SAIMM
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Presidential Address
Ethics, Morals, and Leadership
by V.G. Duke

Introduction to our new President

Vaughn Glenn Duke

Vaughn was born in 1962 in Boksburg, where his father worked as a geologist on ERPM for Rand Mines. He grew up with two younger sisters and a brother. Geologists move around a lot and so, although Vaughn started his schooling at Christian Brothers College in Boksburg, he completed grade 7 at Princess Primary in Krugersdorp before attending Krugersdorp High to grade 11. Circumstances changed again and he ended up matriculating at Glenwood High School in Durban, where he made a wise choice to stop playing rugby and instead achieved a university exemption. Vaughn was accepted by the University of the Witwatersrand to sit for a BSc (Hons) Mining Engineering and graduated in 1986.

He then worked for COMRO for six months before moving to Zimbabwe to work for his father in a small business that researched, 'pegged', and packaged small mining prospects for lease and/or sale to larger companies. Geologists and mining engineers traditionally disagree, so Vaughn accepted a job with the Cementation Company (Zimbabwe) Ltd to work on a shaft sinking contract at Dalny Mine, which was Zimbabwe's deepest mine at the time. He was 25 years old and had to learn quickly because it was a 6 m diameter shaft that was being sunk without the help of a 'boesman' or a cactus grab. He did learn fast and was soon appointed to run the site as a Site Manager, after which he was transferred to a large civil tunnelling contract at Darwendale near Harare.

Vaughn's marriage to his university sweetheart Lynne in 1988 was followed by his return to South Africa in 1989, where he joined AngloVaal Pty Ltd to work at its Hartebeesfontein Gold Mine. He was moved to Loraine Gold Mines in 1992 to work as a Section Manager at the No 1 Shaft and for AngloVaal's new Target Project. He returned to Harties on promotion in 1995, and was transferred to head office in 1997. He worked as the Group Mining Engineer, General Manager: Technical Services, and Project Manager for the Northern Free State Expansion Project until AngloVaal was absorbed by African Rainbow Minerals in 2004. Vaughn also completed an MBA with distinction at the Gordon Institute of Business Science of the University of Pretoria during this period.

Vaughn established Sound Mining Solution (Pty) Ltd in 2004, and after working out of rented offices in Marshalltown, purchased a building in Rivonia in 2008. Sound Mining is now well established and Vaughn is privileged and blessed to be working with a fantastic group of colleagues who have chosen to remain with Sound Mining. They have been successfully working together for many years.

Vaughn has been associated with the Southern African Institution of Mining and Metallurgy since his university days. He is now a Fellow and has contributed to numerous committees and working groups. Vaughn is a recognised Project Management Professional and a registered professional with the Engineering Council of South Africa.

Vaughn has been married to Lynne for 32 years. Their eldest son Michael is studying chemistry at Wits, their daughter Stephanie is currently completing her degree at Lisof (London Institute of Fashion), and the youngest, Daniel, is in his second year at the University of Pretoria studying mechanical engineering. Lynne and Vaughn cherish and prioritize their time together and are often found cycling in the Cradle.

Presidential Address

Members of the SAIMM are facing a rapidly changing global environment and uncertain times at home.

The world is being increasingly challenged by climate change, shortages of arable land and potable water, sustainable energy needs, food security, and a huge growth in urban populations.

Closer to home, South Africa finds itself entrenched in a climate of continual political and economic uncertainty. South Africans are experiencing unemployment, corruption, civil unrest, xenophobia, polarization, increasing debt, and poverty. The reputations of numerous businesses, including Steinhoff, KPMG and McKinsey, have been compromised over allegations of corruption.

These dynamics, together with a growing digital economy, are changing the way we communicate, absorb information, and interact with one other. This in turn creates new challenges for the SAIMM, including increasing competition for our members' attention with respect to attendance at conferences, meetings, and working groups.

So, to remain relevant, the SAIMM needs to transform itself. A suitable strategy has been developed and the code of ethics updated to provide ethical guidance to members as they face today's challenges and issues. This update entailed a move away from a more rules-based code to a less prescriptive, values-based code of ethics that is more suited to today's world, and which emphasises the SAIMM's principles and values.

Effective leadership is clearly needed, now more than ever. The SAIMM can and should contribute to the enormous challenges facing South Africa, and this can be achieved by urging members to act ethically and lead effectively when needed.

In this address, on Ethics, Morals, and Leadership I will expand on the above, clarify the differences between ethics and morals, and discuss the essence of leadership and how an appreciation of the difference between ethics and morals can impact our performance as professionals and leaders.

Hopefully, these insights will contribute positively to members as they continue to influence and impact developments across Southern Africa by leading ethically, whenever and wherever required.

Vaughn and Lynne with their sons Michael and Daniel, and their daughter Stephanie

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PAPERS OF GENERAL INTEREST

As we find ourselves in a world dominated by COVID-19, it is clear that the mining industry is not exempted from the impact of the pandemic. The industry will change in ways we cannot yet quite appreciate. This comes on top of the other challenges that mining (and indeed the wider world) is faced with, such as climate change, uncertain trade relations, and the imperatives of sustainable development. The coal mining industry faces a further challenge in that the need to reduce carbon emissions will inevitably lead to a reduction in the use of coal, particularly for the generation of power. It is accepted that the transition to renewable energy will proceed, and that the use of coal will decline in the medium to long term. A transition is needed that will result in security and affordability of electricity supply, while at the same time allowing the industry and all its stakeholders to adjust to the disruptions that such a transition will cause. For this to happen, stakeholders need to find new and innovative ways to operate, and the input of the scientific community is a crucial part of this process. In this edition of the *Journal*, general papers are published. Some touch directly on the issues the coal industry faces; others more indirectly. All, in their own way, will assist in meeting the challenges the mining industry is faced with.

H. Lodewijks
Coaltech Research Association NPC

Press Release: Into a gold mine: How South Deep achieved its dramatic turnaround in 12 months

Like many other sectors in South Africa, the mining industry was severely impacted by the national lockdown. Minister of Mineral Resources and Energy Gwede Mantashe showed great understanding of the need to strike a balance between ensuring the health of employees and maintaining some degree of business continuity for the mining industry, which remains a significant contributor to South Africa's economy and a major direct employer. In line with the measures announced by the Minister on 25 March 2020, mining operations – particularly deep-level mining, which is generally considered labour intensive – were scaled down significantly.

Gold Fields' South Deep Mine, located on the West Rand, was placed on care and maintenance in April and in compliance with government regulations, operated well below its full labour complement for the remainder of the lockdown period. Despite this, South Deep continued to show progress on most of its operational measures during the first half of 2020 compared to the same period in 2019, largely due to an organizational culture and capability alignment process.

Prior to the outbreak of the COVID-19 pandemic, Gold Fields announced two consecutive cash-positive quarters at South Deep Mine at the end of 2019 and recorded a full year of cash flow positive results, while meeting production guidance.

Since acquiring the mine in 2006, Gold Fields has experienced a number of organizational challenges and setbacks, preventing it from operating South Deep as a modern, bulk, mechanized and profitable mine.

To address these challenges, Gold Fields embarked on a strategic transformation journey which included an organizational restructuring exercise, followed by a broader cultural and capability alignment process.

South Deep engaged Cape Town-headquartered business consultancy OIM Consulting to support the cultural and operating aspects of the process.

Run by a core management team with a wealth of experience across various sectors, including mining, retail, financial, and manufacturing, OIM Consulting services various blue-chip companies, specializing in enhancing organizational performance through operational optimization and people management and development.

OIM Consulting's four-pillared process is centred on what it considers to be the 'beating heart' of any organization – its frontline leaders. As Arjen de Bruin, OIM Consulting's Managing Director, explains: 'We've realized that the successful execution of any business plan relies on supervisory effectiveness, yet organizations typically do not place enough effort on building this capability and capacity, and changing frontline leader behaviour.

'Our process addresses cultural change, the identification and building of new capabilities, and performance assessment, management, and improvement, with a pivotal focus on the supervisor as key to sustaining this improvement.'

De Bruin notes the establishment of a 'coaching culture' as integral to the process. 'We maintain that 80% of our time needs to be spent on the shop floor, mentoring line managers so that they continue driving change upon our eventual exit. This is essential to entrench new skill-sets through ongoing reinforcement.'

The initial results at South Deep are extremely encouraging, with the mine reporting a profit at the end of the initial 12-month period. More revealing are the metrics that demonstrate significant operational improvement; the mine saw a 41% increase in gold production when comparing H1 2019 with H2 2019. Its overall productivity in 2019 improved by 30% to 26.7 tons per employee, costed from 20.5 tons per employee in 2018. The overall efficiencies for development and destress improved to 60 m per rig per month in 2019 from 39 m per rig per month in 2018, all contributing to turning the net loss made in 2018 to a net profit of over R104 million in 2019.

De Bruin points out, it is important to note that 'these results were achieved with approximately 30% less staff and equipment than the year before.'

Says Martin Preece, Executive Vice President at Gold Fields, 'We've seen a remarkable improvement in most production metrics during 2019, resulting from a culmination of initiatives centred around our people, including organizational culture, processes, systems, and technical improvements; a process supported by OIM.'

De Bruin attributes the integration of culture, capabilities, and practices as the foundation that underpins all operational improvements. 'In order to improve output and efficiencies, one needs to start with changing mindsets, through developing appropriate and relevant skill-sets and toolsets.

'As a world-renowned leader with operations that span three continents, we are very proud to partner with Gold Fields, and are honoured to have been granted the opportunity to support them in driving their organizational goals.'

EDITORIAL:
Into a goldmine: How South Deep achieved its dramatic turnaround in 12 months
OIM Consulting contributes to transforming South Deep Mine to cash-positive

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Johannesburg Branch

Daniel F. Jensen

Danie has more than 35 years' success driving comprehensive strategic planning, project governance, business analysis, tender/bid development, and general management for renowned, world-class organizations, all aimed at cultivating business and revenue growth and optimal performance. He has been involved in the mining industry specifically from 2003 onwards. While consulting to re-establish a diamond and a coal mine in eSwatini (Swaziland), he initiated the eSwatini 'Mining Rights' Project with a potential market capitalization in excess of US$1 billion. Danie was employed on a contract basis between 2007 and 2009, at RSV ENCO as Project Management Office Manager (PMO), where he was instrumental in the establishment of major engineering, procurement, commissioning, and management (EPCM) coal mining projects such as Thubelisha and Impumelelo at Sasol Secunda. He also spearheaded key efforts to implement an innovative documentation management systems (DMS) and a new PMO system (using PRISM Management Suite and ProjectWise software packages) to effectively support the delivery of cutting-edge EPCM mining projects with values of over R5 billion. He spent the period from 2010 to 2012 in the IT industry, where he was instrumental in compiling numerous tenders and initiating projects worth over R2 billion. He returned to his home ground of engineering project development and management in 2012, and played an integral role in assisting the companies he worked with in pioneering 93 project portfolio concept business plans, about 35% of which are mining-related, to be executed on a global scale with a value of over US$20 billion if implemented.

Danie has been an active member of the Johannesburg Branch of the SAIMM since 2007 and is currently proudly serving as Chairperson since the untimely passing of John Luckmann, stepping from his role as Branch Secretary to Acting Chairperson, and subsequently elected Chair. He holds a Master of Science degree in Project Management, certificates in Project Risk Management and Senior Management Programme (University of Pretoria), Senior Management and Staff Course (South African Air Force), and Bachelor of Technology, Engineering (Technical University of Tshwane). ◆

Namibian Branch

Nikowa Mabvuto Namate

Nikowa Namate is Deputy Head and lecturer at the Department of Mining and Process Engineering, Namibia University of Science and Technology (NUST). He holds a Batchelor's degree in Mining Engineering from the University of Zimbabwe and a Master's in Industrial Engineering (MIE) from NUST, and has over fifteen years of industrial and academic experience. His major research interests are in industrial engineering processes and ergonomics. Nikowa is also a certified energy auditor. He has an in-depth understanding of the mining industry in Namibia, and plays an active role in the development and diversification of the sector by providing consulting services to small-scale enterprises. ◆

SAIMM Branch Chairpersons

Northern Cape Branch

Itumeleng Lute

Itumeleng Lute began his mining career as a Learner Official at AMCOAL's New Denmark Colliery.

He obtained his National Diploma Coal Mining at the Technikon Witwatersrand (now the University of Johannesburg) in 1999 and joined Loxton Exploration as Junior Mining Engineer Projects and planning. During his three years with the company he was involved in the Bellsbank Diamond Mine-Speed Development Project and Feasibility Study.

In 2001 Itumeleng moved to Loxton Exploration, where he worked as a Shift Boss and acting Mine Overseer. He obtained his National Diploma in Mining Engineering in 2001, followed by a BTech Mining Engineering a year later, both from Technikon Witwatersrand.

From 2002 to 2004 Itumeleng worked for Rex Mining as Mining Engineer at their head office, where he participated in the mine design project for Rex Diamond Mine. He was appointed Mining Engineer: Projects in 2004, and obtained his Mine Overseer's Certificate the same year.

In 2004 Itumeleng left the corporate world to strike out on his own. He co-founded Botswerere Mining CC and served as the Chairman and Consulting Mining Engineer until April 2006, when he established Lute Mining CC and Lute Diamonds CC. He currently serves as Chairperson/Senior Mining Consultant and factory manager of these companies, respectively.

Itumeleng is very active in the Black Management Forum (BMF), serving as Provincial Secretary of the Northern Cape Branch in 2006/2007, Provincial Treasurer from 2007-2009, and Northern Cape Provincial Chairperson in 2009/2010 He was appointed as a member of the BMF National Board in 2009/2010.

