste streams can in fact be considered as secondary sources of valuable minerals and metals. In this paper we review some of the research and emerging trends in the recycling and re-use of mining and metallurgical wastes. A brief overview is provided of how some of the key aspects of sustainability can be integrated into the teaching and research space in engineering sciences.

Keywords

mining and metallurgical wastes, circular economy, sustainability, recycling, re-use.

Introduction

Mining is a complex process involving activities that range from exploration through mine development, mineral beneficiation, metal extraction, smelting, refining, reclamation, and remediation (Bian *et al.*, 2012; Ndlovu, Simate, and Matinde, 2017). In the process of extracting the metal values, these activities produce significant amounts of wastes, typically consisting of (1) solid wastes in the form of waste rock, dusts, sludges, and slags, (2) liquid wastes in the form of waste water and effluents, and (3) gaseous emissions. In South Africa, mining and metallurgical wastes constitute one of the biggest challenges to the environment. If not managed properly, the anthropogenic effects of these mining and metal extraction activities can result in irreversible damage to the environment and a hazard to humans. In South Africa, in particular, these types of wastes are usually disposed of in landfills, thereby creating serious environmental and health challenges for communities. Mitigating the effects of such mining, metallurgical, and metal manufacturing processes requires a holistic waste management approach that incorporates reduction in the amount of waste produced, in-process recycling, and finding new markets and applications in other sectors of the economy (Lottermoser, 2011; Environmental Protection Agency, 2015; World Steel Association, 2015; Ndlovu, Simate, and Matinde, 2017). As such, increasing the recycling and re-use of the different types of wastes is a potential panacea to the environmental and health challenges posed by these waste streams. Despite the environmental challenges associated with these types of wastes, the mining and metal extraction industries can be integrated to form a circular economy model that promotes zero waste through the re-use and recycling of these waste materials (Lottermoser, 2011; EPA, 2015; World Steel Association, 2015; Ndlovu, Simate, and Matinde, 2017; Flanagán, Grail, and Johnson, 2016). The different waste streams can in fact be considered as secondary sources of valuable metals and other resources.

Circular economy, recyclability, recycling, and re-use have been identified as some of the emerging paradigms that can drive the multi-dimensional aspects of sustainability in the mining and metal extraction industries. In this context, a sustainable circular economy is defined as a transition where the value of products, materials, and resources is maintained in the economy for as long as possible, and the generation of waste minimized (European Commission, 2015; World Steel Association, 2015). The broad

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objective of a circular economy model is to reduce the linear flow of materials through recycling and re-use in order to rejuvenate the life-cycle of a material (Ndlovu, Simate, and Matinde, 2017). Lottermoser (2011) defined the recycling of mine wastes as a practice that extracts new valuable resource ingredients, or uses the waste as feedstock, and/or converts the entire waste stream into a new valuable product. Recyclability refers to the amenability of waste materials to be captured and converted into a new material and/or re-used in the same capacity (Ndlovu, Simate, and Matinde, 2017). According to Ndlovu, Simate, and Matinde (2017), the recyclability of a waste material is driven by technological and economic factors. In essence, the recyclability of materials depends on the availability of methods and technologies as well as the existence of markets for the recycled products. Finally, the World Steel Association (2015) broadly defined the term re-use as using an object or material again, either for its original purpose or for a similar purpose, without significantly altering the physical form of the object or material. Although most of these terms are used loosely and interchangeably by various stakeholders, some of the short- and long-term benefits of recycling and re-use of mining and metal extraction wastes include (Lottermoser, 2011): (1) creating financial assets, (2) increasing resource efficiency by reducing the linear consumption of natural resources, (3) reducing waste production and accumulation, (4) encouraging innovation and growth of local industry spin-offs, (5) creating employment, and (6) shared responsibility and ownership over the environment.

In line with these emerging paradigms of environmental responsibility and sustainable development, the broad objective of this paper is to explore sustainable alternatives to the primary supply of metals from mined ores through recycling and re-use of mining and metal extraction wastes. In detail, this paper provides a critical review of the emerging body of knowledge on recycling and re-use of these waste materials. In addition, we propose potential approaches that can contribute to unlocking the economic value of mining and metallurgical wastes by integrating some of the key sustainability drivers into the teaching and learning of engineering sciences.

Categories of mining, metallurgical, and industrial wastes

Basically, waste is a complex, subjective, and sometimes a controversial issue and, in fact, a review of literature reveals an ongoing international debate on the definition of waste (Ndlovu, Simate, and Matinde, 2017). Nevertheless, waste is generally defined by many environmental bodies as any discarded, rejected, abandoned, unwanted, or surplus matter, whether or not intended for sale or for recycling, reprocessing, recovery, or purification by a separate operation from that which produced it (Ndlovu, Simate, and Matinde, 2017). Waste is generated in all sorts of ways, and its composition and volume depend largely on consumption patterns and the industrial and economic structures in place. Waste may exist in solid, liquid, or gaseous form. However, waste, in whatever form, is one of the world's largest concerns and gives rise to both public health and environmental concerns (Bian *et al*., 2012). Environmental threats may include contamination of groundwater and surface water by leachates, as well as air pollution from burning of waste that is not properly disposed of. Therefore, in the recent past, the effects of waste, particularly mining and metal extraction wastes, have been the focus of critical attention (Lottermoser, 2011; Bian *et al*., 2012; Reck and Graedel, 2012; Ndlovu, Simate, and Matinde, 2017).

Inasmuch as the generation of mining and metallurgical wastes is inevitable in the production of industrial materials, conventional waste management practices to date have focused on how to manage the way in which the waste is generated and disposed of (Hering, 2012). In fact, mining, metallurgical, and industrial wastes are considered to have little or no apparent economic use, and are thus usually discarded and/or landfilled (Lottermoser, 2011; Bian *et al*., 2012; Ndlovu, Simate, and Matinde, 2017). Despite protracted efforts to reduce the amount of waste produced by the mining and metallurgical industries, these types of wastes still constitute one of the world's largest waste streams (Bian *et al*., 2012). Table I provides an overview of some of the types of wastes produced in the mining and metals extraction industry (Lottermoser, 2011; Rankin, 2011; Ndlovu, Simate, and Matinde, 2017).

Legislative framework governing mining and metallurgical wastes

Over the years, environmental laws and social awareness programmes have been promulgated in order to mitigate the potential threats of mining and metallurgical wastes (Table II). Basically, these environmental and social regulations are of universal importance, and revolve around environmental protection, protection of local communities, and promotion of business ethics (Phadke *et al*., 2014; Ndlovu, Simate, and Matinde, 2017). Despite the lack of consensus, and varying degree of legislative scope and effectiveness, various regulations and policies have been enacted so as to control the disposal and/or recycling of these types of wastes (Kumar and Singh, 2013; EPA, 2015; Ndlovu, Simate, and Matinde, 2017). Table II shows some of the selected environmental policies and regulations from the USA, EU, and South African jurisdictions (European Commission, 2006, 2008, 2010a, 2010b; Environmental Protection Agency, 2015; Ndlovu, Simate, and Matinde, 2017).

Inasmuch as there are no agreed-upon quintessential guidelines or legislation on the management of mining and metallurgical wastes, the different legislations converge towards one objective of environmental protection and sustainability. Ironically, the Bill of Rights, as enshrined in Chapter 2 of the South African Constitution (Act No. 108 of 1996), clearly stipulates the universal rights of citizens to environmental protection while promoting economic and social development. Furthermore, the USA's RCRA (1976) and the European Commission's Extracting Waste (Mining) Directive (2006/21/EC) have a strong emphasis on sustainability. These legislations specifically require operators to draw up waste management plans for the minimization, treatment, metal recovery from, and disposal of mining and extractive wastes.

In addition to the specific directives, the European Commission has also drawn up extensive industrial emissions directives (IED, 2010/75/EU) (European Commission, 2010b). The principal focus of these directives

Table I

Overview of some of the types of wastes produced in the mining and metals extraction industry

Type of waste	Comment
Overburden	Soil and rock material removed to access mineral deposits, and which is usually stockpiled. Overburden generally has low potential for environmental contamination, but may be associated with acid rock drainage.
Waste rock	Contains minerals in concentrations considered too low to be economically extracted. Waste rock has a heterogeneous mineralogical composition, chemical, and physical characteristics due to deposition of the wastes from different mine sources. Depending on the mineral being extracted, waste rock is problematic due to the formation of acid rock (mine) drainage.
Mineral beneficiation tailings	Fine barren rock that remains after extracting the valuable components. Tailings can also contain residual chemicals and are usually deposited in the form of water-based slurry into tailings ponds.
Metallurgical slags	Glass-like or amorphous materials produced as by-products in the smelting and refining of metals. Although some slags such as blast furnace slags can be environmentally benign and have found widespread applications in the construction and agricultural industries, the presence of entrained and/or dissolved heavy metals presents challenges to re-use.
Waste water	Emanates from a number of processes, with varying degrees of quality and environmental contamination potential. Typical examples include minewater, process (mill) water, leachate (containing dissolved minerals, chemicals and/or metals), effluent (process water discharged into surface water, often after treatment), and mine drainage water (surface or groundwater with the potential to flow off the mine or industrial site).
Water treatment sludge	Semi-solid residue or slurry remaining after on-site treatment of mine and industrial water and waste water. Sludge may be contaminated with heavy metals and other residual chemicals. The recycling and recovery of valuable metals from mining and industrial sludges is being extensively explored. Depending on the processes, waste treatment sludges are classified as hazardous materials.
Acid mine drainage (AMD)	Generated from mine waste rocks, tailings, and/or mine structures such as closed, active, or abandoned pits and underground workings.
Gaseous and particulate emissions	Atmospheric emissions in the form of particulate dusts and toxic gases such as SO_x, NO_x, CO, CO_2, organometallic compounds, polychlorinated-p-dibenzodioxins and dibenzofurans (PCCD/Fs) emitted during the high-temperature chemical processing of metals. These emissions are classified as hazardous materials in most jurisdictions due to the presence of entrained toxic heavy metals, toxic organometallic compounds, and PCCD/Fs.
Post-consumer waste	Waste material generated by households or commercial, industrial, and institutional facilities in their role as end-users of the products, which can no longer be used for its initial purposes, *e.g.* e-waste.

is to provide the best available techniques (BATs) and emerging techniques towards the reduction, recycling, and re-use of various waste streams across various industries (European Commission, 2010a, 2010b). In South Africa, for example, the various Acts clearly mandate the need for environmental responsibility with respect to air, water, and ground contamination, and further emphasize the mandatory rehabilitation of the affected environment. Globally, the convergence of the discrete legislations in these various jurisdictions has since resulted in a standardized ISO 14000 series, the main objective of which is to provide a standardized and effective management system.

Recycling and re-use: ongoing research and emerging trends

As discussed earlier, the circular economy model mandates the reduction, recycling and re-use of mining and metallurgical wastes. Based on the categories of wastes highlighted in Table I, the following sections outline the typical recycling and re-use opportunities of selected categories of mining and metal extraction wastes based on the emerging paradigms of converting wastes to valuable resources.

Waste rock, overburden, and beneficiation waste

As indicated in Table I, waste rock and overburden originate from materials overlying the area to be mined and which are moved in order to gain access to the orebody. These materials range from being barren (*e.g.*, waste rock) to containing minerals at concentrations that are considered too low to be economically extracted (*e.g.*, overburden). Mineral beneficiation waste, on the other hand, refers to tailings and/or residual materials generated from the mineral beneficiation processes (Ndlovu, Simate, and Matinde, 2017). Basically, mineral beneficiation wastes consist of rock, soil, loose sediment, and fine to ultrafine particles (Lottermoser, 2011; Bian *et al.*, 2012; Ndlovu, Simate, and Matinde, 2017). Corollary to waste rock and overburden, the mineralogical characteristics of beneficiation wastes are usually highly heterogeneous as a result of the deposition of waste streams arising from different stages in the processing chain(Bian *et al.*, 2012; Edraki *et al.*, 2014). Furthermore, the amounts and complexity of waste rock, overburden, and beneficiation wastes vary significantly depending on the type of commodity, physical, and chemical composition of the mineralization, and the mining and processing methods used (Lottermoser, 2011; Bian *et al.*, 2012; Flanagán, Grail, and Johnson, 2016; Ndlovu, Simate, and Matinde, 2017).

Millions of tons of waste rock, overburden, and beneficiation wastes are produced by the global mining industry. Due to their low intrinsic value, and the remote location of most mining operations, over 95% of these materials end up being disposed in landfills (Lottermoser, 2011; Bian *et al.*, 2012; Flanagán, Grail, and Johnson, 2016; Ndlovu, Simate, and Matinde, 2017). However, disposal is associated with environmental challenges such as acid rock drainage, airborne dust emissions, and contamination of surface- and groundwater sources. Although these materials are generally classified as non-hazardous, increasing their

Table II

Overview of relevant waste management policies and regulations from selected jurisdictions

Jurisdiction	Legislation	Content
USA (Environmental Protection Agency, 2015)	National Environmental Policy Act (1970)	Mandates environmental impact assessments (EIAs) for all economic activities that can impact the environment. Applies to mining operations that require Federal approval.
	Resource Conservation and Recovery Act (1976)	Conserving of natural resources and reducing the amounts of wastes generated. Ensures holistic waste management practices to protect the environment, and applies to categories such as municipal solid waste landfills, hazardous waste generation, and transportation, storage, and disposal facilities.
	Comprehensive Environmental Response, Compensation and Liability Act (1980)	Requires operations to report inventory of chemicals handled and release of hazardous substances to the environment, and mandates clean-up of sites where hazardous substances are found. Amended to include mining, milling, and smelter wastes that were not covered by the RCRA (1976).
	Clean Air Act (1970)	Addresses airborne pollution that may potentially cause harm to humans or natural resources, *e.g.*, dust emissions from operations or tailings disposal, exhaust emissions from heavy equipment, and emissions from processing facilities such as smelters.
European Union (European Commission, 2010a, 2010b; 2015)	The European Commission (EC) has promulgated various environmental directives governing mining, metallurgical, and industrial processes in member states	Environmental Assessment Directive – similar to NEPA (1970).
		Water Framework Directive for protection of surface and groundwater sources.
		Waste Framework (2006/12/EC), Hazardous Waste (91/689/EEC) and Landfill (99/31/EC) directives – similar to RCRA (1976).
		Extracting Waste (Mining) Directive (2006/21/EC) – similar to RCRA (1976). Operators mandated to draw up extractive waste management plan (EWMP) for the minimization, treatment, recovery, and disposal of extractive waste (emphasis on sustainable development) (European Commission, 2006).
		Industrial Emissions Directive (IED, 2010/75/EU) – similar to RCRA (1976). Principal focus on best available techniques (BATs) and emerging techniques in waste management across various industries (European Commission, 2010a, 2010b).
South Africa	Bill of Rights, Chapter 2 of the Constitution (Act No. 108 of 1996)	Universal right to environmental protection through reasonable and other measures that prevent pollution and secure ecological sustainable development and use of natural resources while promoting economic and social development (Republic of South Africa, 2018)
	National Environmental Management Act (No. 107 of 1998)	Framework and principles for sustainable development, and imposes duty of care and remediation of environmental damage. Inclusive of sections that provide legal authority to enforce environmental laws and private liability/prosecution (Department of Water and Environment, 2010).
	National Water Act (No. 36 of 1998)	Integrity of water resources. Provides a hierarchy of priorities for mine water management in terms of pollution prevention, water re-use or reclamation, and water treatment and discharge (Department of Water and Environment, 2010).
	Minerals Act (Act 50 of 1991)	Statutory instrument for enforcing environmental protection, management of environmental impacts, and the rehabilitation of affected environments (Republic of South Africa, 1991).
	Air Quality Act (No. 39 of 2004)	Prescribes measures to control air quality, including, but not limited to, the emission of respirable and non-respirable dusts and their control and minimization through cleaner technologies and cleaner production practices.
Global (Ndlovu, Simate,	ISO 14000 series	Guidelines and standards for environmental management systems, environmental auditing, and Matinde, 2017)environmental performance and evaluation, environmental labels and declarations, and life-cycle assessment. Established through consensus by national standards bodies globally.

recycling and re-use potential can provide sustainable and cost-effective alternatives to mitigate the disposal and waste management challenges.

To date, extensive research has been conducted in order to mitigate the environmental impacts, as well as increasing the valorisation potential, of waste rock, overburden, and beneficiation wastes (Lottermoser, 2011; Bian *et al.*, 2012; Lébre and Corder, 2015; Flanagán, Grail, and Johnson, 2016; Ndlovu, Simate, and Matinde, 2017; Gorakhki and Bareither, 2017). In addition to the widely adopted uses such as feedstock for cement, concrete, and aggregates in construction industry, waste rock and overburden can also be re-used as low-grade resources of valuable minerals and metals, as backfill materials for open voids, as landscaping materials, as capping materials for waste repositories, and as substrates for mine revegetation (Lottermoser, 2011; Bian *et al.*, 2012). Other waste streams such as mine drainage sludges can be economically reprocessed to extract metals and mineral compounds, or can be re-used as flocculants and/or adsorbents to remove phosphates from sewage and agricultural effluents, as well as soil additives in agriculture. Beneficiation tailings, on the other hand, can be reprocessed to extract metals and metals compounds, while sand-rich tailings can be mixed with cement and used as backfill in underground mines. Clay-rich tailings can be used as amendments to sandy soils and for the manufacture of bricks, cement, floor tiles, sanitary ware, and porcelains (Lottermoser, 2011). Furthermore, Lottermoser (2011) proposed the potential use of ultramafic tailings in the production of glass and rock wool, and phlogopite-rich tailings for use in sewage treatment. Nevertheless, the proposed re-use opportunities entail their own challenges. For example, the heterogeneous composition and complex mineralogy of these materials may affect the physical and

chemical properties of intended products and processes. Furthermore, mining and beneficiation wastes may contain elevated amounts of transition metal elements, toxic elements, and reactive minerals, which can contaminate the environment (Lottermoser, 2011; Bian *et al.*, 2012). The low intrinsic value may further be exacerbated by the remote location of most mining operations and distance from mine site to potential markets (Lottermoser, 2011; Bian *et al.*, 2012).

Metal recovery from acid mine drainage

Acid mine drainage (AMD) is an environmental problem related to the release of acidic, sulphate- and metal-containing waste water into the environment. AMD is generated when sulphide-bearing minerals such as pyrite (FeS_2) are exposed to both oxygen and water, as well as the presence of acidophilic chemolithotrophic microorganisms. In general, the oxidation of FeS_2 in the presence of oxygen and water takes place as depicted in Equation [1] (Simate and Ndlovu, 2014):

$$2FeS_2 + 7O_2 + 2H_2O \rightarrow 2Fe^{2+} + 4SO_4^{2-} + 4H^+ \qquad [1]$$

AMD may also be generated from various sources such as mine waste rock, tailings, or mine structures such as active, closed, or abandoned pits and underground workings (Simate and Ndlovu, 2014; Ndlovu, Simate, and Matinde, 2017).

The effects of untreated AMD on the health of humans, wildlife, plants, and aquatic species are well documented (Akcil and Koldas, 2006; Ma and Banfield, 2011; Simate and Ndlovu, 2014). Consequently, much research has been dedicated to finding remediation solutions for AMD. Basically, remediation techniques focus on the treatment of the already-produced AMD before it is discharged into water bodies (Chowdhury, Sarkar, and Datta, 2015). In principle, remediation technologies can be broadly categorized as active or passive. Further details of the remediation techniques are well documented by Johnson and Hallberg (2005). The challenge with the conventional remediation processes is that they generally result in new waste streams that would require further treatment and/or disposal. Furthermore, the treatment residues may still contain the elements and compounds removed from the mine drainage, as well as the additives dosed during the treatment processes (Simate and Ndlovu, 2014).

As already stated, AMD is rich in dissolved metal sulphates, which are formed in a complex process from the oxidation of pyrite and other sulphide minerals like pyrrhotite, marcasite, chalcocite, covellite, arsenopyrite, and chalcopyrite. Due to the need for metal-containing resources and increasingly stringent environmental conditions, AMD is currently being considered for metal recovery as part of the larger AMD remediation strategy (Nordstrom *et al.*, 2017). In fact, there are several such techniques used to recover metals from AMD, and these are primarily driven by incentives such as (Simate and Ndlovu, 2014): (1) reducing the amounts of waste sludge and brine products that require handling and disposal and incur potential long-term environmental liabilities, (2) generation of revenue streams to partly or fully offset the ongoing treatment and metal recovery costs, and (3) contributing to the long-term sustainability of mine water treatment projects.

To date, the removal and/or recovery of metals as hydroxide precipitates from AMD using alkaline reagents (*e.g.* NaOH) has been the most widely adopted method (Johnson and Hallberg, 2005; Balintova and Petrilakova, 2011). Unfortunately, this technique has several drawbacks, including the generation of large volumes of hazardous concentrated sludge (Macingova and Luptakova, 2012; Ndlovu, Simate, and Matinde, 2017), and the fact that the selective extraction of metals is very difficult (Simate and Ndlovu, 2014; Ndlovu, Simate, and Matinde, 2017). There are several other methods used to treat AMD and/or recover metals. These techniques, which will not be discussed in detail in this paper, include solvent extraction, reverse osmosis, ultrafiltration, electrodialysis, ion exchange/adsorption, and wetland treatments (Ndlovu, Simate, and Matinde, 2017).

It must be noted that, in the recent past, a paradigm shift has taken place in the way AMD is viewed. Basically, research interests have shifted towards the recovery of valuable materials from AMD, in addition to its remediation. Apart from valuable metals, there are other industrially and economically useful products that can be recovered from AMD treatment processes (Simate and Ndlovu, 2014). Typical examples include saleable products such as sulphur, sulphuric acid, pigments, and metal sulphates; production of electricity; alkaline earth compounds such as calcium carbonate and magnesium hydroxides; building and construction materials such as gypsum and cement; agricultural materials (*e.g.* fertilizer), adsorbents used in municipal and industrial wastewater treatment; and pigments (*e.g.* ferrihydrites).

Metal recovery and recycling of metallurgical dusts

Metallurgical dusts consist of heterogeneous mixtures of complex oxides from feed and process materials entrained in the off-gas from smelting and refining furnaces. Large volumes of these waste materials are produced in processes such as the blast furnace ironmaking process, raw material agglomeration (*e.g.*, coke, sinter, and pellet plants), electric arc furnaces (EAFs), basic oxygen furnaces (BOFs), stainless steel refining, base metal smelting and converting, and in submerged arc furnaces (SAFs) for ferrochrome and ferromanganese production (Ma and Garbers-Craig, 2006; Ndlovu, Simate, and Matinde, 2017). The physico-chemical properties of metallurgical dusts vary greatly depending on the process design, process parameters, operational procedures, and the type of raw materials used (Nyirenda, 1991; Palencia *et al.*, 1999; Beukes, van Zyl, and Ras, 2010; Tangstad, 2013; de Buzin, Heck, and Vilela, 2017). For example, about 7–45 kg of blast furnace particulate dusts containing 100–150 g of Zn are produced per ton of hot metal (tHM). These materials are classified as hazardous waste in most jurisdictions due to the presence of toxic metal elements such as Pb, Cr, Zn, and Cd, and as such, cannot be disposed of in landfills without some form of pretreatment and/or stabilization (Nyirenda, 1991; European Commission, 2008; 2010a, 2010b; Beukes, van Zyl, and Ras, 2010; Environmental Protection Agency, 2015; de Buzin, Heck, and Vilela, 2017; Ndlovu, Simate, and Matinde, 2017). Furthermore, metallurgical dusts are extremely fine and difficult to handle (Palencia *et al.*, 1999; de Buzin, Heck, and Vilela, 2017). Despite these materials being rich in valuable

metals, their direct in-process recycling is usually constrained by the potential build-up of deleterious and volatile metal compounds in the process (Bian *et al.*, 2012; Remus *et al.*, 2013; Ndlovu, Simate, and Matinde, 2017). Table III summarizes some of the characteristic properties and the recycling options applicable to selected categories of metallurgical dusts (Remus *et al.*, 2013; Lin *et al.*, 2017; Ndlovu, Simate, and Matinde, 2017).

The metallurgical dusts produced in the iron and steel manufacturing processes, for example, consist mainly of heterogeneous mixtures of complex oxides of feed materials entrained in the off-gas. As a result, these materials tend to contain significant amounts of iron, reductants, and alloying elements. To date, several processes have been developed, or are being investigated, in order to recover valuable metallic elements contained in metallurgical dusts while ensuring an environmentally benign barren product. Generally, two process options exist for the recovery of value metals from ferrous metallurgical dusts: pyrometallurgical and hydrometallurgical processes. Pyrometallurgical processes use thermal energy to separate the desired metals from other materials based on the differences between oxidation potentials, melting points, vapour pressures, densities and/or miscibility of the dust components when melted (Abdullah, 2013). In contrast, in hydrometallurgical technologies the desired metals are separated using techniques that capitalize on differences between constituent solubilities and/or electrochemical properties while in aqueous solution (Abdullah, 2013).

Metal recovery and recycling of ferrous metallurgical dusts using pyrometallurgical processes

As indicated in Table III, the direct recycling of ferrous dusts is limited by the presence of volatile toxic metal compounds such as Zn, Cd, As, and Pb. In steelmaking, for example, Zn-bearing compounds such zincite (ZnO) and franklinite ($ZnFe_2O_4$) are particularly problematic due to the heavy reliance on use of galvanized steel scrap as feed to these processes. The pyrometallurgical processing of ferrous dusts revolves around the carbothermic reduction sequence $ZnFe_2O_{4(s)} \rightarrow ZnO(s) + Fe_2O_{3(s)} \rightarrow Zn_{(v)} + Fe_{(s)} \rightarrow$ of $ZnFe_2O_4$ in dusts, and the resultant volatilization of the zinc before recovery in the downstream condensers (Lin *et al.*, 2017; Ndlovu, Simate, and Matinde, 2017).

Table III

Typical properties and the recycling and mitigation processes applicable to metallurgical dusts

Category	Characteristics	Recycling and mitigation	Challenges to recyclability
Blast furnace dusts	7–45 kg of dust/tHM; 100–150 g Zn/tHM	Briquetting and in-process recycling as additional sources of iron and carbon	Recyclability limited by in-process build-up of volatile metal compounds, organometallic compounds, and polychlorinated dibenzo-p-dioxins and dibenzofurans (PCDD/Fs). Intricate off-gas cleaning systems required but fugitive emissions may still be problematic.
Coke plant dusts	Varies (approx. > 1 kg coke breeze/ton coke	Additives in sinter plant as auxiliary fuels and reductants	
Ferrous raw material agglomeration	Varies	Briquetting and in-process recycling as well as auxiliary additives in the blast furnace and EAFs.	
Basic oxygen furnace (BOF) dusts	14–143 kg dust/t liquid steel 0.75–24 kg/t liquid steel	Briquetting and in-process recycling. Metal recovery using pyrometallurgical, hydrometallurgical and plasma processes. Solidification and/or stabilization	Controlled material as the long-term leachability behaviour of toxic metal elements in stabilized/solidified dusts is largely unknown. Recyclability limited by the in-process build-up of volatile metal compounds, variable value-metal thresholds, and the complex chemical and mineralogical characteristics of the dusts. Additional metal recovery processes costly.
Electric arc furnace dusts (EAFDs)	15–20 kg dust/t steel. Over 8.5 Mt are produced per year		
Stainless steel-making dusts	10–70 kg dust/t steel		
Submerged arc furnace (HCFeCr)	18–25kg dust/t HCFe Cr	Process-integrated dust recycling systems. Emissions reduction by mandatory adoption of closedfurnaces, agglomeration of feed, use of intricate wet scrubbing systems, and control of raw material quality and process parameters.	Recyclability limited by in-process build-up of volatile metal compounds and variable value-metal thresholds. Fugitive emissions still problematic. Intricate wet scrubbing systems costly and may not always be available.
Submerged arc furnace (HCFeMn)	Up to 25 wt% Mn and varying amounts of volatile metals.		
Base metals (copper) smelter dusts	Up to 30 wt.% Cu and high amounts of Fe, S, Zn, and metalloids (As, Sb, Pb, Cd, Bi)	In-process recycling of smelter dusts. Metal recovery using conventional pyrometallurgical and hydrometallurgical processes, or a combination of both.	Recyclability of smelter dusts is limited by build-up of deleterious, volatile toxic metals (*e.g.*, (As, Sb, Pb, Cd, Bi), low value-metal thresholds, and the complex chemical and mineralogical characteristics of the dusts.
PGM smelter and converter dusts	Varying amounts of entrained PGMs, Fe, S, Zn, and metalloids (As, Sb, Pb, Cd, Bi)		

/tHM: per ton of hot metal

To date, several pyrometallurgical approaches have been extensively applied globally in the economic recycling of ferrous metallurgical dusts. Table IV summarizes some of the processes for metal recovery and recycling of common metallurgical dusts (Palencia *et al.*, 1999; Lin *et al.*, 2017; Ndlovu, Simate, and Matinde, 2017).

Metal recovery and recycling from metallurgical dusts using hydrometallurgical processes

The major advantages of the pyrometallurgical processes developed so far lie in their ability to economically process metallurgical dusts containing high amounts of Zn existing as $ZnFe_2O_4$ (Ndlovu, Simate, and Matinde, 2017; Lin *et al.*, 2017). However, these processes are sensitive to economies of scale, have high thermal energy requirements, and require elaborate dust collection systems and additional steps to recover volatile metals from the flue gas (Ndlovu, Simate, and Matinde, 2017). As a result, the application of hydrometallurgical processes in the recovery of metals from metallurgical dusts is slowly gaining prominence due to their greater flexibility of operation and the required economies of scale. Hydrometallurgical processes also have lower capital costs and few or no environmental challenges associated with flue gases, dusts, and noise. Notwithstanding some of these advantages, the technical and economic feasibility of hydrometallurgical processes requires careful management of water, waste water, and process solutions (Palencia *et al.*, 1999; de Buzin, Heck, and Vilela, 2017; Ndlovu, Simate, and Matinde, 2017). In addition, the widespread adoption of hydrometallurgical processes has largely been constrained by the complex physical, chemical, and mineralogical characteristics of the dust materials. For example, most of the zinc in typical EAF dusts occurs as franklinite ($ZnFe_2O_4$), which is stable and insoluble in most acidic, alkaline, and chelating media under conventional conditions (Palencia *et al.*, 1999; Miki *et al.*, 2016; de Buzin, Heck, and Vilela, 2017; Ndlovu, Simate, and Matinde, 2017). Furthermore, hydrometallurgical processes using conventional leaching and chelating media also suffer from poor selectivity and co-dissolution, thereby increasing the complexity of downstream solution purification processes. As a result, several alternative leaching and pretreatment processes have been proposed, or are being investigated, in order to improve the kinetics and economics of the leaching processes (Leclerc, Meux, and Lecuire, 2002; Abott *et al.*, 2006, 2009; Steer and Griffiths, 2013; Bakkar, 2014; Chairaksa-Fujimoto *et al.*, 2015; Miki *et al.*, 2016; Chairaksa-Fujimoto *et al.*, 2016; Binnemans and Jones, 2017; Xing, Wang, and Chen, 2017).

Emerging trends in the recycling and re-use of metallurgical dusts and other residues

Pretreatment and leaching of ferrous dusts—As discussed in the preceding sections, the presence of the refractory $ZnFe_2O_4$ spinel presents challenges to the conventional leaching of these materials in aqueous media. As such, several studies have focused on the thermal pretreatment of Zn-bearing dusts before the leaching process (Chairaksa-Fujimoto *et al.*, 2015; Miki *et al.*, 2016; Chairaksa-Fujimoto *et al.*, 2016; Yakornov *et al.*, 2017). Chairaksa-Fujimoto *et al.* (2015) investigated the effect of CaO addition on the conversion of $ZnFe_2O_4$ in industrial EAF dust. Their findings confirmed that the $ZnFe_2O_4$ spinel can be decomposed by thermal treatment in the temperature range 900–1100°C and in the presence of a stoichiometric amount of reactive CaO to produce ZnO and $Ca_2Fe_2O_5$, as shown in Equation [2] (Chairaksa-Fujimoto *et al.* (2015):

$$\begin{aligned} &ZnFe_2O_{4(s)} + 2CaO_{(s)} = Ca_2Fe_2O_{5(s)} + ZnO_{(s)} \\ &\Delta G^0(\mathrm{J/mol}) = 32114 + 11.329T + 4.201 \times 10^{-3} \\ &T^2 - 5.46 \times 10^5 T^{-2} - 3.373T \ln T = -41006.722\mathrm{J/mol} \end{aligned} \quad [2]$$

Based on the thermodynamic feasibility of reaction [2] at the temperatures investigated, the authors have demonstrated that 100% of the ZnO can be selectively leached in conventional acidic or alkaline media after 4 hours, compared to 40% leaching efficiency in the case of the as-received sample (Chairaksa-Fujimoto *et al.*, 2015, 2016).

Table IV

Summary of processes for metal recovery and recycling of ferrous metallurgical dusts

Process	Process description	Products
Rotary hearth furnace (Zn-containing dusts), *e.g.* FASTMET®, INMETCO®	Solid-state carbothermic reduction and de-zincification of pelletized steelmaking dusts using rotary hearth furnaces (RHFs) in the temperature range 1200–1400°C. Secondary dust is processed to recover zinc.	Crude ZnO-rich off-gas and metallized direct reduced iron.
Premus®	Multi-hearth furnace solid-state reduction (temperature range 1000–1100°C) of steelmaking dusts using coke or coal fines and the EAF melting of the resulting hot briquetted iron (HBI).	Crude ZnO-rich off gas (Zn > 55 wt.%), direct charging of HBI into EAF.
Shaft furnace	Coke packed-bed shaft furnace for smelting-reduction of steelmaking dusts in the temperature range 1500–1600°C.	Crude ZnO-rich off-gas, hot metal/ crude steel, and environmentally benign barren slag.
Oxyfines™	Oxy-fuel process for smelting of fine dusts in the temperature range 1600–1700°C.	
Enviroplas™	DC -arc plasma furnace for smelting-reduction of steelmaking and ferroalloy fine dusts in the temperature range 1600–1700°C.	
OxyCup® Process	Shaft furnace for reduction and melting of agglomerated ferrous metal dusts in the temperature range 1500–1600°C.	
Waelz process	Waelz kiln carbothermic reduction of pelletized metal oxides (1100-1200°C), volatilization of Zn and Pb compounds and further processing of ZnO- and PbO-rich off-gas dust.	

Since the solubilities of iron and calcium were negligible under the experimental conditions investigated, the $Ca_2Fe_2O_5$ residue can be used as a dephosphorization fluxing agent in the steelmaking process.

Application of solvometallurgy in the recovery of metals from secondary resources—In order to increase the leachability and selectivity, the use of non-aqueous solvents in the form of molecular organic solvents, ionic liquids, deep eutectic solvents, organic solvents, and supercritical carbon dioxide, among others, has been explored in detail (Leclerc, Meux, and Lecuire, 2002; Abbott *et al*., 2005, 2006, 2009, 2011; Nockemann *et al*., 2006, 2008; Steer and Griffiths, 2013; Kilicarslan *et al*., 2014, 2015; Kilicarslan and Saridede, 2015; Davris *et al*., 2016; Amarasekara, 2016; Wang *et al*., 2016a, 2016b; Abbott *et al*., 2006a, 2006b, 2009; Park *et al*., 2014; Binnemans and Jones, 2017). Of particular interest in recent years is the use of ionic liquids, in the form of protic ionic liquids and carboxyl acid group functionalized ionic liquids, in the leaching of metals from a variety of ores and secondary resources (Abbott *et al*., 2005; 2006a, 2006b, 2009, 2011; Kilicarslan *et al*., 2014; 2015; Kilicarslan and Saridede, 2015; Amarasekara, 2016; Wang *et al*., 2016a, 2016b; Park *et al*., 2014; Binnemans and Jones, 2017). Ionic liquids are low-temperature molten salts comprising cations and organic/inorganic anions, and are increasingly gaining traction due to their desired properties such as non-volatility, low toxicity, good ionic conductivity, and wide electrochemical potential window (Park *et al*., 2014).

Kilicarslan *et al.* (2014) investigated the use of 1-butyl-3-methyl-imidazolium hydrogen sulphate ($bmim^+(HSO_4^-)$) ionic liquid in the presence of hydrogen peroxide (H_2O_2) and potassium peroxymonosulfate (oxone) as the oxidants for leaching brass wastes containing 3.4% Al, 5.81% Ca, 22% Cu, 12.17% Si, and 53.92% Zn. The results show that 50 vol.% $[bmim^+]HSO_4^-$ in aqueous media is an efficient ionic liquid for brass ash leaching, and resulted in dissolution efficiencies of 99% for Zn and 24.82% for Cu, even without the use of any oxidant. Other studies also indicated that pure zinc in brass waste could be recovered by leaching in Bronsted acidic ionic liquid and electrowinning (Kilicarslan and Saridede, 2015; Amarasekara, 2016).

Due to their similar solvent properties to ionic liquids, the use of deep eutectic solvents has also gained appreciable attention in the dissolution of metals oxides from secondary resources (Abbott *et al*., 2005, 2006a, 2011). Abbott *et al.* (2005) investigated the solubility of various metal oxides using low-temperature choline chloride ($HOC_2H_4N(CH3)_3^+$ Cl-)-based deep eutectic solvents. Their findings, based on the solubility of metal oxides in 2:1 urea/choline chloride eutectic at 60ºC, indicated that oxides such as ZnO, PbO_2, and Cu_2O exhibit appreciable solubility, whereas the solubility of iron and aluminium oxides is low (Table V). The selectivity of the chosen deep eutectic solvents indicates the potential to selectively leach and separate the metals from complex metallurgical dust mixtures using electrochemical methods (Abbott *et al*., 2005, 2006, 2011).

Although the commercial adoption of solvometallurgy processes is still in its infancy, these techniques provide several advantages such as high selectivity, ability to recover the valuable metals in their pure states, and the ability to detoxify and produce environmentally benign residues.

Table VI summarizes some of the research that has been, or is being, conducted using hydrothermal treatment and leaching of metallurgical dusts and residues in non-aqueous solvents in order to enhance the leachability of metallurgical dusts and residues.

Synthesis of structural and functional materials—To date, the recovery of metals from metallurgical dusts using conventional pyrometallurgical and hydrometallurgical approaches has been implemented with varying degrees of success. In parallel to the ongoing efforts to recover valuable metals using these processes, extensive research is also being conducted to increase the valorisation potential by incorporating these materials in the synthesis of structural and functional materials such as glasses, ceramics, and soft magnetic ferrites (Barbieri *et al*., 2002; Rashad, 2006; Machado *et al*., 2011; Vieira *et al*., 2013; Stathopoulos *et al*., 2013; Wang *et al*., 2017a; Salamati, Younesi, and Bahramifar, 2017; Chinnam *et al*., 2017). Most of these studies have demonstrated the potential to synthesize high-value functional soft magnetic spinel ferrites (MFe_2O_4, where M is Ni, Cr, Zn, Mn, *etc*.) from the solid-state reaction, sintering, hot pressing, and/or densification of metallurgical dusts containing these materials. For example, Wang *et al.* (2017a) proposed an innovative one-step process for the synthesis of Ni-Zn spinel ferrites with high saturation magnetization (M_s approx. 60.5 emug^{-1}) and low coercivity (H_c approx. 49.8 Oe) from the solid-state reaction of EAF dust with $NiCl_2{\cdot}6H_2O$ at 1100ºC for 2 hours. In other studies, Salamat, Younesi, and Bahramifar (2017) proposed a method to synthesize magnetic core-shell $Fe_3O_4@TiO_2$ nanocomposites from EAF dusts, with high degree of superparagnetism and photocatalytic activity for the decomposition of chemical oxygen demand (COD) in steel mill wastewater. The proposed processes practically demonstrated the potential to transform waste streams such as EAF dusts from problematic solid wastes to high-value-added products (Vieira *et al*., 2013; Stathopoulos *et al*., 2013; Wang *et al*., 2017a; Chinnam *et al*., 2017; Salamati *et al*., 2017). Table VII highlights some of the innovative approaches currently being employed to increase the valorisation potential by incorporating these materials in the synthesis of structural and functional materials.

Table V

Solubility of selected metal oxides in a 2:1 urea/choline chloride eutectic at 60ºC (Abbott *et al*., 2005)

Metal oxide	Melting point (°C)	Solubility (ppm)
Al_2O_3	2045	< 1
CaO	2580	6
CuO	1326	470
Cu_2O	1235	8725
Fe_2O_3	1565	49
Fe_3O_4	1538	40
MnO_2	535	493
NiO	1990	325
PbO_2	888	9157
ZnO	1975	8466

Table VI

Summary of emerging trends in pretreatment and use of non-aqueous solvents

Approaches	Key findings	References
Leaching of blast furnace dust using carboxylic acid and non-aqueous solvents	Use of non-aqueous solvents to achieve selective extraction of Zn of up to 85.7% using prop-2-eonic acid. Minimal levels of iron dissolution (approx. 8.5% Fe) Detoxification of the iron oxides matrix leach residue.	Steer and Griffiths, 2013
Selective leaching of rare earth elements (REEs) from bauxite residue using functionalized hydrophobic ionic liquids.	Selective leaching of metal oxides by carboxyl-functionalized task-specific ionic liquids. Approx. 70-85% dissolution of REEs from bauxite residue using functionalized betainium bis(trifluoromethylsulsonyl)imide (HbetTf2N) ionic liquid (150°C, 4 hours leaching time, and 10% pulp density) followed by stripping in acidic solution at ambient temperatures. Dissolution (T >55°C): $6HbetTf2N + REE_2O_3 \longrightarrow 2REE(bet)_3(Tf2N)_3 + 3H_2O$ [3] $REE(bet)_3(Tf2N)_3 + 3HCl_{(aq)} \longrightarrow REECl_{3(aq)} + 3HbetTf2N$ [4] Solubility of metals enhanced by temperature and pH control. Co-dissolution of Ca, Na, and Al and limited solubility of Fe, Si, and Ti, thereby enhancing the selectivity of leaching.	Davris *et al.*, 2016; Nockemann *et al.*, 2006, 2008
Stepwise hydrometallurgical treatment and recovery of Pb, Se, and Hg from lead smelting flue gas scrubber sludge using sodium chloride (NaCl) and sodium chlorate ($NaClO_3$)	NaCl leaching to extract Pb and precipitation using CaO to produce Se- and Hg-enriched residue, followed by stepwise leaching of Se and Hg in HCl and sodium chlorate and precipitation to recover individual elements of high purity (Pb, Hg, and Se)	Xing *et al.*, 2017
Hydrothermal extraction of Zn from $ZnFe_2O_4$ in EAF dust using ferric chloride hexahydrate mixtures.	Selective dissolution of Zn and Pb from EAF dust using ferric chloride hexahydrate mixtures followed by simple aqueous leaching of $ZnCl_2$: $2(FeCl_3)\cdot 6H_2O_{(s)} + 3ZnFe_2O_{4(s)} \longrightarrow 4Fe_2O_{3(s)} + 3ZnCl_{2(s)} + 12H_2O$ [5] Resultant detoxification renders the iron oxide residues usable as fluxes and/or raw materials in steelmaking.	Leclerc, Meux, and Lecuire, 2002; 2003; Wang *et al.*, 2016a, 2016b;
Hydrometallurgical extraction of Zn from $ZnFe_2O_4$ in CaO-treated EAF dust in NH_4Cl media.	Over 80% Zn extraction efficiency from CaO-treated $ZnFe_2O_4$ in 2M NH_4Cl, (70°C, S/L ratio 1:300, 2 hours). The ZnO formed from the thermal dissociation of $ZnFe_2O_4$ (Equation [8]) is solubilized in NH_4Cl solution according to the reaction: $ZnO_{(s)} + 2NH_4Cl_{(1)} \longrightarrow Zn(NH_3)_2Cl_{2(1)} + H_2O_{(1)}$ [6] Limited leachability of Ca and negligible solubility Fe to produce inert $Ca_2Fe_2O_5$ leach residue usable as flux in iron- and steelmaking.	Miki *et al.*, 2016
Recovery of metals from EAF dust using (deep eutectic) ionic liquids	Application of choline chloride and urea ionic liquids in the selective dissolution of Zn and Pb from mixed metal oxide matrices and production of recyclable insoluble iron oxides and aluminosilicates.	Abbott *et al.*, 2006, 2009; Bakkar, 2014; Binnemans and Jones, 2017
Selective leaching of and recovery of Bi_2O_3 from copper smelter converter dust.	Over 90% leaching efficiency of Bi as Bi_2O_3 from pretreated Cu dust using H_2SO_4 and NaCl, followed by hydrolysis of the $BiCl_3$ from solution to form pure solid Bi_2O_3: $Bi_2O_{3(s)} + 3H_2SO_{4(aq)} \longrightarrow Bi_2(SO_4)_{3(s)} + 3H_2O_{(1)}$ [7] $Bi_2(SO_4)_{3(s)} + 6HCl \longrightarrow 2BiCl_{3(aq)} + 2Na_2SO_{4(s)}$ [8] $BiCl_{3(aq)} + 3H_2O_{(1)} \longrightarrow BiOCl_{(s)} + 2HCl_{(1)}$ [9] $2BiOCl_{(s)} + 2NaOH_{(1)} \xrightarrow{Heat} Bi_2O_{3(s)} + 2NaCl + H_2O$ [10]	Ha *et al.*, 2015

Metal recovery, recycling and re-use of metallurgical slags

Metallurgical slags play indispensable roles in the efficient extraction of metals, and are produced in large volumes in pyrometallurgical smelting and refining processes (Pretorius and Nunnington, 2002; Reuter, Xiao, and Boin, 2004; Durinck *et al.*, 2008). In fact, the solidified slag constitutes a high-volume by-product from most pyrometallurgical processes (Reuter, Xiao, and Boin, 2004; Durinck *et al.*, 2008). During smelting and refining processes, slag separates from the metal/alloy and is tapped from the furnace before being granulated or slow-cooled. Depending on process design and conditions, solidified metallurgical slags contain significant amounts of entrained and/or dissolved metals, which can cause serious long-term environmental harm. For example, the presence of entrained and/or dissolved toxic metal species, such as chromium in stainless steel and ferrochromium alloy slags, can cause serious environmental problems due to the high leachability, mobility, and toxicity of the higher valence chromium Cr(VI) species (Durinck *et al.*, 2008; Ndlovu, Simate, and Matinde, 2017). As a result, the current practice of disposing process slags in landfills presents environmental challenges, hence the need for protracted efforts to increase the recyclability and re-use potential of these materials (Ndlovu, Simate, and Matinde, 2017).

As indicated in Table VIII, there are several opportunities, as well as constraints, in the recycling and re-use of metallurgical slags. Nevertheless, the pyrometallurgical industry continues to explore opportunities in order to increase the valorisation potential of metallurgical slags. Several researchers have focused on the applications of metallurgical slags in other sectors of the economy (Reuter, Xiao, and Boin, 2004; Euroslag, 2017; Ndlovu, Simate, and Matinde, 2017). In particular, extensive research has been conducted on the application of the various types of process slags as construction materials (Euroslag, 2017; World Steel Association, 2015), in the manufacture of ceramics and other functional materials (Quijorna, Miguel, and Andres, 2011;

Table VII

Examples of ongoing research to increase the valorization potential of ferrous metallurgical dusts

Approaches	Comments	References
Thermal and chemical behaviour of different glasses containing steel fly ash and their transformation into glass-ceramics	Synthesis of chemically inert and coloured glasses from mixtures of up to 10 wt.% steel fly ash, CaO-MgO-Al_2O_3-SiO_2 glass-ceramic systems, and other kinds of inorganic matrices.	Barbieri, Corradi, and Lancellotti 2002
Stabilization of nickel constituents of dusts in aluminate and ferrite spinels and investigating their leaching behavior.	Sintering (600-1480°C; 3 hours) of nickel-laden waste sludge to bind the nickel in nickel aluminate ($NiAl_2O_4$) and nickel ferrite ($NiFe_2O_4$) stable spinel phases with low toxicity characteristic leaching properties (TCLPs).	Shih, White, and Leckie, 2006
Synthesis and magnetic properties of manganese ferrite from low-grade smanganese ore	Synthesis of manganese ferrite with magnetic properties (M_s approx. 27.24 emug^{-1}) from low-grade Mn ore using oxidative acid leaching, co-precipitation, and calcination (1000–1200°C).	Rashad, 2006
Structural ceramics made with clay and steel dust pollutants	Synthesis of brick and hollow brick materials meeting TCLPs standards from sintering clay blends containing up to 20 wt.% EAF dust at 1100°C.	Machado *et al.*, 2011
Structural ceramics containing EAF dust	Pilot-scale study of the stabilization properties of EAF dust/clay ceramic structures and the synthesis of structural ceramic blocks containing up to 5 wt.% EAF dust with good chemical, macroscopic, and mechanical properties and low TCLPs.	Stathopoulos *et al.*, 2013
Recycling of EAF dust into red mud	Synthesis of fired ceramics from mixtures of EAF dust and natural clays with desirable mineralogical and physico-mechanical properties for civil construction applications.	Vieira *et al.*, 2013
Synthesis of magnetic core-shell Fe_3O_4@TiO_2 nanoparticles from EA dust	Synthesis of magnetic core-shell Fe_3O_4@TiO_2 nanocomposites from EAF dust with high degree of superparagnetism and photocatalytic activity for the decomposition of chemical oxygen demand (COD) in steel mill wastewater.	Salamati *et al.*, 2017
Functional glasses and glass-ceramics from iron-rich waste and industrial residues	Review of processes and technologies for the synthesis of functional glass-based products with suitable catalytic activity, magnetic, optical, and electrical properties from iron-oxide-containing residues.	Chinnam *et al.*, 2017
Facile synthesis of metal-doped Ni-Zn ferrite from treated Zn-containing EAF dust	One-step process for the synthesis of Ni-Zn spinel ferrites with high saturation magnetization (M_s approx. 60.5 emug^{-1}) and low coercivity (H_c approx. 49.8 Oe) from the solid-state reaction of EAF dust with $NiCl_2{\cdot}6H_2O$ at 1100°C for 2 hours.	Wang *et al.*, 2017a

Table VIII

Properties, recycling, and re-use opportunities of selected metallurgical slags

Category	Typical mineralogical composition	Recycling and re-use opportunities
Blast furnace slags	CaO-MgO-Al_2O_3-SiO_2 system. Crystallized mineral composition consisting mainly of melilite ($Ca_2MgSi_2O_7$–$Ca_2Al_2SiO_7$), and merwinite ($Ca_3MgSi_2O_8$)	Granulated and used as additives in Portland cement, geopolymers, and other absorbents. Slow-cooled and used as and construction and soil aggregates.
EAF slags	$Ca_3Mg(SiO_4)_2$; β -Ca_2SiO_4; $(Mg,Mn)(Cr,Al,Fe)_2O_4$ spinel solid solution; $CaAl_2SiO_6$; $(Fe,Mg,Mn)O$ wüstite-type solid solution; $Ca_2(Al,Fe)_2O_5$	Dissolved and/or entrained alloying elements. High level of impurities for refining slags. Recycled as pre-melted fluxes. Metal recovery and slag cleaning processes. Presence of elevated amounts of toxic alloying elements, *e.g.* Cr and Ni, limits their direct uptake in other sectors.
Steelmaking slags	Ca_3SiO_5; α-Ca_2SiO_4; $Ca_2Fe_2O_5$; β-Ca_2SiO_4; FeO–MnO–MgO solid solution; MgO; wüstite solid solution.	
Stainless steel (AOD) slags	$FeCr_2O_4$; $FeFe_2O_4$; Ni–Cr–Fe solid solution; Ca_2SiO_4; CaF_2; $Ca_{14}Mg_2(SiO_4)_8$; Ca_2SiO_4; $Ca_4Si_2O_7F_2$; MgO; Fe–Cr alloy; Fe–Ni alloy	
PGM smelting and converting slags	Fayalitic ($2FeO{\cdot}SiO_2$) slags with some dissolved magnetite. Chromium present as spinels $(Fe,Mg)Cr_2O_4$), particularly in furnaces smelting UG2 concentrates.	Slag cleaning and metal recovery. In-process build-up of Cr_2O_3 as spinels $(Fe,Mg)Cr_2O_4$) in UG2 concentrates limits recyclability and re-use.
Base metal slags (copper and nickel)	Fayalitic ($2FeO{\cdot}SiO_2$) and FeO-CaO-SiO_2 slag systems with some dissolved magnetite.	Slag cleaning and metal recovery. Converter slag recyclable as pre-melted fluxes.
Ferroalloy slags	HCFeCr slags: Al_2O_3-MgO-SiO_2-Cr_2O_3 system containing various phases such as MgO–$MgO{\cdot}Al_2O_3$–$2MgO{\cdot}SiO_2$–$2CaO{\cdot}SiO_2$, $MgO{\cdot}Cr_2O_3$, $(Mg_2(Cr,Al,Si)_2O_6$.	Slag cleaning and metal recovery. High leachability, mobility, and toxicity potential of entrained and/or dissolved Cr (VI) species limit the alternative applications.

Ponsot and Bernado, 2013; Karayannis *et al.*, 2017), and as geopolymeric materials (Kalinkin *et al.*, 2014; Huang *et al.*, 2015). However, the presence of entrained and/or dissolved toxic metal elements, as well as the build-up of deleterious elements in the slag, is still a major constraint in the recycling and re-use of these materials.

Emerging trends in the recycling and re-use of metallurgical slags

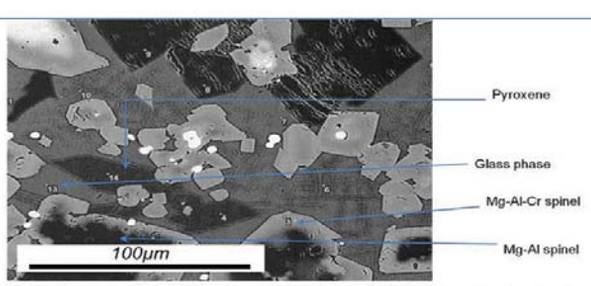
In the past, research in pyrometallurgy focused mostly on improving process performance by optimizing the properties of slags (Mills, Yuan, and Jones, 2011), but largely ignored

their environmental efficacy once the slags were tapped from the furnace. In other words, not much emphasis was placed on the holistic integration between the required properties of slags for process performance and their environmental performance after solidification and disposal. However, the drive for sustainability and environmental stewardship mandates the need to take into account the environmental effects of the slags. To date, emerging research in the pyrometallurgical industry is shifting towards both the in-process and post-process engineering of slag properties in order to achieve metallurgical and energy efficiency in the furnace while producing an environmentally benign slag.

Engineered slag structure-process-properties for recycling and environmental performance—The stabilization of dissolved and/or entrained toxic metal species in stable phases in various slags has drawn considerable attention in recent years (Barbieri *et al*., 1994; Kühn and Mudersbach, 2004; Tanskannen and Makkonen, 2006; Durinck *et al*., 2008a, 2008b; Albertsson, Teng, and Björkman, 2014; Santos *et al*., 2012; Albertsson, 2013; Cabrera-Real *et al*., 2012; Liapis and Papayiani, 2015; Piatak, Parsons, and Seal, 2015). In particular, Kühn and Mudersbach (2004) investigated the effect of MgO, Al_2O_3, and FeO additions to high-temperature slags. Their findings indicate that the addition of these compounds prior to the crystallization of high-temperature slags decreased the leachability of chromium species from the solidified slags. Cabrera-Real *et al*. (2012) investigated the effect of basicity (CaO/SiO_2) and MgO on the stability of the mineralogical phases formed during crystallization of CaO-SiO_2-Cr_2O_3-CaF_2-MgO slags. Mineralogical characterization of the solidified slag highlighted the predominance of $MgCr_2O_4$ spinels, and minor presence of $CaCr_2O_4$ and $CaCr_2O_4$ phases at $CaO/SiO_2 = 1$. Increasing the CaO/SiO_2 ratio to 2 resulted in the predominance of $MgCr_2O_4$, $CaCr_2O_4$, and the Cr (V)-containing complex $Ca_5(CrO_4)_3F$ (Cabrera-Real *et al*., 2012). Based on the comparative TCLPs of chromium from the different phases, the findings indicated the highest leachability of Cr at $CaO/SiO_2 = 2$, owing to the presence of potentially soluble $CaCr_2O_4$ and $Ca5(CrO_4)_3F$ phases. The low leachability of MgO-stabilized slag phases was attributed to the predominance of $MgCr_2O_4$ spinel phases, which in essence, function to bind the Cr in a stable spinel (Cabrera-Real *et al*., 2012). The effect of basicity on the stabilization of chromium species in stable phases was also investigated by Albertsson (2013). Based on high-temperature engineered parameters such as alumina addition, heat treatment, and oxygen partial pressure, Albertsson (2013) proposed that the addition of alumina to molten slags was effective in binding the chromium in the $MgAl_2O_4$-$MgCr_2O_4$ stable spinel phases. Slow soaking of solidified slag at low temperatures and low oxygen partial pressure also improved the formation of the spinel phases and reduced the amount of chromium dissolved in water-soluble matrix phases. Conversely, the heat treatment of slags with CaO/SiO_2 ratio greater than 1.4 under high oxygen potential accentuated the formation of leachable Cr-bearing phases (Albertsson, 2013).

In earlier studies, Barbieri *et al*. (1994) investigated the solubility, reactivity and nucleation of Cr_2O_3 in a CaO-MgO-Al_2O_3-SiO_2 glassy system. The slag system, containing up to 5 mol % Cr_2O_3, was melted at 1400ºC and the effect of Mg content on the spinel formation was investigated. At the glass melting temperature, the reaction of Cr_2O_3 and MgO was proposed to proceed to form stable $Mg_2Cr_2O_4$ spinels (Barbieri *et al*., 1994). Tanskanen and Makkonen (2006) also investigated the mineralogical and petrological characteristics of CaO-SiO_2-Al_2O_3-MgO-Fe-Cr slags from high-carbon ferrochrome production. Their findings indicated that the slag solidified with a partly crystalline and porphyritic structure with hypidiomorphic spinel ($(Mg,Fe)(Fe,Al,Cr)_2O_4$) crystals enclosed in a homogenous glass matrix, as shown in Figure 1.

In line with the structure-process-property hypotheses, Durinck *et al*. (2008a, 2008b) investigated the hot stage processing of molten steelmaking slags in order to enhance the environmental performance of the solidified slags. Based on case studies on $2CaO{\cdot}SiO_2$ (C_2S)-driven disintegration and chromium leaching, Durinck *et al*. (2008a) proposed that the functional properties of solidified slags can be significantly enhanced by controlled additions to the high-temperature slags and/or by variations in the cooling path. In other studies, Liapis and Papayianni (2015) investigated the high-



Point	Phase/zone	O	Mg	Al	Si	K	Ca	Cr	Fe
2	Glass	47.51	7.02	10.08	28.69	0.98	3.61	1.11	
3	Mg-Cr-Al-Sp	40.08	15.59	27.59				15.82	0.94
4	Pyroxene	45.79	21.94	3.09	26.11			2.04	0.50
5	Mg-Al-Sp	43.67	17.33	38.53					0.47
7	Glass	47.13	6.84	10.18	28.80	0.91	3.54	1.57	
8	Fosterite	43.10	34.01		20.30			1.60	0.99
13	Glass	47.46	5.70	11.11	28.42	1.20	3.73	1.28	
14	Pyroxene	45.50	21.62	3.46	26.05			2.24	0.53

Figure 1—Phase composition and mineralogy of air-cooled ferrochromium slags (Tanskanen and Makonnen, 2006; Ndlovu *et al*., 2017)

temperature in-process modification of high-carbon EAF slags using perlite (with non-normalized composition 76 wt.% SiO_2-14 wt.% Al_2O_3-3.4 wt.% Na_2O-2.7 wt.% K_2O-1.2 wt.% CaO-1 wt.% FeO_x), ladle furnace slags (with non-normalized composition approximately 62 wt.% CaO-14.5 wt.% SiO_2-3.6 wt.% MgO-2.5 wt.% FeO_x-2 wt.% Al_2O_3) and olivine (with non-normalized composition approximately 50 wt.% MgO-36 wt.% SiO_2-8.7 wt.% FeO_x-1.2 wt.% CaO) additives. Based on tailored chemical and mineralogical compositions, the modified EAF slag was found to be suitable for use in the production of normal weight cement concrete. Interestingly, the addition of slag modifiers had minimal interference on the process performance of the EAF (Liapis and Papayianni, 2015).

The studies highlighted so far support the hypothesis that the physicochemical properties of slag in the molten state can be engineered in order to obtain the desired solidified and crystal structure properties that are essential for improving the recycling and environmental attributes of metallurgical slags. In essence, the domain of slag engineering is a promising approach to controlling the solidification and crystallization properties of slags in order to improve their environmental compatibility. Slag engineering can improve the environmental performance of slags through binding the toxic metal species in stable spinel phases. In short, the objectives of slag engineering can be achieved by integrating the following aspects: (1) the post-tapping processing, solidification, and crystallization pathways, mineralogy, and speciation of entrained alloys in the slag, (2) evaluation of the efficacy of the proposed post-treatment methods based on TLCPs of the disposable and/or re-usable slag materials, and (3) correlating the structure, process and properties of slags to the broad recyclability and environmental performance.

Engineered structure-process-properties of ferrous slags for metal recovery—Inasmuch as slag cleaning using pyrometallurgical furnaces was the major focus to recover entrained and/or dissolved metals from slag, attention in recent years has shifted towards the recovery of metals from solidified slags by physical beneficiation methods. As discussed earlier, the engineered solidification and crystallization behavior of slags can result in the evolution of certain phases with desired properties that are essential for the recovery of metals from these materials. This means the speciation behaviour of alloys in discrete phases in rapidly cooled slags can be exploited in the physical beneficiation of these slag materials (Engström *et al.*, 2010; Burja *et al.*, 2017). In fact, the recovery of dissolved and/or entrained metals and alloys can be enhanced by hot-stage engineering of the microstructural properties of slag materials during crystallization and solidification (Durinck *et al.*, 2008a, 2008b; Liapis and Papayianni, 2015). This approach is based on the hypothesis that certain phases with desired properties in solidified slags can be engineered and maintained during the solidification process in order to facilitate their separation from the solidified slag. Based on studies of the crystallization behaviour of steelmaking slags during rapid cooling, Engström *et al.* (2010) observed the formation of wüstite-type solid solution (Mg,Fe,Mn)O enclosed in β-Ca_2SiO_4 phases. Burja *et al.* (2017) investigated the speciation of chromium in rapidly cooled CaO-SiO_2-Cr_2O_3 and CaO-SiO_2-MgO-Al_2O_3-Cr_2O_3 stainless steel slags. Based on X-ray diffraction analyses, the authors confirmed the presence of metallic Cr and FeCr phases, as well as the presence of calcium chromates, pseudowollastonite ($CaSiO_3$), and larnite (Ca_2SiO_4) (Burja *et al.*, 2017). The formation of metallic Cr and FeCr phases was particularly observed in low-basicity slags containing high CrO_x levels (Burja *et al.*, 2017). In other studies, Norval and Oberholster (2011) investigated the effect of granulation on the recovery of manganese from re-melted FeMn slags. Their findings indicate a significant difference in the residual MnO content of granulated and air-cooled slags, wherein the MnO content decreased to 6.1% in granulated slags compared with 21.8% in the air-cooled slags.

The formation in slags of certain phases with high magnetic susceptibilities can be engineered using post-furnace solidification processing, and such phases can be maintained during solidification to facilitate the separation of the alloys from solidified slag by magnetic separation. Based on the knowledge of soft magnetic properties of spinel ferrites synthesied by various methods (Viart *et al.*, 2000; Sugimoto, 1999; Lakshmi, Kumar, and Thyagarajan, 2016; Zhang *et al.*, 2016), the magnetic ferrite phases (MFe_2O_4, where M is Mn, Ni, Cr, Mo, *etc.*) in crystallized slags can be exploited in the magnetic separation from the bulk nonmagnetic components (Semykina and Seetharaman, 2011; Semykina, 2013; Shatokha *et al.*, 2013; Sharma, Sharma, and Shah, 2014; Menad, Kanari, and Save, 2014; Ma and Houser, 2014; Li *et al.*, 2015; Lakshmi, Kumar, and Thyagarajan, 2016; Zhang *et al.*, 2016; Jiang *et al.*, 2018). In other words, the transformation of nonmagnetic compounds of transition metals in slags to ferrites with high magnetic susceptibilities by controlled solidification and crystallization of molten slags is a promising method to increase the recovery of values from steelmaking slags.

As highlighted earlier, certain phases, such as $MnFe_2O_4$ and (Mg,Fe,Mn)O, in crystallized slags possess magnetic properties that can potentially be exploited in their magnetic separation from the bulk non-magnetic components of steelmaking slags (Semykina and Seetharaman, 2011; Semykina, 2013; Shatokha *et al.*, 2013; Menad, Kanari, and Save, 2014; Ma and Houser, 2014; Jian *et al.*, 2017). Semykina (2013) proposed a slag utilization technique that involves the ambient air oxidation of molten CaO-FeO-SiO_2 and CaO-FeO-SiO_2-MnO-Cr_2O_3 slags in order to precipitate ferrite phases that can then be recovered by magnetic separation. The proposed oxidation reactions are shown in Equations [11]–[13] (Semykina, 2013):

$$2(FeO)_{slag} + MnO_{(slag)} + \frac{1}{2}O_{2(air)} \rightarrow MnFe_2O_{4(s)} \quad [11]$$

$$3(FeO)_{slag} + \frac{1}{2}O_{2(air)} \rightarrow Fe_3O_{4(s)} \quad [12]$$

$$FeO_{slag} + 2CrO_{(slag)} + \frac{1}{2}O_{2(air)} \rightarrow FeCr_2O_{4(s)} \quad [13]$$

Based on the thermodynamic possibility of phase transformations in liquid slags highlighted in Equations [11]–[13], Shatokha *et al.*, (2013) also investigated the selective recovery of iron and manganese values from oxidized CaO-SiO_2-FeO and CaO-SiO_2-FeO-MnO slags. In recent studies, Jiang *et al.* (2018) investigated the oxidation behaviour of BOF slag (CaO-SiO_2-MgO containing 10–

35 wt.% FeO_x) under ambient air and selected temperature (1000–1100ºC) conditions. Their findings also proved the potential to form magnetic spinel phases, in the form of magnetite (Fe_3O_4) and magnesio-ferrite ($MgFe_2O_4$), from the Fe-containing species in the slag. The studies highlighted so far substantiate the underlying hypotheses that the stable transition metal ferrites produced in liquid slag during solidification are amenable to separation from the bulk of the solidified slag by magnetic separation (Semykina and Seetharamn, 2011; Shatokha *et al.*, 2013; Semykina, 2013; Jiang *et al.*, 2018).

Recovery of sensible energy in molten slags—The pyrometallurgical extraction of metals is associated with high energy consumption. For example, the iron and steel manufacturing industry is one of the most energy-intensive industries, accounting for about 5–6% of global energy consumption (Duan *et al.*, 2017). According to Barat *et al.* (2011), the energy consumption in high-temperature metal extraction processes is distributed between metal, slag, off-gas, and environmental losses. Depending on the slag/metal ratio and tapping temperatures, the slag thermal requirements in these processes can vary from 10–90% of the output energy (Barati, Esfahani, and Utigard, 2011). Due to the high volumes of iron and steel production, the production of ferrous metals accounts for over 90% of the available energy associated with slags. Slags also account for a significant amount of energy output from the energy-intensive production processes of ferroalloys and other commodities. Barati, Esfahani, and Utigard (2011) summarized the historical global production of selected metallurgical slags, and the estimated energy content associated with these materials (Table IX).

Since the molten slags are tapped from smelting and refining furnaces at high temperatures, they contain a lot of sensible energy that can be harnessed. In order to increase sustainability in the high-temperature extraction of metals, several researchers have focused on the recovery of energy from the dry granulation of these molten materials (Bisio, 1997; Barati, Esfahani, and Utigard, 2011; Zhang *et al.*, 2013; Liu *et al.*, 2015, 2016; Rycroft, 2014). To date, several energy recovery processes have been investigated and developed, based primarily on the physical and chemical processing of these materials. Typical physical energy recovery processes include (Bisio, 1997; Rycroft, 2014; Barati, Esfahani, and Utigard, 2011):

(1) Dry granulation processes, wherein liquid slag is continuously broken down into droplets and the heat is recovered, firstly by the solidification of the droplets passing through a dry cooling stage, followed by the further recovery from the solid granules as they cool to ambient temperatures.
(2) Granulation by air blast, where a stream of liquid is broken down into droplets by high-pressure air jets. The air granulation process allows for the concomitant rapid cooling of slag and heat recovery from the air blast via a heat exchanger.
(3) Granulation by solid slag impingement, where the liquid stream is broken up into particles by a stream of recycled solid slag particles, and the energy is transferred to air for steam generation by convection in a multi-step fluidized bed.
(4) Centrifugal granulation, encompassing processes such as the rotating drum process, rotary cup atomizer, spinning disc atomizer, and rotating cylinder atomizer. The processes highlighted so far integrate the physical phenomena of slag solidification using centrifugal force or impinging jets, and heat transfer using media such air and steam in fluidized bed and/or countercurrent heat exchangers.

Extensive research has also been conducted on the recovery of sensible energy in liquid slags using chemical methods (Maruoka *et al.*, 2004; Purwanto and Akiyama, 2004; Barati, Esfahani, and Utigard, 2011; Duan *et al.*, 2014a, 2014b; Sun *et al.*, 2015a, 2015b, 2016b; Duan, Yu, and Wang, 2017; Sun *et al.*, 2017). Several researchers have proposed the use of molten slags in the gasification of coal and biomass to produce synthetic fuels and/or using the waste heat in liquid slags in methane-steam reforming reactions. In this case, it is proposed that the hot slag not only acts as a thermal medium but also as a catalyst for the gasification and reforming reactions. Using blast furnace slag waste heat as a thermal agent, Duan *et al.* (2014a) developed a technological and exergy analysis model based on gasifier system using coal/waste gas gasification. The findings indicated a recovery efficiency of 85% for the blast furnace slag waste heat, and an 80% conversion of the CO_2 to syngas in the waste gas. In similar studies, Duan *et al.* (2014b) conducted a thermodynamic analysis of steam gasification of coal to produce hydrogen-rich gas using blast furnace slags

Table IX

Historical global production and the estimated energy content of selected metallurgical slags (Barati, Esfahani, and Utigard, 2011)

Process	Tapping temp. (ºC)	Slag rate (kg/t metal)	Slag production (Mt) (2008)	Slag enthalpy (GJ/t) (2008)	Heat value (TWh/a) (2008)
	1500	250-350	236	1.6	105
Steel (BOF)	1550	126	112	1.3	40
Steel (EAF)	1600	169	69	1.8	35
HCFeCr	1650	1.1–1.6	58	1.6	26
Copper	1250	220	34	1.2	11
Nickel	1350	530	8.4	1.4	3.3
PGMs	1350	n.a	1.5	1.5	0.6

n.a: not available

as the heat carrier. The findings indicated that: (1) the amount of hydrogen at a steam-to-coal ratio of 2 increased until peak temperatures of 450–550ºC, (2) increasing the pressure from 0.1 MPa to 10 MPa at 775ºC adversely affected the yield of hydrogen, and (3) an increase in slag basicity had a positive effect on the yield of hydrogen, particularly at low temperatures.

Sun *et al.* (2015a) also proposed a process to recover waste heat and iron from high-temperature (1450–1650ºC) steelmaking slags by integrating the gasification of coal with the treatment of steel slag. The proposed process resulted not only in an improved yield of syngas, but also enabled the potential recovery of iron units by magnetic separation due to the oxidation of FeO to Fe_3O_4 in the slag. Sun *et al.* (2015c) investigated the kinetics of low-temperature (250–500ºC) gasification of biomass combined with heat recovery from slag. Based on the catalytic effect of slags, the findings indicate that the addition of slag increased the production of syngas in the temperature range 425–500ºC. In earlier studies, Maruoka *et al.* (2004) demonstrated significant financial benefits of recovering waste heat in the steelmaking industry from the CH_4-H_2O reforming process using a chemical recuperator. Similarly, Purwanto and Akiyama (2006) studied the production of hydrogen by introducing a CH_4-H_2O gas mixture into a packed bed of steelmaking slag. Based on the conditions investigated (700–1000ºC, constant flow rate, and atmospheric pressure), the findings indicate that the slag acted not only as thermal medium but also as a catalyst to the reforming reactions (Purwanto and Akiyama, 2006).