Itumeleng is an Associate of the South African Institute of Mining and Metallurgy, a Member of the Mine Managers Association of South Africa, and a member of the Institute of Directors, NOCCI, and St Peter Church Finance Committee. He has been a Director of Northern Cape Economic Think Tanksince July 2020. ◆

Pretoria Branch

Sezer Uludag

Sezer Uludag holds a BSc (Mining Engineering) and an MSc (Mining Engineering) from the University of the Witwatersrand. She is currently a lecturer in the Department of Mining Engineering at the University of Pretoria, where she presents three undergraduate and two postgraduate modules on underground and surface mining methods. Her research interests include drilling and blasting, automation, application of systems thinking and system dynamics to mining-related problems, rock cutting, and spontaneous combustion. She is currently busy with a PhD study on the quantification of the impact of disruptive technologies in an open pit mining environment using system dynamics methods.

Sezer was previously as a consultant at Anglo American Technical Solutions, creating innovative value-adding solutions to drilling and blasting related issues, as well as working on support governance and supply chain within the Group on drilling and blasting quality and standards. Prior to that, she was a lecturer at the University of the Witwatersrand, where she presented a full-year course on Excavation Engineering, involving principles of rock cutting, drilling and tunnel boring machines; and drilling and blasting principles and design. She also lectured and coordinated a Graduate Diploma Engineering course on blasting technology. ◆

SAIMM Branch Chairpersons

Western Cape Branch

Alan Nesbitt

Dr Nesbitt holds an N.D. Chemical Engineering (Cape Technikon, 1985), an N.H.D. in Chemical Engineering (Vaal Triangle Technikon, 1989), an M. Tech. (Cape Technikon, 1996), and a PhD (UCT, 2016). Dr Nesbitt started his consultancy Tibsen Technologies in 2008 and has conducted research projects through direct contact with various mineral processing companies/organizations including, De Beers, BHP Billiton, Alcoa, Gekko Systems, Namdeb, and UCT. In addition to research he teaches water purification and hydrometallurgy to graduate staff of Anglo Platinum, Debswana, and other personnel on an ad hoc basis. Dr Nesbitt's main research interests are in resin ion exchange technology for the mineral processing industry and water purification.

Prior to Tibsen Technologies Dr Nesbitt was a lecturer at the Cape Technikon (1998–2009) and worked as a technical officer on the Chamber of Mines collaborative initiative in mineral processing education in the Western Cape while based at the University of Stellenbosch (1992–1997). Prior to 1992 Dr Nesbitt worked for a number of companies including South African Breweries, Caltex South Africa, and Mintek. ◆

Zambian Branch

Darius Muma

Darius Muma graduated from the University of Zambia (UNZA) in 2004 with a BSc Chemistry. He worked as an Environmental Project Assistant at UNZA School of Mines with the Advocacy for Environmental Restoration Zambia (AREZ), before joining Konkola Copper Mines' (KCM) Nchanga Mine in Chingola as a Graduate Chemist. He remained with KCM for 8½ years, progressing through the ranks to Senior Chemist, Sectional Chemist, and Acting Head of Analytical Services. During this time he gained a Diploma in Business Management (Association of Business Executives, UK). In February, 2014 Darius moved to Mopani Copper Mines' Mufulira Mine to take up his present position of Assistant Superintendent Technical.

While at Mufulira, he completed a Bachelor of Education in Environmental Education at UNZA, and also a MSc in Sustainable Mineral Resource Development under the Education for Sustainable Development in Africa (ESDA) joint programme of the University of Cape Town's Faculty of Engineering and Built Environment and the UNZA School of Mines. Darius is married to Priscilla Chola, a professional nurse and tutor at Nchanga School of midwifery and nursing in the Zambian Ministry of Health, and the couple have four children; Martin Kazandwe, Precious Mwaba, Gracious Kunda, and Joseph Kaindu.

Darius has a wide range of specialized skills and training in mineral sampling and assaying, including X-ray fluorescence, fire assay and related pyrometallugical analysis techniques, laboratory auditing, chemometrics and intelligent laboratory systems, metallurgical accounting, copper metallography sample preparation, laboratory information management systems, and project management. He is also a BSI Certified Auditor for BSI ISO 14001 and Lead Auditor for BSI ISO 9001. He is a Member of the SAIMM (currently Chairperson of the Zambian Branch), the Royal Society of Chemistry (RSC), the Chemical Society of Zambia (CSZ) (currently Northern Region Vice President), the Engineering Institution of Zambia (EIZ), and a member of ABE, UK. ◆

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Zimbabwean Branch

Clara Petronella Sadomba

Clara Sadomba graduated from the University of Zimbabwe in 1990 with a Bachelor of Technology (Applied Chemistry & Chemical Technology), and went on to complete a Master's degree in Metallurgical Engineering at McGill University, Montreal, Canada in 1996. After a brief spell in the chemicals and plastics industries, she joined Zimbabwean integrated ferrochrome producer Zimasco (Pvt) Ltd under their graduate learnership programme and has been with the company since then, progressing through the operational, middle, and executive managerial levels. She is currently General Manager - Marketing and Administration, with key responsibilities that include marketing strategy and planning; market analysis; marketing, sales, and logistics process optimization; corporate communication; and administration of head office operations.

Clara has extensive international business experience with cross-sector and cross-cultural exposure. She is registered as a Professional Engineer with the Engineering Council of Zimbabwe and a member of the Zimbabwe Institute of Engineers. She is also Chairperson of the Zimasco Pension Fund.

Clara is single, with one son aged nineteen. ◆

Zululand Branch

Christo Mienie

Christo Mienie obtained his Dip Tech (T5) in Metallurgical Engineering from the Vaal Triangle Technikon. He began his career with Iscor (Mittal Steel) in their research and development department, where his area of focus was iron manufacturing, with particular interest in the evaluation and characterization of iron ore, coal, and coke for the blast furnace, direct reduction, and Corex processes.

In 1992 he joined Richards Bay Minerals (RBM), and gained valuable metallurgical and production experience over the next 14 years at various RBM plants including the roaster, char plant, iron processing, slag processing, and the smelter. His responsibilities included increasing the production of prime grade slag and pig iron, furnace rebuilds, establishing new markets and customers in conjunction with the marketing department, and the introduction of best practices in process metallurgy. His HR responsibilities included the management, training, and development of metallurgists, and he obtained valuable experience with regard to business restructuring ('right-sizing') while the team leader responsible for the SHEQ and technical departments. He completed his Management Development Programme through the University of Durban Westville during 1997.

Christo left the corporate world when he joined Spectrum Technical (Pty) Ltd in 2006 as a director and shareholder – his current position. Services to clients includes plant feasibility studies, flow sheet design, process equipment selection and supply, pilot-scale test work, plant commissioning, troubleshooting, and cost analyses in the metals and minerals industry.

Christo is married to Marietjie, and they have two boys – Francois and Christo Junior. He is a keen jogger. ◆

Presidential Address: Leadership, Morals, and Ethics

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Synopsis

South Africans have experienced material change since the 1980s. The country saw the end of apartheid and the beginning of a new era under the leadership of a man who raised expectations of a Rainbow Nation and an equal society. Unfortunately, this has not since transpired. There has been a dearth of effective leadership, with daily reports of unethical behaviour in the media. Productivity is low, the economy has deteriorated, unemployment is higher than ever, and South Africans are finding themselves becoming more polarized as a society.

At the same time the South African mining industry, which has the potential to be the catalyst for national economic growth, has shrunk significantly. This is partially due to both an evolving and increasingly turbulent global environment, and a result of international divestment as the legislative and socio-economic environment becomes increasingly difficult.

This industry traditionally supported the activities of the Southern African Institute of Mining and Metallurgy (SAIMM), but this has also changed. The SAIMM is now finding it difficult to retain members and as a consequence is in the process of repositioning itself around six pillars of value to arrest this decline. One of these pillars is about enhancing ethical awareness, and by extension, establishing a climate where ethical leadership can thrive, because this is now urgently needed in South Africa.

This address briefly describes the challenges facing South Africa, the mining industry and the SAIMM, and how the organization is responding to remain relevant in today's rapidly changing world. Morals, ethics, and leadership are explained in support of a call for higher levels of ethical awareness, and more ethical leadership, in all areas of South African society.

As professionals, SAIMM members are already committed to ethical behaviour, and accordingly, are well placed to assist in making this happen.

Keywords
ethical awareness and leadership.

The SAIMM in turbulent times

The Southern African Institute of Mining and Metallurgy (SAIMM) has a proud history that extends from its inception in 1894 following the discovery of gold in Johannesburg in 1886, through to today, spanning a period of over 125 years. The objectives of this non-profit organization are:

- To identify the needs of its members
- To ensure that their requirements for knowledge are satisfied
- To represent and promote the interests of its members.

The SAIMM has been achieving these objectives by

- Providing mentorship and promoting the professional development of professional members and continuing education of other members, including student members
- Working together with other professional associations when engaging with government and mineral industry forums where appropriate
- Representing the interests of members on statutory bodies, including the registration of professionals linked to the international reporting codes and the Engineering Council of South Africa (ECSA)
- Facilitating networking and the exchange of technical information *via* SAIMM-hosted conferences
- Providing free access to an internal technical library and to an external library containing papers from around the world (*via* OneMine.org)

- Assisting needy students with tuition, meals, supplies, travel, and accommodation through a Scholarship Trust Fund established for the purpose;
- Publishing a globally recognized and respected monthly technical Journal
- Fostering international contact and cooperation between similar institutes for professional recognition by peers internationally.

However, despite these and other initiatives, the SAIMM is shrinking. Figure 1 shows that membership numbers are dwindling, a sign that members are becoming less interested in the activities and offerings of the Institute.

Further analysis (Figure 2) reveals that it is primarily the male members from the Associate and Member categories that are leaving, and that the SAIMM is failing to attract more female members.

Reduced membership means fewer volunteers to participate in the Technical Programme Committees, which are fundamental to the organization's fundraising efforts and to its commitment to disburse information and knowledge to members. The SAIMM is also experiencing reduced attendance at conferences, which translates to lower levels of income. Figure 3 shows that the SAIMM has been spending more money than it has managed to raise over the past seven years, and approximately R18.5 million has been withdrawn from savings to cover the deficit.

This is clearly not sustainable and the SAIMM has accepted that it cannot continue to function as usual, or to do the same things that it has always done and expect different results.

Numerous megatrends (Figure 4) have been simultaneously impacting how people interact, communicate, operate, or simply exist, with one another, and these global dynamics are responsible for a world that is now changing faster every day.

The mining industry is not immune from the effects. It has also been steadily and continuously changing in order to remain globally competitive, and the SAIMM has had to start innovatively adjusting and adapting to this new landscape, because of its close links to this industry. A series of workshops was conducted in 2019 to examine what SAIMM needed to do to:

- Ensure that it remain current and relevant to its members
- Embrace modernization of the mining industry
- Support and enhance the professional needs and development of its members
- Extend its reach and influence
- Enhance networks and increase membership.

These workshops culminated in a new vision statement with general alignment on the following six 'Pillars of Value' upon which the strategy to revitalize the SAIMM is now based:

- Professional development
- Emerging professionals
- Networking and strategic relationships
- Diversity and inclusion
- Conferences and events
- Improving ethical awareness.

Vision

To be an independent and globally recognized platform for the development of the African minerals industry of the future.

(A.S. Macfarlane and S. Ndlovu with the support of Office Bearers of the SAIMM, 2018)

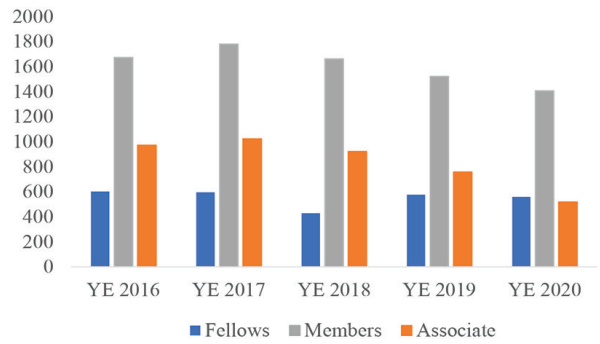


Figure 1—Gradual reduction in fee-paying members (i.e. excluding students)

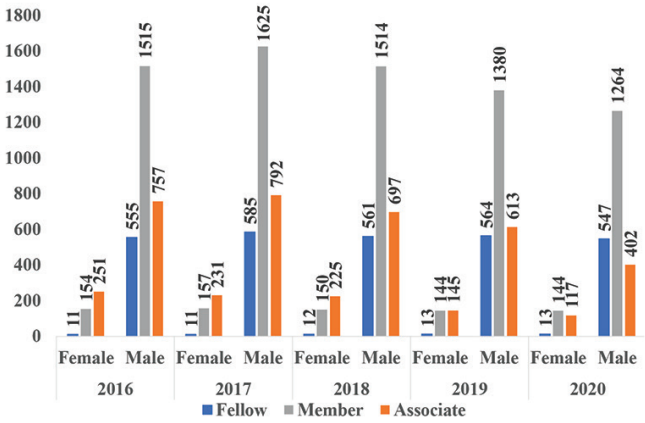


Figure 2—SAIMM membership: gender mix

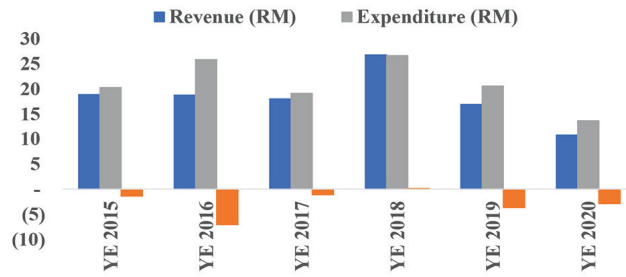


Figure 3—SAIMM losses since 2015

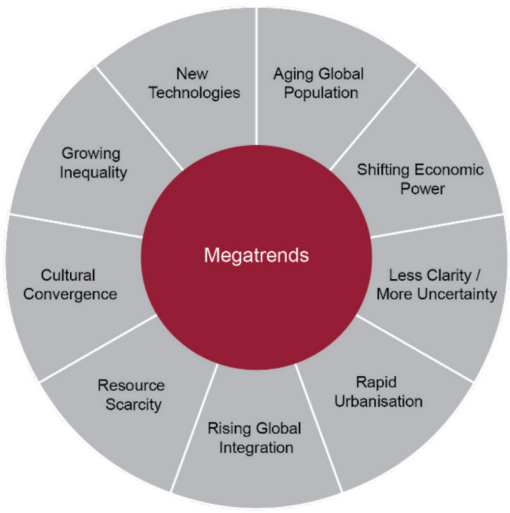


Figure 4—Global dynamics (megatrends)

Various Council members volunteered to take responsibility for entrenching these Pillars of Value, and to ensure deliberate and sustained progress in delivering such value to members. Value proposition have been clearly articulated for each Pillar of Value, and these are now being professionally tailored to enable SAIMM representatives (the volunteers) to effectively engage with members and all other stakeholders to communicate the organization's strategic intent, and to secure their support and increased involvement with the SAIMM. Covid-19 has forced the SAIMM to accelerate its efforts on the necessary marketing material and on the roadmap for implementing the strategy.