The findings highlighted so far present a strong business case for further optimizing the processes to recover the sensible energy from molten metallurgical slags. The kinetic models developed in the various studies also demonstrate the multiple roles of metallurgical slags in coal/biomass gasification and methane-steam reforming processes; that is, not only as a thermal carrier but also as a catalyst and reactant (Maruoka *et al.*, 2004; Purwanto and Akiyama, 2004; Duan *et al.*, 2014a, 2004b; Sun *et al.*, 2015a, 3025b, 2015c, 2016b, 2017; Duan, Yu, and Wang, 2017).

Synthesis of functional materials from solidified slags—Metallurgical slags have a great potential as feedstocks in the production of novel functional materials such as sintered glass-ceramics (Rawlings, 1994; Gorokhovsky *et al.*, 2002; Rawlings, Wu, and Boccaccini, 2006; Zanotto, 2010; Ponsont and Bernado, 2013; Bai *et al.*, 2016; Liu, Zong, and Hou, 2016), porous ceramic materials (Tanaka, Yoshikawa and Suzuki, 2009; Nikitin, Kol'tsova, and Beliy, 2013; Suzuki, Tanaka, and Yamasaki, 2014; Sun and Guo, 2015), ceramic bricks (Nel and Täuber, 1970; Shih, White, and Leckei, 2006; Quijorma, San-Miguel, and Andrés, 2011; Karayannis *et al.*, 2017), functional zeolites for waste water treatment (Kuwahara *et al.*, 2010; Chen *et al.*, 2012; Li *et al.*, 2016), and refractory materials (Gu *et al.*, 2018), among other applications. Of particular interest in the context of this paper is the synthesis of functional glass-ceramics and zeolites from metallurgical slags.

Glass-ceramics are fine-grained polycrystalline materials formed when glasses of suitable composition are heat treated to undergo controlled crystallization to the lower energy crystalline state (Rawlings, 1994; Bai *et al.*, 2016). To date, sintered functional glass-ceramics have attracted considerable attention as low-cost alternatives to conventional glass-ceramic manufacturing (Ponsot *et al.*, 2015). These materials have novel applications in a wide variety of industries, ranging from catalysis to thermal insulation, lightweight structural support, cooking ware, ceramic tiles, and military applications (Rawlings, Wu, and Boccaccini, 2006; Zanotto, 2010; Ponsot *et al.*, 2015). In that regard, the manufacture of sintered glass-ceramics with functional properties such as catalytic activity, optical and electrical properties, and desirable mechanical properties, using CaO-Al_2O_3-SiO_2-MgO-$FeOx$- based metallurgical slags has also drawn considerable attention in recent years (Gorokhovsky *et al.*, 2002; Chinnam *et al.*, 2013; Ponsot and Bernado, 2013; Bai *et al.*, 2016; Liu, Zong and Hou, 2016). Liu, Zong and Hou (2016) investigated the effect of MgO/Al_2O_3 addition on crystallization behaviour, microstructure, mechanical properties, and chromium solidification performance of glass-ceramics synthesized using EAF slag. Their findings indicate that MgO/Al_2O_3 ratios of 1:1 and 1:2 resulted in improved mechanical and chromium fixation properties. Ponsot and Bernado (2013) investigated the manufacture of self-glazed glass-ceramic foams from iron-rich metallurgical slags and recycled glass. Homogenous foaming and specific mechanical properties comparable to those of conventional porcelain stoneware sintered above 1100ºC were obtained. Bai *et al.* (2016) investigated the synthesis of glass-ceramics using high-carbon ferrochromium (HCFeCr) slag and waste glass. In addition to microstructural and mechanical properties comparable to other slag-based glass-ceramics, the findings demonstrated the evolution of stable $(Mg,Fe)_2SiO_4$, $MgAl_2O_4$, $Mg(Al,Cr)_2O_4$, and Cr_2O_3 phases which are desirable in reducing toxicity of the HCFeCr slags (Bai *et al.*, 2016). As demonstrated by the various studies, the incorporation of metallurgical slags into functional glass-ceramics not only increases the low-cost recyclability and re-use opportunities of the slags, but also results in the formation of stable phases with low TCLPs.

Synthesis of functional zeolites from slags—Zeolites are crystalline porous solids with intricate pore and channel systems in the molecular sieve range of 0.3–3 nm (Li *et al.*, 2016). Basically, zeolites are crystalline aluminosilicates, with group I or group II elements as counter-ions (Simate *et al.*, 2016). The structure of zeolite consists of a framework of $[SiO_4]^{4-}$ and $[AlO_4]^{5-}$ tetrahedra linked to each other at the corners by sharing oxygen atoms (Simate *et al.*, 2016; Li *et al.*, 2016; Mallapur and Oubagaranadin, 2017). In general, these materials are commonly used as commercial absorbents and catalysts (Simate *et al.*, 2016; Li *et al.*, 2016; Mallapur and Oubagaranadin, 2017). Functional zeolites produced by the hydrothermal treatment of silicate-rich slags have widespread applications in waste water treatment and in the remediation of AMD (Kuwahara *et al.*, 2010; Lin *et al.*, 2016; Simate *et al.*, 2016). For example, Li *et al.*, (2016) investigated the synthesis of sodium aluminate ($6Na_2O\cdot 6Al_2O_3\cdot 12SiO_2$) and sodalite ($4Na_2O\cdot 3Al_2O_3\cdot 6SiO_2$) zeolites using Ti-bearing EAF slag as a precursor. The performance of the crystalline zeolites, synthesized based on the design parameters such as molar ratios of SiO_2/Al_2O_3 (ratio 2:1) and H_2O/Na_2O (ratio

100:1), and hydrothermal temperature (140°C) and time (3 hours), was evaluated based on the removal of Cu^{2+} ions from aqueous solutions. In other studies, Kuwahara *et al.* (2010) proposed an acid leaching and NaOH precipitation method to synthesise hydrotalcite-like compounds ($Mg_3Al(OH)_8Cl\cdot 2H_2O$ and $Mg_6Al(OH)_{16}CO_3\cdot 4H_2O$) and zeolites from blast furnace slags. Based on the absorptive capacity of the phosphate ions, the comparative performance of synthesised hydrotalcite and zeolite materials was significantly higher at 40 mg and 333 mg of phosphorus per gram, respectively, compared to 1.5 mg of phosphorus per gram for raw slag. Due to the novel heavy-metal absorption properties and the environmental challenges associated with heavy-metals-laden waste water and AMD, the synthesis of functional zeolites using metallurgical slags as precursors thus provides low-cost opportunities for the utilization of these materials.

Recycling and recovery of value metals from post-consumer wastes

As highlighted in Table I, post-consumer waste refers to different types of waste generated by households or commercial, industrial, and institutional facilities in their role as end-uses of the products, which can no longer be used for their initial purposes. The increasing global population, coupled with advances in technology and increased consumption of natural resources, has led to dramatic increases in the flow of these anthropogenic materials (Johnson *et al.*, 2007; Dodson *et al.*, 2012; Jin, Kim, and Guillaume, 2016; Park *et al.*, 2017). As a result, post-consumer wastes present environmental challenges in modern society as any leftover and/or obsolete products are often discarded. Typically, these anthropogenic materials are disposed in landfills, thereby creating environmental challenges (Birloaga *et al.*, 2013; Corder, Golev, and Guirco, 2015; Ongondo, Williams, and Whitlock, 2015; Ndlovu, Simate, and Matinde, 2017). On the other hand, increasing the recovery of metals from post-consumer products in anthropogenic spaces has intrinsic business, socioeconomic, and technological advantages (Hagelüken *et al.*, 2016; Sun *et al.*, 2016). In other words, considering these materials as anthropogenic sources of secondary resources that can be recycled and re-used can result in reduced environmental damage while supporting sustainable development goals through the efficient use of resources (Ongondo, Williams, and Whitlock, 2015; Corder, Golev, and Guirco, 2015; Ndlovu, Simate, and Matinde, 2017).

The recycling and re-use of post-consumer waste provides incentives to shift away from the traditional linear use of resources to a closed-loop, cyclical utilization of resources that allows for maximum recovery of resources from waste (Cossu, 2013; Ongondo, Williams, and Whitlock, 2015; Sun *et al.*, 2015d, 2016). Of particular interest in the context of this paper is the recovery of precious metals from spent autocatalysts and electronic waste (e-waste). Spent autocatalysts and e-waste contain significant amounts of high-value metals such as Pt, Rh, Pd, Cu, Ni, and Au, among others. To date, extensive research based on pyro-metallurgical and hydrometallurgical processing, or combinations of both, has been conducted in order to recover precious and other valuable metal elements from these types of wastes (Johnson *et al.*, 2007; Cui and Zhang, 2008; Birloaga *et al.*, 2013; Binnemans *et al.*, 2013; Itoh, 2014; Yang, Zhang, and Fang, 2014; Akcil *et al.*, 2015; Sun *et al.*, 2015c; Corder, Golev, and Guirco, 2015; Ongondo, Williams, and Whitlock, 2015; Jadhav and Hocheng, 2015; Jin, Kim, and Guillaume, 2016; Sun *et al.*, 2015d, 2016; Lu and Xu, 2016; Hagelüken *et al.*, 2016; Ndlovu, Simate, and Matinde, 2017; Wang *et al.*, 2017b).

Pyrometallurgical processes involve heating the waste materials at high temperatures to recover the valuable metals. Typical unit processes such as plasma smelting, conventional submerged arc, top submerged lance, chlorination, and volatilization, among others, have widely been adopted in the processing of these wastes (Cui and Zhang, 2008; Jadhav and Hocheng, 2015; Wang *et al.*, 2017b). Despite the fast kinetics and favourable thermodynamics as a result of high-temperature conditions, pyrometallurgical processes have several inherent constraints such as high energy footprints and the need for intricate off-gas cleaning systems (Tuncuk *et al.*, 2012; Jadhav and Hocheng, 2015). Due to the presence of polymers, ceramics, and halogenated flame retardants in some of these waste streams, the formation of highly toxic volatile compounds such as polychlorinated dibenzo-p-dioxins and dibenzol furans (PCDD/Fs), polybrominated diphenyl ethers (PBDEs), and polycyclic aromatics, among others, is also a major challenge in most of these processes (Johnson *et al.*, 2007; Tuncuk *et al.*, 2012; Jadhav and Hocheng, 2015; Lu and Xu, 2016; Cui and Anderson, 2016). As a result of the structural challenges associated with conventional pyrometallurgical processes, hydrometallurgical processes have been developed as alternatives in order to improve the economics of extraction of valuable metals from spent autocatalysts and e-wastes.

To date, extensive research has been conducted to recover precious metals from spent autocatalysts and e-waste using both conventional aqueous and emerging non-aqueous leaching and chelating media (Tuncuk *et al.*, 2012; Birloaga *et al.*, 2013; Binnemans *et al.*, 2013; Itoh, 2014; Park *et al.*, 2014; Yang *et al.*, 2014; Sun *et al.*, 2015d; Jadhav and Hocheng, 2015; Corder, Golev, and Guirco, 2015; Ongondo, Williams, and Whitlock, 2015; Sun *et al.*, 2015e; 2016; Lu and Xu, 2016; Cui and Anderson, 2016; Ndlovu, Simate, and Matinde, 2017; Wang *et al.*, 2017b). So far, several processes have been developed to recover metals using acidic media (Syed, 2006; Park and Fray, 2009; Yang, Zhang, and Fang, 2014; Jadhav and Hocheng, 2015; Cui and Anderson, 2016), complexing agents such as cyanide, halides, thiosulphate, and thourea (Jadhav and Hocheng, 2015; Cui and Anderson, 2016; Akcil *et al.*, 2015; Lu and Xu, 2016) and biohydrometallurgy (Cui and Zhang, 2008). Recently, attention has focused on the recovery of precious elements from spent autocatalysts and e-waste using ionic liquids (Binnemans *et al.*, 2013; Sun *et al.*, 2015d; Park *et al.*, 2014; Rzelewska *et al.*, 2017). As discussed earlier, ionic liquids are low-temperature molten salts comprising cations and organic/inorganic anions, and are increasingly being investigated for the extraction of metals due to their desired properties such as non-volatility, low toxicity, good ionic conductivity, and wide electrochemical potential window (Park *et al.*, 2014). The recovery of valuable metals from spent autocatalysts and e-waste using ionic liquids provides

several advantages such as high selectivity, ability to recover the valuable metals in their pure state, and the ability to detoxify and produce environmentally benign residues.

Although research into the recycling of spent autocatalysts and e-waste for metal recovery and the generation of other valuable products has been widespread, large scale production plants dedicated solely to the processing of these wastes material are still to be universally adopted. This is mainly due to the fact that metal recycling plants are capital-intensive and expensive to set up. Furthermore, the success of these interventions is also predicated on a consistent and uniform supply of waste materials, thereby competing with similar products from primary raw materials (Johnson *et al*., 2007; Dodson *et al*., 2012; Corder, Golev, and Guirco, 2015; Sun *et al*., 2016). As a result, the major driver for the success of recycling initiatives so far has been to blend and incorporate the recycled materials with other materials into the existing primary and/or secondary processes for metal extraction.

Integration with engineering education

The previous sections discussed in detail the technical aspects of recycling and re-use of mining and metallurgical wastes. As discussed earlier, the recycling and re-use of these waste materials is important from an economic, environmental, and industrial point of view. Therefore, in order to maximize the benefits, new systems thinking approaches are crucial to leverage on the circularity and recyclability of some of these materials. For example, the sustainability of the mining and metallurgical industries can be greatly enhanced by incorporating an environmentally friendly way of treating the process residues and/or by-products generated in the extraction of metals and mineral commodities. In other words, improving the circularity of the various waste materials requires multidisciplinary recycling and re-use initiatives that take into account the initial stages of developing a new material and/or process (Person, 1971; Pech-Canul and Kongoli, 2016). Based on the aforementioned discussions, several researchers have highlighted the early career education and training of design and process engineers as essential components in the dynamics of environmental protection and sustainable consumption of natural resources (Bishop, 2000; Gutierrez-Martin and Hüttenhain, 2003; Boyle, 2004; Hering, 2012; Pech-Canul and Kongoli, 2016; UNESCO, 2010. 2017).

To date, the integration of aspects of sustainability into the engineering curriculum has broadly been driven by the need to address complex societal and socio-economic problems arising from the negative externalities of industrial processes and products (Bishop, 2000; Gutierrez-Martin and Hüttenhain, 2003; Boyle, 2004; Kastenhofer, Lansu, and van Dam-Mieras, 2010; Pech-Canul and Kongoli, 2016; UNESCO, 2017). Obviously, the protracted industrial growth experienced in the 21st century has not only led to the unsustainable use and consumption of natural resources, but has also resulted in unstainable levels of environmental degradation and pollution (Kastenhofer, Lansu, and van Dam-Mieras, 2010). To a greater extent, the emerging paradigms on environmental stewardship have ignited a debate on the role of engineering education in preparing graduates' interdisciplinary and transdisciplinary ethos of sustainable development (Tilbury, 1995; ABET, 2009; Kastenhofer, Lansu, and van Dam-Mieras, 2010; Vanderburg, 1999; Guerra, 2017; UNESCO, 2010; 2017). For example, UNESCO (2017) explicitly proposed the need for responsive education systems based on pedagogies that empower learners and include sustainability principles. In particular, UNESCO (2017) clearly stipulates the need for institutions to initiate and disseminate engineering curricula that integrate sustainability and sustainable development.

In view of these emerging paradigms in engineering education, extensive debate is now centred on the holistic integration of these issues into the broad engineering teaching and learning space. In principle, the various education systems globally have converged on explicit sets of statements and principles on learning outcomes, desired graduate attributes, and professional competencies relevant to sustainable development. Possible approaches include facilitating active learning and development of higher order cognitive skills on sustainability by:

(1) Incorporating environmental aspects in the design tasks of the various courses covering the unit processes producing the waste materials (Vanderburg, 1999; Boks and Diehl, 2006; Zimmerman and Anastas, 2009; Zoller, 2013)
(2) Research-based teaching in order to develop cognitive enquiry skills through experiential learning (Kolb, 1984; Grimson, 2002; Healy, 2005; Zoller, 2013; Hedden *et al*., 2017)
(3) Alignment of individual academics' agency into the teaching and learning of sustainability principles (Cebrián and Junyent, 2015; Colombo and Mattarolo, 2017; Hedden *et al*., 2017).

Conclusion

The increasing global population, coupled with advances in technology and increased consumption of resources, has led to a dramatic increase in the flow of anthropogenic materials. However, the linear model of resources consumption obviously results in high levels of waste from the mining, metallurgical, and industrial processing of virgin raw materials. Inasmuch as the production of mining and metallurgical wastes is inevitable in the production of industrial materials, waste management practices to date have focused on how to manage the way in which the waste is generated and disposed of. However, the traditional approach to managing mining and metallurgical waste is not only unsustainable, but also discounts the circularity of most of these anthropogenic materials. Based on the emerging paradigm of a circular economy model that mandates the reduction, recycling, and re-use of wastes, this paper provided a critical review of current and emerging research on the recycling and re-use of mining and metallurgical wastes. Firstly, the paper categorized the various types of wastes in the mining and metallurgical industries and introduced some of the legislative framework governing these types of wastes. Secondly, the paper highlighted the ongoing research and emerging trends in the recycling and re-use of selected mining, metallurgical and post-consumer wastes. With a view of igniting debate, the paper provided a brief overview of the converging principles on the integration of the key aspects of sustainability into the engineering education curriculum.

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Limits to artisanal and small-scale mining: evidence from the first kimberlite mines

by W.P. Nel

Synopsis

The number of people involved in artisanal and small-scale mining (ASM) has grown quickly to about 40.5 million, compared to 7 million in industrial mining. Furthermore, the ASM sector is contributing significantly to global mineral supply and new opportunities are arising for ASM in an evolving mining ecosystem. Given this growth trend, it is important to ask whether ASM is likely to be successful in the mining of all types of orebodies. The history of early South African diamond mining suggests that the mining of a massive ore deposit by numerous artisanal and small-scale miners is likely to result in poor safety conditions as the depth of mining increases. Early photographs taken at the Kimberley mine showed a very uneven pit floor with leads-lags between the claims. This raises the question of why neighbouring miners did not ensure safer working conditions for each other. Two models described in the paper illustrate why there is likely to be a lack of cooperation and coordination between miners to address this and other safety-related problems. The dynamics of multiple claim holders mining next to one another at increasing depths are analysed, and it is shown that a consolidation of claims into a single firm per kimberlite pipe was required for improved planning, coordination, safety, efficiency, and sustainability.

Keywords

artisanal and small-scale mining (ASM), coordination, mine management and economics, mine safety, rules, Theory of the Firm.

Introduction

Diamonds are mined from alluvial deposits, kimberlite pipes, and lamproites by artisanal miners and firms of different sizes. Humanity has known diamonds for thousands of years, and diamonds were first mined in countries such as India and Brazil from alluvial deposits. It was, however, the 1867 discovery in South Africa that resulted in a big increase in the global supply of diamonds and the launch of the modern diamond market (Janse, 2007; Levinson, 1998). 'Dry diggings' in kimberlite pipes located in South Africa during the second half of the 19th century led to Emil Cohen's suggestion that such diggings were conducted in cylindrical 'pipes' that represented volcanic conduits for diamonds that were brought up from many kilometres below the Earth's surface.

Kimberlitic and related magma types such as lamproites are the primary sources of diamonds (Levinson, 1998; Robb, 2005). Because this fact was initially unknown to the diggers, they approached such diggings in a similar manner to 'wet' (alluvial) diggings, expecting to reach bedrock after a few feet (Turrel, 1987; Meredith, 2008). The dynamics of these diggers, mining next to one another at increasing depths, are analysed in this paper and an attempt is made to illustrate that the consolidation of claims into a single mine per kimberlite pipe was required for improved planning, coordination, safety, and efficiency. History shows that the mining of a single kimberlite pipe at increasing depths by multiple claim-holders next to one another is not sustainable over the full potential life of such an orebody.

In its early years, before the existence of South African mining companies such as De Beers, the diamond mining industry consisted of hundreds of individual diggers and claim owners who were initially self-regulated by rules that, for example, opposed concentration of ownership and thus kept barriers for new diggers to enter the industry at very low levels. The diggers elected persons to represent them on 'Miners' Committees', which were responsible for making and enforcing rules. These Miners' Committees, representing hundreds of individual digger-entrepreneurs, were the first form of organization on the South African diamond fields. The focus of this paper is on how it became necessary, despite the initial anti-consolidation ('anti-monopoly') rules, for ownership at the level of a single kimberlite pipe to be consolidated. The result was that all the claims on a specific kimberlite pipe became consolidated under a few large owners, including companies, and eventually by a single company.

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The mining of claims at increasing depths resulted in increased safety risks, associated loss of life, and water inside the pit, requiring greater cooperation and coordination between the hundreds of individual claim-holders. The fortunes of claim owners varied and some diggers may have had little or no working capital to address such problems.

Claim and concessionary holders did not cooperate much, as can be seen from the duplication of equipment such as windlasses and 'stages' in Figure 1. The photograph was taken at a time when the roadways were removed because they had become unstable and posed a safety risk. The photograph also illustrates that digging at the different claims proceeded at different rates, giving rise to leads-lags and an uneven pit floor. Williams (1905) describes the scene at Colesberg Kopje as a 'jumble of holes, pits and burrows with no attempt to secure any system or union in mining'.

This case study illustrates that, under certain conditions, large-scale mining may result in greater efficiencies and lower safety risks than small-scale mining. This paper is about organizational change on the early South African diamond fields, the dynamics of numerous owners mining side by side, and contributing to the still incomplete Theory of the Firm (Demsetz, 1988).

The growth of informal mining and the need for formalization

The purpose of this section is to provide some background information on the latest developments in artisanal mining and the associated challenges.

The increased number of people involved in artisanal mining can be ascribed to a number of causes. One is that not enough jobs are created in the formal economies of certain countries. This is the situation in Zimbabwe, where many citizens left the country and others had to find a way of making a living in the informal sector of the economy because of depressed conditions in the formal sector. According to one source the informal economy of Zimbabwe, estimated to be larger than 60% of the gross domestic product (GDP), is now the second biggest in the world. This can be compared with the most developed economies, those of Switzerland and Austria, where the informal sector comprises only 7.2% and 8.9% respectively of the GDP (Medina and Schneider, 2018).

Figure 1—Little cooperation between diggers is illustrated by the numerous ropes from 'stages' to claims (http://www.milnerlodge.co.za)

In South Africa many job losses have occurred in the minerals industry and this may be one reason why illegal mining by the so-called *zama-zamas* has increased. Today, infrastructure such as shafts created by large-scale miners is used by artisanal miners to extract some value from orebodies that large mining companies, with high fixed costs, are no longer able to do. The identification of such niches by artisanal miners has resulted in the expansion of the mining ecosystem. According to Omarjee (2017), South Africa has more than 6 000 abandoned mines, some of which are currently been exploited by artisanal miners.

Although artisanal mining has advantages such as the generation of income for its participants, it also has certain disadvantages. Apart from the various potential problems arising from informal mining not adhering to some or all of government's safety- and environmental-related laws and regulations, there is also an impact on government income because it is relatively easy for the informal sector not to pay taxes and get away with it. There is, therefore, a need to formalize informal mining, not only to broaden the tax base of government but also to improve the working conditions of such miners.

Two recent initiatives to legalize and/or formalize informal mining are those by the Department of Mineral Resources (DMR) of South Africa and by Birrell Mining International, who re-opened the Klipwal gold mine in KwaZulu-Natal. The DMR recently announced an initiative to legalize the extraction of diamonds from 'floors' at Ekapa, and at Klipwal former *zama-zamas* are now working for the mine (Wood, 2017).

There are, however, limitations to ASM. At the first kimberlite mines consolidation and formalization of mining took place to improve safety and working conditions. The mining of large, massive orebodies at depth by artisanal miners is, therefore, not recommended based on arguments in this paper.

From alluvial to kimberlite diamond mining: 'The orebody dictates'

The purpose of this section is to illustrate that whereas alluvial diamond deposits could be successfully mined by artisanal miners, that was not the case for the mining of kimberlite pipes at depth.

Alluvial diamonds were mined at the Deccan diamond fields of India, in Brazil, and the Urals before the exploitation of alluvial fields in South Africa started in the 1860s (Williams, 1905). The diggers in South Africa initially mined diamonds from claims along the banks of the Orange and Vaal rivers at places such as the Mission Station of Hebron, the kopje (hill) near the Klip-drif camp (later called Barkley-West), and Pniel (Joyce, 1988; Williams, 1905). These were all alluvial diggings.

At the time when alluvial diggings first started in South Africa, nobody knew that the rock that later came to be called kimberlite was the primary source of diamonds. Diamond-bearing kimberlites were soon found at a number of places such as the De Beers, Kimberley, Dutoitspan, and Bultfontein mines and were called 'dry diggings' because they were not closely associated with rivers. It was at Dutoitspan that a digger discovered that the kimberlite rock was, unlike alluvial diggings, not restricted in depth down to the bedrock of a

river. The mining of diamond-bearing ground at Dutoitspan at deeper levels resulted in ground slides and rockfalls. It soon became clear that the artisanal and small-scale way of mining claims situated at alluvial deposits were not suitable for the mining of kimberlite pipes at deeper levels. The differences and similarities between the two types of ore deposits and mining methods are summarized in Table I.

Table I briefly describes some of the characteristics of alluvial *versus* kimberlite mining. When the first 'alluvial' miners started to mine kimberlite pipes they had no prior knowledge of the nature of the orebody, the implications for mining, and how mining practice had to be changed as depths increased beyond those they were used to at alluvial claims. It could be argued that these changes led to the transition from ASM to large-scale, capital-intensive mining associated with the firm as organizational structure over the years, as the depth of mining increased and underground mining methods had to be used to exploit the massive vertical orebodies at ever-deeper levels. Such a situation is well encapsulated by the slogan 'the orebody dictates', which is a basic tenet of the so-called 'Harmony Way' (Lanham, 2006).

Self-governance and regulation: the rules of the game

The hundreds of digger-entrepreneurs on the South African diamond fields were initially self-governed through organizations such as 'Diggers' Committees' (Williams, 1905, p. 146) and/or the Diggers' Mutual Protection Association (Worger, 1987).

Such organizational structures served the diggers' interests as a group through a system of voting and rules. According to Williams (1905) these rules worked remarkably well despite being simple in nature. However, as the depth of mining increased and safety and working conditions deteriorated, some of the rules had to be changed in an attempt to address such conditions. Some of the rules on the South African diamond fields may have been proposed by diggers who were involved in previous rushes, for example the Ballarat and Californian gold rushes.

In this paper the word 'rule' is broadly used to include not only laws and regulations promulgated by government, but also self-regulating type rules introduced by a group or industry. Most, if not all, self-governing diggers had rules related to claim size. The sizes of claims varied; at Colesberg Kopje (The Big Hole), for example, it was 31 by 31 feet. Colesberg Kopje was divided into more than 400 claims as a result of this. Some of the claims at Colesberg Kopje were later split up by concessions, bargains, and sales (Williams, 1905). Another related rule was that a digger was allowed one claim only (Worger, 1987). At Dutoitspan it was two claims per digger, probably because Dutoitspan was a poorer mine and, therefore, the demand for claims must have been lower (Payton, 1872; Turrell, 1987). Combined, these rules had the purpose of accommodating as many diggers as possible and of opposing concentration of ownership. They initially helped to keep entry barriers for new diggers (or artisanal miners) at very low levels. Any person who wished to do so could become a digger because they were not excluded by high capital barriers – only simple equipment such as shovels, picks, and sieves was used and, therefore, the extraction process was initially labour- rather than capital-intensive (Worger, 1987). Another objective of the rules may have been to keep down rivalry for claims. It seems that the rules worked well on the early diamond fields in comparison to the situation of illegal miners on Gauteng's East Rand today, where groups of illegal miners are turning on one another (Payton, 1872; Magwedze, 2017; TimesLIVE, 2017).

Another rule, referred to here as the 'Use it or lose it' rule, was applied when a claim was not worked for three days (Worger, 1987), or eight days at Dutoitspan (Payton, 1872).

Table I

Comparison of shallow-level claim mining and deeper level pit and underground mining

Alluvial mining by digger-entrepreneurs	Kimberlite mining
Secondary orebody is mined.	Primary orebody is mined.
Diamonds are irregularly distributed (Williams, 1905).	Diamonds are not homogenously distributed throughout kimberlite pipes.
Digging was difficult because it occurred in thick heavy gravel, which included some boulders.	The weathered kimberlite (yellow ground) was easier to dig (Payton, 1872).
The first diamond claim mining occurred along the banks of rivers (wet diggings)	Claims were situated next to one another in kimberlite pipes (dry diggings).
2D mining. Diggers mined diamond-bearing gravels for a few feet down to the river bedrock.	3D mining. Kimberlite pipes are vertical structures that originate from deep down in the Earth's crust and mantle.
Claim mining at shallow depths involves fewer safety risks.	Large pits (and underground mining) are more complex from a geotechnical and rock mechanics perspective.
Ease of digging is an advantage.	Competency of rock is important for slope stability and underground excavation integrity.
Each digger-entrepreneur decided how to mine his/her own claim. There was little coordination of activities with neighbouring claim owners.	Prior information about the orebody is important for mine design and planning. Management functions such as planning, organizing, leadership, coordination, and control are important for large-scale mining. Operations design and capacity planning are important aspects.
Form of business ownership: sole proprietor.	Preferred form of business ownership: listed company.
Marketing and sales: most diggers sold their diamonds to the representatives of European diamond merchants (Worger, 1987).	The sales of diamonds were later channelled through the Central Selling Organisation after the amalgamation of mines. De Beers launched clever advertising campaigns in the 1900s.

This rule may have had more than one objective, one related to the opening up of ownership while another may have been an attempt to eliminate excessive leads-lags between claims, which are clearly visible in Figure 1. One exception, and one change of rules, are listed in Table II.

The focus of this paper is on how it became necessary, despite the initial anti-consolidation (anti-monopoly) rules and the inertia that opposes change, for ownership at the level of a single kimberlite pipe to be consolidated. The result was that all the claims of a specific kimberlite pipe passed into the hands of a few large owners, including companies, and eventually to a single company.

Three types of consolidation on the South African diamond fields

Three types of consolidation occurred fairly early in the South African diamond industry. They are briefly discussed below.

- The consolidation of ownership of numerous claims at a specific kimberlite pipe, for example at the Big Hole (Kimberley Mine). This form of consolidation resulted in a transition from ASM to large-scale mining (LSM), the latter being closely associated with the firm as form of enterprise and organizational unit.
- The consolidation of ownership of various kimberlite mines, for example the consolidation of the De Beers, Kimberley, Bultfontein, and Dutoitspan mines into De Beers Consolidated Mines (DCM). This resulted in diamond production being confined to fewer producers. A number of persons linked this type of consolidation of ownership to monopolistic practices (see, for example, Montpellier, 1994).
- The consolidation of diamond sales through, effectively, one selling organization. At one time the Central Selling Organisation (CSO) controlled 90% of the world's diamond sales. This consolidation of sales had been described as 'cartel-like behaviour' (Bergenstock, 2001, p. 2; Reekie, 1999; Montpellier, 1994)

It is important to differentiate between these types of consolidation because different reasons are behind each type and the impact ranges from local to global. The cartel-like behaviour that DCM and the CSO were accused of is linked to the aim of controlling global diamond supply. Although the monopolistic behaviour that DCM was accused of cannot be achieved without the first type of consolidation, this paper illustrates that other, completely different reasons and dynamics were behind the first type of consolidation. It is a type of consolidation that should not raise any red flags with a Competition Commission or when anti-competitive laws are designed, as there are sensible reasons behind it. One of the reasons why this type of consolidation is often overlooked in the general literature is because it is of a more technical nature, whereas the second and third forms of consolidation are linked to issues related to market structure, which is much more commonly reported on and more widely applicable to all types of industries.

Differences in diggers' incomes gave rise to an uneven pit floor

Differences in the income and expenses of diggers may explain a number of things, for example, the uneven pit floor illustrated in Figure 1 and the associated safety risks. The incomes of diggers probably differed because of the non-homogenous diamond distribution in a specific kimberlite pipe. Williams (1905, p. 150) confirmed that neighbouring claim owners often had different budgets and some diggers could not sustain themselves for long on the diggings, which were described as 'gambling speculation'.

Diamond quality and grade varied greatly between the kimberlite pipes (Williams, (1905) but, more significantly, the distribution of diamonds in a single kimberlite pipe could be highly erratic, with little or no consistent evidence of a decreasing relationship between grade and depth (Nixon, 1995; Robb, 2005). Clement (1982) studied the De Beers, Wesselton, Dutoitspan, Bultfontein, Finsch, and Koffiefontein kimberlite pipes and reported on grade variations within and among discrete intrusions in the pipes, each pipe being made up of multiple magma intrusions over time. It seems that during each intrusion, different degrees of mixing with the host rock occurred, probably as a result of turbulence and convection. Evidence shows grade differences between different intrusions at a specific depth in a pipe. The DB3 kimberlite intrusion at De Beers was of a higher grade than the other intrusions over a significant part of the pipe, for example (Clement, 1982).

The non-homogenous distribution of diamonds in the first kimberlite pipes probably explains, largely, why some claim owners were luckier or more successful than others – some claims generated better cash flows than others and therefore some diggers received a better return on their labour. Such claims would have been valued higher than those that generated lower cash flows. It is known that some claims were subdivided, and it is therefore quite possible that labour was more concentrated in those areas of the pit, resulting in a faster extraction rate and thus giving rise to an uneven pit floor as illustrated in Figure 1. It is probable that

Table II

Exceptions and changes to rules

Rule	Exceptions and changes
One to two claims per digger	One known exception to this rule was made, for example, in the case of a digger named Rawstone, who was granted an additional two claims as a reward for his discovery of Colesberg Kopje in 1871 (Payton, 1872). This was intended as an incentive for the discovery of new deposits.
One to two claims per digger	In 1874 permission was granted for the holding of ten claims by a single owner (Williams, 1905). By 1881, the limit of ten claims per owner was abolished (Turrell, 1987). The reasons behind this change are described in this paper.

high-income claims were in demand and that some owners subdivided and sold them as another source of income. High-income diggers would have been more able to afford the hiring of additional labour and introducing some labour specialization. In contrast, other claims may have been worked by a single digger who had to climb out of the pit once a container was filled with ore to process it somewhere outside the pit. Diggers who earned more could afford better equipment over time as technology evolved. In 1877, hauling at the Kimberley Mine was done by a mixture of steam winding engines, whims, whips, and windlasses, pointing to the fact that many diggers did not have the capital to acquire the latest technology (Turrell, 1987). Figure 2 illustrates why diggers who could afford the latest technology had a better chance of mining at higher rates. The different depths at which neighbouring claims were worked created a safety risk, somewhat similar to that created initially by the roadways. There was no financial incentive for one digger to wait for a neighbouring, lagging digger to catch up and therefore that type of cooperation and coordination did not take place. Much of the information in this paragraph has been used to construct model 1 in Figure 3.

The causes listed in Figure 3, together with a lack of adequate incentives and the type of authority exercised within a firm to enforce a higher level of planning and coordination between neighbouring workings, resulted in the uneven pit floor illustrated in Figure 1. A model to illustrate the impact of erratic diamond distribution in a kimberlite pipe and how that contributed to this situation is depicted in Figure 3. Diggers did not sample their claims and therefore the average grades of the hundreds of individual claims are unknown. Figure 3 is therefore just a schematic of how grades may have varied between claims, based on the fact that kimberlite pipes generally consist of several discrete bodies that differ in diamond grade and quality (Bliss, 1992). In the model illustrated in Figure 3 grades are divided into just high or low. Other categories such as intermediate grade could be added. That was not done because an additional grade category may not significantly improve the model's value in explaining the uneven pit floor.

As the claims at Colesberg Kopje progressed deeper, a number of problems were experienced. The pit had to be dewatered, slope stability problems increased, and falls of waste rock, which had to be removed from some of the claims, contributed towards increasing costs. Furthermore, the weathered kimberlite (yellow ground) found on the surface turned into blue ground below about 30–60 feet in depth (Meredith, 2008, p. 26). Diamonds were not as easily extracted from the unweathered blue ground. The solution to this problem was to spread the ground out for a few weeks on pads (also called 'floors') to decompose. Some claim owners did not, however, have the working capital to let the blue ground lie in the open for a few weeks. Williams (1905) described the situation as follows:

'The blue ground exposed to the air crumbled away by degrees, but the miners were rarely patient enough to wait for this disintegration, preferring quick returns by pulverising the ground with their shovels and mallets. This was hard work and costly, from the loss in imperfect pulverisation. But the diamond seekers were poor men who could scarcely afford to hold any stock of blue ground for the sake of increasing returns, even if they had been able to guard it.'