Improving ethical awareness

Improving ethical awareness is recognized as a key means of ensuring that members, and indeed all of the SAIMM's stakeholders, both see and experience the value of being associated with the organization. The value proposition for this 'Pillar of Value' also makes specific reference to promoting ethical leadership within the SAIMM.

Sustained ethical conduct by members will ensure that they can work, and live, in an environment where they are inspired by one another and respected by others, including the broader South African society. This will be possible if each and every member actively promotes ethical awareness and ethical behaviour, by making ethics real and visible to all stakeholders, both internally and externally.

To this end, work started on revisiting the SAIMM's Code of Ethics, which, although sufficient to address past ethical concerns, did not necessarily provide sufficient guidance regarding the SAIMM's values and principles in the context of today's evolving and increasingly complex world. The Code has, as a consequence, now been modified and reads that members must:

- Have due regard for the welfare, safety, and health of the general public (including employees and fellow professionals) and the environment in all activities
- Discharge their duties to employers and/or clients effectively and competently, with fidelity and honesty as well as respect their employer's confidentiality
- Uphold the dignity and standing of their respective professions and the objectives of the Institute
- Only undertake work that they have been adequately trained for, have the necessary experience of, and are therefore competent to perform
- Strictly avoid advertising their professional services in a self-laudatory way or in a manner that would undermine the dignity of the Institute
- Refrain from malicious or negligent conduct that would injure the professional reputation or business of others
- Continue their professional development throughout their careers
- Comply with the Constitution of the Institute and, where applicable, those of legislated Codes of Professional Conduct.

The revised Code of Ethics is essentially a short-list of specific principles and value statements to guide members' behaviour and their decisions, both at work and in the broader South African community. The underlying intention was that it should avoid being specific so as to prevent inadvertent limitation of the application of the ethical and behavioural principles contained therein. In other words, the revised Code of Ethics now has more of a value-based leaning than that of a rules-based code of ethics, and when interpreting it, both the ethical and behavioural principles must be applied to advance and promote the spirit and the letter of the SAIMM's Code of Ethics.

The SAIMM's Code of Ethics also provides for the suspension or expulsion of members, but this does not prevent the organization from negative exposure as a consequence of unethical behaviour, and members may need to proactively act to prevent possible violations, which can both damage the SAIMM brand and lower the levels of trust and cohesion within the Institute.

Values-based codes of ethics

This type of code emphasises the principles and values of a common ethical culture. It is intended to be less prescriptive and relies on continuous communication between people for ongoing alignment in an ever-changing world.

Rules-based code of ethics

This is more of a prescriptive and controlling approach that defines specific rules about what is acceptable and unacceptable.

(D. Rossouw and L. van Vuuren, 2020)

Ethical and moral decay

In addition to having to cope with a rapidly evolving global environment, South Africans have also had to deal with the consequences of money laundering, bribery, fraud, and corruption, often involving high-profile companies and prominent political or business leaders.

Numerous businesses in the private sector have had their reputations damaged by allegations of corruption, or corporate collusion, including:

- Enron, 2001. The media revealed that the energy company's accounting practices were amiss, resulting in bankruptcy. The share price dropped from a high of US$90.75 in June 2000 to less than US$1 by the end of November 2001. Many of the executives and employees were charged with fraud or conspiracy
- KPMG, 2016. South Africa's Companies and Intellectual Property Commission (CIPC) accused the auditing firm of sub-standard quality controls while reporting on the South African Revenue Service. KPMG was also accused of helping the Gupta family write off a wedding as a business expense
- Steinhoff, 2017. South African pension funds incurred huge losses due to 'accounting irregularities' that resulted in the furniture company's share price dropping by 85%. The auditors responsible for the 2014, 2015, and 2016 financial statements are being investigated by the Independent Regulatory Board for Auditors (IRBA)
- CIPC accused McKinsey & Company of contravening the Companies Act by misleading Eskom in dealings with the Gupta-aligned Trillian Capital Partners
- PwC was cited in an IRBA investigation for not disclosing irregularities in the procurement practices of South African Airways.

These types of governance failures indicate a lack of ethics and ethical leadership. It appears that most of the people encountering unethical behaviour are doing little about it.

Unfortunately, the stream of regular reports on fraud and corruption (Figure 5) keeps chipping away at the resolve, and efforts of those few people and business actually trying to make a meaningful difference.

Ethics in business and politics has fortunately been receiving increased attention because of the many large corporate failures (Rhodes, 2006), and today most of the bigger companies are incorporating ethics and codes of conduct into their employee development programmes. However, reports of fraud and corruption do not seem to have slowed, despite these efforts. Studies have suggested that the 'internal bias' or 'situational influences' of individuals may be where attention needs to be directed.

Enron culture was about increasing the short-term share price without concern for the longer-term consequences, and everyone turned a 'blind eye'. This is an example of 'situational influence', where people are less likely to intervene when others are around to do so. Individuals need to learn to recognize the threat of 'internal bias' or 'situational influences' when facing a situation where their personal values seem to be in conflict with corporate culture. This is important.

Internal Bias

Some people believe that their ethical standards are higher than the average and as a consequence are more likely to overestimate the morality of their own behaviours, particularly in situations that they may not have encountered before.
(Ross and Nisbett, 1991)

Situational Influences

Bonuses, promotions, prestige, loyalty to an employer or even cultural aspects, can impact a person's behaviour or decision making, where even people with honourable motives can be influenced to do unethical things when finding themselves in difficult situations.
(Milgram, 1974)

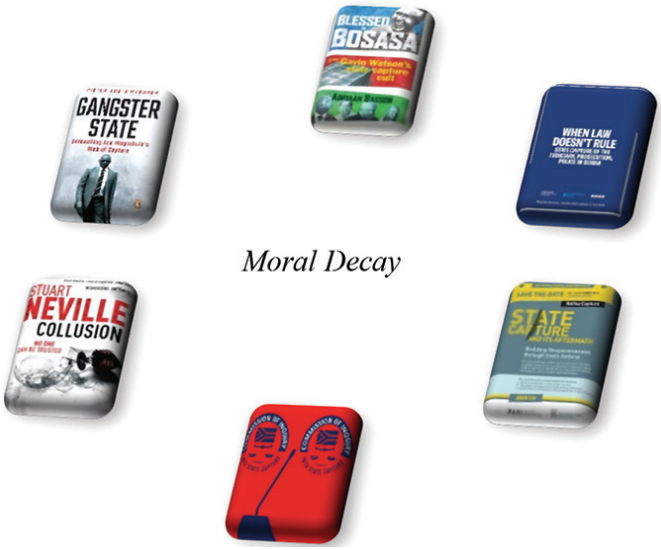


Figure 5—Fraud and corruption

Repeated reports of moral and ethical transgressions contribute to increasing the incidence of unethical behaviour across all levels of South African society. They also negatively influence investor confidence due to growing perceptions of the state of corruption in South Africa. In fact, South Africa has been placed squarely among countries deemed to have a serious corruption problem. Many investors, intent on avoiding political infighting, legislative uncertainty, skills shortages, infrastructural bottlenecks, and increasing taxes are looking elsewhere to grow their wealth.

Transparency International, a global coalition against corruption, publishes an annual Corruption Perceptions Index (CPI). The 2019 survey used feedback from thirteen executives on their perceived levels of public sector corruption, to rank 180 countries on a scale of zero (highly corrupt) to 100 (fairly clean). The 2019 CPI (Figure 6) shows that South Africa has slipped to the 70th position and now ranks lower than eight other sub-Saharan African countries.

Business for South Africa (B4SA), which comprises the Black Business Council and Business Unity South Africa, has been developing plans for economic reform. These plans include the mining industry, which should be growing and not contracting. The sector contributed about 8% to South Africa's GDP and about a third of our exports last year. It also employed some 450 000 people directly, and when the multiplier effect is taken into account, is responsible for many more jobs.

Mining is a significant industry and could play a materially larger role in economic reform if it were able to attract the necessary investment to explore more and to address infrastructural bottlenecks. Unfortunately, this may not be possible in the light of current investor perceptions.

There are of course numerous other problems that have emerged as a result of unethical behaviour across South Africa. Unemployment is increasing, while at the same time the country is having to deal with a dwindling skills base. Covid-19 has exacerbated the current situation. It is unlikely that people and businesses will continue to work and engage in the same way after the pandemic. South Africans are generally concerned with what the future has in store for them, and one consequence of this is that they are becoming increasingly polarized, with higher levels of prejudice evident everywhere.

Reversing the trend of moral decay

South Africans are now facing serious political and economic uncertainty and leadership is sorely needed. People at all levels in our society need to consider taking the time to influence those around them to make better choices and to do what is right, and the chances of this actually happening would be materially better if there were higher levels of ethical awareness prevalent amongst the general public in South Africa.

SAIMM members are well positioned to contribute and make a difference in this regard. They are already obligated by their profession to maintain the highest ethical standards, and are 'part and parcel' of South Africa and its mining industry. Each and every member should be mindful of this, and consider the implications of their choices and actions, or lack of actions, on society, and perhaps ask even themselves:

- If it is enough to simply follow a list of ethical guidelines
- If they care enough and are courageous enough to try and influence a better outcome

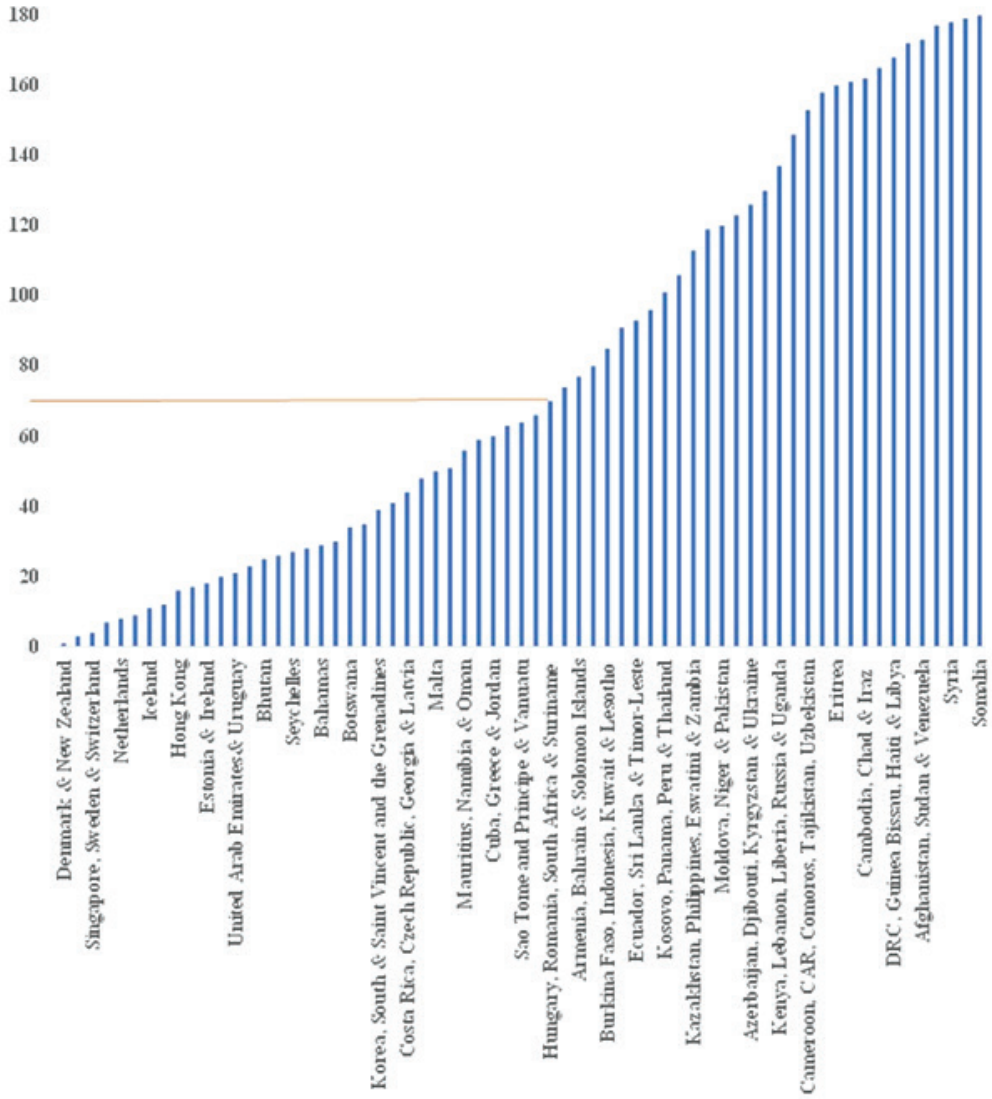


Figure 6—South Africa's position in the 2019 Corruption Perceptions Index (CPI)

- If they would be comfortable if they failed to act when knowing that they could and should have
- If they are sufficiently attuned to what is 'right and wrong' from a moral point of view
- If they are prepared to stand up and speak up, when they see something wrong?

It is proposed that a Code of Ethics, although necessary, is simply not enough. While useful as a general guide for how people should act, codes don't seem to be sufficient to ensure ethical conduct. There simply needs to be significantly more awareness around the importance of morals and ethics, in addition to following the abundance of ethical rules already available.

Members should not simply adopt a 'checklist' mentality when making ethical decisions. There are clearly people who are capable of easily circumventing rules for personal gain. Complacency around the importance of personal morals or ethical principles will not help in reversing this trend of moral decay. By talking and living ethics in the workplace and elsewhere, SAIMM members can contribute to building trust and developing partnerships, which will encourage the broader community to start working collaboratively for a better future. This type of behaviour will also contribute to enhanced governance, not only in the mining industry, but also among South Africans in general.

There will, however, be situations where the rules are unclear, nonexistent, not applicable, or even in conflict with one another. When a member encounters someone making, or about to make, a poor choice, he or she may feel obligated to act, but unsure of what to do. In such instances, members are encouraged to contact SAIMM's office manager for assistance, which could be through:

- *Peer advice*—appropriate contact can be facilitated with either a SAIMM Fellow or other suitable mentor.
- *A hotline*—where a question or a concern can be raised through an internal link, hosted by an external service provider, to keep information confidential to the extent permitted by law.
- A frequently asked questions link to a list of answers to the more common questions by members uncertain of the usual procedures or processes relating to the issue at hand.

Note: These links have been contemplated to serve solely as a means for members to obtain guidance as opposed to ignoring a situation. The intention is not to 'police' but to try and proactively prevent violations before they occur.

The ability to care and the courage to actually take a stand and 'buck the trend' are traits of a leader. However, many leaders acquire influence through the authority and power that comes with their position in an organization. This places them in a space where they can make a real difference, but many do not. Some of them actually hold the view that they need do only as much as what is contractually or legally required and no more. This type of thinking, or mindset, is not consistent with that of leaders who are concerned about the needs of their followers.

Effective leaders understand the importance of morality in today's mix and are capable of influencing the right ethical behaviours while also accommodating moral differences. They have an appreciation of what it takes to lead and understand the difference between morals and ethics.

> *The best way to protect the ethical culture of an organisation is to actively promote it, practice it, train in it, update it, and make it real and visible to external and internal stakeholders.*
> (J.D. Sullivan, 2008)

Morals *versus* ethics

Cognisance needs to be taken of the difference between morals and ethics (Figure 7) to effectively and discerningly weigh up right from wrong.

These words both relate to 'right' or 'wrong' and are often used interchangeably, but they actually mean different things (Table I).

Morals are principles upon which our judgements regarding right and wrong are based. They are mostly shaped by our social, cultural, or religious beliefs. While unlikely to be the same for people from different social, cultural, or religious environments, they don't usually change when a person operates in a different context. A decision on right or wrong from a moral perspective may not always appear rational. We need to remember that when people act according to their morals, they are doing what they personally believe to be right. For example:

- Eating meat may be immoral in some cultures, but perfectly acceptable in others
- Public displays of affection may be a problem in some places, but not in others
- Certain attire in one place of worship may not be acceptable in another, where different views of morality exist.

While our personal notions of right and wrong are often inherited and accepted without question, we still need to understand that diversity is here to stay and that we all should to try to understand one another's views.

Ethics, on the other hand, are more practical than morals. They are a set of values and principles of conduct that tell us how to behave or act in a social system, like a workplace, organization, or profession. They are consistent in the same context, but can vary depending on the social system to which they apply. Ethics are also not always moral and conflict can occur. For example:

- A doctor may agree with views on euthanasia, but cannot act due to his code of ethics
- A person chooses not to steal because he could go to jail, not because he actually believes that stealing is wrong
- A lawyer must defend his client despite knowing that the client is guilty.

We usually want to act ethically, even if the 'rules' do not agree with our personal views, because we don't want our peers to judge us, and so there will be times when we will need to compromise on our personal morals. Ethics can therefore be considered as a rational way of choosing between competing morals or when there is not a clear right or wrong answer.

Globalization has added a new dimension to ethics. There are more frequent conflicts between cultures, values, and beliefs as a result of the changes around communication and deregulation. However, while there are differences, these are not significant. Western views on ethics around motivation, character, and integrity, which are inherited from the likes of Aristotle, Plato, Socrates, and Niccolò Machiavelli, are not dissimilar to those based on the philosophical traditions of Confucius and Sun Tzu from the East (Resick *et al.*, 2006).

Ethical decision-making requires leaders to consider a decision from different perspectives so that they can clearly see the bigger issues and make decisions that also minimize unintended consequences. This type of awareness should traverse every level of an organization.

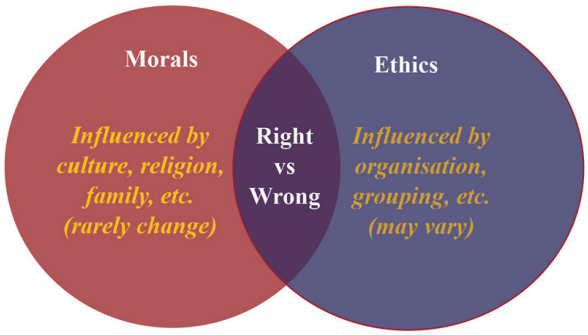


Figure 7—Morals *versus* ethics

Table I

Morals *versus* ethics

	Morals	Ethics
Definition	Latin word '*mos*' (custom)	Greek word '*ethos*' (character).
	Internal personal principles regarding right and wrong.	External guiding principles of conduct for a group or society.
Description	Unrelated to professional work.	Can relate to professional work
	Linked to personal values and influenced by society, culture, or religion.	For group alignment and influenced by profession, organization, *etc.*
	Generally unwavering as long as beliefs remain unchanged.	Can vary from one group or place to another.
	A function of culture or religion and can go beyond societal norms.	Reasonably uniform within a given context.

Although ethics is very subjective to an organization, the practice of good ethics in business leads to improved productivity and good morale.

> *Ethical leadership is leadership that is directed by a respect for ethical beliefs and values and for the dignity and rights of others.*
> (T. Watts, 2008)

Ethical leadership

There are several leadership labels in the literature, including:

- Courageous leadership
- Situational leadership
- Decisive leadership
- Servant leadership, and more recently
- Ethical leadership.

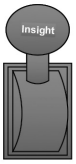
Much has been written about leadership and there are numerous descriptions of the various types of attributes that academia ascribes to leaders who are effective. For example, these types of leaders:

- Make the hard decisions
- Are courageous, have integrity, and lead by example
- Are mindful of the impact of their views on others
- Create a sense of community and team spirit
- Are caring, fair, honest, and respectful
- Embrace diversity and develop social cohesion
- Connect with their employees
- Do what they truly believe to be right
- Do not tolerate ethical violations.

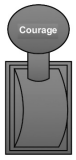
A question often asked is whether people need first to be appointed, or otherwise provided with authority, before trying to lead. Everyone, however quiet or reserved, has the potential to lead, but only if they care enough about the people around them. They would need the courage to act if an opportunity, or a threat, materializes which could impact the people around them. Leadership is tied to purpose and has nothing to do with position.

Leaders exist in the moment, and it is only that very moment when leadership is actually required. There are three simple levers that people can pull to successfully lead in the moment.

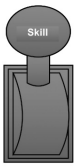
(1) **Insight**—The first lever relates to knowing something that the people around you don't. It may be something that could cause them harm or an opportunity for them to choose an alternative course of action for their benefit.



(2) **Courage**—The second lever relates to having the courage to act, because you effectively run the risk of losing the support of the people you are trying to lead. This assumes that you actually care for the people that you are with.



(3) **Skill**—The third lever is about having the ability to change the minds of those people who do not know what you know, at a rate that they can absorb. If you don't get this right you are likely to fail in your efforts to influence their decisions or actions. Many people do fail but at least they had the conviction and courage to stand up when it mattered.



Another consequence of the prevailing global dynamics is the greater attention now paid by leaders to ethics with a moral dimension. More particularly, a respect for moral differences is an essential part of ethical leadership, where decisions and actions are guided primarily by principles of respect, fairness, honesty, integrity, and accountability. These leaders, *inter alia*:

- Serve others and not strictly narrow individual interests
- Create spaces where people are more tolerant, understanding, and respectful of each other
- Seek to increase ethical awareness as this encourages people to self-regulate
- Are more able to embrace the levels of diversity that have emerged with globalization
- Are concerned with moral development, virtuous behaviour, and will usually contribute to developing a more cohesive society.

Ethical leaders are generally in touch with their own values, but are also capable of considering the morals of others when leading. They are not afraid to do what they truly believe to be right, even when it is unpopular or inconvenient. They have the courage to act for the common good of their followers, in the moment, when it counts.

Conclusion

South Africans are in desperate need of leadership, and SAIMM members can contribute in this regard. South Africa exists in a rapidly changing global environment and its citizens are facing uncertain times at home. The world is being increasingly challenged by climate change, shortages of arable land and potable water, sustainable energy needs, food security, and a huge growth in urban populations. Closer to home, South Africa's GDP is dropping dramatically. South Africans are experiencing unemployment, corruption, civil unrest, xenophobia, and polarization, in a climate of continual political and economic uncertainty. This has occurred as a result of many years of endemic corruption and a dearth of effective leadership.

Despite all of this, South Africa's constitution is standing firm and positive progress is being made against corruption. It is time for South Africans to start standing together to contribute to an accelerated economic recovery in the national interest, and effective leadership is required to boost the performance of South Africans by elevating accountability at all levels in our society.

It is specifically ethical leaders, capable of caring for the interests and aspirations of people around them, while remaining true to their personal sense of what is right and what is wrong, that are sorely needed. These leaders will enhance the prevailing levels of trust and respect in their communities and this will, in turn, lead to the standards of integrity and loyalty that South Africa requires of its citizens.

The SAIMM is already adapting to change to make sure that it is able to effectively serve both its members and a mining industry that needs to keep pace with the best in the world. The Institute is committed to increasing ethical awareness amongst its members, but remains reliant on the collective effort and support of all of its members to lead by example and, whenever possible, to influence their peers to make the right choices.

Developing and maintaining this type of culture of integrity within the SAIMM may arguably be the single most important thing that its professionals can do to promote the real value of the SAIMM, and the benefits that it and its members can bring to the mining industry and to South Africa. Indeed, as professionals,

they are already well placed to actively contribute by leading ethically to make a difference and promote ethical awareness and the importance of ethical conduct. They should do this whenever and wherever the opportunity arises.

> *Everyone is responsible for creating ethical leadership*
> (P. Kariuki, 2019)

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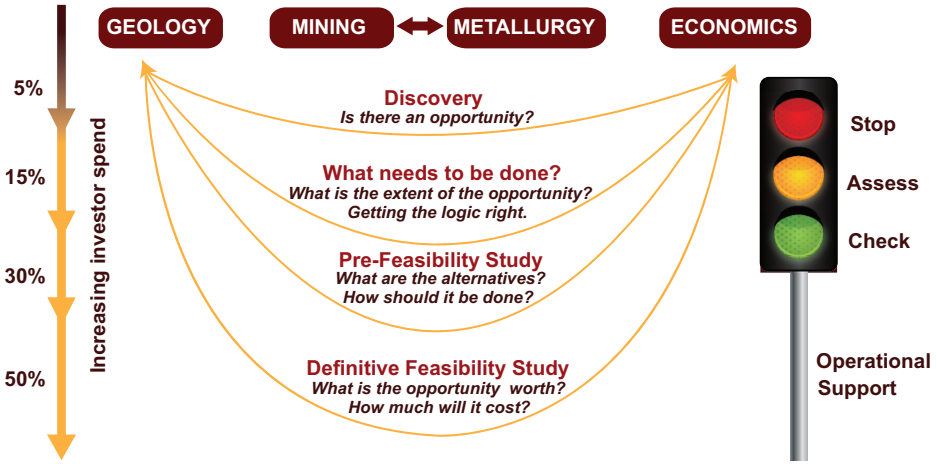

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RPEEE (Reasonable Prospects for Eventual Economic Extraction): The critical core to the SAMREC Code

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Synopsis

Reasonable Prospects for Eventual Economic Extraction (RPEEE) is the critical basis for effective implementation of the SAMREC Code as it applies to Mineral Resources, and implicitly to Mineral Reserves that are derived from Mineral Resources. This implementation follows the founding principles of materiality, transparency, and competence.