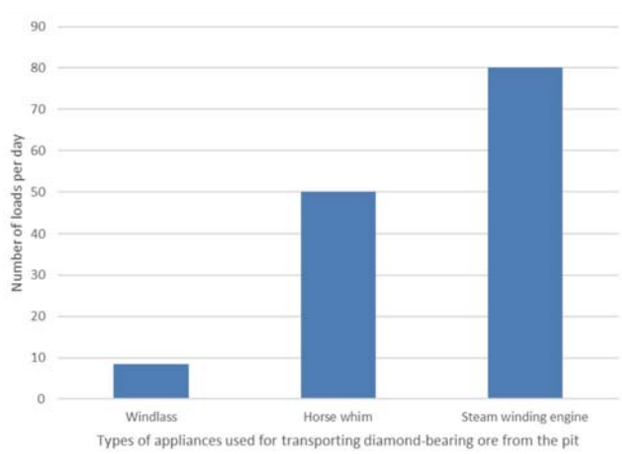
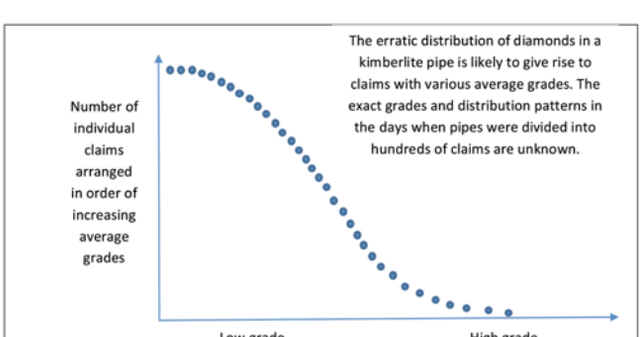


Figure 2—The evolution of hauling technology and associated improvement in productivity (Turrell, 1987, p. 12)



Grade of a specific claim	Low	High
Diggers' sources of income	Diamond sales	Diamond sales and the subdivision and selling sections of claims.
Diggers' income and working capital	Low	High
Diggers' capacity to hire labour	Low	High
Likely concentration of labour	Low	High
Diggers' capacity to invest in the most advanced equipment	Low	High
Likely rate of mining	Low	High
Likely overall result	Uneven pit floor	

Figure 3—Model 1 : a 'two-grade' claim model to illustrate some of the causes of an uneven pit floor

Although diggers did not need much capital, initially, to enter the diamond industry they had to generate income to cover certain expenses. One of the costs was related to a shortage of water at the diamond fields, which was addressed by sinking more pits (Williams, 1905). In addition to running costs, all diggers incurred an economic cost as well, called opportunity cost – the cost of forgoing income that could have been earned by spending their time, energy, and skills on another venture or working for wages.

From the above it is clear that the problems that diggers experienced increased with depth, and so did their expenses. The higher expenses were not necessarily compensated for by increasing grades as the depth of mining increased (Robb, 2005). This was probably one of the reasons why more efficient ways had to be found to mine kimberlite pipes at deeper levels.

Other safety risks and the struggle to solve them

A number of different safety risks arose at the open-pit workings of the Big Hole (Kimberley Mine). One of the earlier risks was associated with weathered kimberlite, which is loose and friable. Loaded carts travelling on the early roadways sometimes toppled over and plunged with the driver, cart, and mule into the pit. This led to the removal of the roadways between the claims as the average mining depth increased and the introduction of other means of transportation such as windlasses, 'stages', whims, and later steam winding engines.

Another safety risk was due to poor slope stability and loose rock that fell into the open pit. Some of the rock surrounding the kimberlite pipe, called 'reef', consisting of decomposed basalt and shale (not to be confused with gold-bearing reef), was prone to breaking loose and falling down the pit (see Figure 4. When diggers started to mine the kimberlite pipes they had no prior knowledge about the depth of the orebody and did not intend mining it at great depths. As a result, at the Kimberley Mine they did not pay attention to slope angles and stability, as is done today during the creation and operation of a large open pit mine.

Reasons have been proposed in the previous section as to why an uneven pit floor existed and conditions deteriorated with depth of mining to such an extent that rules had to be changed. In this section, another reason and mechanism, that of 'different priorities', is briefly discussed and proposed as another cause of organizational change at the Kimberley Mine. In this second model, claim owners are divided into two groups namely the 'rim' and 'centre' owners, as illustrated in Figure 5.

Solving the problems of poor slope stability, falling 'reef', and water in the pit would have required a major undertaking because of the hundreds of separate holdings and different priorities, which complicated coordination of efforts at a pit-wide scale. One of the main tasks of the Mining Board appointed in 1874, and which replaced the original Diggers' Committee at the Kimberley Mine, was to solve the in-pit water and falling reef problems (Williams, 1905). One of the reasons why it was difficult to solve such problems is the conflicting interests, as illustrated by the simple model in Figure 5. Rock falling into the Big Hole affected diggers who had claims closer to the rim ('rim claims') of the pit much more severely and more often than those at the centre ('centre claims'). It seems that one of the largest rockfalls affected only about half the claims inside the pit and, therefore, incidents of rockfalls and rock accumulation were much less likely to affect persons who had claims in the middle of the pit (Williams, 1905). If each claim holder had to pay a levy to the Mining Board and each had a vote on how such money was to be used to address problems, then 'centre' miners would have allocated a significantly lower priority to reef removal compared to 'rim' miners because they were less affected. 'Centre' miners had rational reasons to 'free-ride' because 'rim' diggers had to remove the 'reef' from their claims anyway in order to get to the diamond-bearing blue ground and generate income.

The need for consolidation of ownership and organizational change

The numerous windlasses and stages for the transportation of ore from the pit to the surface (Figure 1) point to a lack of

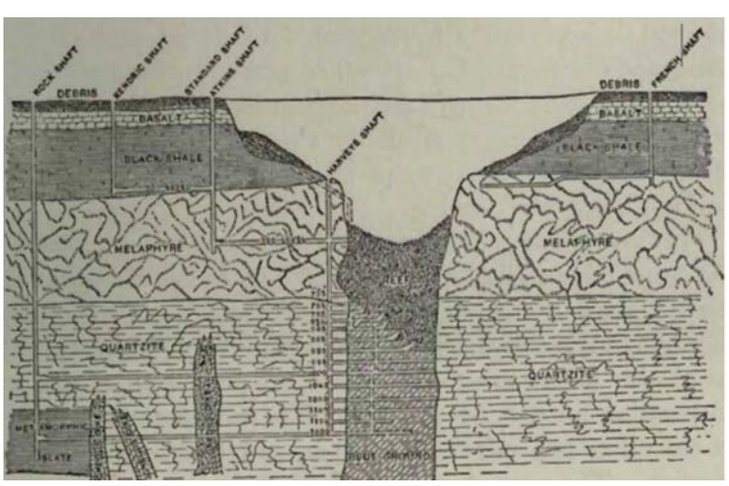
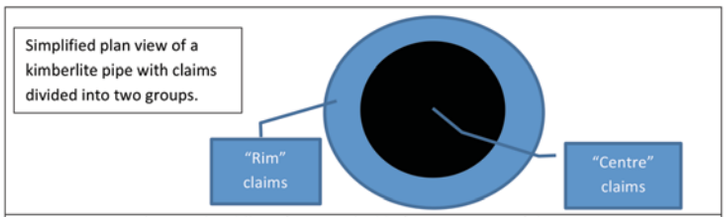


Figure 4—Different rock types surrounding the kimberlite pipe (https://en.wikipedia.org/wiki/Big_Hole)



'Priority' and 'benefit-to-cost' table for two groups of diggers

Problems to be solved	'Rim' miners	'Centre' miners
Rock ("reef") removal	More affected by rock falling into the pit. High priority Benefit to cost ratio – high because of impact on production	Much less affected by rockfalls. Low priority Benefit to cost ratio low
Slope stability	High to medium priority and safety risk. Benefit to cost ratio high to medium	Low to medium priority

Figure 5—Model number 2 illustrates the different priorities of claim owners

Table III

Number of claims and holdings at the Kimberley Mine over time

Year	Number of separate claims and holdings	Form of organization
1872	About 430–470 claims were split up by concessions, bargains, and sales into about 1600 holdings of claims (Williams, 1905, p. 197; Turrell, 1987)	Digger-entrepreneurs and 'Miners' Committee' (Payton, 1872)
1877	About 400 separate holdings (Williams, 1905)	Digger-entrepreneurs and Mining Board
1885	Eleven companies (*e.g.* Standard Diamond Co. Ltd, the 'French Company', W.A. Hall & Co., and Kimberley Central Company) and eight private holdings (Williams, 1905; Chilvers, 1939)	Digger-entrepreneurs and 11 companies
Exact date not known	Three companies only, namely the Kimberley Central Company, the 'French Company', and W.A. Hall & Co. (Chilvers, 1939)	Companies
1888	One single owner, De Beers Consolidated Mines Ltd.	Company

cooperation and coordination between claim holders with regard to transportation. Furthermore, the leads-lags in the same figure also point to coordination failure which resulted in poor safety and working conditions and associated incidents as claims were dug to deeper levels. In 1874 the Mining Board gave permission for the holding of up to ten claims by a single owner in order to address some of these problems, including that of poor economics (Williams, 1905). This relaxation of the first 'antimonopoly' rule of 'one claim per person' resulted in the combination and consolidation of claims, as illustrated in Table III. The 'ten claims per owner' rule was abolished later, by 1881 (Turrell, 1987). The required consolidation of ownership was an enormous task, which was made easier by the poor conditions at some claims and, therefore, the willingness of some claim owners to sell. Increasing costs with depth and increasing opportunity costs related to the discoveries of gold at Barberton and on the Witwatersrand also helped with the consolidation. Some diggers sold their claims and left for the goldfields.

The reduction in the number of entities that held claims at the Kimberley Mine to only three still did not result in holistic and optimal mine design because of jealousy, antagonism, obduracy, and a lack of cooperation (Chilvers, 1939). This resulted in a working arrangement that Chilvers (1939) described as 'most costly' to both the Central and French companies. After further consolidation and amalgamation processes De Beers Consolidated Mines Ltd finally became the single owner of the Kimberley Mine in 1888 (Turrell, 1987; Chilvers, 1939). Once above-ground operations became too dangerous and unproductive, mining proceeded by underground methods. The surface and underground sections were mined to depths of about 240 m and 1 100 m, respectively. The surface section of the Kimberley Mine is thought to be the largest hand-dug excavation on Earth.

The Theory of the Firm

The firm plays a central role in modern economic activity. Despite this and the contributions of numerous researchers, the Theory of the Firm (https://en.wikipedia.org/wiki/Theory_of_the_firm), which explains the nature of the firm, its behaviour, structure, and relationship to the economy, is still incomplete. A well-capitalized firm with competent management and a skilled workforce could have solved many of the problems experienced by the diggers. Such a firm could, for example, afford to build up a stock of blue ground that could be sufficiently exposed to the air before being processed in order to increase processing efficiencies. It could also appoint guards and take other measures to secure the ore while it lay on pads in the open air.

Today it is evident that the public-listed company has a huge role to play in large-scale mining (LSM) because of its capital intensiveness. The case study illustrates, however, that the role of the (mining) firm entails much more than just funding. It is also about centralized mine design and planning, the safety improvements that systematic mining

offers, the economics of consolidation and central control, and so forth. It is therefore today unlikely that hundreds of diggers or claim owners would each have a small part of a deep-level, massive orebody. It is more likely for persons in developed countries to own shares in large listed mining companies either directly or through a pension fund, or to be employed by such a company.

The situation at the first kimberlite mines analysed in this paper points to the impossible task of successfully coordinating the actions of hundreds of individual claim owners as mining progresses to deeper levels. Even after a degree of consolidation took place and a few companies operated the Kimberley Mine, a number of problems persisted, such as the duplication of shafts. The mechanism proposed in Figure 3 applies not only to diamonds mined from kimberlites but all orebodies where grades vary throughout the orebody as indicated, for example, by grade-tonnage relationships. The finding that a massive orebody should be controlled by one firm may even apply to minerals such as some industrial minerals, where the ore quality may be fairly uniform. If such a massive orebody were to be mined by two or more owners then it is unlikely that they would be able to optimally use and share infrastructure and standardize work hours, incentives, technologies, labour complement, and other variables that impact on the mining rate.

In summary, one of the main contributions that the analysis of the case study makes to the Theory of the (mining) Firm is the finding that centralized mine design, planning, control, and authority is required, effectively requiring ownership of a whole massive orebody, such as a kimberlite pipe, by a single firm to overcome the numerous problems described in this paper.

Conclusion

This paper has a number of objectives. One is to demonstrate that consolidation of mine ownership at the (massive) orebody level was required for reasons other than to create monopolies or cartels. It has also been shown that mechanisms exist that will undermine coordination between artisanal miners working adjacent claims in a massive orebody. The dynamics of artisanal miners working at the world's first kimberlite mines are drawn upon to illustrate the central thesis of this paper, which is that it is highly unlikely that massive orebodies can be mined safely and optimally by ASM at depth.

Experience gained from the changes in mine organization and scale of mining at the first kimberlite mines contributed significantly to accumulated knowledge in the areas of mining practice and mine management. The physical and economic conditions at a kimberlite pipe, or any other massive orebody, may result in various problems if such an orebody is subdivided into claims and mined by digger-entrepreneurs at deep levels. A single firm having management control over such an orebody can solve many of the problems related to ASM, especially that of coordination. The large-scale mining of a massive orebody at depth is very likely to result in greater mining and extraction efficiencies and improved safety, and also reduce unnecessary duplication of equipment. The author is not aware of any massive orebody currently being mined at depth by artisan miners unless centrally controlled. A rule for massive orebodies such as kimberlite pipes is therefore proposed: that the authority to oversee the implementation of a centrally designed mine plan for a massive orebody should be the responsibility of a single organization.

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Post-mining excellence: strategy and transfer

by J. Kretschmann

Synopsis

In 2007, a political understanding was arranged to phase out German hard coal mining (GHCM) in a socially acceptable manner by the end of 2018, after more than 200 years of production. This decision required a new strategy in order to prepare the GHCM industry for the post-mining era. To create a sustainable development strategy (SDS), the long-term impacts of GHCM activities concerning the environmental, economic, and social dimensions were analysed systematically and forward-looking. High technical standards in environmental protection and post-mining technologies, and experiences in the development of former mine sites to create jobs, are significant from an international point of view. Besides, the cultural heritage of the mining era was preserved and developed for new purposes. The SDS of the GHCM industry can serve as a role model for mining regions facing similar challenges.

Keywords

sustainable development strategy, German hard coal mining, post-mining era.

Introduction

Principally, mining is a finite business that reaches its end if the deposits are depleted or the raw materials can no longer be mined in an economically profitable way. However, the impact of mining is potentially an infinite one – at least if measured at a human time-scale. Mining always means interfering with geological and ecological systems that cannot be returned to their original condition.

In general, the mining cycle can be divided into three phases:

1. The exploration phase, in which the deposit is investigated to determine its technical feasibility and economic profitability. The period of these undertakings is relatively short and can lead to the launch of mining operations.
2. What follows is the actual mining phase, which usually lasts for a long time and ends at the latest when all deposits are fully depleted. This lifespan may be shortened if the economic conditions (production cost or market price) deteriorate. Nevertheless, mining may be resumed if those conditions become favourable once more (examples are silver mining in Germany or rare earth mining in the USA).
3. The longest phase, however, is the post-mining phase as human interference in geology and nature is usually intensive and irreversible. Mining can lead to consequences that have a permanent adverse impact on people and the environment; therefore, they have to be managed optimally (Kretschmann and Hegemann, 2012).

In the past, mining companies paid the most attention to the first two phases of the cycle, as those were the ones in which they could act profitably in the market. Moreover, many mining nations had not created a legal framework that would oblige companies to handle the impact of active mining in the post-mining phase. As a result, the medium- and long-term impacts that the environment and the people living in the mining regions had to endure were often neglected – and even the economic impact was neglected. Remediating mining damage that occurred years later, such as subsidence damage or the restoration of land and water bodies, represents costs that the entire society has to cover if the funding by the mining companies is not properly defined. In the end, such impacts can lead to a loss of acceptance in politics and among the population. In Germany, the mining companies or their legal successors have to cover the costs. If the former owner is unknown, the federal state has to cover them (Kretschmann 2015).

The objective of this paper is to describe the end of the German hard coal mining industry and the strategy to successfully

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control and manage the challenges, risks, and opportunities of the post-mining era. This strategy is based on a concept of sustainable development of the mining region. Possibilities for knowledge transfer to South Africa are outlined, as well as the necessity of specialized post-mining education.

Sustainability and post-mining

If mining is to be as sustainable as possible, all three phases of the entire cycle have to be integrated, and the requirements of the post-mining phase have to be included in the strategic planning and the operations. As mining cannot be carried out without impact, it is necessary that negative consequences are minimized.

Sustainable development is defined as development that 'meets the needs of the present without compromising the ability of future generations to meet their own needs' (United Nations, 1987, p. 16).

Sustainability is about three basic aims: the sharing of wealth among as many people as possible (social sustainability), a durable and positive economic development (economic sustainability), and the preservation of nature (environmental sustainability) (Kretschmann, 2014).

This three-dimensional approach to sustainability can easily be transferred to post-mining activities. Sustainable management of mining impacts means that environmental damage caused by emissions, subsidence, and such factors should be limited (environmental dimension); the cost of managing both mining damage and permanent tasks should be kept as low as possible (economic dimension); and the population living in the mining regions should be offered future prospects to ensure their standard of living and their well-being after mining has come to an end (social dimension).

Hard coal mining in Germany: From industrialization to post-mining

In Germany, industrial underground mining of hard coal began in the early 19th century. In the Ruhr area, the main coal mining area in Germany, one of the biggest industrial agglomerations in Europe, with more than 5 million inhabitants, was built on coal between 1830 and 1930 (Figure 1).

For more than a hundred years, hard coal from domestic production had been the basis for industrial success in Germany, even after World War II, when Germany started its 'economic miracle', becoming one of the leading economies in the world. However, since the late 1950s, hard coal mining in Germany has been in a state of continuous decline. Because of high production costs due to the challenges of mining conditions at depths up to 1500 m, hard coal mining in Germany could not continue to compete on the world coal market. More than 170 mines have been closed. The number of employees has decreased from about 500 000 to 6 000. Coal production fell from 150 Mt/a in 1957 to 4 Mt/a in 2016 (RAG, 2017, p. 6; Statistik der Kohlenwirtschaft, 2017, p. 26).

In 1968, the remaining mining companies in the Ruhr area merged their coal activities under the supervision of the so-called Ruhrkohle AG. Hard coal production received state subsidies to ensure supply for domestic power generation, the supply of coking coal, and to maintain jobs in mining. In 1997, Ruhrkohle AG was restructured and received a new name, RAG Aktiengesellschaft (RAG). The operation of the mines was step-by-step separated from the other business units in order to optimize funding opportunities and to draw a line between profitable and subsidized business activities.

In 2007, the German government decided to end financial support for the coal mining industry in 2018 because the European Union no longer allows such subsidies. This has led to a final mining closure programme.

The German government has passed a law on funding the termination of hard-coal mining. Based on this law, the 'old' RAG was split into three parts (Figure 2): a newly set up foundation, the RAG-Stiftung (RAG-foundation); the subsidized coal mining unit plus coal trade, land management, site development, and a few other coal-related service companies, still named ('new') RAG; and profitable business units, mainly the subsidiary Evonik Industries, one of the world´s leading specialty chemicals companies, beside other interests. RAG-Stiftung holds 100% of the shares of

Figure 1—The Ruhr, one of the biggest industrial agglomerations in Europe, was built on coal (City Archive of Bochum/Compilation Christoph Dahm)

Figure 2—Company structure of the RAG-Stiftung/RAG (RAG-Stiftung, 2015)

RAG and 68% of the shares of Evonik Industries. The foundation has to ensure that the proceeds from the profitable business units will be used to provide sustainable funding for post-mining tasks so that German taxpayers will not have to pay for them. In addition, the RAG-Stiftung has the responsibility for promotion and support of education, sciences, and culture in the mining regions (RAG-Stiftung 2017).

Germany has long experience with mine closure, starting in the 1960s. At that time, the German government decided on an adjustment process that would reduce the social consequences of mine closure. To do so, the mining companies worked in close collaboration with political institutions and trade unions. The objective, until today, has been to achieve the economically necessary downsizing of staff without resulting in excessive unemployment in the mining regions. This objective could be achieved by means of a number of measures, such as the different ways that collective bargaining offered (*e.g.* agreements of early retirement), by attracting new employers to the manufacturing industry, and by establishing universities. This structural change in the economy was accompanied by the assignment of new land use functions to closed mining areas. Many of these areas became monuments of industrial heritage that are now open to the public, and the brownfield sites were restored for recreational and environmental purposes, and even for new commercial use.

Since the end of mining has been definitely decided, RAG is transforming from an active hard-coal producer to a post-mining developer group. RAG has to master the organizational and technical challenges of mine closure in such a manner that any harm to people or damage to the environment is avoided or offset.

RAG has published a sustainability reporting structure that emphasizes the three columns of its sustainable development strategy: environment (post-mining), society and finances (employees and company), and humanity (cultural and social life). A very important aspect of developing a sustainability strategy is a continuous, open, and transparent dialogue with stakeholders and the general public. In the course of this dialogue, the challenges of post-mining should be intensively discussed. In order to summarize the outcome of this dialogue as the basis of its sustainable development strategy, RAG has defined ten spheres of activity (Figure 3).

The CEO of RAG Group, Bernd Tönjes, has summarized the motivation of RAG to develop its sustainable development strategy as follows:

'At the end of the year 2018 the German hard coal mining will end. [In] more than 200 years it has transformed villages into cities and industrial centress. For many decades coal provided economic power for the mining regions and the whole federal republic. Mining has shaped visibly the landscapes, even after the end of the production. Its impacts are noticeable. But the unavoidable impacts can be regulated and mitigated in a responsible way. We are meeting the challenge to take responsibility for the impacts of coal mining – even when the time of an active mining is over.' (RAG, 2015).

One of many examples where all three dimensions of sustainability have been combined in order to reshape a former mining site is the 'Kreativ.Quartier Lohberg' (Figure 4). This development project illustrates the transformation of a former industrial centre into the first CO_2-neutral suburb in Germany. An important feature of this site is the combination of modern and listed architecture that is completely supplied by renewable energy resources like photovoltaic plants, heat from minewater, biomass, wind turbines, and geothermal energy (Steinkohle, 2016).

Risk management in German post-mining

The general public perceives only the 'tip of the iceberg' (Figure 5) when it comes to the impact of underground mining, as the challenges of the development of the entire post-mining underground environment remain hidden.

With particular regard to post-mining the mining companies, in close collaboration with the mining authorities, have developed a risk management system that facilitates the recognition of all risks and the definition of suitable mitigation measures.The risk areas of abandoned mine sites can be divided into the following categories:

- Near-surface cavities
- Surface openings, old shafts
- Subsidence, uplift
- Discontinuous faults

Figure 3—Ten spheres of activity in the sustainable development strategy of RAG (RAG 2015)

Figure 4—Kreativ.Quartier Lohberg. Overview of the former industrial centre that is being transformed into the first CO_2-neutral suburb in Germany (Steinkohle, 2016. Photo by Dietmar Klingenburg)

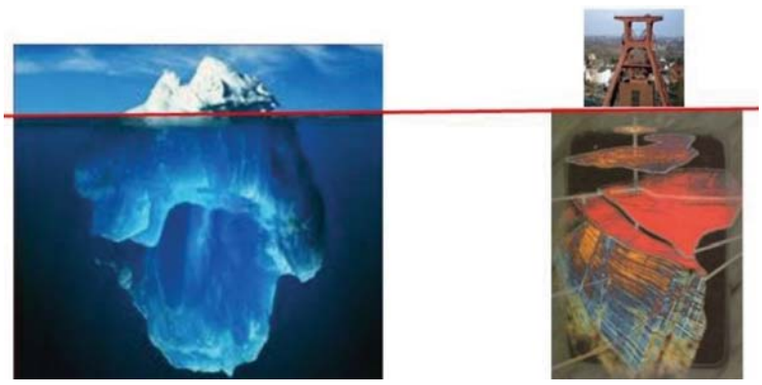

Figure 5—Challenges in German post-mining (the iceberg model)

- Mine gas emission at the surface
- Surface water
- Pumping of minewater
- Refuse dumps
- Operation areas (brownfield).

According to the post-mining risk areas, RAG has defined a number of environmental challenges, which have been divided into two groups: legacy and perpetual tasks. Legacy includes the old mine structures like former operation areas (collieries, coking plants), slag and coal heaps, *etc.*; the remediation of mining damage caused by *e.g.* subsidence or uplift; and the securing of abandoned shafts and former near-surface mining panels. Perpetual tasks include measures to handle minewater drainage, landfilling where mining subsidence occurs, and the cleaning up of groundwater at former mining locations (Figure 6). The distinction between legacy and perpetual tasks is especially relevant for the funding. Whereas RAG has made provisions for the legacy cost, the perpetual tasks are to be paid for by the RAG-Stiftung. From 2019 onwards, approximately 220 million euros will be spent each year by the RAG-Stiftung to handle these tasks (RAG-Stiftung, 2015). In both cases, however, RAG will be in charge of the operating measures.

Minewater drainage is probably the biggest and most expensive challenge among the perpetual tasks. Currently, RAG pumps 70 million cubic metres of minewater per year out of its three coalfields (RAG, 2015). Where all mining operations have been closed, future water drainage is particularly important to avoid contamination of ground and drinking water by minewater in the regions.

Opportunities from post-mining

Until recently, post-mining management has focused on avoiding and preventing risks. However, in the risk management context, sustainability requires a holistic view of the mining cycle, including the opportunities that arise from abandoned mining sites. An effective post-mining strategy provides numerous opportunities for avoiding, mitigating, or even utilizing the risks by reinventing brownfield sites to create new jobs. What principally matters is the successful control and management of post-mining risks and the effective use of opportunities.

In the past few years, the German energy industry has undergone fundamental changes. Some key factors in this context are the finite nature of fossil fuels, global warming, CO_2 emissions, and the risks of nuclear power. Therefore, German politics has incorporated these issues into the mission to move towards renewables without CO_2 emissions, with initiatives such as solar energy, wind power, and

Figure 6—Burdens and perpetual tasks. (1–3 ©RAG; 4. ©THGA; 5. JERS data ©JAXA, SAR and InSAR processing by Gamma Remote Sensing AG, 1998; ©Brüggemann)

geothermal energy. This move towards alternative energy sources brought about a boost of creativity and innovation that also captured post-mining areas, because former mining areas offer many opportunities to generate renewable energy.

There are a number of applications already in use. Research projects work on prototypes, ideas, and visions like:

- Photovoltaic plants on mining areas. Apart from their height, mine dumps have other advantages: there are many free areas and hardly any shade. Therefore, they are ideal locations for photovoltaic systems; the large roofs of factory buildings can be used for these, too (Figure 7.1).
- Heat from minewater. Every year, approximately 70 million cubic metres of minewater is pumped in the Ruhr area. The temperature of this minewater is 35–40°C (95–105°F) and it can be used for supplying heating to buildings by means of heat exchangers, or for accelerating biomass production when generating energy (Figure 7.2).
- Wind turbines on dumps. The dumps in the Ruhr area are often 80–100 m above the ground surface. Thus, they often feature high wind speeds, which allow for the economically viable use of wind turbines, and some wind turbines have already been erected (Figure 7.3).
- Energy production from methane that is released from coal beds.
- Production of biomass on former mining areas, especially dumps.
- Pumped-storage power plants using dumps and underground mine structures.
- Production of geothermal energy (Kretschmann and Hegemann, 2012).

Knowledge transfer to South Africa

Experts in post-mining at TH Georg Agricola University have started knowledge transfer from Germany to the Republic of South Africa. Possible solutions from Germany have been tested for their applicability in South Africa. South Africa has been an exporter of hard coal for over a century. Many abandoned near-surface mines are located in the Mpumalanga Province. According to information from the University of the Witwatersrand (Wits) in Johannesburg, back-calculations from well-documented sales lists of hard coal estimate that there are 1600 km² of undermined territory; this represents an area measuring 40 × 40 km. When more than 160 mines (now abandoned) used the normal production method, more than six million pillars supported this area. The average depth of these mines is expected to be between 40 and 80 m. (Otto and Wolf, 2013).

With room and pillar mining there is a high risk of failure of the supporting pillars or the roof. The estimated six million pillars will fail over time, and most of the 1600 km² area will be disturbed. This poses a latent risk for every kind of surface usage like housing, roads, or farming. Additional

Figure 7—Creating renewable energy, an opportunity of post-mining. (1) Photovoltaic plant on a mud pond; (2) heat from minewater; (3) wind turbine on a dump

risks exist because of possible self-ignition of the remaining coal in the abandoned mines. During their excursion in 2014, researchers and students of TH Georg Agricola University documented springs of minewater with a temperature of about 35°C. The Council for Geoscience in Pretoria testified that there is a fire underground at the transition between active mining and abandoned mining areas. Here, the risk of collapsing pillars is high and the air pollution in the working part of the coal mine has adverse health effects (Figure 8).

In addition, during the rainy season, the mined-out cavities close to the surface are flooded with rainwater (Figure 9). The result is that minerals like pyrite are oxidized and dissolved. This dissolution of oxides, especially the sulphur-containing species, is responsible for an acidic pH of the minewater in the range between 2 and 4. This acid minewater is able to dissolve heavy metals and salts. At the end of the rainy season, acid minewater with many dissolved minerals flows from the mine sites into the rivers (Otto and Wolf, 2013; Otto, 2015).

From the perspective of risk management, the severity of contamination will rise and the post-mining costs will increase in the future. There is therefore a need to restore the areas disturbed by sinkholes and secure other abandoned mining areas against sinkholes and subsidence. The Council for Geoscience wants to proof the near-surface rooms of abandoned hard coal mines against filling. Therefore a suspension should be used that is able to fill the rooms complete in order to support the roof and stabilize the pillars. The concrete formed from this suspension should have a very low permeability to prevent the infiltration of fresh water. Moreover, the concrete should become hard enough to stay in position even with water flowing over its surface.

Figure 8—Collapse pits filled with acid minewater (Photo by Henk Coetzee)

To conserve fresh water the suspension could be mixed with the acid water from the abandoned mines, or other innovative and cost-effective methods could be developed. The TH Georg Agricola University has investigated the use of fly ash. To find the right solution the chosen ash needs to be tested for suitability, and the contaminants in the ash analysed. Then, specimens have to be produced using different water qualities and different water-solid ratios in order to test the hardening behaviour (Otto, 2015). The first results are promising, but more research is needed to establish long-lasting sustainable solutions.

Post-mining education – the basis for post-mining excellence

Post-mining excellence requires high motivation as well as a high level of abilities and skills. Without crucial elements like ideas and vision, research and development, integration of surface and underground challenges, and risk management, post-mining will be inefficient, based on short-term thinking, muddling through, or in the worst case on lip service.

High motivation needs a change of mind-set: post-mining strategy no longer means simply avoiding certain hazards but has to be seen as an evolutionary, sustainable process that is based on the management of risks and the utilization of opportunities. This process should be encouraged by suitable governmental regulations and incentives to promote ideas, to support research and development, and to run high-profile 'flagship' projects profitably. The implementation of such innovative projects at old mine sites is often a milestone for mining communities on their long road to a brighter future. To improve abilities and skills, the existing network of companies, universities, government institutions, mining authorities, and research centres can be used. Its members can promote the transfer of both knowledge and technologies.

In order to competently develop post-mining management, a sufficient number of experts and executives have to be qualified in this field (Figure 10). Therefore, the TH Georg Agricola University offers a unique Master's programme in geo-engineering and post-mining to qualify specialists who will be able to deal with the challenges. The

Figure 9—Decant of AMD (acid mine drainage) from the abandoned Transvaal and Delagoa Bay Colliery close to eMalahleni in Mpumalanga. (Right) at the end of dry season and (left) at the end of rainy season (Photos by Henk Coetzee and Frank Otto)

Figure 10—Researcher from TH Georg Agricola University examining minewater from an old adit (TH Georg Agricola, 2017)

study programme combines natural sciences and technology and prepares new experts to cope with complex interdisciplinary issues, from rock mechanics to mineral economics, from hydrogeology to surveying and legal aspects of mine closure. To date, on average 20 students have been enrolled in this programme every year. Furthermore, the University has established a research centre for post-mining (FZN) where new methods of geo-engineering will be developed and tested for sustainable management of mining impacts (Hegemann and Goerke-Mallet, 2014). However, the research centre does not only look at the risks; it also looks at the abovementioned opportunities of post-mining. The research activities are by no means limited to hard coal mining. They include investigating the impacts of all kinds of mining activities. In the long term, socio-economic and spatial aspects will be added to the more technical issues that are currently investigated. Both the Master's programme and research centre are supported by the RAG-Stiftung (Melchers and Goerke-Mallet, 2016).

Conclusion

RAG, which was established in 1968, will stop producing hard coal after a 50-year period in 2018. To find sustainable solutions for the post-mining era, especially to create good perspectives for the generations to come, RAG has realized a sustainable development strategy that includes numerous measures. The experience and knowledge that the company has gained can be used in many mining regions around the world that will face similar transitions in the future. To develop and transfer knowledge and experience in post-mining, the TH Georg Agricola University has established a specific Master's programme and a research centre. South Africa is one of its first partner countries in international knowledge exchange in post-mining. Clearly, every post-mining impact has to be examined carefully to find the right solutions for the specific issues on surface and underground, so as to properly deal with the risks and to use the possible opportunities.

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I would like to thank my colleague Professor Dr Frank Otto for his support and the insights he gave me into his research in South Africa. As Professor for Geotechnology at the TH Georg Agricola University, he has been one of the first promoters of the Research Institute for Post-Mining. I am convinced that, with his expertise, he can be a promoter to realize post-mining solutions in southern Africa.

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The Southern African Institute of Mining and Metallurgy

is proud to host the

FURNACE TAPPING 2018 CONFERENCE

14 October 2018 - Welcome Networking Cocktail & Entertainment

15–16 October 2018—Conference

17 October 2018—Technical Tours

Nombolo Mdhluli Conference Centre Kruger National Park South Africa

BACKGROUND

A controlled tapping process is essential for all pyrometallurgical smelting furnaces. Draining of metal and slag in a controlled fashion is as important as maintaining a closed tap-hole when required. From a furnace containment perspective, the tap-hole area is one of the high wear, and therefore high risk. Tap-hole failures can have devastating impacts on smelters. Risk mitigation steps therefore addresses safety and health of operators, consequential property damages, and loss of production (business interruptions). Tap-hole life-cycle design and management require development of high quality materials, equipment, and methods that not only take criteria for normal operating conditions into account, but also those of maintenance, repairs, and relines.

Furnace Tapping 2014 was a first of its kind event. The event focused on the challenges associated with furnace tap-hole life-cycle design and management, and on finding ways to address these challenges, for which no miracle one-size-fits-all solution exists. South Africa, which produced 18 commodities at more than 75 sites applying smelter technology, was an ideal breeding ground for such an event. Although it had a strong local focus, people from other parts of the world also gathered to share creative solutions to problems many had in common.

The SAIMM takes pride in announcing a follow-up conference, Furnace Tapping 2018, which will again be hosted in South Africa, in October 2018. The high standard of technical papers compiled in the peer-reviewed proceedings of Furnace Tapping 2014 will be maintained. The SAIMM envisage for Furnace Tapping 2018 further documentation of tapping practices by existing operators, more reviews of current operations, and descriptive case studies in which technologies available for tap-hole design, monitoring, closure, and maintenance were applied. Of special interests are feedback on research conducted in the field, and the impact of safety, health and environmental regulations from various parts of the world, on tap-hole life-cycle design and management.

Looking forward to meeting you at Furnace Tapping 2018!

OBJECTIVES

To provide an international forum for transfer of new knowledge on the design, maintenance, and operating practices surrounding the tapping of pyrometallurgical smelters, and to discuss methods and results of research conducted in the field.