The SAMREC definition of a Mineral Resource specifically includes RPEEE. SAMREC provides guidance for all minerals on what RPEEE means, with some specific additional guidance for diamonds. SAMREC instructs the Competent Person that RPEEE must be demonstrated through reasoned assessment of multiple technical and non-technical parameters, and with disclosure in reporting. A recommended chapter in a CPR includes a sub-section for RPEEE.

SAMREC definitions and guidance are presented with discussion of the factors that must be assessed to demonstrate RPEEE. Comparison with guidance in other reporting countries and examples of actual practice are presented.

Keywords
reporting codes, Mineral Resources, definitions, disclosure, SAMREC.

Introduction

Reasonable Prospects for Eventual Economic Extraction (RPEEE) is perhaps the decisive basis for effective implementation of the SAMREC Code (SAMREC, 2016a). It is the critical core of the work processes leading to the declaration of Mineral Resources and Mineral Reserves, and is mandatory.

Notwithstanding the importance of the assessments required to demonstrate RPEEE, it is a step in the estimation process that in the past has been overlooked by default. This is perhaps because mathematical models play such an important role in our work practices today. Also, there is too often an implicit assessment of economic viability without provision of the supporting assessment that the layperson or investment adviser needs to bolster confidence in their judgement of the public disclosure.

While real-world work practices often fall well short of good practice in this regard, the SAMREC Code sets out clear definitions and provides guidance for the assessment of RPEEE that, if followed as required, results in clear and confident disclosure for the intended readers. The SAMREC Code also now introduces the concept of 'if not, why not' in Clause 6 with respect to the provisions of Table I. These aspects at the core of the code are described and discussed, together with comparison from other national codes and examples from published reports.

RPEEE in the SAMREC Code

What does the acronym mean? While it is easy to state that RPEEE means Reasonable Prospects for Eventual Economic Extraction, as a starting point in the discussion here, it will be helpful to break up the acronym, word by word, for clear and concise understanding.

R: Reasonable – using good judgment
P: Prospects (for) – possibility (not proven) of meeting with success in the future
E: Eventual – happening or existing at a later time
E: Economic – making a profit, or ability to attract investment
E: Extraction – excavation and extraction of minerals (as a practical matter) such as those containing metals or diamonds.

The SAIMM has adopted the Oxford English Dictionary as a standard for spelling; the implication must be that this dictionary also provides a preferred meaning for words. While the SAMREC Code makes no reference to a specific dictionary, the reader is advised that meaning may be of great importance, especially the meaning of the word 'reasonable' that appears multiple times in the code.

'*Reasonable'* means: 'Having sound judgement; fair and sensible; based on good sense.' Put another way: 'Able to reason logically.' All these synonyms suggest or imply a process, and process is a core aspect of the SAMREC Code. The process need not be prescriptive but allows our vastly disparate work practices in the mining industry to strive for a common purpose of conveying public disclosure that is comprehensible to the 'investors or potential investors and their advisers.'

'*Prospects'* means: 'The possibility or likelihood of some future event occurring.' It can also mean: 'Chances or opportunities for success or wealth.' In either case the meaning falls short of something proven. But, fundamentally for the mining industry, it also means: 'A place likely to yield mineral deposits.'

'*Eventual'* means: 'Occurring or existing at the end of or as a result of a process or period of time.' The crux of the meaning for our industry is the word 'time.' The SAMREC Code provides discussion in the guidelines, in relation to the timeline for development, which is often very different for bulk commodities compared to other commodities, like gold, for which significantly shorter timelines may be expected.

'*Economic*' means: 'Relating to economics or the economy' or 'Justified in terms of profitability.' SAMREC is very clear that 'A Mineral Resource is not an inventory of all Mineralisation …' Reasonable assessment of the information is required to move a Mineral Deposit (as defined in the SAMREC Glossary) to a Mineral Resource, with economics being at the centre of that assessment. Economic in this context, in the author's opinion, should not mean just one dollar over a threshold, but extends to the realm of ability to attract investment.

'*Extraction*' means: 'The action of extracting something, especially using effort or force', with mineral extraction provided as example. In our industry the something being extracted is minerals, and extraction is the holistic process from the source (rock) to the market. Nevertheless, there are inevitably different methods of extraction for different minerals, each of which may impact comparative economic viability.

The SAMREC Guidelines provide pertinent discussion, specifically following the definition of a Mineral Resource. This will be elaborated further below.

Definitions

What does SAMREC tell us? Clauses 24 and 67 provide the definitions for the general case and for diamonds.

A 'Mineral Resource' is a 'concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade, or quality and quantity that there are reasonable prospects for eventual economic extraction.' This definition excludes brines containing soda ash as in Botswana, or lithium as in Argentina, but these types of deposit were encompassed in SAMREC 2007 when the single word 'solid' was omitted. It is possible liquids may be included again in the future.

A 'Diamond Resource' is a concentration or occurrence of diamonds of economic interest in or on the Earth's crust in such form, quantity (volume/tonnage), grade and value that there are **reasonable prospects for eventual economic extraction**."

These clauses continue, 'Any (Diamond) Mineralisation that does not have demonstrated r**easonable prospects for eventual economic extraction** may not be included in a Mineral (Diamond) Resource.'

Guidelines or guidance

Guidelines, or guidance, as discussed here, is as defined in the Introduction to the SAMREC Code (Clause 2). Clause 24 continues after the definition of a Mineral Resource with several paragraphs of Guidelines text to discuss context and meaning that support and explain the very clear-cut statement that 'A Mineral Resource is not an inventory of all Mineralisation.'

The Competent Person preparing a public report must absorb these Guidelines and make them central to everyday work practices. It is not the purpose here to include those Guidelines in full; however, some key points can be made to aid understanding and implementation.

RPEEE 'should be demonstrated through the application of an appropriate level of consideration of the potential viability of Mineral Resources.' The word 'appropriate' is used liberally throughout the SAMREC Code but is not defined generally, or in each case of usage; in this instance it should be taken to relate to risk or confidence where a qualitative low, medium, or high confidence corresponds to Inferred, Indicated, or Measured Resource categories respectively. The consideration must be based on a reasoned assessment, as will be elaborated below. While this may include technical assumptions, these must be presented and justified.

'The determination of RPEEE should be based on the principle of reasonableness and should be justifiable and defendable. The assumptions used to test for reasonable prospects should be reasonable and within known/assumed tolerances or have examples of precedence.' While SAMREC Table II, Guidelines for Technical Studies, relates to Scoping, Pre-Feasibility, and Feasibility Studies, the tabulation may provide some insight into understanding limits or ranges for precision, accuracy, and confidence. For example, the Capital and Operating Cost accuracy ranges are given as ±25–50%, ±15–25%, and ±10–15%, and the Risk tolerance as High, Medium, and Low respectively. Precision (and accuracy) in resource estimation should include reference to the quality of the analytical data as assessed through a quality assurance and control programme.

Where untested practices are applied, their use must be justified by the Competent Person. To use a hyperbolic example, an iron meteorite with nickel content perhaps as high as 25% would require a unique 'mining' practice to recover and extract the nickel. While no longer just science fiction, the practical solutions are far from economically viable.

As an example of another scenario, even though nickel is normally sourced from sulphide or laterite deposits, consideration of silicate-hosted deposits (*e.g.* olivine) is becoming a technical possibility (Santos *et al.*, 2015) and similar processes investigating CO_2 sequestration may give these investigations prominence in the foreseeable future. However, until such time, these laboratory experiments would qualify as untested practices requiring justification, not just chemically but also at a production scale to demonstrate the economics.

Interpretation of the word 'eventual' may vary depending on the commodity, mineral involved, or legal tenure. Bulk commodities such as coal and iron ore are typically planned and mined with very long timelines in mind, perhaps extending over many decades. This would be in stark contrast to most precious and other metals projects, for which short timelines of up to about two decades may be more appropriate. In discussing the importance of 'eventual' for these contrasting commodities, the impact of price and markets over the intended project timeline is almost certain to be paramount and deserving of qualifying comment.

For bulk commodities that require large infrastructure investment and possible government involvement, the mineralization may be more effectively described and reported within the United Nations Framework Classification for Resources 2009 and 2019 (UNFC) (UNECE, 2013, 2019). This scheme facilitates classification in a Rubik's Cube with axes of geological knowledge, project feasibility, and socio-economic viability. Mapping between SAMREC and other CRIRSCO Codes, and UNFC-2009 (UNECE, 2015) demonstrates the linkage between what is described as Exploration Results in SAMREC and the UNFC-2009 equivalents of 'Additional Quantities in Place', 'Exploration Projects', and 'Non-Commercial Projects'. The Additional Quantities may relate, for example, to the greater coalfield, beyond the current extent of drill delineation, for which it is not unreasonable to anticipate expanded formal resources in the future; the fuzzy logic bridging Inferred Mineral Resources and Additional Quantities may aid visualization of the need and understanding of the word 'eventual.' In any or either case it is imperative that there is a full discussion to the extent this is needed to support the concept of 'eventual'.

Diamond Guidelines

The particulate nature of diamond mineralization is one feature among many that differentiates diamond mineralization from that of most other commodities. The definition of a Diamond Resource in Clause 67 is a modification of the definition of a Mineral Resource in Clause 24, in recognition of these special circumstances for diamonds that are documented in the SAMREC Diamond Guidelines (SAMREC, 2016b). Specifically, for diamonds, the phrase 'grade or quality and quantity' is substituted by 'quantity (volume/tonnage), grade and value.'

Further guidance for Clause 67 states that 'In order to demonstrate that a Diamond Resource has reasonable prospects for economic extraction, some appreciation of the likely stone size distribution and value is necessary, however preliminary. '

Inferred Diamond Resources (Clause 68) are qualified in the SAMREC Diamond Guidelines as of 'Low' confidence, not 'no confidence', nor a 'guesstimate'. At an absolute minimum, RPEEE must be demonstrated. It is noted that although most Inferred Mineral Resources (including diamonds) are reasonably expected to be upgraded to Indicated Mineral Resources consequent on additional exploration, it must not be assumed that such upgrading will always occur.

For most metallic minerals the concept of 'eventual economic extraction' is normally restricted to 20 to 30 years, and frequently to much shorter periods. For Diamond Mineralization (especially alluvial deposits), as defined in Clause 65 of the SAMREC Code, the development timeline is typically short, and the word 'eventual' must not be an excuse to apply an unreasonably long time frame for the assessment of RPEEE. The Competent Person shall discuss this.

How do we achieve RPEEE?

Any achievement is the consequence of taking certain actions, often predetermined actions. In this case, the SAMREC Code Clause 24 provides the guidance, following the definition of a Mineral Resource, that a 'Reasoned Assessment' of a specific list of techno-economic assumptions likely to influence economic extraction should be the subject of the Competent Person's opinion. These are enumerated as geological, engineering (mining and processing), metallurgical, legal, infrastructural, environmental, marketing, socio-political, and economic issues.

SAMREC Table I, Section 4 details more specific guidance for the Estimation and Reporting of Exploration Results and Mineral Resources. Section 4.3 deals with Reasonable Prospects for Eventual Economic Extraction and subsections (i) to (vii) state that for each issue the Competent Person must 'Disclose and Discuss'. In addition, Section 4.3 (viii) requires that any material risks should be discussed. As an example of a material risk, Lock and van der Merwe (2003) described a portion of the Mineral Resource that was traversed by a rural gravel road, thus sequestering the resource. However, provisional approval had been granted by the relevant authorities to reroute the road, and thus access to the resource was restored. Although the risk still existed at the date of the report, the disclosure discussion provided the necessary solution. No doubt the reader could point to other projects with more significant risks.

Appendix 1 of the SAMREC Code includes a Recommended Table of Contents for a Competent Person's Report. Although the Table of Contents is provided only as a guide, it is intended to include all the requirements of Table I of the Code. The Table of Contents, Chapter 7 Mineral Resource Estimates, includes as subsection 7.3, Reasonable Prospects for Eventual Economic Extraction. Therein is the opportunity to present all the disclosure and discussion. It is quite possible that many aspects for discussion have already been referred to elsewhere in the CPR, but drawing the issues together in summary form and in a single place makes comprehension of the assessment process so much easier for the reader.

Australian JORC Code

The JORC Code (JORC, 2011) and its guidelines are very similar to the SAMREC Code. The JORC Code is perhaps more precise in providing the guidance that 'The basis for the reasonable prospects assumption is always a material matter, and must be explicitly disclosed and discussed by the Competent Person within the Public Report using the criteria listed in the JORC Table 1 for guidance.' The JORC requirement for commentary on an 'if not, why not' basis in the JORC Table I Report Template reinforces the mandatory nature of the reporting of reasonable prospects.

Emphasis in the JORC Code and the 2001 *AusIMM Bulletin* 23 (Mineral Resource and Ore Reserve Estimation – The AusIMM Guide to Good Practice) appears to be on the cut-off grade as the primary basis for the mathematical assessment of economic prospects. It is beyond the scope of this paper to discuss the sometimes complex formulations for estimation of cut-off grade, but both the SAMREC and JORC Codes also specify the inclusion of non-geological parameters in the assessment.

Reasonable Prospects is not just derived from a mathematical equation; in the previous section the author discussed the case of diamond-bearing gravels under a public road, and argued for resource declaration because provisional authority was granted. As an example of a situation where a matter beyond the geology,

mining, and processing has great importance in a contrary sense, Border and Butt (2001) advise that 'Ownership may determine whether a volume of mineral is a Mineral Resource.' They continue, 'An example of [this point] is where a limestone is suitable only for cement manufacture, and the local cement market is being accommodated by an existing production facility. If the local kiln owner chooses to use only limestone produced by an associated company, any other limestone deposit, even if technically superior, may not be a Mineral Resource, as it may not have the required "reasonable prospects for eventual economic extraction". It may have prospects in the distant future (when the kiln owner's limestone resource is exhausted or when the economic parameters change) but unless these events are considered likely to occur within the next 20 years or so, the deposit has no significant present value and therefore, in the opinion of the authors, should not be quoted as a Mineral Resource under the JORC Code'. This of course is the opinion of Border and Butt (2001); there may well be other opinions or options for the example described but the point being made is that a thought process should always be applied. There may be a fine balance between attributing present value through a resource declaration, and possible future value when opportunity opens. In this example, the emphasis was placed on the relationship between deposit owner and kiln owner, whereas the possibility that the deposit owner had more expansive plans is almost disregarded. However, if the latter were the case the planned alternative market would need explicit discussion.

Canadian CIM definition standards

Canada has taken a different path to public report disclosure and code compliance. The CIM (Canadian Institute of Mining, Metallurgy and Petroleum) Definition Standards (CIM, 2014) are relatively simple in comparison with the SAMREC and JORC Codes. However, CIM Definition Standards are incorporated by reference into the Canadian National Instrument 43-101 (National Instrument 43-101, 2014), a part of statutory law in Canada. When working to prepare a report for Canadian compliance, familiarity with these two documents is essential.

RPEEE is included in the definition for a Mineral Resource in the CIM Definition Standards where guidance for the arithmetic estimate of cut-off is provided. However, other parameters are not explicitly mentioned and RPEEE is not mentioned once in National Instrument 43-101, although it is implicit in the Section 3.4 Requirements Applicable to Written Disclosure of Mineral Resources and Mineral Reserves.

CIM has provided considerable guidance away from the formal Code and National Instrument documents. Discussion articles have been published in the *CIM Magazine* and by the CIM Reserve Definitions Committee, including specific guidance on RPEEE by Gosson and Smith (2007) and CIM (2009, 2015). This guidance covered the following matters that are discussed in the next section with project examples:

- Cut-off grade
- Long-term metals/commodity prices
- Lerchs-Grossman (LG) pit captures the required considerations of location, deposit scale, continuity and assumed mining method, metallurgical recovery, operating costs, and reasonable long-term metal prices
- An LG pit is not a regulatory requirement, but good for advanced projects. It is popular in Canada but care should be exercised for possible pitfalls from misuse of the parameter metrics
- The grade shell method for underground mineralization
- Qualified person (QP) can demonstrate RPEEE by comparison with analogous mine operations, for early stage assessments
- QP opinion on margin of revenue over potential capital as discussed by Waldie (2009)
- QP should consult other QPs to augment experience in determining RPEEE. This could be provided with internal peer review in a company.

These matters all fall within the geological, engineering, and economic parameters and assumptions listed for reasonable assessment under the SAMREC Code, even though not explicitly stated there.

Project examples

In providing several project examples, it is intended that the main matters for reasoned assessment can be illustrated in as simple a way as possible. Much of the work that is required to fulfil the obligation to demonstrate reasonable prospects is already part and parcel of good practice in estimating Mineral Resources and Reserves. However, too often this work is scattered across, or hidden away in the body of, a report in such a way that the reader may have difficulty in finding the critical inference. If the reader is, as will often be the case, an investor or adviser with limited time for little more than the executive summary, then bringing RPEEE assessment together in a single body of text will make an average report into a good report. A specific subsection in a report chapter on the estimation of Mineral Resources (and Reserves) is a recommendation in Appendix 1 of the SAMREC Code; among the three codes discussed here, SAMREC is the only Code to be explicit in this way, although in practice many companies reporting to other codes also follow this practice – for example Johnson *et al.* (2010) below.

This is an opportune moment to briefly discuss commodity prices, whether these be for diamonds, gold, copper, or nickel. Although the selection of a specific commodity price might seem to demand the adoption of objective criteria, this is often not the case. Commodity prices used for the estimation of Mineral Resources are often more optimistic than for the rigorous approach in determining a price for Mineral Reserve estimation. The estimator's discretion and experience are paramount, even when the most optimistic outcome may be preferred.

For diamonds, valuation requires considerable skill and expertise because diamond value/price is deposit specific. It is not intended to discuss this further here; however, it is worth noting that the discussion for metals is not practicable for diamonds. The alternative for diamonds that has been followed in many cases is to reference the advice of diamond market experts, either through published predictions or specific contracted opinion.

For most metals, prices can be found from multiple freely available online website sources based on the London Metal Exchange or other marketing organizations. Prices change by the minute but daily, weekly, or monthly records may be adequate to use for assessments of RPEEE. How to use the price data in the public record is discussed in CIM (2008, 2015) and Waldie (2009).

McDonald (2016) has provided an important recent perspective from a South African viewpoint. Some of his analysis

overlaps with the discussion here. The main point he makes is that the uncertainty in forecast prices and exchange rates may impact seriously on the expectation for ±10% confidence limits in Feasibility Studies. Although his study is directed towards Mineral Reserves, his conclusions are equally valid for cut-off grades and Mineral Resource estimates.

Figure 1 is a chart of the ten-year gold price from 2009 to 2019. It is presented here to illustrate some of the points discussed by Waldie (2009). If a gold mineral deposit were estimated in March 2012 using the then current gold price of about US$1 674 per ounce, the Mineral Resource declared would be significantly overestimated because of the fall in price over the following three years. Although this fall may not have been expected, the estimate clearly was at risk, and suitable qualifying statements should be applied if a current metal price is adopted.

An estimate at the same date but using a three-year trailing average of US$1 320 per ounce, a US Securities and Exchange Commission guideline, would more closely match the future of the project estimate. However, a similar trailing average estimate for March 2015 would overestimate resources compared to the March 2015 current price of $1 179 per ounce. In general, a three-year trailing average in a rising market will underestimate resources; in contrast, in a falling market this method will overestimate resources. Note this guideline should be used advisedly and with careful qualifying statements.

A long-term average (say ten or twenty years) may remove much of the annual price volatility, but comparison using current and long-term average prices will highlight material deviation in asset value for producing mines as long as all prices for comparison are in real terms.

Waldie (2009) also suggests the application of a 'Margin over Cash Cost of Production', but this may only be appropriate for producing mines when resource or reserve updates are undertaken and where short-term fluctuations in metals prices encourage mining of lower grade areas during price upturns, and high-grading during price declines. Producing mines will generally have their own benchmarks for this purpose, but there are several global research and consulting firms that provide cost analysis research that major companies may avail themselves of (*e.g.* CRU https://www.crugroup.com/, AME Research https://www.uk.amegroup.com/, or Wood Mackenzie https://www.woodmac.com/).

A very simple method may be to consider peer group consensus. A Competent Person with access to a database of contemporary reports can quickly find this time-dependent consensus as the average of the prices used in a group of reliable estimates. There are commercial services available that can undertake this consensus research, for example Consensus Economics (https://www.consensuseconomics.com/about/).

It is lastly worth mentioning that certain commodities and/or projects rely on contract pricing. A good example is uranium, where the project timeline and financing hurdles require the consideration of eventual price to be locked in at an early stage of development.

Discovery Metals Limited

Discovery Metals was an ASX-listed company operating the Boseto mine in the Kalahari Copper Belt of northwest Botswana. Reference is made here to a 2013 news release of the company (Discovery Metals Limited, 2013) although, sadly, the operations have closed since then.

In line with the requirements of the revised JORC Code of 2012, Discovery's news release included a Table I attachment that provides a detailed technical summary of the project with required comments on an 'If not, why not' basis. Table I here is an extract from the JORC Table I, Section 3 Estimation and reporting of Mineral Resources, as an example of the open pit cut-off parameters used to define Mineral Resources.

Table II is an extract from JORC Table I Section 4 Estimation and Reporting of Ore Reserves.

While the many other matters for a reasoned assessment were not included in this assessment of cut-off grade, the core requirement for economic extraction was established.

Mountain Province Diamonds Inc - Gahcho Kué Kimberlite Project

It is common practice in Canada (and sometimes elsewhere) to make use of mining software to quickly create an open pit mine model with several simple techno-economic parameters that define potential viability (Gosson and Smith, 2007). The resulting 3D clipping of mineralized blocks inside the defined frame are then cumulated to give a resource tonnage and grade. While this approach to assessing RPEEE may well be fitting for advanced exploration projects close to Feasibility Study, application to early

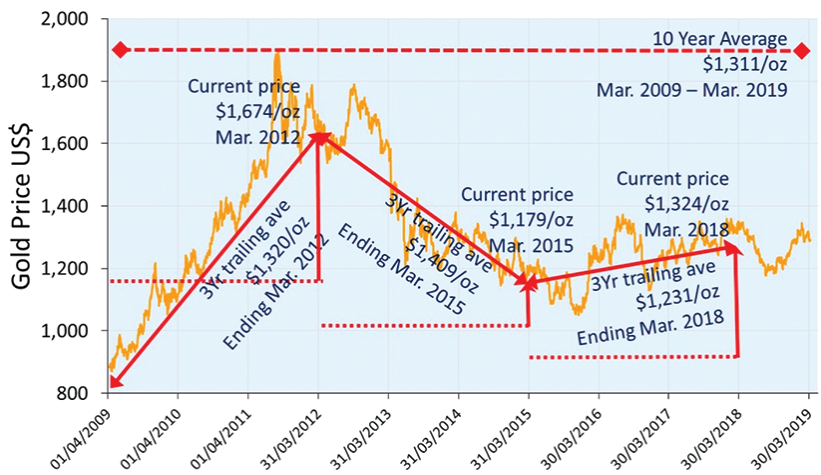

Figure 1—10-year gold price (US$) April 2009 – March 2019 (www.macrotrends.net)

Table I

Cut-off parameters. The basis of the adopted cut-off grade(s) or quality parameters applied

Criteria	Commentary
Cut-off parameters	For open pit resources, a variable cut-off is applied on Cu grades depending on oxidation state (1% Cu in oxide, 0.7% Cu in transition material, and 0.5% in sulphide ores). These cut-offs were calculated based on application of a simple economic model (Cu price $5700/t, mining cost $2/t, processing cost of $25/t and Cu recovery of 45% in oxide, 65% in transition and 90% in fresh).

Table II

Cut-off parameters. The basis of the adopted cut-off grade(s) or quality parameters applied

Criteria	Commentary
Cut-off parameters	Owing to the relationship between the metallurgical copper recovery and the block S:Cu ratio, no traditional cut-off grade was applicable. The determination of ore was made by calculating the cash flow that would be produced by processing material and the cash flow which would be produced by mining it as waste. If the cash flow from processing was higher, the material was considered ore. If not, it was considered waste. The calculation of cash flow was based on: • A copper price of US$7 250 per tonne; • A royalty of 3% of gross revenue; • A total transport, smelting and refining cost of US$378 per tonne of concentrate produced; • Payable copper metal of 96.65%; • A variable metallurgical recovery dependent on the S:Cu ratio; • A variable process cost dependent on the material type, with the majority being sulphide with a process cost of $7.00 per tonne; and • Minor contribution from silver. However, the Ore Reserve for Zeta is >99% sulphide and the average S:Cu ratio for this material results in the application of the maximum allowable copper recovery of 93%. This would result in a cut-off grade (ignoring silver contribution) of 0.5%.

stage projects may be questionable because the techno-economic parameters will likely be generic, thus introducing an element of risk that becomes masked by the perception of computer capability. The adage of 'garbage in, garbage out (GIGO)' must be avoided.

Gahcho Kué is a kimberlite diamond mine located in northern Ontario, Canada, jointly owned by De Beers and Mountain Province Inc. The Inukshuk Capital Corp./Mountain Province Mining Inc. joint venture began exploration in the area in 1992 and found the first kimberlite (5034) in 1995. De Beers Canada Inc. entered the joint venture in 1997, now on a 51/49 equity basis, and three new kimberlite discoveries (Tesla, Tuzo, and Hearne) joined 5034 to form the main Gahcho Kué kimberlite cluster. A maiden Diamond Mineral Resource was reported in 2003.

A Diamond Resource estimate for the Definitive Feasibility Study on the Gahcho Kué Kimberlite Project, Northwest Territories, Canada, prepared for Mountain Province Diamonds Inc. and De Beers Canada Inc. by JDS Mining & Energy Inc. (Johnson *et al.*, 2010) applied the LG Pit method. The techno-economic parameters used are listed in Table III. Note that the range in diamond price is the consequence of summarizing the parameters for three separate kimberlite pipes, each with distinct diamond populations.

It is also worthy of note that the parameters for the Diamond Resource estimate were not the same as those for the Diamond Reserve estimate; this was partly the consequence of the requirement to exclude all Inferred Resources for the Diamond Reserve estimate, but also a consequence of improved understanding of other factors as the engineering study progressed. This distinction is important, not as an impediment to future extraction of the Inferred Resources, but as a matter of code compliance limiting Resource to Reserve conversion only to the higher confidence resource categories.