KEYNOTE

Prof. P.C. Pistorius
(Carnegie Mellon University)

J.J. Sutherland
(Transalloys)

Photographer ©Joalet Steenkamp

It doesn't interest me who you know or how you came to be here. I want to know if you will stand in the centre of the fire with me and not shrink back. (from the poem, The Invitation, by Oriah Mountain Dreamer)

WHO SHOULD ATTEND

The conference is aimed at delegates from the pyrometallurgical industry operating smelters or those who support them, and includes:

- First line, middle, and senior management
- Plant/production engineers
- Process/development engineers
- Refractory engineers
- Design engineers
- Maintenance engineers
- Safety practitioners
- Researchers in the field

SAIMM
THE SOUTHERN AFRICAN INSTITUTE OF MINING AND METALLURGY

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http://dx.doi.org/10.17159/2411-9717/2018/v118n8a8

Enhancing sustainability of minerals engineering programmes through the development of a Minerals Resource Centre

by H. Musiyarira, T. Hollenberg, and P. Shava

Synopsis

Depressed mineral commodity prices have a significant impact on the minerals industry, particularly from a sustainability perspective. In this regard, tertiary institutions across the world offering minerals education face enormous sustainability challenges. This problem is not limited to the developing world. The Department of Mining and Process Engineering (DMPE) at the Namibia University of Science and Technology (NUST) was established in 2009 with the objective of developing a critical human resource base for Namibia's mining industry. Despite the current challenges, the DMPE has the potential to make a significant contribution to the future of Namibia and the country's Vision 2030 skills imperatives. This paper presents the DMPE's strategy to ensure sustainability through mapping out a trajectory to provide quality minerals engineering programmes and strengthen capacity for the local industry through consultancy, research, and development. In the short to medium term, the DMPE aims to enhance its research capacity through the development of postgraduate programmes and the establishment of a national centre of excellence – the Namibia Resource Engineering Centre (NREC). The NREC's envisaged mission is to be an internationally recognized centre of excellence for research, consultancy, innovation, and technology transfer in resource engineering. The NREC will become a key enabler of national development through partnerships with the resource industry, government organizations, and communities. The sustainability of institutions offering minerals engineering programmes depends on their ability to deliver value to local industry.

Keywords

sustainability, mining education, research and development, centre of excellence.

Introduction

Depressed mineral commodity prices have a significant impact on the minerals industry, particularly from a sustainability perspective. In this regard, tertiary institutions offering minerals education across the world face enormous sustainability challenges (Phillips, 1999; Galvin and McCarthy, 2001). This is a global problem that is not limited to the developing world.

Mining education, like the mining industry, is typified by 'boom and bust' cycles. The deep 'bust cycle' that spanned the 1990s through 2011 catalysed renewed efforts to eliminate, consolidate, restructure, rename, reinvent, or otherwise change mining education to make it (for some) more economically or socially acceptable (Isaacs, 2011).

Mining education programmes globally have traditionally seen recruitment and survival at risk during industry down-cycles. For example, at the turn of 2000 many North American, as well as Australian, minerals education programmes were close to extinction (Scoble, 2003). It is unrealistic to believe that the minerals education sector can be relevant and sustainable while remaining in isolation from these dramatic developments.

The Department of Mining and Process Engineering (DMPE) at the Namibia University of Science and Technology (NUST) was established in 2009 with the objective of developing a critical human resource base for Namibia's mining industry. In spite of the current challenges, the DMPE has the potential to make a significant contribution to the future of Namibia and the country's Vision 2030 skills imperatives. This paper presents the DMPE's strategy to ensure sustainability through mapping out a trajectory to provide quality minerals engineering programmes and strengthen the capacity of the local industry through consultancy, research, and development.

Background

Namibia is endowed with mineral resources and talented people. It is a vibrant part of Africa, with a stable political system and a burgeoning middle class. Namibia produces uranium oxide, Special High Grade zinc, and acid-grade fluorspar, as well as gold bullion, blister copper, lead concentrate, salt, and

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dimension stone. Rio Tinto and Vedanta produce and export uranium oxide and Special High Grade zinc, respectively. Moreover, Namibia is a primary source of gem-quality diamonds, mined both on- and offshore. De Beers works with the government of the Republic of Namibia through Namdeb Holdings in a 50:50 joint venture, producing some of the world's finest gem diamonds. Namibia's diamond output comes increasingly from the marine environment, reflecting the technical expertise of Debmarine Namibia. Value addition is boosted by eleven cutting and polishing factories, supplied with rough diamonds from the Namibia Diamond Trading Company, worth approximately US$300 million annually (Ralston *et al.*, 2015).

For any country to develop technologically and economically there must be strong links between its industry, government, and academic institutions. All courses and programmes offered by such institutions derive their relevance from the needs of the nations they serve, and hence should promote development of existing and future industries (Tesh *et al.*, 2014). The direction of research at the DMPE was defined based on a thorough mapping of the Namibian minerals industry. This has the advantage of improving the synergy between the DMPE and the companies in the Namibian mineral industry. There is close collaboration with the Ministry of Mines and Energy, Chamber of Mines, Ministry of Industry and Trade, and the broader Namibian mining industry through the DMPE's Advisory Board. Furthermore, the Department has active memoranda of understanding (MOUs) with Namdeb, Swakop Uranium, Areva, and Gecko Namibia.

Nowadays, engineers and scientists required to lead the minerals industry into a competitive position have to emerge from innovative educational environments and from institutions that are forming global partnerships and understand the need to collaborate and share resources, irrespective of their location (Tesh *et al.*, 2014). This vision of meeting the industry's needs for technically competent graduates requires a major restructuring of the system. A close collaboration with stakeholders is essential if minerals tertiary education is to be sustainable in Namibia in the long term. Through stakeholder engagement, society gains a wide range of benefits. Interaction with universities leads to enhanced human and social capital development, accelerated economic growth, improved professional and intellectual infrastructure in communities, progress towards sustainability, and research outcomes that can benefit the social, economic, environmental, and cultural dimensions of society (Winter, Wiseman, and Muirhead, 2006). Full stakeholder engagement would imply that the university can identify relevant research topics and develop courses that employers need and students want.

NUST Department of Mining and Process Engineering

The Department of Mining and Process Engineering has been in existence for nearly ten years. It has grown into a vibrant and proactive unit within the Faculty of Engineering. Founded in 2009 following a needs analysis, and through the efforts of two expatriate professors, it initially offered one degree course – the Bachelor of Engineering in Mining – with two specializations, namely Mining Production and Mineral Processing. The Department experienced operational setbacks when the two founding members were reassigned to higher responsibilities in their countries of origin and were subsequently available to the Department only for limited periods. It was indeed a challenging time. The period 2011–2012 was characterized by limited research output as well as very restricted laboratory space and infrastructure. Unsurprisingly, staff and student morale was low. Industry engagement was limited and the Department faced the possibility of closure. The mining companies traditionally preferred older-established South African and overseas universities to the younger Namibian universities for their research needs. This was understandable, since minerals engineering education institutions in Namibia were still in their infancy and had to prove their capabilities before they could be accepted like longer-established institutions.

There were several threats that weighed on the DMPE and which had the potential to impact on its success. One of these threats was the prevailing global economic crisis, coupled with dwindling government financial support. This trend has also been observed in other universities in Africa and the western world, and has become a major constraint in attaining sustainability of education in general. Dwindling state funding has the biggest impact on new departments at universities that are trying to take their first steps towards building sustainable academic programmes regarding both teaching and research (Tjivikua, 2012). There was a need to tackle this threat before it eroded the benefits of offering minerals education programmes. Fortunately, there was a lot of experience available from the northern hemisphere and Australia, where measures have been taken in order to guarantee the sustainability of the minerals education programmes.

The Resource Engineering Centre

The wealth to be gained from Namibia's natural resources will play a key role in the country's future development. The nation can leverage its natural advantages by embracing the science, engineering, and digital revolution; adapting and developing unique technologies and capabilities. Sharing a common time zone with many of its trade partners, Namibia is poised to make unique contributions to the 21st century. These resources are being developed with consideration to the overarching Harambee Prosperity Plan (HPP) and National Development Plans (NDPs), in step with Constitution Articles 95 and 100. Given the importance of natural resources to the nation, it is critical that the university establishes the Namibian Resource Engineering Centre in order to maintain, and increase, the international competitiveness and long-term sustainability of Namibia's resources, as shown in Figure 1.

Functions of the Centre

The vision of the Namibia Resource Engineering Centre is to become the premier centre in Southern Africa for resource engineering, with the mission of becoming an internationally recognized centre of excellence for research, innovation, technology transfer, and consultancy in resource engineering, as shown in Figure 2. Through partnerships with the

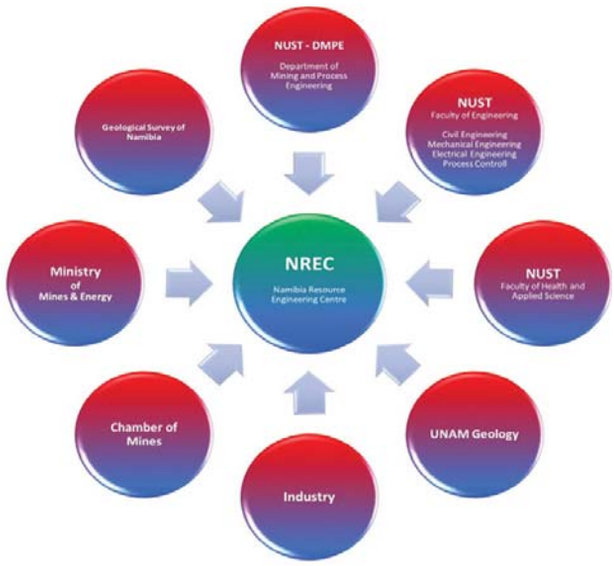

Figure 1—Structure of the Namibia Resource Engineering Centre

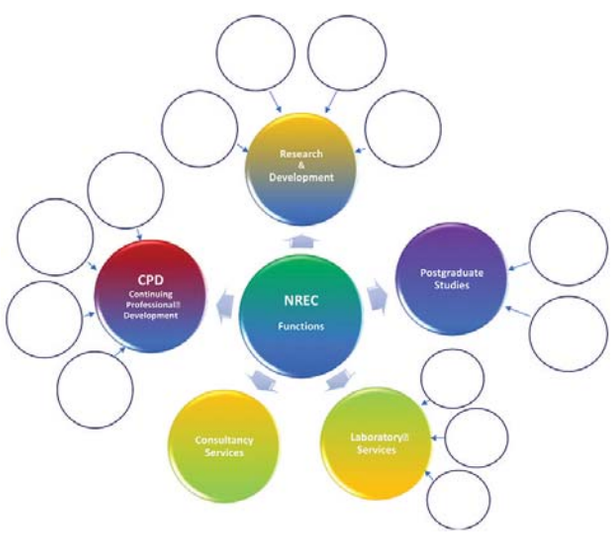

Figure 2—Functions of the Namibian Resource Engineering Centre

Namibian minerals industries, along with government organizations and communities, the NREC will become a key enabler for the nation. The NREC will become a driver of national development through partnership with resource and associated industries, coupled with government organizations and communities. The sustainability of institutions offering minerals engineering programmes depends on their ability to deliver value to local industry.

Research and development services

The main targets of the NREC are to perform excellent multidisciplinary research and development, engage and collaborate with the Namibian industry and government as well as across the Southern African Development Community (SADC), and produce the science and technology outputs needed to enable Namibia to leverage its natural advantage in resources. The DMPE conducts research in mining, minerals, and metallurgy, as well as safety and the environment, with the emphasis on relevance to industry and other stakeholders. The core research and development activities involve a combination of strategic basic and applied research, incorporating theory and experimentation, based upon principles of engineering, technology, and science. The DMPE will actively seek to enhance the funding base for minerals-related research with the government and industry. Minerals research today requires interdisciplinary approaches and talents. Minerals research should be driven by the enabling technologies that are fueling the high-technology revolution. To this end, the DMPE will promote interdisciplinary research and joint research initiatives with other departments at the University and its international partners. The Department has

established symbiotic partnerships with leading universities offering mining and process engineering programmes and mining companies within the Southern African region and beyond. These partnerships have allowed the DMPE to identify key research areas, which dovetail well with the strategic intent of establishing a National Centre of Excellence, the Namibia Resource Engineering Centre, in 2019.

Postgraduate studies

Over the next five years, the Department of Mining and Process Engineering aims to enhance its research capacity through the development of postgraduate programmes. The DMPE will continue to educate and train a new generation of metallurgical and mining engineering graduates, and plans to offer PhD and Master's degrees (by research) in applied science and engineering. These programmes will provide training, offering a choice of fundamental or applied research projects, and lead to national and international careers within the mining and allied industries. The introduction of postgraduate courses and a centre of research excellence is one option that the Namibia University of Science and Technology is pursuing as a way of sustainably producing experts in the field, from whom qualified staff will be recruited (Mischo, 2011).

Continuing professional development (CPD)

The Department also aims to deliver short and continuing education courses to industry and associated groups. These include employee training and qualifications to improve skills, performance, and capabilities required to meet professional and career development goals. Duty-of-care and business performance dictate that employers must ensure that the education, training, and skills of their employees remain current. The DMPE has staff with appropriate profiles and cultures to provide this service to industry. Virtually all universities and research institutions run short courses, and some offer Master's programmes with extensive coursework components. The DMPE is positioning itself to be the leader in minerals education, within Namibia and beyond. Excellent short courses, in-house training, and technology transfer sessions for professional engineers and scientists are aimed at providing training in modern techniques and capabilities.

Consultancy and bureau services

The Centre will offer first-rate consultancy services to industry and government, ensuring the rapid application of new technologies and approaches. It will offer a specialized bureau service to industry and government, using its skilled staff and extensive equipment to solve immediate short-term problems. From its humble infrastructure and equipment base of eight years ago, the DMPE is now housed in a N$200 million, purpose-built building and equipped with multimillion-dollar modern laboratory facilities, equipment, and instrumentation. The Department has standard laboratory equipment as well as specialized equipment like a scanning electron microscope (SEM), Magotteaux mill, flotation units, an XRF facility, thin-section machine, rock sample preparation equipment, and uniaxial/triaxial testing machine, which are essential in catering for the research and development needs of the local mining industry. The laboratory equipment is critical for teaching, research, and for consultancy purposes. The Centre will build and maintain productive partnerships with industry, business, government, tertiary institutions, and community stakeholders within and beyond Namibia.

Conclusions

The current depressed commodity prices threaten to have a significant impact on the sustainability of minerals education. This is further compounded by the inability of governments to provide sufficient funding. The situation extends beyond the developing world and has become characteristic of institutions offering minerals engineering programmes across the globe. In order to address this challenge and ensure the sustainability of minerals education, the DMPE intends to leverage the nexus that exists between government, industry, and other stakeholders by establishing a centre of excellence in minerals resource engineering. Among its key objectives, the Centre aims to expand research activities and postgraduate studies, and offer specialized bureau services to industry and government. Through this innovative vehicle, the DMPE envisages continued growth into the future, thus contributing to the national development agenda and securing its position as a recognized centre of excellence for mineral resources engineering.

Acknowledgments

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Mine Planning and Design for Mining Engineering learners at Wits University

by B. Genc and R. Mitra

Synopsis

The School of Mining Engineering (Wits Mining) at the University of the Witwatersrand (Wits) has a final-year course called 'Mine Design'. The objective of this course is to ensure that students utilize the knowledge gained not only during the four years of coursework, but also during their vacation work, to conduct a mine planning and design exercise at the pre-feasibility study level.

The overall objective of the course is to enable the students to follow a rigorous method of ascertaining the technical and economic viability of a project. Any conclusion arrived at, either positive or negative, is acceptable provided that logical and quantifiable reasons are given. The students have to use mine planning software in order to complete the study. The setting up of a practical layout for a proposed mining operation, with the associated expenditure, will permit the students to make a substantiated recommendation regarding the viability of mining the deposit. In order to include ventilation aspects, only underground design projects are made available. Mine planning and design is done in groups, and each group prepares a coherent and professionally consolidated report and presentation that are presented to staff members, external examiners, and their peers.

To assist the students to complete the project, the fully-equipped Mine Design Laboratory is made available. This paper looks into the steps that are involved for a successful completion of the Mine Design course.

Keywords

mining education, mine planning and design, Wits Mining.

Introduction

Mine planning and design involves compiling and incorporating suitable geotechnical, geochemical, geological, mining, engineering, and economic data in order to establish an approach to exploit a specific mineral deposit within the legal and regulatory requirements (Fourie and van Niekerk, 2001). The aim of planning and design in mining operations is to enable extraction and processing of a mineral deposit at the desired market specification, at a minimum unit cost, and under the existing economic conditions (Fourie and van Niekerk, 2001). A number of professions such as engineering, science, and finance contribute either directly or indirectly to the completion of the mine planning and design process (Department of Minerals and Energy, 1997; Fourie and van Niekerk, 2001). Smith, Pearson-Taylor, and Andersen, (2006) reported that centralizing of strategic long-term planning with the aim to enable efficient involvement with the operations (strategic long-term planning) and the Executive Committee (strategic alignment, scenario development and evaluation) and capital investment prioritization was initiated at Anglo American Platinum to standardize ways of carrying out business and running operations. Smith and Pearson-Taylor (2006) illustrated, with a decision tree, a strategy from which a mining right plan is generated and the long-plan is extracted. The concepts of mine planning and design has been reported in long term production scheduling in both opencast and underground coal mines (Campell, 1994; Department of Minerals and Energy, 2001; Fourie and van Niekerk, 2001; Smith and Pearson-Taylor, 2006; Steffen, 1996).

According to Brophy *et al.* (2008), there is a particular focus on design-oriented teaching in engineering education. Design-oriented teaching deals with solving complex problems, which entails a range of technical and generic skills as opposed to traditional engineering education. Saydam and Mitra (2013) discussed in detail the implementation of project-based courses in mining engineering education.

Wits Mining offers the BSc degree in Mining Engineering, which is a fixed curriculum programme, *i.e.* there are no elective courses offered. The four-year degree programme offers a number of courses covering various aspects of mining engineering. The Mine Design course, one of the fourth-year courses, has two components:

- Mine design report
- Mine design presentation.

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In this course, the students have to use knowledge gained from courses in previous years and their vacation work to design a mine at a pre-feasibility study level. For this reason, the Mine Design course is also known as the 'capstone' course.

In line with the requirements of the Engineering Council of South Africa (ECSA), the Mine Design course addresses several Exit Level Outcomes (ELOs). The students are tested for report writing as well as presentation skills. The ELOs related to the Mine Design course are:

- Perform creative, procedural, and non-procedural design and synthesis of components, systems, works, products, or processes (ELO3)
- Design and conduct investigations and experiments (ELO4)
- Use appropriate engineering methods, skills, and tools, including IT (ELO5)
- Communicate effectively, both orally and in writing, to their peers, engineering audiences, and the community at large (ELO6)
- Demonstrate critical awareness of the impact of engineering activity on the social, industrial, and physical environment (ELO7)
- Work effectively as an individual, in teams and in multi-disciplinary environments, and take responsibility for their own work (ELO8)
- Engage in independent learning through well-developed learning skills (ELO9)
- Act professionally and ethically, exercise judgement, and take responsibility for their own work (ELO10)' (ECSA, 2018).

Furthermore, the students are also tested on the following graduate attributes for this course:

- Analysing information and working with information that may not be complete or working in unfamiliar design contexts
- Integrating a number of subjects into a single, coherent technical report
- Analysing alternatives and selecting the best option based on sound engineering and economic principles
- Applying mathematical analysis in deriving engineering designs
- Applying appropriate software and computer applications in an engineering field.

To help achieve all the above requirements, Wits Mining established a Mine Design Laboratory (MDL) in 2010. Although the MDL serves a number of purposes within the Wits Mining facilities, it also plays an important role in assisting students to complete the Mine Design Project (Genc and Cawood, 2012).

Mine design course

As part of the prerequisite for this 'capstone' final-year course, students have to pass all the subjects in their previous years of study. Furthermore, Wits Mining has established a strong foundation to conduct the Mine Design exercise since 1998. To assist students in their preparation for the Mine Design exercise, in 1997 a new course, 'Computerised Mine Design' was introduced in the third year in partnership with some of the leading mine planning software providers. In this course, the students are taught to use specific mine planning software so that it can be used in the following year during the Mine Design exercise.

Mine design exercise

As part of this assignment, students are provided with actual mine design projects from the industry taking into consideration the planning and design process during the life of the mine.

Studies reported by Fourie and van Niekerk (2001) and Steffen (1996) indicated that the mine planning and design process during the life of any mine involves five stages:

- Obtaining data
- Data evaluation, mine planning and design
- Construction and development of the mine
- Operations
- Mine closure.

The planning and design process related to the above stages involves factors such as identification of expected outcomes, planning and design expectations, risk identification, identification of planning and design limitations, data gathering, risk assessment, assessment of options, *etc.* (Fourie and van Niekerk, 2001).

The purpose of evaluation, planning, and design of a new mine is to analytically investigate the prospecting area with respect to the technical and economic viability of the project. Several investigations may be required as a mine project develops from the initial exploration and conceptual economic phase to the stage where a management decision is made to exploit a particular mineral deposit. Each stage involves an increasing amount of data, which may require a lot of time to prepare, and a higher degree of accuracy. According to the SAMREC Code (2016), for a mine planning and design process to be established, the investigation of the mineral deposit needs to progress through the following stages:

- Scoping study
- Pre-feasibility study
- Feasibility study.

The outcome of each study must be documented in such a way as to support the decision on whether to continue to the next stage of the mine project or not. The following required information pertaining to the studies mentioned above must be included in order to meet this requirement:

- Description of the project: Information regarding area of interest (AOI), roads, topography, weather, development plans of the mine and any processing amenities
- Geology: Full report of the AOI in terms of regional geology, initial reserve estimation, and target valuation
- Mining: Information regarding the depth of the mineral deposit, deposit geometry, proposed plan regarding how to mine the deposit, as well as plant and equipment requirements
- Processing: Description of the processing facilities
- Other operating needs: infrastructure (power and water), equipment, and spare parts availability
- Transportation system: Infrastructure of transportation facilities (roads, harbours, railway lines *etc.*).
- Towns and related facilities: Accommodation for workers, medical facilities, schools, *etc.*

- Labour requirements: Availability of work force based on qualifications and skills
- Environmental protection: Detailed environmental impact assessment to minimize environmental damage in line with the environmental legislation
- Legal considerations: Compliance with the mining laws, taxation, foreign investment regulations *etc*.
- Economic analysis: Full cost analysis including labour, equipment, and other factors.

Every year, different commodities such as gold, coal, platinum, diamonds, and base metals are used in the Mine Design exercise. The benefit of having real design projects stems from the good relationship the School has with the mining industry. The Wits Mining team starts looking for possible projects a few months before the starting date of the exercise. After a project is identified, the exercise commences immediately after the fourth-year students' last examination, so that the students can focus solely on the design project. The Course Coordinator briefs the students about the project and divides the class into groups of five or six students. The groups are selected in a random manner. The students are briefed about the project location and given the necessary information such as mining boundaries, 3D orebody models, and geostatistical information, considering the fact that the students have limited time and manpower. Historically, the students used to create the 3D model using borehole data, and also used to do the geostatistical data analysis. However, since 2017, Wits Mining has decided to provide such information after considering the fact that the mining engineers do not create 3D orebody models in a workplace environment.

As part of the briefing, the students are also provided with a list of topics showing what is expected of the project report. The topics are listed in Table I. The students distribute the workload within their team and also are required to select a group leader, who is responsible for the overall management of the project (Table I).

The Course Coordinator also nominates a staff member for each of the topics in Table I, based on his/her expertise. The staff member then provides a lecture to the students informing them what is expected in that section. This is helpful for the students to recapitulate their learnings from previous years. Table I also shows the mark allocations for each of the topics. The total mark allocated for the project report is 300. The students are advised to consult with the lecturers during the project. The students spend eight weeks in the MDL and complete the project. During the eight-week period, the students make use of the mine planning software that is available in the MDL as well as other related software to complete the task. The students are required to submit weekly progress reports to the Course Coordinator, in order to ensure that they are on track to complete the project.

After the project reports have been submitted, the staff members have five days to mark all the projects (a total of 19 projects in 2017). This is due to the Faculty of Engineering requirements that all the final-year marks have a submission deadline so that the graduation list can be completed. Each topic or section is marked by the lecturer who was assigned for helping the students, as discussed previously. During marking, the lecturers check that the ELO requirements have been met. Furthermore, one of the staff members, who is the Writing and Communication Coordinator in the School, looks after the overall presentation and referencing of the report. This is an important ELO from ECSA's point of view.

The marking has to be completed before the oral presentations take place due to the time constrain. The presentation also contributes a total of 50 marks to the final marks (altogether a total of 350 marks). The distribution of marks for the presentation is shown in Table II. After randomly selecting the groups for the presentations, each group presents their findings to staff members, the external examiners, and their peer groups at the MDL. All the groups are required to upload the final copy of their PowerPoint presentation before the first day of the presentations. This is done to prevent students from making any changes to the presentations based on feedback from the other groups. The external examiners include people from the mine from where the data was obtained. This is very helpful, as not only are they fully aware of the issues at their mine, but they also look for innovative designs that the groups come up with that they can use at their mine site. Each group is given half an hour to present their findings. This is followed by 15 minutes of question time. At the end of each presentation, all marks from the staff members and the external examiners are recorded and an average group mark is calculated. All students in the same group receive the same mark.

Conclusion

This paper outlines how Wits Mining runs the Mine Design course projects. Furthermore, it establishes the connection between the ELOs and Mine Design projects. The paper emphasises the importance of using real data, along with having good facilities such as the MDL for a successful completion of the project. Crucially, successful completion of the project is an outcome of the teamwork with the various staff members in the School. This course also helps students to learn to work in groups. This study reviews the stages involved during the mine planning and design processes that were incorporated by the students during the writing up of their final report and presenting it as a group.

Recommendations

To enhance the effectiveness and use of this study, it is recommended that mine planning and design should be introduced into the curricula of universities offering Mining Engineering programmes in a way that can be applied by a multidisciplinary mine planning and design project team. This paper will assist other programmes that are trying to develop such courses. Wits Mining is currently in the process of redesigning the Mining Engineering curriculum. This will lead to further improvements in the Mine Design course.

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Table I

Mine Design – allocation of marks to areas (2017)

Topic	Marks
Executive summary	10
History, location, and description of project	10
Mining policy, rights, licenses, ownership, and community issues	10
Evaluation of the markets	10
Review of geology and stratigraphy, with implications for mining	15
Technical evaluation (grade-tonnage curves; Resource and Reserve statements)	15
Geotechnical environment, rock engineering requirements, and design	20
Review of previous design and mining method selection	20
Mine design criteria, layout, and blast designs	20
Production scheduling	20
Equipment selection, transportation of ore, people, and materials	15
Ventilation requirements and design	15
Mineral processing plant requirements and design	15
Ore flow accounting (incl. mine call factor - MCF)	15
Surface infrastructure and load-out facilities	10
Manpower requirements and productivity	10
Environmental impacts, rehabilitation, and mine closure	10
Economics of South African gold mining and gold mining taxation	10
Financial valuation (discount rate, capital and operating costs, cash flow analysis)	20
Risk analysis and key residual risks	10
Conclusions and recommendations	10
Referencing and overall presentation	10
Total	300

Table II

Mine Design – allocation of marks to presentations

Area of assessment	Marks
Visual presentation quality	5
Audio presentation quality	5
Technical content/quality	10
Recommendations and conclusions	10
Originality and innovation	5
Appearance of presenters	5
Timing	5
Ability to answer questions	5
Total	50

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A model to calculate blasting costs using hole diameter, uniaxial compressive strength, and joint set orientation

by A. Ghanizadeh Zarghami*, K. Shahriar†, K. Goshtasbi‡, and A. Akbari§

Synopsis

Calculation of the blasting costs plays a significant role in blast pattern design and reduction of the final extraction cost of minerals. Blasting costs include drilling costs, blasting materials costs, and additional costs of blasting operations. We assessed information from three copper mines in Iran, and found that there is a significant relationship between blasting costs and hole diameter. A relationship was derived to calculate blasting costs per cubic metre as a function of hole diameter, bench height, uniaxial compressive strength, joint set orientation, the cost of drilling, and the unit cost of explosives. This model will enable engineers to estimate blasting costs prior to designing the blast pattern. Based on the model, an increase in the rock strength and the angle between the bench face and the main joint set will increase the blasting cost. On the other hand, the costs will decrease when the hole diameter increases for every range of uniaxial compressive strength.

Keywords

Blasting cost, hole diameter, uniaxial compressive strength, joint set, Iran copper mines.

Introduction

Reduction of the operating costs is of great importance with respect to the final costs of the product. The ability to estimate blasting costs before designing blast patterns enables design engineers to choose suitable blast-hole diameters and other crucial parameters of the blast design (Ghanizadeh Zarghami, 2005). Specific charge and specific drilling are two substantial factors concerning blast pattern design that influence blasting costs (Ghanizadeh Zarghami *et al.*, 2017). The hole diameter is generally regarded as a crucial parameter in designing blast patterns (Ostovar, 2013). In the same vein, this study, proposes several models to estimate blasting costs as a function of hole diameter and other parameters, including uniaxial compressive strength (UCS).

Research objective

Blasting models have been formulated by applying technical and economic information on blasting operations at three large copper mines in Iran, namely Sungun, Miduk, and Chah-Firouzeh (Figure 1), After determining the hole diameter and rock uniaxial compressive strength, it will be possible to calculate blasting costs for these three mines and similar operations.

Various investigations have been conducted with the aim of reducing blast operation costs. Afum and Temeng (2014) explored various parameters affecting drilling cost and blast optimization in a gold mine in Ghana. At this mine, blasting was done in three different blocks. The blasting and crushing costs were affected by parameters such as the ground conditions and blast pattern. The model was employed in order to regulate the costs by testing suggested patterns. The results indicated a decrease of between 5.3 and 12.2% in ore costs and between 2.9 and 14.8% for waste costs.

Adebayo and Akande (2015) investigated the effects of drilling in terms of blast-hole deviation and muck-pile loading costs for six scenarios at Hwange Colliery, Zimbabwe. The study showed that the drilling and operational costs were in the range of US$0.13–7.53 per m^3. Ancillary costs of drilling increased from US$1.7 to US$4.2 per m^3 with an increase in blast-hole deviation from 7% gradient to 21%.

Adebayo and Mutandwa (2015) evaluated the relationship between blast-hole deviation, fragment size, and fragmentation cost. The use of ANFO, heavy ANFO, and emulsion explosives in holes 191 mm and 311 mm in diameter was compared using six scenarios. The results showed that as blast-hole deviation increases the mean fragment size decreases and the cost of drilling and blasting increases. Increasing the hole diameter from 191 mm to 311 mm increased the blast fragmentation.

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A: Sungun Copper Mine

B: Miduk Copper Mine

C: Chah-Firouzeh Copper Mine

Figure 1—Perspective views of Sungun, Miduk, and Chah-Firouzeh copper mines

Nenuw and Jimoh (2014) designed and optimized the blasting parameters to reduce the damaging effects and blasting costs by using Langfors and other common blast formulae. In this study, which was conducted at four mines, parameters such as burden, spacing, bench height, hole diameter, the number of holes, bottom charge, and total charge per hole were examined and the planned and actual costs calculated. The actual costs of blasting material were higher than the calculated cost, which required modification and revision.

Cunningham (2013) investigated four key parameters that determine the ability to design an effective blast in terms of delay timing and cost. These parameters included heave control and monitoring, hole diameter, and explosive type.

Strelec, Gazdek, and Mesec (2011) designed an optimized blast pattern to reduce drilling costs. The blast fragmentation was optimized by applying the calibration factors in the Kuz-Ram model.

Eloranta (1995) showed, by comparing the loading costs of materials to the blasting costs, that due to the increase in specific explosive charge in large-diameter holes, the blasting costs have a strong inverse correlation with the specific explosives charge. Increasing the specific explosives charge by 15% increased the shovel and crusher efficiencies by 5%, resulting in an overall reduction in operating costs.

Blasting blocks information

More than 4600 records of blasting operations at Sungun, Miduk, and Chah-Firouzeh from 2012 to 2014 were collected. Incorrect and unreliable records were deleted and finally, 2414 blasts with limited back break, air blast, ground vibration, oversize, and destructive effects were selected. Basic information on the blasting operations, including drilling costs, blasting material, and blast block geometry for the three mines is shown in Table I (Ghanizadeh Zarghami, 2017).

In Table I, the mines are categorized according to rock strength. The drilling cost per metre is considered according to the contractor price, and the cost for ANFO is based on the purchase price, transport, and delivery to the mine. The types of rocks blasted are shown in Table II.

The choice of effective parameters

The large number of factors and the complicated iterations make it impossible to determine the theoretical consumption of explosives at the present level of development in blasting theory. Thus, recourse is made either to practical data or to empirical formulae that generalize blasting practice in application to drifting (Pokrovsky, 1980). In the present research, four important parameters: hole diameter, UCS, joint set orientation, and bench height were selected for calculating blasting cost. These parameters could be easily calculated by the engineers and ultimately aid in estimating the blasting costs.

Blasting pattern and cost calculations

The correct ratios between the geometric parameters of blasting patterns and hole diameter in the UCS range between 10 and 250 MPa and for the angle between the bench face and main joint set (γ) <90° and >90° were extracted from the blasting databases at the respective mines. These ratios are presented in Tables III and IV. In the same UCS range, more energy is required when γ is greater than 90° because the joint set dips in the opposite direction to the free face direction. Therefore, the specific charge and specific drilling, and blasting costs are higher for γ>90° than for γ<90° (Ghanizadeh Zarghami, 2017).

According to the rules of block theory, the angle between the bench face and the main joint set is important. This angle is located between the two normal vectors of the planes. In other words, γ is the same angle between the two planes and

Table I

Data of drilling costs, explosives, and bench geometry considering compressive strength

Mine	Chah-Firouzeh	Miduk	Miduk and Sungun	Sungun
UCS (MPa)	10–70	70–120	120–180	180–250
Length of block (m)	70	70	70	70
Width of block (m)	150	150	150	150
Bench height (m)	15	15	15	15
ANFO density (t/m^3)	0.88	0.88	0.88	0.88
Volume of block (m^3)	157 500	157 500	157 500	157 500
Drilling cost (US$/m^3) (6 inches diameter in 2017)	4.5	5.4	6.48	7.77
Price of ANFO (2017) (US$/kg)	0.73	0.73	0.73	0.73

Table II

Rock types at the three case study copper mine

No.	Mine	Rock type	Description	UCS (MPa)
1	Miduk	Waste	Andesite	70–120
2	Miduk	Mixed	Andesite and granodiorite	120–180
3	Miduk	Ore	Granodiorite	120–180
4	Sungun	Waste	Trachyte	180–250
5	Sungun	Ore	Monzonite	120–180
6	Chah-Firouzeh	Waste	Alluvium	10–70

Table III

Ratios between the geometric parameters of the blasting pattern and the hole diameter for γ<90°

UCS	10–70 MPa			70–120 MPa			120–180 MPa			180–250 MPa		
Range	Min.	Max.	Mean	Min.	Max.	Mean	Min.	Max.	Mean	Min.	Max.	Mean
B/Dh (m/in)	0.99	1	0.99	0.915	0.945	0.93	0.882	0.918	0.9	0.821	0.859	0.84
S/Dh (m/in)	1.29	1.31	1.3	1.185	1.195	1.19	1.082	1.118	1.1	0.941	0.979	0.96
T/Dh (m/in)	0.89	0.91	0.9	0.825	0.855	0.84	0.792	0.828	0.81	0.731	0.769	0.75
J/Dh (m/in)	0.24	0.26	0.25	0.255	0.285	0.27	0.282	0.318	0.3	0.281	0.319	0.3

Table IV

Ratios between the geometric parameters of the blasting pattern and the hole diameter for γ>90°

UCS(MPa)	10–70 MPa			70–120 MPa			120–180 MPa			180–250 MPa		
Range	Min.	Max.	Mean	Min.	Max.	Mean	Min.	Max.	Mean	Min.	Max.	Mean
B/Dh (m/in)	0.699	1.161	0.93	0.599	1.201	0.9	0.81	0.87	0.84	0.76	0.8	0.78
S/Dh (m/in)	1.18	1.2	1.19	0.899	1.301	1.1	0.94	0.98	0.96	0.794	0.866	0.83
T/Dh (m/in)	0.828	0.852	0.84	0.78	0.84	0.81	0.725	0.775	0.75	0.47	0.93	0.7
J/Dh (m/in)	0.24	0.3	0.27	0.299	0.301	0.3	0.28	0.32	0.3	0.27	0.33	0.3

it is a necessary factor for writing the equation of plane, dip, and dip direction of the plane. The dip and dip direction of the main joint set and bench face are of importance to present the equation of their plane. Equation [1] demonstrates the plane equation and Equation [2] represents the coordinates of normal vector through dip and dip direction (Dehghan, 2001). Figure 2 shows the layout of the angles and plane.

$$AX + BY + CZ = D \quad [1]$$

$$A = \sin\alpha \sin\beta$$
$$B = \sin\alpha \cos\beta \quad [2]$$
$$C = \cos\alpha$$

In Equation [2], α indicates the dip and β represents dip direction relative to north. Equation [3] is used to measure the angle between the two planes (γ).

$$P_1(\alpha_1, \beta_1) \rightarrow \hat{n}_1 = (A_1, B_1, C_1)$$
$$P_2(\alpha_2, \beta_2) \rightarrow \hat{n}_2 = (A_2, B_2, C_2)$$
$$y = \cos^{-1}\left(\frac{A_1 A_2 B_1 B_2 + C_1 C_2}{\sqrt{(A_1^2 + B_1^2 + C_1^2)(A_2^2 + B_2^2 + C_2^2)}}\right) \quad [3]$$

All blasting costs were modelled in the Comfar technical and economic analysis software and the cost per cubic metre broken was calculated. As presented in Table V, 87% of the blasting operation costs depends on the cost of ANFO and drilling costs. Equation [4] shows the cost of blasting operations according to specific drilling, specific charge, the price per kilogram of ANFO, and drilling cost per metre (Ghanizadeh Zarghami, 2017).

$$C_1 + C_2 = 87\% BC, BC = (1/0.87)(C_1 + C_2),$$
$$BC = 1.15(P_A \times SC + P_D \times SD) \quad [4]$$

In Equation [4], parameter C_1 represents ANFO cost, C_2 represents drilling cost, BC represents blasting cost per cubic metre, P_A the price of ANFO per kilogram, P_D the price of drilling per metre, SC the specific charge (kg/m^3), and SD the specific drilling (m/m^3).

Tables VI to IX show the burden, spacing, stemming, and sub-drilling considering the rock strength with γ<90° and γ>90°. At the studied mines, hole diameters of 6 to 6.5 inches are used. The burden parameter, spacing, stemming, and sub-drilling in zone classification of UCS were calculated according to joint set orientation with a hole diameter of 6 inches (152.4 mm).

Discussion and review

Factors in the blasting operation costs include blasting material costs and auxiliary costs such as staff wages, transportation, storage, and overhead costs. The bulk of the costs includes the blasting costs and consists of the drilling costs and the cost of ANFO. Finally, considering the contractor costs, the blast side cost was equal to 15% of the total cost.

The cost of drilling operations and consumed specific costs were calculated through burden, spacing, stemming, and sub-drilling. Parameter calculations and the operational costs in rocks with UCS of 10 to 70 MPa and hole diameters of 2 to 16 inches are presented in Table X, for γ<90°.

According to Table VI, for γ>90°, the same calculations were carried out based on Table X, the results of which, along with the results of calculations for γ<90°, are shown in Figure 3.

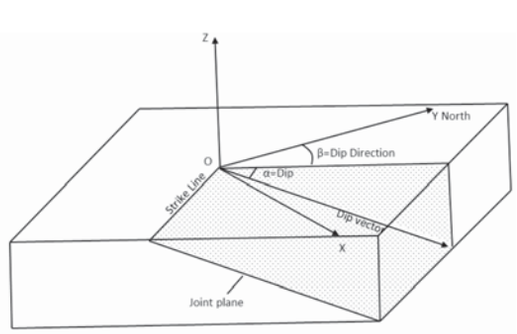

Figure 2—Layout of the angles and plane (Dehghan, 2001)

Table V

The ratio of ANFO costs and drilling costs to the total blasting costs

No.	Mine	Type	Year	Blasting cost (1000 Rials/m³)	Production volumes (m³)	ANFO cost + drilling cost (1000 Rials) (A)	Drilling cost (1000 Rials)	ANFO cost (1000 Rials)	Total blasting cost (1000 Rials) (B)	Ratio A/B
1	Miduk	Waste	2012	13.67	3 931 645	48 863 950	16 528 950	32 335 000	53 739 619	91%
2	Miduk	Waste	2013	14.83	2 460 168	33 736 800	10 861 800	22 875 000	36 487 558	92%
3	Miduk	Waste	2014	14.07	1 021 837	12 349 950	3 627 450	8 722 500	14 377 475	86%
4	Miduk	Mixed	2012	14.32	1 952 261	23 644 850	8 024 850	15 620 000	27 959 571	85%
5	Miduk	Mixed	2013	15.42	2 802 693	38 499 100	11 924 100	26 575 000	43 215 594	89%
6	Miduk	Mixed	2014	13.27	5 981 862	72 757 900	22 107 900	50 650 000	79 380 906	92%
7	Miduk	Ore	2012	15.05	1 430 466	15 393 550	5 176 050	10 217 500	21 529 271	72%
8	Miduk	Ore	2013	21.81	1 010 146	13 553 350	4 679 100	8 874 250	22 027 611	62%
9	Miduk	Ore	2014	18.79	1 002 165	11 914 000	3 939 000	7 975 000	18 831 301	63%
10	Sungun	Waste	2012	32.18	624 178	18 668 700	6 610 950	12 057 750	20 085 600	93%
11	Sungun	Waste	2013	34.62	147 914	3 914 725	1 410 450	2 504 275	5 120 925	76%
12	Sungun	Waste	2014	40.21	315 153	11 390 050	4 014 000	7 376 050	12 672 690	90%
13	Sungun	Ore	2012	13.52	7 698 287	101 487 100	35 356 350	66 130 750	104 078 530	98%
14	Sungun	Ore	2013	14.19	6 777 431	93 713 950	32 721 450	60 992 500	96 188 900	97%
15	Sungun	Ore	2014	14.37	6 562 884	91 971 600	31 918 800	60 052 800	94 300 530	98%
16	Chah-Firouzeh	Waste	2012	18.12	2 702 430	45 956 250	15 142 500	30 813 750	48 973 369	94%
17	Chah-Firouzeh	Waste	2013	15.32	3 742 393	54 760 000	21 303 750	33 456 250	57 345 636	95%
18	Chah-Firouzeh	Waste	2014	13.39	3 098 502	39 060 050	13 063 800	25 996 250	41 480 320	94%
*In 2017: $1 = 37 000 Rials Iran's currency is the Rial									Mean:	87.01%

Table VI

Blast pattern parameters at Chah-Firouzeh copper mine, UCS 10–70 MPa and hole diameter 6 inches

Variable parameters of blast pattern	Ratios UCS = 10–70 Mpa (γ<90°)		Computational v alues Dh = 6 in	Ratios UCS = 10–70 Mpa (γ>90°)		Computational values Dh=6 in
Burden	B/Dh	0.99	5940	B/Dh	0.93	5580
Spacing	S/ Dh	1.3	7722	S/ Dh	1.19	7142
Stemming length	T/ Dh	0.9	5346	T/ Dh	0.84	5022
Sub drilling length	J/ Dh	0.25	1485	J/ Dh	0.27	1618

Dh=Hole diameter B=Burden S=Spacing T=Stemming j=Sub-drilling

Table VII

Blast pattern parameters at Miduk copper mine, UCS 70–120 MPa and hole diameter 6 inches

Variable parameters of blast pattern	Ratios UCS = 70–120 Mpa (γ<90°)		Computational values Dh = 6 in	Ratios UCS = 70–120 Mpa (γ>90°)		Computational values Dh=6 in
Burden	B/Dh	0.93	5580	B/Dh	0.9	5400
Spacing	S/ Dh	1.19	7142	S/ Dh	1.1	6588
Stemming length	T/ Dh	0.84	5022	T/ Dh	0.81	4860
Sub drilling length	J/ Dh	0.27	1618	J/ Dh	0.3	1782

Dh=Hole diameter B=Burden S=Spacing T=Stemming j=Sub-drilling

Table VIII

Blast pattern parameters at Miduk and Sungun copper mine, UCS 120–180 MPa and hole diameter 6 inches

Variable parameters of blast pattern	Ratios UCS = 120–180 Mpa (γ<90°)		Computational values Dh = 6 in	Ratios UCS = 120–180 Mpa (γ>90°)		Computational values Dh=6 in
Burden	B/Dh	0.9	5400	B/Dh	0.84	5040
Spacing	S/ Dh	1.1	6588	S/ Dh	0.96	5746
Stemming length	T/ Dh	0.81	4860	T/ Dh	0.75	4486
Sub drilling length	J/ Dh	0.3	1782	J/ Dh	0.3	1799

Dh=Hole diameter B=Burden S=Spacing T=Stemming j=Sub-drilling

Table IX

Blast pattern parameters at Sungun copper mine, UCS 180–250 MPa and hole diameter 6 inches

Variable parameters of blast pattern	Ratios UCS = 180–250 Mpa (γ<90°)		Computational values Dh = 6 in	Ratios UCS = 180–250 Mpa (γ>90°)		Computational values Dh=6 in
Burden	B/Dh	0.84	5040	B/Dh	0.78	4680
Spacing	S/ Dh	0.96	57468	S/ Dh	0.83	4961
Stemming length	T/ Dh	0.75	4486	T/ Dh	0.7	4165
Sub drilling length	J/ Dh	0.3	1799	J/ Dh	0.3	1778

Dh=Hole diameter B=Burden S=Spacing T=Stemming j=Sub-drilling

Table X

Calculations of blasting parameters and costs with UCS 10-70 MPa and hole diameter of 2 to 16 inches

10-70 MPa		Dh (in)	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Ratio is rounded (m/in)		Dh (mm)	50.8	76.2	102	127	152	178	203.2	228.6	254	279.4	304.8	330.2	355.6	381	406.4
B/Dh	0.99	B (mm)	1980	2970	3960	4950	5940	6930	7920	8910	9900	10890	11880	12870	13860	14850	15840
S/Dh	1.3	S (mm)	2574	3861	5148	6435	7722	9009	10296	11583	12870	14157	15444	16731	18018	19305	20592
T/Dh	0.9	T (mm)	1782	2673	3564	4455	5346	6237	7128	8019	8910	9801	10692	11583	12474	13365	14256
J/Dh	0.25	J (mm)	495	742.5	990	1237.5	1485	1732.5	1980	2227.5	2475	2722.5	2970	3217.5	3465	3712.5	3960
SC (kg/m³)			0.32	0.3	0.29	0.27	0.26	0.24	0.229	0.214	0.199	0.184	0.1694	0.154	0.139	0.124	0.109
SD (m/m³)			0.2	0.09	0.05	0.03	0.02	0.02	0.014	0.011	0.009	0.008	0.0065	0.006	0.005	0.004	0.004
Drilling cost (1000 $)			141	63.5	36.3	23.6	16.6	12.4	9.635	7.724	6.346	5.319	4.532	3.915	3.421	3.02	2.69
ANFO cost (1000 $)			37	35.2	33.5	31.8	30	28.3	26.57	24.83	23.09	21.36	19.618	17.88	16.14	14.41	12.67
The lateral blast costs the equivalent of 13% of the total (1000 $)			26.6	14.8	10.5	8.3	7	6.11	5.43	4.883	4.416	4.001	3.6226	3.269	2.935	2.614	2.304
Blasting operation total cost (1000 $)			204	114	80.3	63.7	53.7	46.8	41.63	37.44	33.85	30.68	27.773	25.07	22.5	20.04	17.67
Blasting operation total cost ($/m³)			1.3	0.72	0.51	0.4	0.34	0.3	0.264	0.238	0.215	0.195	0.1763	0.159	0.143	0.127	0.112

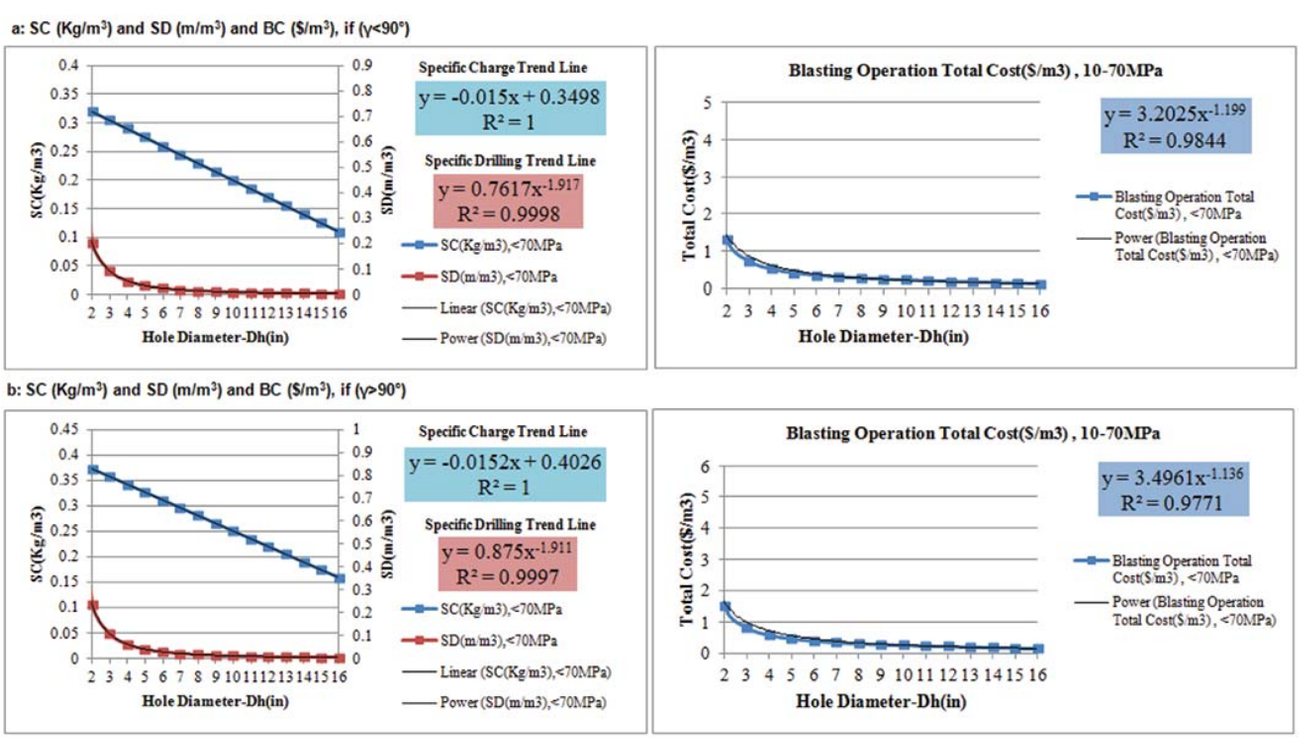

Figure 3—Relationship between of specific charge (SC), specific drilling (SD), and blasting costs considering hole diameter in the UCS range of 10-70 MPa for γ<90° and γ>90°

For other rock strengths in the three mines, the tables of calculated blast parameters and diagrams are presented together. Table XI lists the parameters and blasting costs in rock with a strength of 70–120 MPa and with hole diameters of 2 to 16 inches.

According to Table VII, for γ>90°, the same calculations were carried out based on Table XI, the results of which, along with the results of calculations for γ<90°, are shown in Figure 4.

Table XII shows the parameters and blasting costs in rock with the strength of 120 to 180 MPa and hole diameters of 2 to 16 inches.

According to Table VIII, for γ>90°, the same calculations were carried out based on Table XII, the results of which, along with the results of calculations for γ<90°, are shown in Figure 5.

Table XIII shows calculations of parameters and blasting costs in rock with strength between 180 to 250 MPa and hole diameters of 2 to 16 inches.

According to Table IX, for γ>90°, the same calculations were carried out based on Table XIII, the results of which, along with the results of calculations for γ<90°, are shown in Figure 6.

Table XI

Calculations of blasting parameters and costs with UCS 70-120 MPa and hole diameter of 2 to 16 inches

70-120 MPa		Dh (in)	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Ratio is rounded (m/in)		Dh (mm)	50.8	76.2	102	127	152	178	203.2	228.6	254	279.4	304.8	330.2	355.6	381	406.4
B/Dh	0.93	B (mm)	1860	2790	3720	4650	5580	6510	7440	8370	9300	10230	11160	12090	13020	13950	14880
S/Dh	1.19	S (mm)	2380.8	3571.2	4761.6	5952	7142.4	8332.8	9523.2	10713.6	11904	13094.4	14284.8	5475.2	6665.6	17856	9046.4
T/Dh	0.84	T (mm)	1674	2511	3348	4185	5022	5859	6696	7533	8370	9207	10044	10881	11718	12555	13392
J/Dh	0.27	J (mm)	539.4	809.1	1078.8	1348.5	1618.2	1887.9	2157.6	2427.3	2697	2966.7	3236.4	3506.1	3775.8	4045.5	4315.2
SC (kg/m³)			0.372	0.357	0.342	0.326	0.311	0.296	0.281	0.266	0.250	0.235	0.220	0.205	0.189	0.174	0.159
SD (m/m³)			0.234	0.106	0.061	0.039	0.028	0.021	0.016	0.013	0.011	0.009	0.008	0.007	0.006	0.005	0.005
Drilling cost (1000 $)			195.07	88.20	50.46	32.84	23.18	17.31	13.46	10.80	8.89	7.46	6.36	5.50	4.81	4.25	3.79
ANFO cost (1000 $)			43.09	41.33	39.57	37.81	36.04	34.28	32.52	30.75	28.99	27.23	25.46	23.70	21.94	20.17	18.41
The lateral blast costs the equivalent of 13% of the total (1000 $)			35.72	19.43	13.50	10.60	8.88	7.74	6.90	6.23	5.68	5.20	4.77	4.38	4.01	3.66	3.33
Blasting operation total Cost (1000 $)			273.88	148.96	103.53	81.24	68.10	59.32	52.87	47.79	43.56	39.88	36.59	33.58	30.76	28.09	25.53
Blasting operation total cost ($/m³)			1.74	0.95	0.66	0.52	0.43	0.38	0.34	0.30	0.28	0.25	0.23	0.21	0.20	0.18	0.16

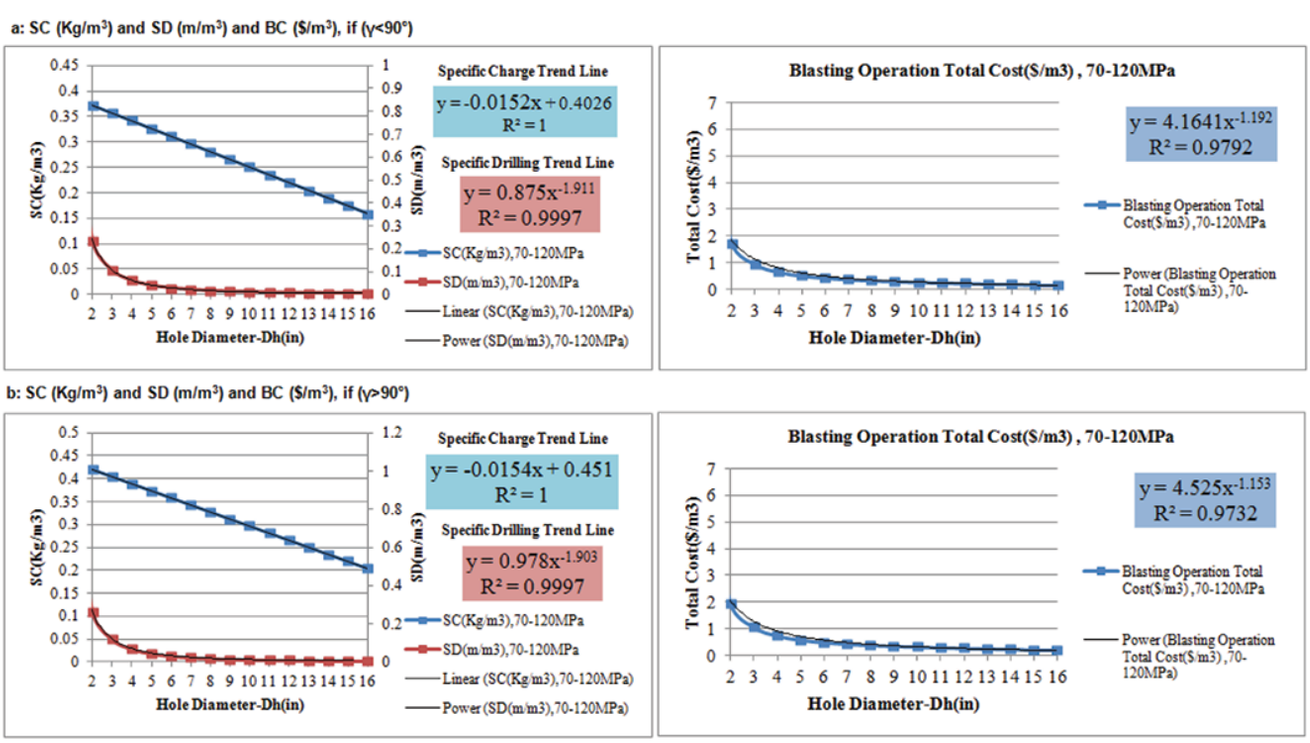

Figure 4—Relationship between specific charge (SC), specific drilling (SD), and blasting costs considering hole diameter in the UCS strength range (70-120 MPa) for (γ<90°) and (γ>90°)

Table XII

Calculations of blasting parameters and costs with UCS 120-180 MPa and hole diameter of 2 to 16 inches

120-180 MPa		Dh (in)	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Ratio is rounded (m/in)		Dh (mm)	50.8	76.2	102	127	152	178	203.2	228.6	254	279.4	304.8	330.2	355.6	381	406.4
B/Dh	0.9	B (mm)	1800	2700	3600	4500	5400	6300	7200	8100	9000	9900	10800	11700	12600	13500	14400
S/Dh	1.1	S (mm)	2196	3294	4392	5490	6588	7686	8784	9882	10980	12078	13176	14274	15372	16470	17568
T/Dh	0.81	T (mm)	1620	2430	3240	4050	4860	5670	6480	7290	8100	8910	9720	10530	11340	12150	12960
J/Dh	0.3	J (mm)	594	891	1188	1485	1782	2079	2376	2673	2970	3267	3564	3861	4158	4455	4752
SC (kg/m³)			0.42	0.40	0.39	0.37	0.36	0.34	0.33	0.31	0.30	0.28	0.27	0.25	0.24	0.22	0.20
SD (m/m³)			0.26	0.12	0.07	0.04	0.03	0.02	0.02	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01
Drilling cost (1000 $)			263.16	119.19	68.30	44.51	31.47	23.53	18.33	14.73	12.13	10.19	8.70	7.53	6.60	5.84	5.21
ANFO cost (1000 $)			48.66	46.87	45.08	43.30	41.51	39.73	37.94	36.15	34.37	32.58	30.79	29.01	27.22	25.44	23.65
The lateral blast costs the equivalent of 13% of the total (1000 $)			46.77	24.91	17.01	13.17	10.95	9.49	8.44	7.63	6.97	6.42	5.92	5.48	5.07	4.69	4.33
Blasting operation total Cost (1000 $)			358.59	190.97	130.39	100.98	83.93	72.74	64.71	58.51	53.47	49.19	45.42	42.02	38.89	35.96	33.19
Blasting operation total			2.28	1.21	0.83	0.64	0.53	0.46	0.41	0.37	0.34	0.31	0.29	0.27	0.25	0.23	0.21

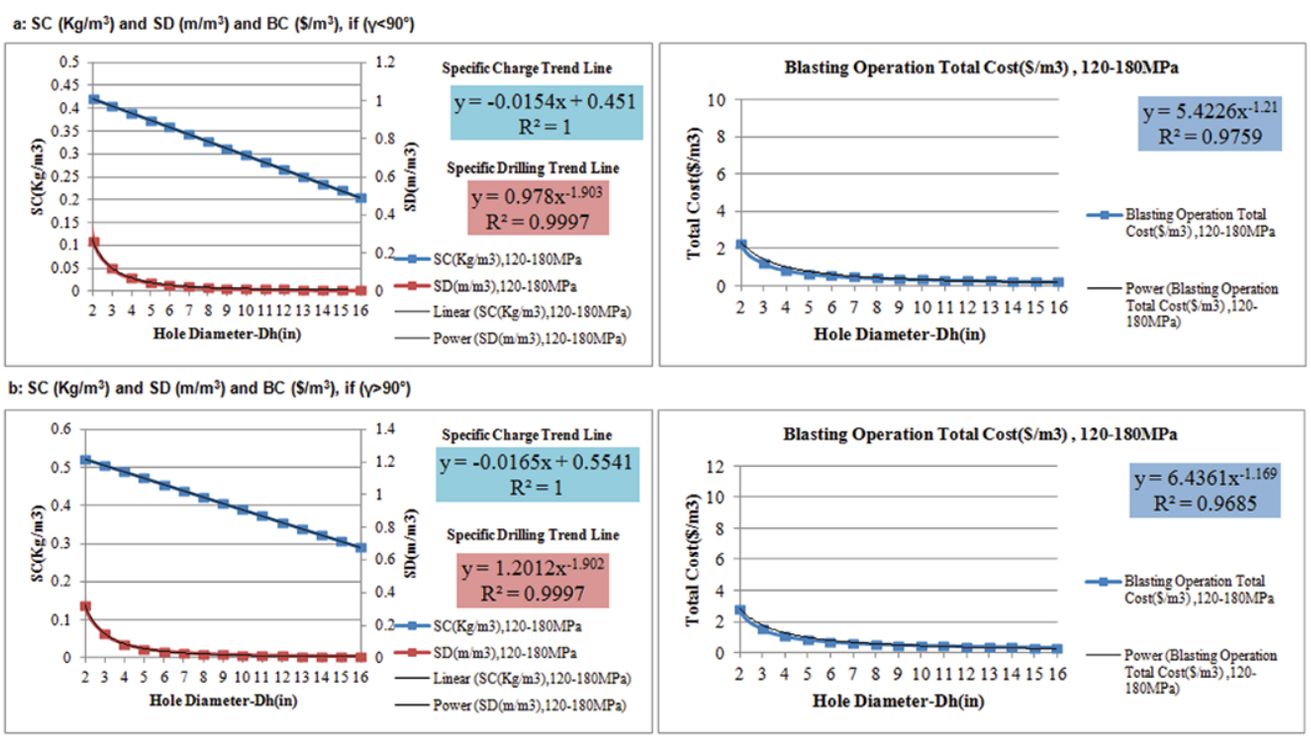

Figure 5—Relationship of specific charge (SC), specific drilling (SD), and blasting costs to hole diameter in the UCS range 120–180 MPa for γ<90° and γ>90°

Results of reviews

According to the research and the proposed models shown in the previous section, the relationship between the hole diameter and specific charge, specific drilling, and blasting costs for bench heights of 15 m in γ<90° and γ>90° are determined. The results are presented in Table XIV for the range of UCS considered.

Equation [5] shows the general equation of blasting cost, which is derived to calculate the blasting cost according to the hole diameter. In this equation, coefficients 'a' and 'b' are functions of bench height, UCS, join set orientation, the cost of drilling per metre, and the cost of ANFO.

$$BC = 1.15(P_A \text{ x } SC + P_D \text{ x } SD) = a(Dh)^{-b} \quad [5]$$

BC_e in Table XIV is the estimated blasting cost during 2017. If the price of ANFO and drilling cost are fixed, blasting engineers can use BC_e in Table XIV; otherwise, they can use BC for calculating blasting cost, which excludes a time-frame. However, they should determine P_A and P_D for every year.

According to Table XIV, blasting cost was calculated using the UCS, hole diameter, and joint set orientation for Sungun, Miduk, and Chah-Firouzeh. In this model the blasting cost was calculated for each blast block, which includes drilling cost, the cost of ANFO, and auxiliary charges for the blasting operation. It should be mentioned that the

Table XIII

Calculations of blasting parameters and costs with UCS 180-250 MPa and hole diameter of 2 to 16 inches

180-250 MPa		Dh (in)	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Ratio is rounded (m/in)		Dh (mm)	50.8	76.2	102	127	152	178	203.2	228.6	254	279.4	304.8	330.2	355.6	381	406.4
B/Dh	0.84	B (mm)	1680	2520	3360	4200	5040	5880	6720	7560	8400	9240	10080	10920	11760	12600	13440
S/Dh	0.96	S (mm)	1915.2	2872.8	3830.4	4788	5745.6	6703.2	7660.8	8618.4	9576	10533.6	11491.2	12448.8	13406.4	14364	15321.6
T/Dh	0.75	T (mm)	1495.2	2242.8	2990.4	3738	4485.6	5233.2	5980.8	6728.4	7476	8223.6	8971.2	9718.8	10466.4	11214	11961.6
J/Dh	0.3	J (mm)	599.76	899.64	1199.52	1499.4	1799.28	2099.16	2399.04	2698.92	2998.8	3298.68	3598.56	3898.44	4198.32	4498.2	4798.08
SC (kg/m³)			0.52	0.50	0.49	0.47	0.45	0.44	0.42	0.41	0.39	0.37	0.36	0.34	0.32	0.31	0.29
SD (m/m³)			0.32	0.15	0.08	0.05	0.04	0.03	0.02	0.02	0.01	0.01	0.01	0.01	0.01	0.01	0.01
Drilling cost (1000 $)			388.10	175.80	100.75	65.68	46.44	34.73	27.05	21.74	17.91	15.05	12.85	11.13	9.75	8.62	7.70
ANFO cost (1000 $)			60.33	58.42	56.50	54.59	52.67	50.76	48.84	46.93	45.01	43.10	41.18	39.27	37.35	35.44	33.52
The lateral blast costs the equivalent of 13% of the total (1000 $)			67.26	35.13	23.59	18.04	14.87	12.82	11.38	10.30	9.44	8.72	8.11	7.56	7.06	6.61	6.18
Blasting operation total cost (1000$)			515.70	269.36	180.85	138.31	113.98	98.31	87.28	78.97	72.36	66.87	62.14	57.95	54.16	50.67	47.40
Blasting operation total cost ($/m³)			3.27	1.71	1.15	0.88	0.72	0.62	0.55	0.50	0.46	0.42	0.39	0.37	0.34	0.32	0.30

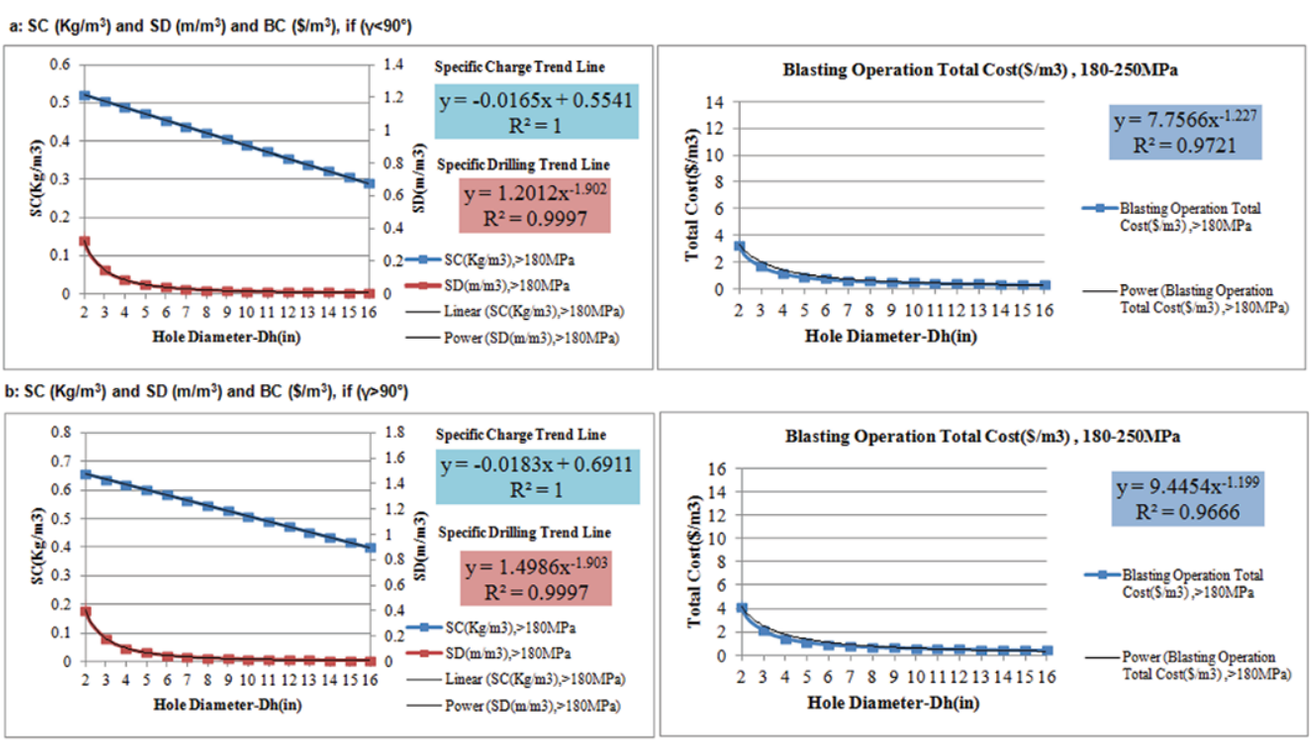

Figure 6 – Relationship of specific charge (SC), specific drilling (SD), and blasting costs considering hole diameter in the UCS range 180-250 MPa for γ<90° and γ>90°

blasting cost will increase with increasing rock strength and γ, and decrease with increasing hole diameter in all ranges of UCS.

Conclusion

Investigation of the blasting cost at Sungun, Miduk, and Chah-Firouzeh copper mines revealed that there is a relationship between hole diameter and blasting cost. Generally, the relationship can be expressed as $BC = a(D_h)^{-b}$, where D_h is the hole diameter in inches, BC is blasting cost in US dollars per cubic metre, and coefficients 'a' and 'b' are a function of bench height, UCS, joint set orientation, drilling cost per metre, and ANFO cost per kilogram. The bench height considered was 15 m. The values of coefficients 'a' and 'b' for various UCS ranges and values of γ (the angle between plane of the bench face and the plane of the main joint set) less than or greater than 90° are as follows.

Table XIV

Models for calculating blasting cost considering *in situ* rock USCS, hole diameter, and γ

UCS	(γ>90°), H = 15 m	(γ<90°), H = 1 5 m
10–70 MPa	$SC = -0.015(D_h) + 0.35$ $SD = 0.76(D_h)^{-1.9}$ $BC = 1.15(P_A \times SC + P_D \times SD)$	$SC = -0.015(D_h) + 0.4$ $SD = 0.87(D_h)^{-1.9}$ $BC = 1.15(P_A \times SC + P_D \times SD)$
	$BC_e = 3.2(D_h)^{-1.2}$	$BC_e = 3.5(D_h)^{-1.13}$
70–120 MPa	$SC = -0.015(D_h) + 0.4$ $SD = 0.87(D_h)^{-1.9}$ $BC = 1.15(P_A \times SC + P_D \times SD)$	$SC = -0.015(D_h) + 0.45$ $SD = 0.9(D_h)^{-1.9}$ $BC = 1.15(P_A \times SC + P_D \times SD)$
	$BC_e = 4.16(D_h)^{-1.2}$	$BC_e = 4.5(D_h)^{-1.15}$
120–180 MPa	$SC = -0.015(D_h) + 0.45$ $SD = 0.97(D_h)^{-1.9}$ $BC = 1.15(P_A \times SC + P_D \times SD)$	$SC = -0.016(D_h) + 0.55$ $SD = 1.2(D_h)^{-1.9}$ $BC = 1.15(P_A \times SC + P_D \times SD)$
	$BC_e = 5.4(D_h)^{-1.2}$	$BC_e = 6.4(D_h)^{-1.17}$
180–250 MPa	$SC = -0.016(D_h) + 0.45$ $SD = 1.2(D_h)^{-1.9}$ $BC = 1.15(P_A \times SC + P_D \times SD)$	$SC = -0.0186(D_h) + 0.7$ $SD = 1.5(D_h)^{-1.9}$ $BC = 1.15(P_A \times SC + P_D \times SD)$
	$BC_e = 7.75(D_h)^{-1.2}$	$BC_e = 9.4(D_h)^{-1.2}$

D_h: Hole diameter (in) BC: Blasting cost ($/m³) BC_e: Blasting cost estimated ($/m³) in 2017 SC: Specific charge (kg/m³) SD: Specific drilling (m/m³)
H: Height of bench (m) P_A: ANFO price ($/kg) P_D: Drilling price ($/m) γ : Angle between plane of bench face and the plane of main joint set

γ <90°:

UCS (MPa)	Coefficient *a*	Coefficient *b*
10–70	3.2	1.2
70–120	4.16	1.2
120–180	5.4	1.2
180–250	7.75	1.2

γ <90°

UCS (MPa)	Coefficient *a*	Coefficient *b*
10–70	3.5	1.13
70–120	4.5	1.15
120–180	6.4	1.17
180–250	9.4	1.2

This relationship shows that blasting costs will increase with increasing rock strength and γ value, but will decrease with increasing hole diameter in every range of UCS.