Vaaldiam Mining Inc. – Braúna Kimberlite Project

The Braúna kimberlites in Bahia State of northeast Brazil were discovered in 1990 as part of a De Beers regional programme that began in 1980. The De Beers work progressed slowly until the project was eventually sold to Majescor for US$500 000 in 2004. Vaaldiam Mining Inc. took over payments for Majescor

Table III

Whittle® input parameters: Gahcho Kué Kimberlite Project

Input	Amount/quantity	Comment
Prices	US$70–116/ct	Range for different sources. BCOS 1 mm
Selling costs	10.0%	
Marketing	2.5%	
Royalty	–	
Costs (C$/t, 4Q 2008)		
Mining	3.41	Same for mineralization and waste
Incremental haulage	0.03	Per 12 m bench below 421 masl pit exit
Process and G&A	42	
Overall slopes (degrees)		
Granite	47–63	
Kimberlite	50	Locally 30 to accommodate ramps
Glacial deposits	25	
Other		
Dilution	8%	
Process recovery	100%	Size-frequency distribution includes allowance for plant losses
Exchange rate	1.17 C$ to US$	

* BCOS = bottom cut of size

and assumed full ownership of the project from 2007. Vaaldiam initiated a more aggressive field programme that ignored the small size and lowish grade which had put De Beers off.

Vaaldiam had completed a bulk sample programme of nearly 5 000 t in 2010 and recovered some 1 000 ct of diamonds. These results were the basis for a maiden Mineral Resource report in late 2010. Vaaldiam moved quickly to undertake a scoping study, or Preliminary Assessment as it was called in Canada at that time. Coffey Mining's Brazil office was contracted to undertake this work (Lock *et al.*, 2011) which was filed in Canada on www.sedar.com in 2011.

The assessment of RPEEE was made primarily through comparison with similar diamond operations familiar to the reporters in Africa. The following text from Lock *et al.* (2011) listed the African operations and the online economic source information.

'… the *in situ* value of the South Lobe kimberlite, based on current bulk sample information, is US$83.1/t.

Brazil does not have a current kimberlite diamond mine in operation to compare with but reference to a selection of projects in Africa point to a much higher *in situ* value for realistic prospects:

- Petra Diamond – Koffiefontein Mine US$25.8/t (http://www.petradiamonds.com/o/p_sa_koffie.php)
- Petra Diamonds – Williamson Mine US$10.8/t (http://www.petradiamonds.com/o/p_tz_williamson.php)
- Gem Diamonds – Letšeng Mine US$29.4/t (http://www.gemdiamonds.com/b/resources.asp)
- Gem Diamonds – Gope Project US$30.7/t (http://www.gemdiamonds.com/b/resources.asp)
- Lucara Diamonds – AK6 Project US$42.7/t (http://www.lucaradiamond.com/s/Botswana-AK6DiamondResource.asp)

The average for these projects and mines is US$27.9/t. The lowest at US$10.8/t is still well in excess of the Braúna 3 North Lobe *in situ* value.

Further confirmation is provided by Lucara Diamonds' statement of US$17.2/t operating cost for the AK6 project.'

The Braúna project is located in Bahia State, where the climate and physiography are comparable to the African projects selected for comparison. Nevertheless, Vaaldiam were not encouraged by the predicted project returns and may have been negatively influenced by the much higher threshold rock value that determines RPEEE in Arctic Canada, where rock value of US$80 per ton or more may be required. Vaaldiam decided to divest of the project and thus missed out on developing Brazil's first hard rock diamond mine, which eventually happened under the new ownership of Lipari Minerãçao Ltda in 2016.

Discussion

The SAMREC Code 2016 contains the clearest statement to date, in both the definitions and guidelines, of the central place that RPEEE holds in achieving the profound objective of the 'required minimum standard for the Public Reporting of Exploration Results, Mineral Resources and Mineral Reserves.'

RPEEE is the critical core of our work processes, for without the assessment of RPEEE the outcome would be no more than an estimate of Mineralization, and the result of bad practice if reported as a Mineral Resource. RPEEE assessments require acceptable appreciation and application of all three founding Principles of SAMREC, namely materiality, transparency, and competence. 'Acceptable' in this context demands an understanding of what industry practice implies and the requirement for a minimum standard. We must all be reminded that perhaps the ultimate test of our work practices is the concept that 'a Competent Person must be clearly satisfied in their own minds that they are able to face their peers and demonstrate competence in the commodity, type of deposit and situation under consideration.'

The SAMREC Code requires that 'a Mineral Resource be not an inventory of all Mineralisation', that 'an appropriate level of consideration' be applied, and that 'the principle of reasonableness' be employed and should be justifiable and defendable. This code guidance can be condensed into three simple aphorisms:

- Declaring Mineral Resources is more than a mathematical model
- RPEEE requires reasoned Thought, Research, and Opinion
- Material, Transparent, and Competent Estimates will support the three 'R's: Reasonable, Robust, and Reliable.

Inappropriate extension of economic parameters to Mineral Resource reporting may lead to premature declarations of economic viability rather than potential for economic viability. Poor or non-adherence to compliance with the SAMREC Code is likely to lead to progressively more prescriptive requirements in our work practices in the future. Self-regulation is an important aspect of the professional associations in South Africa, through which registration is obtained. The Geological Society of South Africa (GSSA) and SAIMM, for example, have formal Codes of Ethics or Conduct with enforceable disciplinary procedures. These associations provide a pathway to Competent Person recognition, but there is also the clear statement in the Foreword to the SAMREC Code that the code 'is binding on members of these organisations'.

The GSSA and SAIMM both recognize that their complaints and disciplinary procedures may lead to serious cases being forwarded to the South African Council for Natural Scientific Professions (SACNASP) or the Engineering Council of South Africa (ECSA), the statutory bodies that link to a more severe legal environment. This linkage will not go away and it is thus incumbent on us to avoid this path by understanding our responsibility in Public Reporting.

As a final thought, it is worth consideration by SAMREC that there are many matters that can be identified and incorporated into workshops or seminars that delve into greater detail on specific subjects. RPEEE is deserving of one such workshop all to itself.

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Use and misuse of historical estimates and data – Examples from diamond projects

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Synopsis

Projects with long histories must be documented in current disclosures with transparency and materiality, using historical data and historical estimates. Historical data may be of great value if it is from a reliable source, and the raw data can be validated and/or duplicated. Historical estimates can and should be reported, but with qualification of the ever-changing economic parameters of 'Reasonable Prospects for Eventual Economic Extraction' (RPEEE). The SAMREC Code requires current sampling results and diamond valuations, without which RPEEE cannot be assessed; consequently, historical estimates cannot ever be declared as current Diamond Resources or Reserves.

The SAMREC Code defines historical estimates and provides guidance on the use of historical data. Examples from real projects and reports in the public domain are reviewed in this paper. Opinions on use and misuse are those of the writer; judgment on good or bad practice is not the intention and is left to the opinion of the reader. Comparison, with both the JORC Code (Australasia) and CIM Definition Standards and National Instrument 43-101 (Canada), is provided. The SAMREC Code appears to be more closely aligned with the Canadian standards.

Keywords
project valuation, historical data, code compliance, diamonds, SAMREC.

Introduction

Mineral exploration is necessarily a process starting with and building on any geological or other relevant information that may be available to the prospector. Such information may be in the public domain in the form of national or international reports or publications concerning any region or area of interest. This information forms the basis for building a database upon which an exploration strategy and target can be developed. With a specific intent in mind, it is certain that an explorer will search for reports on any prior mineral exploration in the area of interest, and about the commodity of interest. These reports contain historical data, and perhaps historical estimates.

It is the scope of this paper to consider and discuss the nature of the historical data and estimates in the context of the SAMREC Code, and modern good practice in the application and usage of such data and estimates. Discussion will consider the definitions and guidance in the SAMREC Code (SAMREC, 2016a). Examples of the use and misuse of historical data and estimates are presented with reference to public disclosure reports.

SAMREC Code definitions

The glossary of terms provided at the very beginning of the SAMREC Code presents a succinct definition of a historical estimate:

> '*... an estimate of the quantity, grade, or metal or mineral content of a deposit that an issuer has not verified as a current Mineral Resource or Mineral Reserve. The estimate predates the issuing of the Code and/or was prepared before the issuer acquiring, or entering into an agreement to acquire, an interest in the property that contains the deposit.*'

In this paper emphasis is placed on mineral exploration and resources, but could and should be understood to also include Mineral Reserves, although such inclusion would undoubtedly require significantly more comment than space here allows.

The word 'issuer' must be taken to mean the current owner of title to the project in question and being reported on by a Competent Person (CP). In the context of the Code and its relationship to the Johannesburg Stock Exchange, 'issuer' should be understood to mean a public company listed on the JSE. However, any Public Report, as that term is defined in the SAMREC Code, may be prepared and presented as compliant with the Code if the technical aspects of such disclosure are the responsibility of a CP. 'Issuer' might safely be defined in this context as the current project owner or operator, where 'current' should be taken literally as meaning compliance with the Code on the date under consideration. Any 'issuer' that is a private company and that elects to invoke compliance with the SAMREC Code is subject to all the responsibilities of the required minimum standards for Public Reporting.

Care should be taken in interpreting this meaning because the SAMREC Diamond Guidelines (SAMREC, 2016b) to the Code include a clear statement that diamond valuation for Mineral Resource estimation 'should be less than six months old.' The Code also states, for all minerals, that estimation cannot be undertaken in the absence of sampling information (Clause 23); diamonds for valuation are recovered from sampling and thus, for diamonds, 'current' has to mean less than six months. More generally 'current' must relate to the validity of parameters and assumptions relevant to the outcomes of work in progress. The most obvious example of this is the commodity price, which may fluctuate over time with either a positive or negative impact on project economics. The Diamond Guidelines include this specific instruction because diamond valuation is deposit-specific, in contrast with most metallic minerals for which there are daily book prices. Where considerations of cost adversely impact a decision to retain a bulk sample diamond parcel, the project operator may be advised to retain the diamonds at least until a specific point along a development timeline.

The original SAMREC Code was issued in March 2000, and this must inform the primary date limitation on a historical estimate. The secondary limitation arises whenever there is a change of ownership of the project in question. In such cases, although the issuer may rely on a historical estimate in the due diligence for project acquisition, once the transaction is completed the implication is that the new owner should take steps to convert or update the estimate to current status. The Diamond Guidelines contain a recommended template Table of Contents, stipulating that Chapter 4 ('Project History') 'may also include a discussion of relevant historical estimates.' This makes it clear there is a difference between a 'Previous Diamond Resource Estimate' (by the current owner) and a historical estimate.

The Diamond Guidelines modified the Clause 23 guidance and include the powerful statement that 'Grade and value data that are not SAMREC compliant may not be used to estimate a Diamond Resource – purely historical or anecdotal estimates are insufficient.' The equally strong statement in the Diamond Guidelines Clause 67 that 'Resource estimates shall not be based on unverified or unverifiable historical or artisanal results only' recognizes the reality that the sometimes shady world of diamonds has a sad record of projects and promoters who wish to appear persuasive with a handful of diamonds, often of undemonstrated provenance or representivity.

Historical estimate

Although the SAMREC definition of a historical estimate does not include any specific reference to age, it is clear that March 2000 may be a threshold date. Alternatively, a change of ownership may define the age, and this could be a matter of days, weeks, or months. However, in either case, the Diamond Guidelines require that any current estimate is based on a diamond valuation no more than six months old. We thus have three defining points on a project timeline that may determine an estimate as a historical estimate: March 2000, date of change of ownership, and/or date of last diamond valuation. The latter may just be a previous estimate that can be fairly easily updated if the diamond parcel has been retained and can be revalued.

The raw data collected, on which an estimate is based, should be immutable, and this will be discussed in the next section. The economic and other parameters and assumptions that support the assessment of viability are not invariable, and it is this reality that is the basis for scrutinizing historical estimates and providing suitable comment.

Diamond valuation has already been mentioned, and it should be no surprise that changes in the diamond value will impact an estimate update, either positively or negatively. For a metals project the commodity price, available as a daily book price, is an integral part of determining a cut-off grade. Diamond value is deposit-specific and not, like metals projects, subject to a daily book price. While diamond projects may not consider cut-off grade in the same way as for metals projects, the diamond value is also still an integral part of the economic outcome, as defined by 'Reasonable Prospects for Eventual Economic Extraction' (RPEEE).

Hard rock diamond mining is most often carried out to the physical limits of the kimberlite pipe, and thus the cut-off grade is zero and is mapped as the kimberlite/country rock contact. Sometimes a kimberlite pipe may contain kimberlite lithologies of differing grade; and these internal contacts may define the cut-off grade, although it is probable that mining would be constrained by visual or other lithological differences. This was the case when De Beers mined the Letšeng Main Pipe in the 1970s (Lock, 1980).

The SAMREC Code makes it very clear that, for diamond projects, the selected bottom cut-off screen size (Clause 60) must be stated. The reason for this is that a larger bottom screen opening will allow more diamonds to pass through as tailings, thus reducing the grade; equally, a larger opening will reduce the number of smaller, lower value diamonds being recovered, thus increasing the average value. Preferred changes over time to the bottom cut-off thus critically affect the economic outcome.

While diamond value may be the most obvious economic parameter that changes with time, we cannot ignore the changes to other technical parameters and matters that individually and collectively can transform a historical estimate into nothing more than Mineralization, as defined by the SAMREC Code, *i.e.* 'A concentration (or occurrence) of material of possible economic interest.' Achieving RPEEE requires the assessment of a range of matters (Lock, 2020, this volume) such as mining and processing costs and constraints of a legal or environmental nature. Each of these may change over time in ways that can impact a historical estimate in a positive or negative way; thus, what was a Mineral Resource may no longer be so, and equally what was just Mineralization may be a Mineral Resource, after current estimation.

The SAMREC Code is clear and firm in its guidance to the definition of a Scoping Study (Clause 44) that historical estimates

may not be included. This exclusion goes with the same restriction in the use of Exploration Results, Exploration Targets, or Mineralisation in a Scoping Study. The nature and reliability of the information from any of these sources is determined to be beneath the threshold acceptable for any Technical Study.