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Effects of oolitic haematite ore additions on the compressive strength of iron ore pellets

by F. Yang, S.P. Zhang, T. Tang, and Sh.Y. Hu

Synopsis

The effects of oolitic haematite ore additions on the compressive strength of iron ore pellets were investigated in a preliminary study. It was found that additions of oolitic haematite at levels of up to 15% by mass decreased the compressive strength from about 3250 N to 1729.1 N, which is below the minimum requirement of 2500 N for use in a blast furnace. As the oolitic haematite content increases, the Fe_2O_3 recrystallizes, forming frangible, unevenly distributed crystallites that are incompletely interlinked, thus reducing the compressive strength of the pellets. At the same time, as the oolitic haematite content is increased from zero to 15%, the porosity of the pellets increases from 13.11% to 19.85%, again resulting in a decrease in strength. It was found that the maximum amount of oolitic haematite should be 10% in order for the pellets to be suitable for blast furnace feed.

Keywords

iron ore, pelletization, oolitic haematite ore, compressive strength.

Introduction

In recent years, the rapid development of China's iron and steel industry has led to an increasing shortage of domestic iron ore resources. China currently imports more than 60% of its iron ore requirements (Xu and Shi, 2015). In order to reduce dependence on foreign iron ore resources, it is necessary to increase the development and utilization of existing iron ore resources in China. Oolitic haematite ore is widely distributed and available in China. The explored reserves of this type of iron ore reach 3720 Mt, which is approximately 11% of the total iron ore resources (Sun, Han, and Gao, 2014). Oolitic haematite ore has a unique structure, as shown in Figure 1, characterized by spherical grains composed of concentric layers. The iron oxides, fluorapatite, and chamosite (a hydrous aluminium silicate of iron) are intimately intermixed; moreover, microfine-grained haematite is disseminated with gangue minerals (Wu, Wen, and Cen, 2011). It is difficult to obtain high-grade iron concentrates with a low phosphorus content using conventional magnetic separation or froth flotation methods following fine grinding of oolitic haematite ores (Li *et al.*, 2013; Nunes *et al.*, 2012; Yu, Yu, and Xu, 2013). However, it is very important for the future of China's steel industry to increase the exploitation and utilization of this kind of ore.

Oolitic haematite ores vary widely in chemical composition. Oolitic haematite is one of the most refractory iron ores, and is characterized by low total iron content (30–55%, although the grade in individual places can reach more than 55%), and high phosphorus (0.4–1.8%) and alumina (2.5–9.0%) contents. After beneficiation, the ore grade is about 58%, the SiO_2 content is above 9%, and the contents of Al_2O_3 and P are higher (0.4%–1.8% P and 2.5%–9.0% Al_2O_3, respectively). Increasing the concentrate grade to more than 64% Fe will not only incur a great increase in mineral processing costs, but the metal yield will also decrease. In this study in order to explore the possibility of utilizing oolitic haematite with a higher grade of iron, which has not been enriched by mineral processing, oolitic haematite ore was used as an addition in pellet production, and a preliminary study of the effect of various additions on the compressive strength of pellets was carried out, based on the changes in the porosity of the finished pellets and the oxidation-recrystallization of the pellets,. The results can provide a theoretical basis and technical guidance for the industrial development and utilization of this kind of iron ore.

Experimental methods

Pellet preparation

The raw materials used to prepare pellets were

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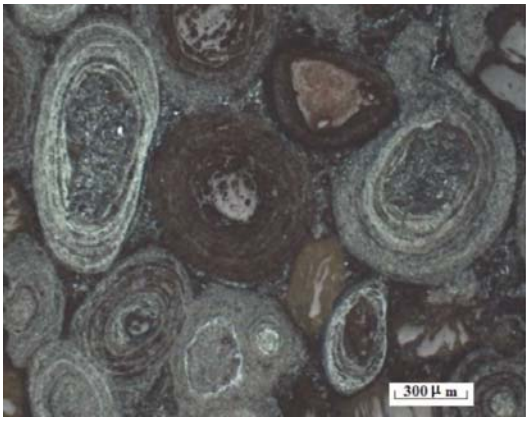

Figure 1—Microstructure of typical oolitic haematite

magnetite concentrate and bentonite from an iron and steel plant, and oolitic haematite ore from Enshi in western Hubei Province, China. The chemical compositions are listed in Table I. As indicated in Table I, the grade of iron in the oolitic haematite ore is lower than that of the magnetite concentrate, and the impuritiy contents are higher.

To investigate the effects of the oolitic haematite on the compressive strength of the pellets, additions of 5%, 10%, and 15% by mass were made to the raw materials mix. Pellets without any oolitic haematite were also prepared as a baseline standard. The bentonite content was 2.0% in all cases. The proportions of raw materials are shown in Table II.

Green pellets were prepared in a disc pelletizer of 1.0 m diameter and 0.2 m rim depth, rotating at 20 r/min and inclined at 45° to the horizontal. The pelletizing time was 30 minutes. The moisture content of the green pellets was about 8%. The green pellets were screened to separate the 8–16 mm size fraction, and dried at 150°C for 2 hours in a drying cabinet. The dried green pellets were placed into a muffle furnace preheated to 900°C and with an air flow rate of 1.5 L/min. The furnace was then heated to 1250°C over a period of 30 minutes, and the pellets were roasted at 1250°C for 20 minutes, then cooled to ambient temperature.

Testing of pellet properties

The compressive strength of finished pellets was tested using a universal test machine (UTM) according to chinese standard GB/T14201-1993. The pressurizing capacity of the device must be greater than 10 kN, and the transmission capacity of the load cell greater than 10 kN. The running speed of the head is 15 ± 1 mm/min, which necessitates constant speed pressurizing. The pellet specimen was placed between two horizontal disks and then slowly pressed by the motor-driven UTM until it developed cracks and eventually collapsed. The load cell on the bottom disk recorded the compressive load versus displacement data. Twelve sample pellets of each pellet recipe were tested; the maximum and the minimum values were deleted, and the average of the rest of the values was considered to be the final compressive strength of the finished pellets. The porosities of the finished pellets were tested by the mercury injection method. The finished pellets were also characterized under a mineralogical microscope.

Experimental results and analysis

Results

The effects of oolitic haematite ore on the compressive strength of the pellets are shown in Figure 2. As the oolitic haematite addition is increased from zero to 15%, the compressive strength of the pellets gradually decreases, from about 3250 N with no oolitic haematite addition to 1729.1 N with 15% addition.

Figure 3 shows that the porosity of finished pellets increases from 13.11% to 19.85% as the proportion of oolitic haematite ore increases from zero to 15%. The microstructures of the pellets are shown in Figure 4

Analysis

As can be seen from Figure 2, the compressive strength of the pellets decreases continuously with increasing

Table I

Chemical composition of pelletizing materials (mass %)

Raw materials	Total Fe	FeO	SiO_2	CaO	MgO	Al_2O_3	P
Magnetite concentrate	67.84	18.40	2.39	0.025	0.016	0.28	0.03
Oolitic haematite ore	58.18	1.97	9.31	0.02	0.26	4.20	0.86
Bentonite	1.67	-	57.44	2.51	2.40	11.20	0.03

Table II

Proportions of raw materials in pelletizing mix (mass %)

Oolitic haematite ore	Magnetite concentrate	Bentonite
0	98.0	2.0
5.0	93.0	2.0
10.0	88.0	2.0
15.0	83.0	2.0

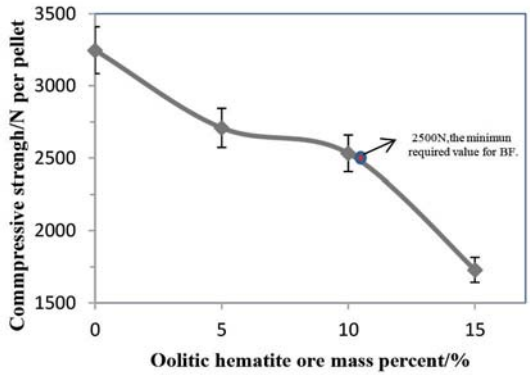

Figure 2—Effects of oolitic haematite ore on the compressive strength of pellets

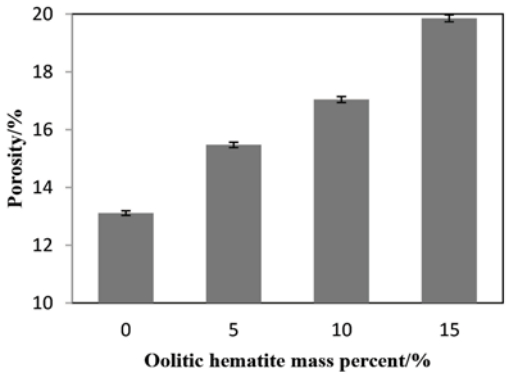

Figure 3—Porosity of the pellets with different proportions of oolitic haematite ore

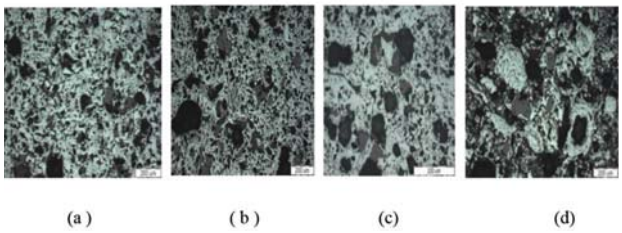

Figure 4—Microstructures of pellets with different proportions of oolitic haematite ore

proportions of oolitic haematite ore. As the oolitic haematite addition increases from zero to 10%, the compressive strength of the pellets decreases from 3247.2 N to 2532.6 N, which still meets the requirement for blast furnace feed. However, at 15% addition, the compressive strength falls to 1729.1 N, which is below the requirements of the modern large-scale blast furnace.

The porosity of pellets has a significant effect on the compressive strength. According to the Griffith microcrack theory, the relationship between the critical rupture stress and the elastic modular ratio (*i.e.*, the ratio of normal stress to the corresponding normal strain) can be expressed as Equation [1] (Shen *et al.*, 2014; Gao *et al.*, 2013a, 2013b):

$$\sigma = Y\sqrt{(E\gamma/C)} \quad [1]$$

where σ is the critical rupture stress (MPa), Y is a correlation coefficient that is determined by the size and shape of the sample and its crack (dimensionless), E is the elastic modular ratio (GPa), γ is the surface energy (J), and C is the half-length of the crack (m).

The empirical formula expressing the relationship between the elastic modular ratio and the porosity is shown in Equation [2)] (Bao and Jin, 2010; Gao *et al.*, 2013c):

$$E/E_0 = 1 - K_1\varepsilon + K_2\varepsilon^2 \quad [2]$$

where E_0 is the elastic modulus ratio of a sample without any pores (GPa), K_1 and K_1 are constants determined by the shape and direction of the cracks (dimensionless), and ε is the porosity. The last term in Equation [2], $K_2\varepsilon^2$. is relatively small and can thus be ignored.

Equations [1] and [2] show that the critical rupture stress σ decreases when the porosity ε of the pellet increases, and the compressive strength of the pellet will consequently be diminished. Therefore, an increase in the proportion of oolitic haematite ore may result in an increase in the porosity of the pellets and subsequently in a decrease in compressive strength of the finished pellets.

As can be seen from Figure 4a, with no addition of oolitic haematite the oxidation and recrystallization of Fe_3O_4 is better, and the crystallite is connecting together compactly and is evenly distributed. The crystallite is interlink-crystallite. It is known that Fe_3O_4 is oxidized to Fe_2O_3 and that Fe_2O_3 recrystallization is the main bonding mechanism for pellets. The better the recrystallization of Fe_2O_3, therefore, the greater the strength of the pellets, thus the compressive strength of the pellets without oolitic haematite ore is the greatest. An increase in the proportion of oolitic haematite ore results in an increase in the Fe_2O_3 content of the pellets and thus a decrease in the degree of Fe_3O_4 oxidation and recrystallization as Fe_2O_3. In this case, the Fe_2O_3 crystallites are frangible and unevenly distributed, which leads to crystallite interlinkages- developing incompletely and a deterioration in the compressive strength of the finished pellets, as shown in Figure 4d. This is another reason for the decrease in the pellets' strength.

Conclusions

The compressive strength of magnetite concentrate pellets with different proportions of oolitic haematite ore was investigated. The porosity of finished pellets was determined to clarify how oolitic haematite ore affects the compressive strength. The main findings are summarized as follows.

- The compressive strength of pellets decreases gradually with increasing mass fraction of oolitic haematite ore from zero to 15%. At mass fractions of oolitic haematite ore more than 10%, the compressive strength of pellets is lower than 2500 N. Therefore, the proportion of oolitic haematite ore in pellets should not exceed 10%.
- The porosity of finished pellets increases gradually with increasing mass fraction of oolitic haematite ore. As the content of oolitic haematite ore increases from zero to 15%, the porosity of pellets increases from 13.11% to 19.85%. The increase of porosity leads to a decrease in the compressive strength of pellets.
- With increasing oolitic haematite content, the degree of Fe_3O_4 oxidation and recrystallization as Fe_2O_3 decreases; the recrystallized grains are poorly interconnected and unevenly distributed, which again decreases the consolidation and strength of oxide pellets.

Acknowledgements

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Evaluation of alternative solid media for coal beneficiation using an air dense-medium fluidized bed

by K.M. Chagwedera, S.O. Bada, and R.M.S. Falcon

Synopsis

A study was carried out to identify alternative dense media with potential for use in coal beneficiation in South Africa. Magnetite, pyrrhotite, silica sand, granulated blast furnace slag, and coal discards were tested. Magnetite and pyrrhotite provided the most uniform and stable bed densities, with bed density variation less than 0.7% and standard deviation less than 0.0025 g/cm^3. A 100% magnetite bed yielded a product with an ash content of about 20.30% and calorific value of about 25.24 MJ/kg from a feed coal of size fraction –13 +6 mm and ash content of 39.27%. A cleaner coal product containg about 14.50% ash and with calorific value of 27.30 MJ/kg was obtained from a bed of pyrrhotite and magnetite blend at a weight ratio of 40:60. The same blend reduced the total sulphur content from 2.49% in the feed coal to about 0.41% at a cut density of 1.63 g/cm^3 at a probable error (Ep) of 0.083. The beneficiation of coal in a pure magnetite bed was not as effective as with the pyrrhotite-magnetite mixed bed.

Keywords

air separation, dense medium, fluidized bed, separation efficiency, bed stability.

Introduction

South Africa has vast coal reserves and its economy is heavily dependent on this abundant resource for primary energy needs. From the estimated 333.6 Mt of RoM (run-of-mine) produced in 2014, approximately 128.2 Mt was used in the generation of electricity (Prevost, 2015). The demand for coal in South Africa as the primary energy source is unlikely to change in the near future.

The Witbank coalfield in Mpumalanga Province supplies 50% of South Africa's saleable coal. However, the reserves are near exhaustion and the remaining coal is of poor quality. This has led to the development of the Waterberg coalfield, which is estimated to contain about 40 to 50% of South Africa's remaining coal (Hancox and Götz, 2014). The main challenge facing the Waterberg region is a lack of water, which might be an obstacle in implementing the conventional wet beneficiation techniques.

This investigation into utilizing dry beneficiation techniques was motivated by the large quantities of water used or lost during coal preparation, and the cost of managing and treating the large volumes of aqueous slurries generated during the wet process. In addition, coal beneficiated by dry methods retains a higher thermal heating value (Dwari and Rao, 2007). Air dense-medium fluidized bed (ADMFB) processes have been investigated for a number of decades and some of the results were patented as early as 1926 (Fraser and Yancey, 1926). The foundation for the industrial application of the ADMFB coal separation was based on studies conducted by different authors over the past decades. Zhao *et al.* (2016) present some of the most recent development in the industrial application of the ADMFB in the form of a large-scale (40–60 t/h) plant.

The solid medium material used in the ADMFB separator is one of the key factors for efficient coal separation. Magnetite has been widely used by other investigators. However, the demand for magnetite has increased because of its use in other industries such as steelmaking and catalysis, besides conventional wet coal beneficiation. In addition, the cost and future supply are of concern to the coal industry (Honaker and Bimpong, 2009). Some of the alternative solid media that have been investigated for use in the ADMFB include silica sand, silica-zircon mixtures, magnetic pearls, and paigeite. Azimi *et al.* (2013) used silica sand to achieve a recovery of 95.63%, and clean coal ash content of 10.6%, from a feed coal of 14.4% ash, at a separation efficiency of 15.29%. Firdaus *et al.* (2012) used a bed of silica-zircon mixture and produced a clean coal product of about 9.71% ash at an Ep of 0.06, with 77.8% recovery. Similarly, paigeite ore has been used to reduce coal ash content from 22.37% to 9.88% with a

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yield and Ep of 60.64% and 0.075, respectively (Zhao *et al.*, 2011). Magnetic pearls used by Luo *et al.* (2007) achieved an Ep value of 0.05, with a coal product of 13.95% ash and yield of about 90.80%.

This study focused on utilizing locally available alternative solid media materials in the ADMFB separator. The media tested include magnetite (4 880 kg/m^3), pyrrhotite (2 983 kg/m^3), granulated blast furnace slag (2 699 kg/m^3), silica sand (2 410 kg/m^3), and coal rejects (2 112 kg/m^3). The use of pyrrhotite, alone or as a blend with magnetite, for a solid medium in an ADMFB application has not been reported before. Pyrrhotite, which is a reject dumped to tailings in nickel concentration and platinum group metals (PGMs) processing, has a density (approximately 3 g/cm^3) suitable for creating a fluidized bed and is known to occur in a magnetic form (Fe_7S_8), which aids in dense media recovery. These properties make pyrrhotite a potentially suitable dense medium for coal beneficiation in an ADMFB separator.

Experimental

Materials

The materials used in this study include the solid media, density tracers, and coal sourced from the Witbank coalfield in South Africa. The solid media were also obtained from different sources within South Africa and prepared in accordance with ISO 3082:2009 into the required size range (–425 +106 µm) for testing. Density tracers were used instead of coal to determine the best operating parameters and separation efficiency for each of the media utilized. The tracers were cube-shaped particles of –13 +6 mm size range, and within the density range 1.2–2.0 kg/m^3. The 100 kg coal bulk sample used for the test was prepared according to ISO 13909:2016 at a particle size range of –13 +6 mm. Batch samples of 1 kg were subsequently used for the float-sink and the dry beneficiation experiments.

Equipment

The fluidized bed (Figure 1) used in this study was constructed from Perspex reinforced with a steel frame of a square cross-section of 40 cm × 40 cm with a height of 60 cm.

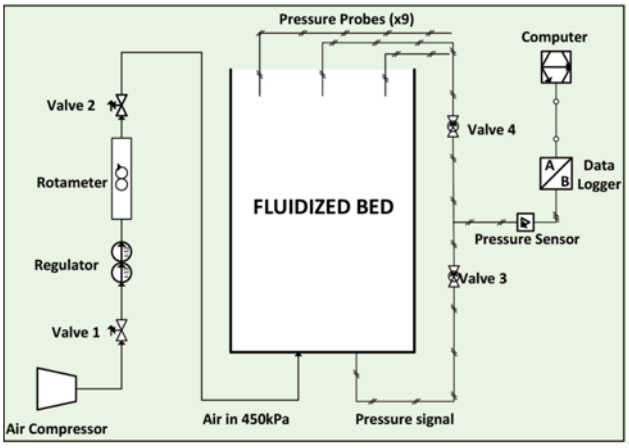

Figure 1—Experimental process flow diagram

Table I

Bed weights and corresponding static bed heights

Bed material (kg)	Static bed height		
	20 cm	25 cm	30 cm
Magnetite	94	117	141
Pyrrhotite	57	72	86
GBFS*	52	65	78
Silica sand	46	58	69
Coal rejects	41	51	61

* GBFS: Granulated blast furnace slag

The distributor was made of a canvas material compacted between two sheets of wire mesh. The inlet air into the system was cleaned and dried by passing it through an air filter, and the pressure into the system was maintained at 450 kPa(g). The pressure drop within the bed's cross-sectional area was measured at nine different points using nine probes, each connected to a pressure transducer. The pressure readings were transformed through the transducers into electrical signal 'data' and recorded using an Agilent 34970A data logger.

Test method

The fluidized bed was loaded with one dense medium material at a time, with three static bed heights (20, 25, and 30 cm) used for each material. The approximate mass of each material in the fluidized bed is shown in Table I.

The operating velocity was determined and pressure drop measurements were taken at different points in the bed cross-section (plan view). A total of nine pressure point (A to I) readings were taken at bed heights of 0, 5, 10, 15, 20, 25, and 30 cm) at a rate ot one reading per 500 milliseconds. Pressure drop profiles and density profiles were plotted, as seen in Figure 2. The process parameters such as superficial air velocity and static bed height were optimized through beneficiation tests using density tracers. A superficial air velocity of 12 cm/s, static bed height of 20 cm, and separation time of 20 seconds were established. The solid medium with the best separation efficiency was selected as the dense medium material for coal beneficiation. The tests involved feeding 1 kg of coal into the fluidized bed and performing the separation at the established optimum conditions. A brush and flat scraper (40 cm length) were used to gently remove the medium material at 1 cm intervals. All the coal particles collected from the top of the bed to 10 cm depth were combined and recorded as 'floats', and particles collected from 10–20 cm depth were considered 'sinks'. Each test run was repeated at least three times in order to improve the precision and accuracy of the results.

Analytical methods

Particle density was measured using pycnometry according to ISO 125154:2014. The particle size distribution of the samples was measured in accordance with ISO 13320:2009 using a Malvern Mastersizer 2000 laser particle analyser. The Davis tube test (ISO 8833:1989) was used to determine the

magnetic content of the medium materials. XRF analysis according to ISO TR 18336:2016 was used to quantify the iron content of the magnetic and nonmagnetic products of the Davis tube test.

Float and sink tests were performed on the feed coal in accordance with ISO 7936:1992 to determine its washability characteristics. The products and discards obtained were subjected to the same test, and the data was used to plot the partition curves for determining the separation efficiency of the process. Ultimate and sulphur analyses of all samples were performed according to ASTM D 5373-02 and ASTM D 4239-05 respectively, using a Leco CHN 628 with add-on 628 S module. Proximate analyses to determine the inherent moisture, ash content, and volatile matter were conducted in accordance with ASTM D-5142, using samples of approximately 1 g. The fixed carbon for the samples is expressed as 100% minus (ash content + volatile matter + moisture content). The calorific value was determined using a Leco AC 500 calorimeter in accordance with ASTM D5865-04.

Results and discussion

Coal and solid medium characterization

The physical properties of the media, including particle size distribution (PSD) and magnetic content, are shown in Table II. The Davis tube was used to separate the magnetic and nonmagnetic fractions in the proposed medium materials at settings ranging between 0.5 and 2.0 A. Pyrrhotite was the only medium out of the four proposed media that had both a magnetic and a nonmagnetic fraction, comprising about 30% and 70% respectively.

The results of the physicochemical analyses of the feed coal are depicted in Table II. The RoM coal sample is classified as a very high-ash, low fixed carbon, D3 grade coal with a low volatiles content (Code 3) coal according to the South African Standard Classification of Coals.

The XRF analysis conducted on the feed pyrrhotite and magnetic and nonmagnetic fractions from the Davis tube test showed that the magnetic fraction has the highest concentration of iron (Table III). This test further supports the utilization of pyrrhotite as a medium that can be used with magnetite for dense medium coal separation.

Bed fluidization characteristics

Fluidization characterization tests were carried out to determine the fluidization behaviour for each solid medium. Figure 2 shows an illustration of the pressure drop profile for two of the nine points in the bed, with magnetite as the dense medium material at 20 cm static bed height.

Table II

Physicochemical properties of coal and solid medium materials

Analysis		Samples				
		Magnetite	Pyrrhotite	GBFS	Silica	Coal Rej.
PSD	D10 (μm)	15	83	98	102	92
	D50 (μm)	113	200	229	232	218
	D90 (μm)	338	390	409	402	410
Davis tube	Magnetic %	100	30	0	0	0
	Nonmag. %	0	70	100	100	100
Feed coal						
		FC (%)	VM (%)	Ash (%)		TM (%)
Proximate	(As-received)	38.77	19.80	39.32		2.11
	(dry basis)	39.61	20.23	40.17		-
Calorific value		18.76 MJ/kg				
Ultimate analysis (dry basis)		C (%)	H (%)	N(%)	S (%)	O (%) *
		49.40	2.57	1.20	2.49	2.06

* By difference [100-(H+C+N+Ash+$H_2$0+S)]
FC: fixed carbon; VM: volatile matter; TM: total moisture; GBFS: Granulated blast furnace slag

Table III

XRF analysis results

Element	Fe (%)	S (%)	Ni (%)	Si (%)	Al (%)	Mg (%)	Ca (%)	Na (%)	Ti (%)
Magnetite	54.96	0.09	0.02	1.11	0.58	5.36	1.52	2.02	0.97
Pyrr. feed	10.66	1.03	0.16	23.93	4.41	19.55	7.05	2.32	0.41
Pyrr. mag	21.77	6.97	0.34	17.80	2.64	23.30	4.01	2.09	0.33
Pyrr. nonmag	8.35	0.67	0.35	24.62	5.61	18.11	7.77	3.52	0.41
GBFS *	0.32	0.78	0.00	21.97	9.90	11.94	23.06	3.37	0.36

* GBFS: Granulated blast furnace slag

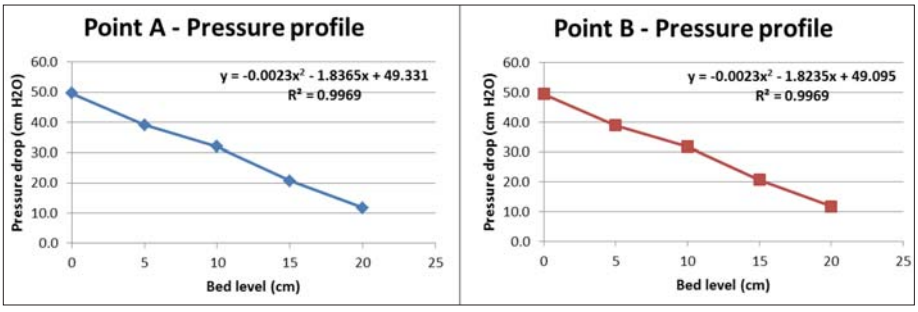

Figure 2—Illustration of pressure drop profiles for two sampling points

The equations below were used to calculate the bed density from the pressure drop profiles (Chikerema, 2011):

$$P = -\rho_{bed} g (h_{top} - h)$$
$$dP = -\rho_{bed} g . dh \qquad [1]$$

where

P is the pressure drop (Pa)
ρ_{bed} is the bed density (kg/m^3)
g = 9.81 m/s^2

The gradient of the pressure drop against bed height plot gives the bed density:

$$\rho_{bed} = \frac{-(gradient)}{9.81} \times 10^4 \qquad [2]$$

where 10^4 is a conversion factor.

The density was determined for each pressure point in the bed and the data was tabulated as shown in Table IV.

Uniformity of the bed density is an important indicator of the stability of a fluidized bed and consequently its ability to provide an efficient separation. Efficient dry separation conditions in an ADMFB are obtained when a stable dispersion, fluidization, and micro-bubbles are achieved. It is very important that the bed density is well distributed in three-dimensional space and does not change with time (Chen and Wei, 2003). The density distribution for different points in the bed utilized in this study was determined by plotting the bed density attained against the bed height. The distribution was analysed vertically per each sampling point and horizontally per each bed level (multiple sampling points).

Figure 3 shows a plot of the vertical distribution of the bed density for two specific pressure points. For pressure point A, the density range was 1871–1965 kg/m^3 with an even distribution (R^2 = 1) from the bottom to the top of the bed, which was the case for point B and other pressure points taken in this study.

Table IV

Bed density distribution for magnetite at 20 cm bed height

	Density (kg/m^3)								
Level (cm)	**A**	**B**	**C**	**D**	**E**	**F**	**G**	**H**	**I**
0	1871	1859	1859	1871	1858	1870	1870	1858	1862
5	1895	1882	1882	1894	1882	1894	1894	1881	1884
10	1918	1906	1906	1918	1905	1917	1918	1905	1905
15	1941	1929	1929	1941	1929	1941	1943	1928	1926
20	1965	1953	1952	1965	1952	1964	1967	1952	1948

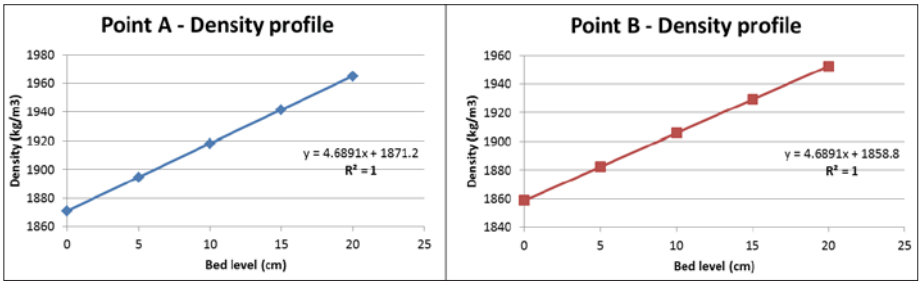

Figure 3—Density profiles for two points in the bed

The stability and uniformity of the fluidized bed were established by determining the standard deviation for the density distribution (S_p) for each point (Zhang *et al.*, 2014). S_p is given by the following equation:

$$S_\rho = \sqrt{\frac{1}{N}\sum_{i=1}^{N}(\rho_i - \bar{\rho})^2} \quad [3]$$

where

N is the total number of sampling points

ρ_i is the density of point i, g/cm^3

$\bar{\rho}$ is the mean density of all points, g/cm^3.

On the other hand, the coefficient of variation (CoV) was also used in this study to further determine the extent of variation of the bed density from its mean at each horizontal bed level. The CoV was calculated by the following equation:

$$\text{CoV} = \left(\frac{\text{Standard deviation of density for a bed level}}{\text{Average density for that level}}\right) \times 100 \quad [4]$$

Table V shows the standard error (S_p), and CoV for bed density at 20 cm bed height of magnetite.

Table V shows that the CoV ranges between 0.33–0.37%, while the S_p is between 2.05–2.41 kg/m^3, indicating a very stable and uniform bed under these operating conditions. The same procedure was applied to all the solid media proposed as an alternative to magnetite in this study. The data obtained was used to plot the graph of CoV *vs.* average bed density, as seen in Figure 4.

Figure 4 was used to select the best solid medium and the static bed height at which the bed density is stable and uniform. Magnetite, pyrrhotite, and GBFS at 20 cm bed height exhibit the most uniform and stable beds, with a coefficient of variation less than 0.7%. However, the GBFS bed had an average density of about 1 000 kg/m^3 which, coupled with its lack of magnetism, makes it unsuitable for coal separation as the sole medium but with some potential if blended with magnetite.

ADMFB separator results using magnetite and pyrrhotite ore (–425 +106 µm)

Magnetite and pyrrhotite were chosen as a solid medium for the initial beneficiation test, using density tracers. Preliminary results with a magnetite bed alone were used as a baseline for comparison. ADMFB separation was conducted using different static bed heights, and a yield of 55% and Ep of 0.40 were achieved with a bed height of 20 cm. This bed height was selected for all subsequent tests. Pyrrhotite alone, with a calculated average bed density of 1300 kg/m^3, could not form a bed with an effective cut density for the density tracers (1.2–2.0 kg/m^3) tested. Therefore, a blend of pyrrhotite and magnetite at different weight % ratios was tested.

Density tracer separation with a blend of pyrrhotite/magnetite

An investigation was conducted to determine the influence of different blends of magnetite with pyrrhotite on the ADMFB separation efficiency. The blends were prepared using from 20% to 50% by weight pyrrhotite, with a bed static height of 20 cm. The best separation results for each blend ratio at the optimum conditions are plotted in Figure 6. The optimum blend was at 40% pyrrhotite plus 60% magnetite, with the yield ranging between 53–58% and probable error 0.053–0.073. At higher proportions of pyrrhotite, the yield and separation efficiency began to decline. Therefore, the established optimum operating parameters of 12 cm/s superficial air velocity, 20 cm static bed height, and 20 seconds separation time were then utilized for the separation of coal at this optimum blending ratio.

Coal separation with magnetite and pyrrhotite/magnetite beds

The RoM material was first subjected to sink/float analysis in order to evaluate the potential of the ADMFB for cleaning

Table V

Bed density distribution showing uniformity of the bed

	Density (kg/m^3)				
Pressure point	**0 cm**	**5 cm**	**10 cm**	**15 cm**	**20 cm**
A	1871	1895	1918	1941	1965
B	1859	1882	1906	1929	1953
C	1859	1882	1906	1929	1952
D	1871	1894	1918	1941	1952
E	1858	1882	1905	1929	1952
F	1870	1894	1917	1941	1964
G	1870	1894	1918	1943	1967
H	1858	1881	1905	1928	1952
I	1862	1884	1905	1926	1948
CoV (%)	0.33	0.34	0.35	0.37	0.37
Sp (kg/m^3)	2.05	2.13	2.24	2.37	2.41

CoV: Coefficient of variation

*S_p: Standard deviation of density distribution

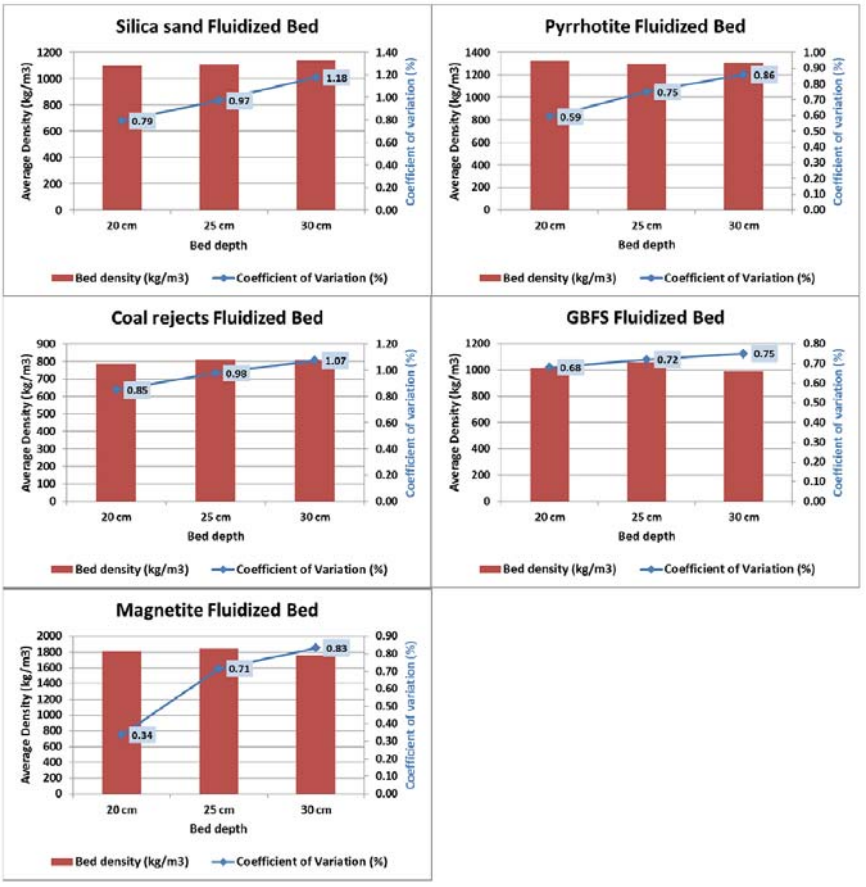

Figure 4—Summary of fluidized bed properties of the solid media

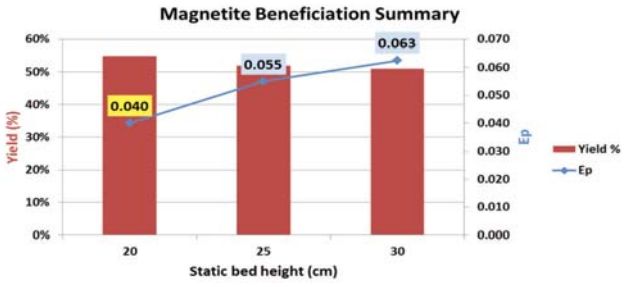

Figure 5—Results of beneficiation tests using magnetite as solid medium

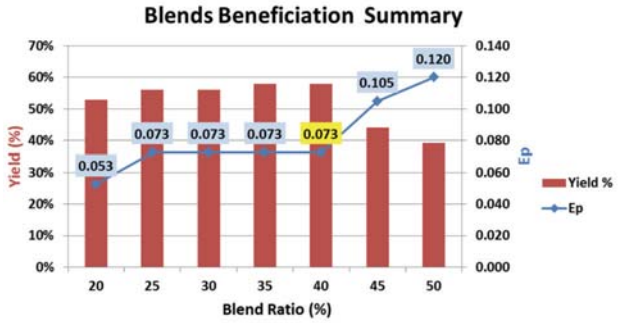

Figure 6—Beneficiation results from the blends of pyrrhotite/magnetite

coal under different conditions. A 100% magnetite fluidized bed at an optimum operating fluidization velocity U_{of} of approximately 1.4 U_{mf} and 20 cm static bed height was utilized, and the result compared with that obtained with the blend of pyrrhotite/magnetite.

From the sink and float analysis (Table VI), it can be seen that clean coal of 5.54 % ash with a yield of 1.18 % is obtainable at 1.3 RD. In addition, at an RD of 1.5, clean coal with an ash content of 13.96% and yield of 23.81% was also obtained.

As depicted in Figures 7 and 8, at a cut-point/separation density (ρ50) of about 1.81–1.82 g/cm^3 for the 100% magnetite bed, coal products with a yield and Ep ranging between 60.26–60.89% and 0.070–0.075, respectively were obtained. The product was a clean coal with an ash content of 19.60–20.95% and calorific value of 25.01–25.46 MJ/kg.

According to Figure 9, a dense medium of 40% pyrrhotite plus 60% magnetite blend produced clean coal with a error probable (Ep) between 0.080–0.083 and yield ranging between 52.04 and 52.67%.

The bed's cut-point/separation density (ρ50) was at 1.63–1.64 g/cm^3 (Figure 10). The ash content of the feed coal was reduced from 39.32% to 14.21–14.75% (an overall ash reduction of 62.49–63.86%, while the feed sulphur content was reduced from 2.49% to 0.41–0.42%. The calorific value of the product was 26.77–27.58 MJ/kg.

The lower ash products obtained with the 40% pyrrhotite bed were a result of the lower ρ50 of the bed compared to 100% magnetite, for which the ρ50 was 1.81–1.82 g/cm^3. These results indicate that pyrrhotite has potential for use in mixed solid media for ADMFB beneficiation.

Table VI

ADMFB separator and float-sink products characteristics

Sample		Ash (%)	CV (MJ/kg)	VM (%)	Yield (%)	Ash reduction (%)	S (%)
Feed		39.32	18.76	19.80	100	0.00	2.49
ADMFB 40% pyrrhotite	Product 1	14.75	27.60	24.14	52.36	62.49	0.42
	Product 2	14.21	26.77	23.55	52.18	63.86	0.41
	Product 3	14.54	27.58	25.09	52.20	63.12	0.42
ADMFB 100% magnetite	Product 1	19.60	25.46	22.50	60.26	50.15	
	Product 2	20.20	25.24	21.99	60.67	48.63	
	Product 3	20.95	25.01	22.24	60.89	46.72	
Float and sink test	F @ 1.30	5.54	31.47	28.31	1.18		
	F @ 1.40	8.29	30.47	26.72	13.99		
	F @ 1.50	13.96	27.70	23.06	23.81		
	F @ 1.60	22.05	24.10	22.00	10.04		
	F @ 1.70	32.06	19.83	18.60	6.90		
	F @ 1.80	38.56	17.73	18.84	6.55		
	F @ 1.86	42.79	15.81	16.14	2.45		
	S @ 1.86	76.53	4.08	10.70	35.08		

* CV: calorific value; VM: volatile matter; F: float; S: sink

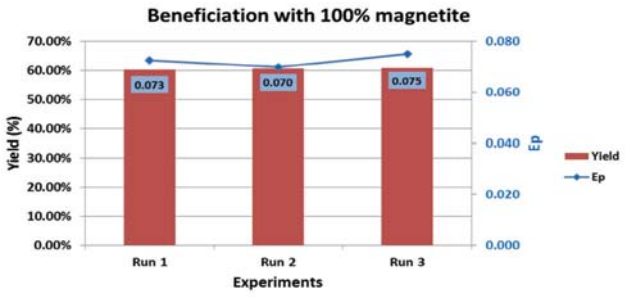

Figure 7—Coal beneficiation results with 100% magnetite bed

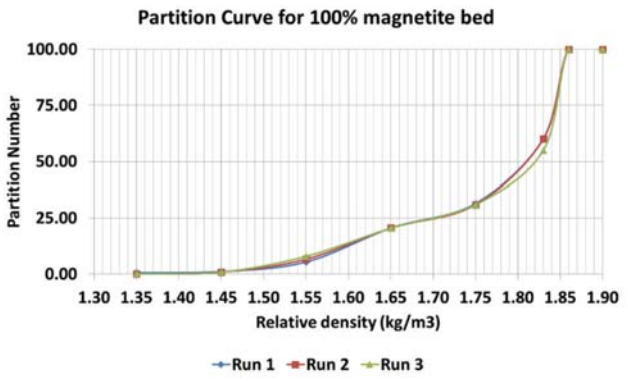

Figure 8—Coal beneficiation partition curve with 100% magnetite bed

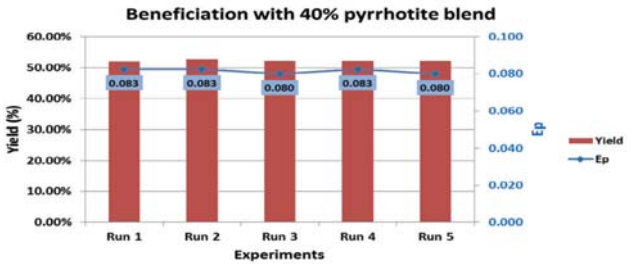

Figure 9—Coal beneficiation results with 40% pyrrhotite blend

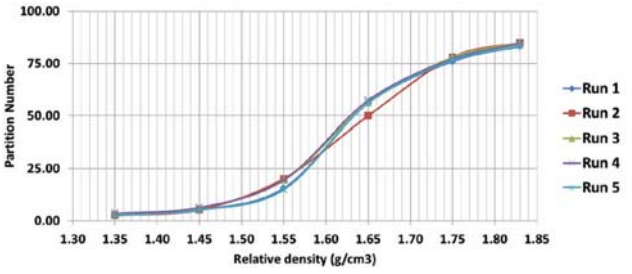

Figure 10—Coal beneficiation partition curve with 40% pyrrhotite blend

Conclusion

Pyrrhotite has proved to be the superior dense medium material among the materials proposed as an alternatives to magnetite. The most uniform and stable bed densities were achieved using a blend of pyrrhotite (40%) and magnetite (60%). A cleaner coal product with less ash and sulphur content was obtained from this blend compared to using magnetite alone. Using a 40% pyrrhotite bed, feed ash content was reduced from 39.32% to about 14.50% and the product coal had a calorific value of about 27.30 MJ/kg. The 100% magnetite bed produced a product with an ash content of about 20.30% and calorific value of about 25.24 MJ/kg.

The blending of pyrrhotite, which is considered a reject 'no-cost' material, with magnetite could reduce the cost of operating a coal ADMFB process significantly.

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A preliminary qualitative evaluation of a hydraulic splitting cylinder for breaking rock in deep-level mining

by W.W. de Graaf and W. Spiteri

Synopsis

Hydraulic rock-splitting cylinders have proved successful in numerous civil and construction applications. The purpose of this study was to conduct a preliminary qualitative evaluation of the applicability of the hydraulic splitting cylinder in deep-level mining with the aim of recommending equipment modifications and operational practices. The particular instrument used in the study was the DARDA® hydraulic splitter.

Conventional drill-and-blast practice in deep-level mining often impacts adversely on the immediate environment. Alternatives are periodically sought for efficient, continuous, and safe rock-breaking in situations where conventional blasting is undesirable. A considerable amount of investigation work has been conducted by mining companies, equipment manufacturers, and research institutions on numerous methods of non-explosive mining, including the use of the hydraulic rock-splitter.

Several trials were conducted underground. The most challenging aspect of in mechanical rock-splitting is to create a second free face in the stope, and the trials evaluated four different 'cut' layouts to achieve this objective. The trials highlighted the limits of the equipment in its current phase of development, as well as the importance of quality drilling in terms of collaring the hole, hole length, and directional accuracy. In the presence of a second free face the splitter becomes far more effective. The unit is simple in design and is easily integrated into existing mining operations. It also does not require a technically skilled workforce or expensive maintenance.

Rock-breaking with the use of a rock splitter could have a place in niche applications in an underground mining operation, with some equipment modifications and further development of the process to establish a free-breaking face. General operational difficulties experienced underground during the trials are summarized and possible solutions recommended.

Keywords

rock-breaking, mechanical splitting, hydraulic splitter.

Introduction

For several decades research has been undertaken to develop viable alternative methods to the use of high-energy explosives for breaking rock and general mining (Haase and Pickering, 1991; Murray, Courtley, and Howlett, 1994). The motivating factors have predominantly been related to allowing continuous mining operations without interruption, and reducing the environmental impacts such as blast-induced ground vibrations, air blast, post-blast noxious fumes, flyrock, and damage to the surroundings (Singh, 1998). In underground mining operations, safety considerations such as the triggering of falls of ground, damage to the side- and hangingwalls, and blast-induced seismic activity also feature prominently. Res, Wladzielczyk, and Ghose, (2003) and Ramezanzadeh and Hood (2010) reviewed and summarized progress to date on the subject. There are, however, minimal literature references on the use of hydraulic splitters in underground mining.

In South Africa particularly, underground mining has now reached unprecedented depths with numerous associated challenges, and the cost of mining at these depths has increased correspondingly. As a result, the major mining companies are conducting serious investigations looking at all aspects of modernization, including mechanization, automation, and robotics, to alleviate the challenges of mining underground at great depths. Drill-and-blast mining is difficult to automate, presenting yet another reason for the continuing search for a viable alternative.

The work conducted in this investigation was aimed at a preliminary qualitative evaluation of the use of a hydraulic splitter in underground mining and to recommend possible modifications to equipment and to operating methods. There was no intention to evaluate the efficiency of the system as a method of continuous mining. Furthermore, as the object was to investigate the suitability of the equipment, no time-and-motion studies were carried out. Rather, the splitter is seen as a potential candidate for non-explosive rock-breaking in niche applications where conventional blasting techniques are not possible or desirable. The choice of the mechanical splitter, as opposed to other non-explosive rock-breaking systems, for this study was motivated by the following: the device has been used successfully in the civil and construction industries; the systems are

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readily available and affordable; the application of the splitter is similar to the application of drilling equipment, hence available infrastructure can be used and minimal training of personnel is needed; and the equipment is relatively simple to maintain.

As described by Murray, Courtley, and Howlett (1994), there are two subgroups of mechanical splitters – the wedge-and-feather splitters and the radial axial splitters. The radial axial splitter simply differs from the normal wedge/feather splitter, such as the one used in this study, in that the feathers are moved outwardly by a retracting rod as opposed to a protruding rod. The particular instrument used in this work was the wedge-and-feather splitter supplied by DARDA®. The choice was based solely on commercial availability.

The DARDA® splitting cylinder

The detailed description and specifications of the various DARDA® splitting cylinder device is given on the company's web site (http://www.darda.de). Figure 1 shows the DARDA® splitting cylinder and power supply.

Essentially, the concept makes use of the wedge principle, whereby a borehole is initially drilled into the material (concrete, rock), then the DARDA® splitter wedge set (two counter-wedges or feathers and a central wedge) is inserted into the hole. The central wedge is then driven forward between the two feathers under hydraulic power, forcing the feathers outward against the walls of the hole. Radially loaded stress build-up is created that fractures the rock. The DARDA® hydraulic unit will apply a pressure of up to 400 t

Figure 1—Hand-held DARDA® splitting cylinder and power supply (DARDA® GmbH)

Figure 2—The rock-splitting concept. (DARDA® GmbH)

Figure 3—Typical application of the DARDA® splitter (DARDA® GmbH)

to force the feathers apart (see Figure 2) (Anderson and Swanson, 1982; Duncan and Langfield, 1972; Paraszczak and Hadjigeorgiou, 1994). A detailed mathematical analysis of the stresses induced on the rock by the splitter was done by Chollette, Clark, and Lehnhoff (1976).

According to the DARDA® web site, the benefits of hydraulic splitting include controlled breaking without the adverse side-effects seen with impact breakers or conventional explosives.

The DARDA® splitter has been successfully applied in the general civil demolition field for breaking and splitting concrete and rock. A typical application of rock-splitting on surface is shown in Figure 3.

The splitting unit is manufactured from steel and aluminium to reduce the weight. The control valve will either extend or retract the centre wedge. The wedges are manufactured from steel coated with a carbide layer for increased durability.

The hydraulic pump delivers a maximum pressure of 50 MPa to the splitter. The pressure is controlled through a pressure limiting valve. Different drive systems are available for various applications and can be either electric, air, diesel, or fuel motors.

During these trials, a 220 V electric motor was used to drive the hydraulic power pack.

The DARDA® splitter is constructed for use in robust environments. The material properties of the components, and the extremely high forces exerted to fracture the rock mass, impose certain restrictions on the equipment size and weight, which in turn impose limits on the operability, such as the minimum hole diameter. The C12L model splitting cylinder was used during these underground trials. This unit weighs 32 kg and is approximately 1.3 m in length, which is not very different to the weight and proportions of a rock-drill. The applicable hole diameter ranges between 45 mm and 48 mm. The two feathers, or counter-wedges, are 450 mm in length. However the minimum hole length required is 680 mm, to accommodate the extending centre wedge.

Field work

Trials were conducted over a period of five months at the then Gold fields KDC West Gold Mine - Pitseng shaft. A non-

Figure 4—Layout of stope, A and B indicating where trials were conducted

production stope was established on 24 level where several new mining initiatives could be trialled and tested. The stope width was on average 1.5 m in unfractured footwall quartzite with a uniaxial compressive strength (UCS) of 160 MPa (personal communication – Professor F. Malan, University of Pretoria). The stope was equipped with the necessary services, including water, compressed air, electricity, and WiFi connections for communication.

Figure 4 shows where the tests were carried out on the rock face: position A in the absence of a second free face, and position B adjacent to the gully, where a second free face was present. During the trials, drilled holes were grouped into one of two categories, either 'cut' for the creation of a second free face or 'slicing' for breaking the rock along the second free face.

All the proposed cut designs were carefully marked on the face and the drilling accuracy was measured and noted. The accuracy of drilling was determined with the use of a clino-rule for measuring the angles, hole depth, and horizontal and vertical distances between the holes. A simple hand-held spring scale was used to measure the amount of rock broken. Still photography and video footage of all splitting operations was taken.

The 'cut' holes:

The hole layouts for splitting were based on conventional drilling and blasting knowledge and experience, as well as established techniques for the use of the splitter in surface mining and demolition applications. In mining terms the 'cut' is a pattern of drilled holes in the rock face, used to create a second breaking face. In conventional drill-and-blast operations, the 'cut' drill pattern differs from the production drill pattern in hole layout, drilling angle, and initiation timing of the blast-holes. The sole purpose of the 'cut' is to create a second breaking face into which successive production holes will break outwards to the perimeter of the excavation. A large variety of 'cut' designs have been proposed for mechanical splitting; for example the US Bureau of Mines (Anderson and Swanson, 1982) suggested the spiral-shaped round drill pattern, whereas Clark and Maleki (1978 recommended a similar method to the V-cut used in an explosive round.

Breaking rock using the DARDA® splitter is a much slower process than blasting. The rock around each hole has to be individually fractured until it breaks into an adjacent drilled hole or a void. Holes must be concentrated in the 'cut' area to facilitate the creation of the second free face. In this investigation, the void area was drilled according to several proposed hole layouts. These layouts included a few designs with holes drilled at an angle, a few combinations of straight and angled holes, and designs where larger holes were included in the cut pattern. The intention was that the larger holes would act as a 'mini-cut' into which the splitter holes would break.

A number of drill patterns to create the cut were carefully considered. As the holes have to be drilled to a minimum depth to accommodate the fully extended central wedge, holes drilled perpendicular to the advancing face had the advantage that they could be deepened and used again for splitting.

In conventional drill-and-blast tunnel development, creating the cut requires a high concentration of explosive energy (Atlas Powder Company, 1987) hence the high concentration of blast-holes. Once the 'cut' has been created, hole burdens can be increased because a second free face is now present. In order to reduce drilling time during these trials, an attempt was made to create the most efficient hole layouts for the 'cut' area in terms of the least number of holes for effective breaking using the DARDA® splitter.

As mentioned, the holes were drilled to a length that would accommodate the protruding centre wedge. After the first round of breaking the remains of each hole would be deepened for the next round. Holes drilled at an angle could not be deepened, as they would extend beyond, and break outside, the perimeter (see Figure 5).

The one area where angled holes would be necessary would be in the case of perimeter holes, *i.e.* holes on the edge

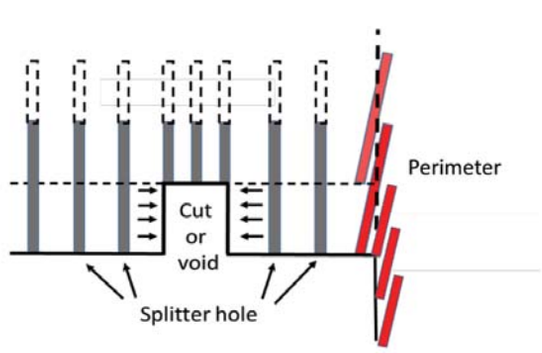

Figure 5—Plan view of splitting rock into newly created void and perimeter holes drilled at an angle

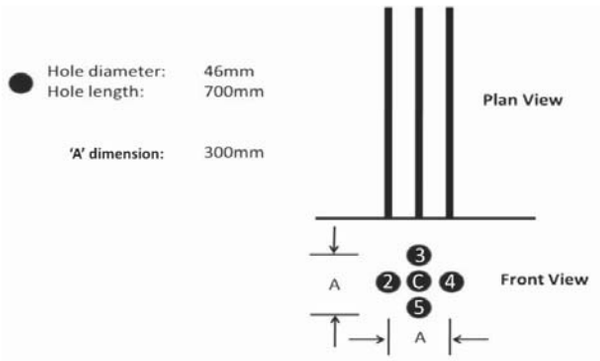

Figure 6—The five-hole diamond pattern

of the stope. These holes would have to be drilled at a slight angle, referred to as the 'look-out' angle (Cooper, Berlieand, and Merminod, 1980). The holes would be collared inside the planned perimeter and angled outwards with the toe of the hole situated on the outside of the perimeter (see Figure 5) to accommodate the centre wedge of the splitter and also to retain the contours of the stope. In these trials, perimeter holes were not drilled or investigated.

During this investigation four cut designs were considered and tested underground. These four trial patterns were chosen based on simplicity of the drilling pattern and the minimum holes drilled per cut, and are described below.

Cut-hole layout 1

The cut-hole layout 1 drilling pattern consisted of five parallel holes to create the 'cut'. The four outer holes were drilled on a 300 mm diamond pattern (Figure 6). The fifth hole was drilled in the centre of the diamond. The holes were drilled to a minimum depth of 700 mm.

Cut-hole layout 2

The second pattern trialled consisted of seven holes. All the holes were drilled to a depth of 700 mm, parallel to one another and perpendicular to the stope face. The outer holes were drilled in a hexagon pattern with dimensions of 450 mm (Figure 7). One hole was drilled in the centre. Breaking was initiated by inserting the splitter in the centre hole, hole 'C'.

Cut-hole layout 3

Figure 8 shows a drilling pattern of 11 holes. The eight outer holes were drilled in a rectangular box pattern to a minimum length of 700 mm. The outer holes converged to the centre row of holes at an angle of between 3° and 6° to the perpendicular. Three holes were drilled to a depth of 350 mm, and parallel to one another, between the two outer rows of holes. The splitter was inserted in the outer holes and the inner holes acted as 'break' or 'relief' holes.

Cut-hole layout 4

Figure 9 shows another combination of perpendicular holes and angled holes. The perpendicular 'relief' holes were drilled parallel to one another, and perpendicular to the rock face to a short depth of 350 mm. The angled holes (700 mm depth), into which the splitter was inserted, were drilled as a rectangular nine-hole pattern converging towards the perpendicular short holes at an angle of between 19° and 23° to the perpendicular, the angles being dictated by the ease of collaring and to minimize the burden between the first row of angled holes and the adjacent perpendicular holes (dimension C in Figure 9). The splitter was initially inserted in hole number 2.

The production or 'slicing' holes:

Two trials were carried out in an area of the stope where the second free face had previously been created by blasting. The

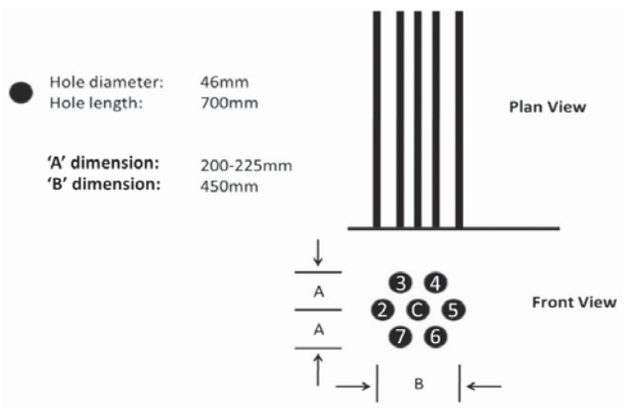

Figure 7—The seven-hole hexagon pattern

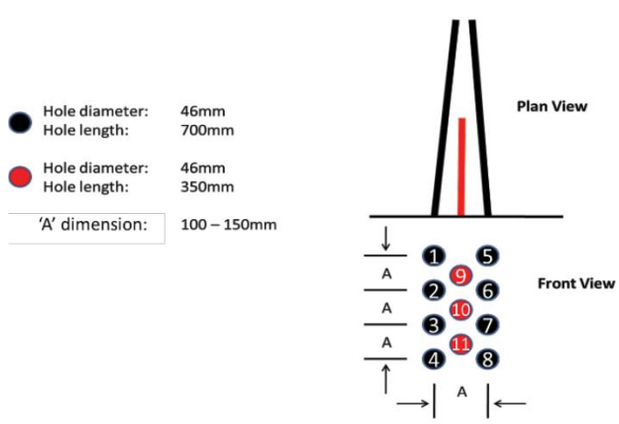

Figure 8—Rectangular box pattern with staggered (non-splitter) centre holes

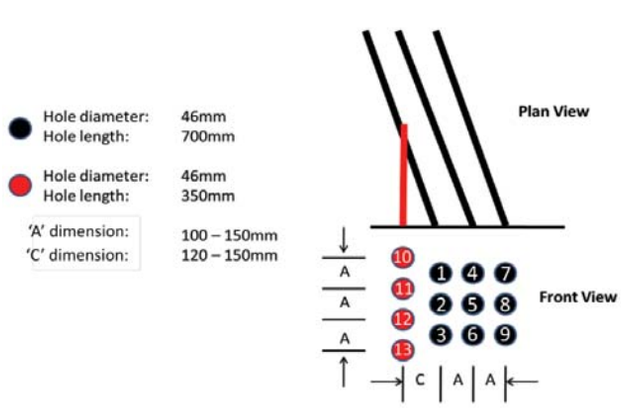

Figure 9—Nine-hole square pattern and four non-splitter holes

rock face was scrutinized and the position of each hole was carefully selected and marked and drilled parallel to the second free face (as per Figure 4, position B and as shown in Figure 10). No plans were presented for the production holes; instead, the pattern was determined according to the face conditions. The distance between any production hole and the free face (burden) varied from 300 mm to a maximum of 500 mm. The results are discussed in the next section.

Results and observations

The observations for each individual test were noted during the trials and further analysed in conjunction with the video footage. One critical common observation from all the tests was the dependence on accurate drilling in terms of the collar position and direction. With divergence of hole direction from the planned pattern, the amount of rock that has to be cracked and broken may increase, hence reducing the performance of the splitter.

Cut-hole layout

Hole layout 1

The DARDA® splitter was inserted into the centre hole ('C' in Figure 6). As the wedge was extended, no visual cracking or breaking of the rock mass was observed. The wedge was retracted and inserted in hole 2 of the pattern. This hole was randomly selected, and as the splitter's centre wedge started extending, some minor cracking developed between the splitter hole and the centre hole. The DARDA® splitter was then removed from the hole and inserted into hole number 3. As the wedge started protruding, more visible cracks developed towards the left and centre holes. The diameter of the centre hole decreased due to the fracturing in the centre of the 'diamond' and the 'swelling' of the rock mass. The same effect was observed when the splitter was inserted into holes 4 and 5. During the splitting process some small rock fragments fell to the footwall; however, the majority of the fragments had to be removed manually. The cut advance produced 30 kg of rock with an average depth of 36 cm (compared to the splitter's feather length of 45 cm).

Figure 10—Free face slicing with visible drill barrels parallel to the free face

Hole layout 2

The DARDA® splitter was similarly inserted into the centre hole ('C', Figure 7). As per the five-hole diamond pattern (layout 1), initially no visual cracking or fracturing occurred. The splitter was then inserted into hole 2 of the pattern. As the wedge was extended, visible hairline cracks developed towards the centre hole. Inserting the splitter into hole 3 caused increased cracking, but not sufficiently for fragments to easily fall to the footwall. The splitter was inserted into all the holes and random hairline cracking was noted. However, attempts to break and loosen the rock were unsuccessful. The splitter was then re-inserted in turn into each of the holes. During this process visible fracturing of the rock was noted and some small fragments fell to the footwall. The DARDA® splitter was removed and the void area was cleaned by removing the loose fragments using a steel rod. This cut design produced 35 kg of rock fragments with an average depth of 38 cm.

Hole layout 3

In hole layout 3 the drilling was slightly more complex than in layouts 1 or 2.

The DARDA® splitter was inserted into hole number 2 (Figure 8) of the drilled pattern. Immediately after the centre wedge started protruding a fracture developed diagonally across the centre hole, hole 9, and hole 5. Upon closer inspection it was evident that there was an existing fracture

Figure 11—The five parallel hole cut

in the area where the holes were drilled. This existing fracture assisted the splitting process, consequently all the holes into which the splitter was inserted generated cracks. Even in this case, however, fragments had to be removed by inserting a steel rod into the cut area and wedging the piece out until the entire area was cleared. This cut design produced 38 kg of rock fragments to a depth of 35 cm.

Hole layout 4

In this particular test, the uneven nature of the face created problems during the drilling of the nine-hole pattern, and the holes were not collared on the marked positions. This was, however the only suitable area for the trial.

The splitter was initially inserted into hole 2 of the nine-hole grid (Figure 9). As the centre wedge was extended, fractures developed between hole 2 and 11 and between hole 2 and hole 12. The same result was observed when the DARDA® splitter was inserted in holes 1 and 3. Again, small rock fragments fell to the footwall but the balance of the fragments had to be removed by hand. The splitting of the grid holes, holes 4 to 8, was difficult due to the large variation between the marked position and actual drilled position of each hole. Hole 9 was abandoned due to the splitter not being able to create any fractures during the splitting process. Some 65 kg was removed from the void area. The larger amount of rock removed compared to the other cuts was due to the larger cross-sectional area mined. The average depth of the cut was 28 cm.

Production or 'slicing' holes

The distance from the free face to the splitter hole position varied between 200 mm and 500 mm. Any existing cracks visible in the rock mass were used to assist in the breaking process. Compared to the 'cut-hole' trials, a relatively higher success level was achieved in breaking rock chunks towards the free face. The majority of the fragments broke on existing fractures. Initially the distance of the holes from the free face was set at 200 mm but as the trials progressed and breakage was achieved easily, the distance was incrementally increased to 500 mm. Spalling of the rock into the second free face was quick and effortless for the splitter and at the same time cracks appeared behind the row of holes, which were exploited in the second round of drilling and splitting.

Discussion

The underground trials identified the limits as well as the potential of the DARDA® splitter in this application. A number of lessons were also learned, including the need for drilling accuracy. Operational aspects of the DARDA® splitter and the results achieved are discussed in the following sections.

Cut-hole trials

As mentioned above, creating the cut or second free face is a crucial step allowing for efficient subsequent rock-breaking into the newly established free face. Within the confines of the various cut patterns trialled in this investigation, difficulty was found with the use of the splitter. Of the four patterns tested, none was shown to be significantly better than the other three in terms of ease of establishing the initial cut. Certain factors need to be considered in the design of the cut, particularly the need to minimize the number of holes required so that drilling time is kept to a minimum. The five-hole burn cut was drilled in the shortest time, but produced the least amount of broken rock. The greater the initial cross-sectional area of the cut, the easier it is to split the following holes (which now become 'slicing' holes). The pattern of holes extending radially outwards around the cut area can then be wider, *i.e.* with holes further apart. Furthermore, by increasing the cross-sectional area of the cut, broken fragments are removed more easily from the created void. In general, however, creating an initial second face or 'cut' using the splitter has been found difficult to achieve and none of the four cut patterns tested gave satisfactory results. A possible solution would be to increase the number of holes and decrease the distance between them to create the initial cut. An alternative would be to increase the diameter of non-splitting holes. This field needs further investigation.

Slicing hole trials

The splitter was found to be more effective for the slicing operation. These tests proved that once a second free face has been established, the use of the splitter becomes more viable and efficient.

General discussion

Drilling accuracy

Drilling of the holes in the rock face for the use of the splitter posed some specific challenges compared to drilling for blasting operations. These issues could, however, be fairly easily rectified. Hole length is crucial as the DARDA® splitter extends the centre wedge into the hole and it should be able to move freely and not be obstructed by the toe of the drilled hole. A hole that is drilled too short will destroy the centre wedge. The hole diameter should be equal to or slightly greater than the specified diameter of 46 mm. Inserting the splitter into a hole with a diameter less than 46 mm is not possible. Holes need to be drilled as straight as possible. This was highlighted when an early test hole deviated from the planned direction and the splitter centre wedge was consequently bent. Furthermore, as mentioned above, inaccurate drilling can drastically affect the performance of the splitter.

Breaking cycle

Although, no detailed time studies were done during the trials, several relevant time periods were noted for interest and future reference (see Table I).

Table I

General time periods required for various DARDA® functions as experienced during the trials

Manhandling of DARDA® from one hole to the next	17–50 seconds
Protruding wedge	53–60 seconds
Retracting wedge	44–47 seconds
Greasing blades	20–55 seconds

DARDA® splitter operational learnings

Several challenges were experienced with the equipment during the initial trials, causing the cycle to take longer than planned.

- Firstly, proper lubrication of the feathers and wedge is critical, and in these trials the feathers and wedge were lubricated approximately every fifth hole. The feathers were manually opened and the lubricant was squeezed between the moving parts. This was time-consuming and slowed down the entire process. It is suggested that this process be automated by installing a greasing device on the DARDA® splitter, which may also increase the usable life of the moving parts.
- The DARDA® splitter also had to be supported when inserted into the hole. During the fracturing process, it was found that as the rock broke away from the hole perimeter, the feathers were exposed and the unit fell to the ground. The unit was supported by tying a rope to the handle of the splitter and suspending it from the safety netting used in the stope. A supporting rig would be advisable.
- On a few occasions some small rock fragments were caught between the feathers and the wedge as the splitter was withdrawn from a hole. These fragments needed to be removed manually before the wedges could be inserted into the next hole.
- When the rock mass did not give way during fracturing, removing the DARDA® splitter from the hole was extremely difficult and a pinch bar was needed to open fractures and allow the wedges to be dislodged.
- On a few occasions the splitter's feathers did not 'grip' against the perimeter of the hole and as the wedge was extended the DARDA® splitter moved backwards.

Other operational learnings

- During the fracturing process the rock fragments on the free face generally fell to the footwall. However the fragments deep inside the void area had to be removed using a steel rod and the last fragments were removed by hand. An improved cut design could alleviate this issue.
- The splitting process should start from the footwall of the stope face and progress towards the hangingwall. In this way the drilled holes are not obscured by rock fragments.
- During most of the trials fractures did not develop immediately after the DARDA® splitter wedge was extended. In some cases it took between two and three attempts before fracturing was observed between the splitting hole and an adjacent hole or the second free face.

Conclusions

With some improvements to the equipment and technique, rock-breaking with the use of a rock splitter could well have a place in an underground mining operation. It is suitable for niche applications, such as (but limited to) areas where conventional drilling and blasting is prohibited, in removing safety pillars and shaft pillars, and areas where seismic events are rife.

The static hand-held tool can easily be manhandled in restricted spaces. It is also simple in design, can be easily integrated into existing mining operations and infrastructure, and does not require a technically competent or skilled workforce, or expensive maintenance.

The trials showed the shortcomings of the equipment in developing the initial cut or second free face. Furthermore, the use of the splitter is highly dependent on accurate drilling. The splitter becomes more effective once a second free face is present. Future work should therefore concentrate on developing more effective techniques for creating an initial cut.

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Revisions to the High Education Qualifications Framework (HEQF) have altered how the tertiary education institutes can package their postgraduate qualifications. This has resulted in a gap between the needs of the South African extractive industry in terms of training and skills development and the manner in which tertiary education institutions are currently delivering and packaging their qualifications. Those tasked with ensuring that their employees are provided with the necessary training in order for them to progress through the company hierarchy are of the opinion that training and skills development are not easily accessible to them. It is therefore essential that the progression for Differently Qualified Applicants (DQA) also be addressed and catered for in any proposals regarding the development of qualifications. The opportunities around dual education and work-integrated models are also to be explored.

TOPICS:

The following topics will be covered:

- Progression of operators into Higher Education
- Recognition of non-formal skills and learning
- Progression of Diplomates
- Industry, experience, qualifications, progression from lower degrees to higher degrees, and addressing the qualification gaps
- Balancing industry needs and Higher Education Institution outputs:
  - Future career paths for graduates
  - The role of work-integrated learning/dual education in Higher Education.

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- MQA representatives
- Department of Higher Education and Training Representative
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KEYNOTE SPEAKER
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14–17 October 2018 — Furnace Tapping 2018 Conference
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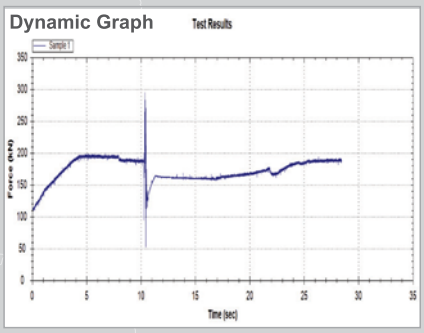

OMNI 127

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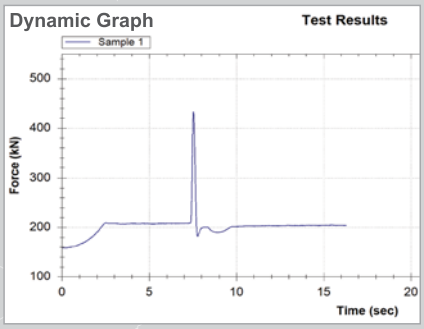
- 20 ton support resistance
- Blast on dynamic load bearing
- Covers stoping widths up to 4.5m



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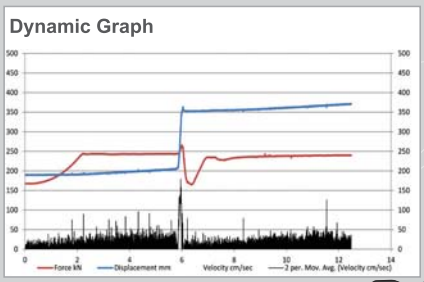

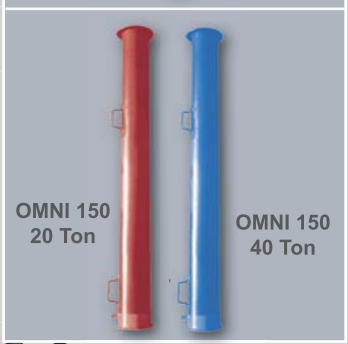

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