The Diamond Guidelines (SAMREC, 2016b) provide a report template that includes a chapter entitled 'Project History' with subsections on Previous Mineral Resource and Reserve Estimates. These may be valid and compliant if the estimates postdate the first SAMREC Code in 2000 and the ownership has not changed during this period. However, any historical estimate disclosed in this chapter should be appropriately discussed.

Historical data

In all cases, in line with Clause 20 of the main Code and the Diamond Guidelines defining and guiding Exploration Results, 'Historical data and information may also be included if, in the considered opinion of the Competent Person, it is relevant and reliable, giving reasons for such conclusions.' Care should be taken in applying this guidance as it is explicitly limited to Exploration Results. Conceivably, historical data might be incorporated into a statement of an Exploration Target, but extreme care would be required to avoid implying the discovery of any potentially economic mineralization; Public Reporting of any such Exploration Target requires the support of an explicit exploration programme.

There are many reasons why historical information should be treated with care, and not assumed to be acceptable for a current Mineral Resource estimate. While historical information may have been accumulated under the work practices of the day, times have changed and standards have generally evolved with a more rigorous approach to the collection and storage of data, whether this be rock samples or drill core, analytical results or reports. As an industry, we continue to aspire to improving our practices with the objective of transparency and reliability of our data, such that future generations may place greater confidence in historical data than is usually possible today. Our exploration database management practices are central to the achievement of this aspiration.

Current exploration project work practices and data management should be undertaken with an attitude of compliance with the SAMREC Code, so that Mineral Resource and Reserve estimation can be undertaken in a timely manner. These compliant work practices provide guidance in understanding the value of historical data. A form of due diligence, as might be undertaken for a buyer in a property acquisition, can be envisaged as necessary in evaluating the worth of historical estimates, and demonstrating why they cannot be simply adopted as a current estimate. In other words, a review of the historical estimate in a test of compliance with modern practices is necessary as a starting point. There are several typical problem areas that could be reviewed, including a broken audit trail, non-representivity of geology and/or grade, contamination affecting grade, and database issues. Any problems identified should be flagged and discussed in reporting of historical estimates to ensure a balanced comparison with current practices.

The conclusion is almost certain to be an understanding that historical data cannot generally be accepted into a current estimation. There may be exceptions to this guidance based on a reasonable assessment of the nature and source of the information, as will be discussed with two project examples.

Primary or interpreted

In a reliable exploration database the raw, or primary, data will be stored and archived in such a way that a new analysis and estimate can be undertaken from first principles. To appreciate the possible difference between primary and interpreted data, it is sufficient to show that drill-hole information recording only lithological intersections or sample grade interpretations of an economically mineralized section incorporates bias or assumptions from earlier times. While this may not be true for all commodities, kimberlite descriptive terminology and classification has undergone fundamental revision over several decades, necessitating careful cross-referencing into the modern scheme of Scott Smith *et al.* (2013). These interpretations may change with changing terminology, technology, or economics. Access to primary data allows for an unbiased new analysis.

Source/reliability

It is surely good practice when reviewing historical estimates and data to at least consider the reputation of the practitioner or company that generated the historical report. Despite changing times, it should be reasonable, in a first parse of old reports, to consider this and to place the report in a virtual spectrum of reliability. It ought to be possible to distinguish the extremes of 'good' and 'bad' in this way, even if there is no immediate acceptance or rejection from this simple parsing. Well-known public companies with a long and reputable history should cluster towards the 'good' part of the spectrum, in contrast to little-known or tainted practitioners who may cluster towards the 'bad' part. While this may seem unfair, closer inspection should allow for a revision of first impressions of either 'good' or 'bad'.

In either case a review of historical estimates and data should always lean towards cautious conclusions, and move to an inclusive approach in any current estimation only in exceptional circumstances, as will be illustrated.

Diamond valuation data

The use or misuse of old diamond valuation reports is perhaps the single most problematic aspect of bad practice. Unlike other commodities where the price is date-specific, diamond value is deposit-specific as well as date-specific. Indeed, any diamond valuation is also subject to the effect of bottom cut-off screen size, as has been discussed.

Bringing old diamond project reports into current reporting may be one challenge, but the copying and pasting of old diamond valuations will always be fraught with serious issues of reliability. Fortunately, SAMREC (2016a) explicitly requires that a diamond valuation no older than six months is used in any current estimate. Notwithstanding this serious constraint, there may be good reason to access and apply historic diamond size distribution data together with current, perhaps limited, data, to build a size-value model that can become the first step in developing a reliable new diamond price estimate, as will be shown for the Letšeng Mine case.

Duplication and validation

While direct use of historical data may be unacceptable in almost all circumstances, the integration of old data with new could be considered as a path to improving the confidence in an estimate. If an initial test of source and reliability is passed, a programme of duplication and validation may provide the confidence for this integration to be agreed. Again, this mimics some typical detailed

due diligence studies that demand limited duplicate drilling and assaying of selected parts of a project. A twinned/duplicate drill-hole should be expected to closely match in lithology even if the sample assays differ to some extent, dependent on the type of mineralization and nugget effect.

The scope of the duplication and validation work that is undertaken may help determine the degree of integration that may be tolerable. It is probable that cost considerations will encourage as little duplication work as possible, but this factor should be carefully balanced against the desired technical objectives.

Project examples of 'use'

Braúna kimberlite, Bahia State, Brazil

The Braúna kimberlites are a cluster located in the interior of Bahia State of northeast Brazil that were discovered by De Beers in the early 1990s, although artisanal mining of diamonds was known in the area from 1927. A long period of exploration and evaluation has progressed to the definition of the Braúna 3 kimberlite South Lobe as a viable diamond resource. Mining, of what is Brazil's first kimberlite diamond mine, commenced in 2016 under the ownership of a private company named Lipari Mineração Ltda.

Vaaldiam Mining Inc., a Canadian public company, undertook a Preliminary Assessment of the Braúna project in 2010 and 2011 (Lock *et al.*, 2011).

The long history of the project can be summarized thus:

1927–1953	Artisanal 'garimpeiro' mining
1980–1998	De Beers: exploration and discovery of primary kimberlite bodies
2000–2004	Artisanal activity
2004	De Beers data sold to Majescor
2005–2008	Vaaldiam 60% buy-in and operator, 100% buy-in
2010	Vaaldiam bulk sample and maiden diamond resource by ACA Howe (Leroux, Roy, and Masun, 2010)
2011	Vaaldiam Preliminary Assessment, after resource audit by Coffey Mining (Lock *et al.*, 2011).

Despite the indicated viability, Vaaldiam declined to progress the project and ownership passed in a private transaction to Lipari Mineração Ltda; this company progressed project development leading to a production decision and mine construction in 2015.

Our preliminary assessment was based on current information generated by Vaaldiam during their ownership period, and is thus not historical data, or estimates, as defined by the SAMREC Code. Additionally, the diamond valuation used in the maiden resource estimation was dated 1 November 2010, and thus less than six months prior to the report date; it was compliant with then current SAMREC requirements even though this was not specified in the SAMREC Code or NI 43-101 at that time.

Despite the sale to Majescor of the De Beers data for Braúna, the records of their surface trenching were patchy and considered too unreliable for integration into any future resource evaluation work. The results were in any case essentially duplicated and validated by the more extensive Vaaldiam programme. All the core drilling undertaken for deposit modelling was undertaken by Vaaldiam, and was not historical data. New microdiamond and mini-bulk sample diamond recovery was also undertaken by Vaaldiam, but while of some value in a comparative sense, this data was not used in the estimation.

Our report described the history in a separate chapter, as required for NI 43-101. Reporting of all the exploration undertaken, including the earlier work of De Beers, was presented chronologically with the current work of Vaaldiam. Although this may have been misleading with regard to a compliant description of the De Beers historical data, this information was not used (and therefore not misused) in resource estimation. The diamond resource estimate was based on the 2010 bulk sample of just under 5000 t that produced 1057 ct. Geology, volume, density, and diamond value were all based on the Vaaldiam work.

Thus, there was no historical estimate, and the historical data was reported in a transparent and material manner that aided comparison, where appropriate, with the current data. It is also worth noting that new geological models, based on reinterpretation of primary lithological logging and relogging, or drill core, have been significant in improving the estimates for this kimberlite.

Messina and Star diamond mines, South Africa

Snowden Mining Industry Consultants (Retter and Snowden, 2003) were commissioned in 2003 to prepare valuations of the Messina and Star diamond mines in the Northern Cape and Free State Provinces of South Africa, respectively. The valuations were needed in connection with a transaction involving Majestic Resources NL, an Australian public company and Messina Investments Pty Ltd (Minvest), a South Africa company, whereby Majestic would pay cash and shares for 100% of the shares of Minvest.

Both mines presented significant challenges to valuation due to their long and chequered history. In this paper, only the Messina story will be discussed, although the methodology applied was the same for both mines.

The earliest mining of the Bobbejaan Fissure atop the Ghaap Plateau west of Barkley West was in the 1930s. This progressed to more systematic small-scale mining through the 1950s to 1981, when operations were consolidated under Minvest as Messina Mine. Messina Diamond Corporation acquired Minvest in 1996 and it was this corporate entity and subsidiaries that carried renewed production through to 2003.

Clearly, much of the information on the mining operations should be considered as historical data because it pre-dates SAMREC. However, the ownership bridges the SAMREC Code threshold, and the valuation report is post-SAMREC 2000. Finally, as the mine was in operation on the valuation date, the diamond value was current and based on actual production sales.

The Bobbejaan Fissure is a kimberlite dyke that, by its nature, has geological, grade, and density continuity along strike and to depth. However, the nature of the deposit and ownership, as well as the history bridging the publication of the first SAMREC Code, are such that current mining operations are based on historical experience, both geological and mining. Formal Mineral Resources of a compliant nature probably never existed over much of the life of the mine. A typical exploration database almost certainly did not exist, but detailed production records did.

Retter and Snowden (2003) took the important decision to use discretion in developing a Mineral Resource estimation methodology, using historical data that may have been viewed as entirely unacceptable if it were not for the high regard in

which Retter and Snowden are held globally. They made the bold assumption that historical mining performance can be used as a guide to future performance (Figure 1).

The down-dip continuity of the kimberlite fissure and its assumed diamond grade was drill-tested with a very limited underground drilling programme. Retter and Snowden (2003) then considered the following:

- Fissure thickness past and present
- Mine's historical diamond grade and grade over the last twelve months
- Evidence for an increase or decrease in grade with depth
- Likely grade in the next 240 m below the lowest current mining level.

This comprehensive and complex analysis allowed them to declare Mineral Resources on the following basis:

- Measured Resource: one mining level (40 m) below the base of the current working levels
- Indicated Resource: two mining levels (80 m) below the base of the Measured Resource
- Inferred Resource: three mining levels (120 m) below the Indicated Resource.

The result of this analysis was a preferred value in the range R54 million to R109 million.

It could well be argued that, by 'the letter of the law' (read SAMREC Code), this analysis was highly irregular and could have been rejected by the regulatory authorities. Perhaps so. Before jumping to this conclusion, consider the record of recent historical resource estimates reported by le Roux (2017), who tabulates four combined estimates for Messina (Sedibeng) and Star by Petra Diamonds after the Retter and Snowden (2003) report and before the le Roux (2017) report. In all cases the diamond grade falls in the range 73.3 to 75.6 cpht (carats per 100 t) based on an undiluted diamond grade derived from real production data, and not new sampling data. The very fact of consistency from integration of old and new production data lends considerable confidence to the method applied and the declared Diamond Resources reported. The production data is in fact bulk sample data under another name. This JORC (2012) compliant report was prepared to support the listing of Frontier Diamonds Ltd on the Australian Stock Exchange in October 2017. Although the Retter and Snowden report may no longer be available online, it is referenced in the le Roux report as a revised 2004 document, and is the basis, with updates, for his resource declaration.

There was no published or known historical estimate for the Messina Mine, but the current resource estimate is profoundly dependent on historical data that has been used with discretion in difficult circumstances to fulfill the valuation need to consider an arm's length transaction based on 'willing buyer, willing seller'. The transaction was accepted by the parties and endorsed by the Australian regulators.

Letšeng, Lesotho

The Letšeng kimberlite pipe has proved to be the most significant of the Lesotho Maluti Mountains diamond discoveries. From the moment that Peter Nixon almost stumbled over a weathered outcrop in a stream bed in 1957, the progression through government diggings in 1959 to the recovery of the Lesotho Brown, the 601 ct stone found by Ernestine Ramaboa, and the history of bringing this deposit to production in the 1970s, there has been a sense of inevitability that production would be renewed, even after the mine first closed as a De Beers operation in 1981.

The author worked with De Beers at Letšeng from 1973 to 1976, undertook research for a PhD immediately thereafter (Lock, 1980), and led a technical team that assisted Gem Diamonds through the due diligence and valuation process for their bid to JCI in 2005. This knowledge provided an exceptional experience of what historical estimates and data are in reality. The Letšeng Diamond Mine has been back in continuous production since 2004. During this time the operation has been both profitable and has produced several of the world's largest diamonds ever recovered, including the 910 ct Lesotho Legend, the 603 ct Lesotho Promise, the 550 ct Letšeng Star, and the 493 ct Letšeng Legacy.

Despite this very positive story, there is no doubt that the unusual, if not unique, combination of low diamond grade and high diamond value may have presented insurmountable obstacles to compliant estimation of mineral resources if the SAMREC Code had been operative in the 1970s when RTZ and De

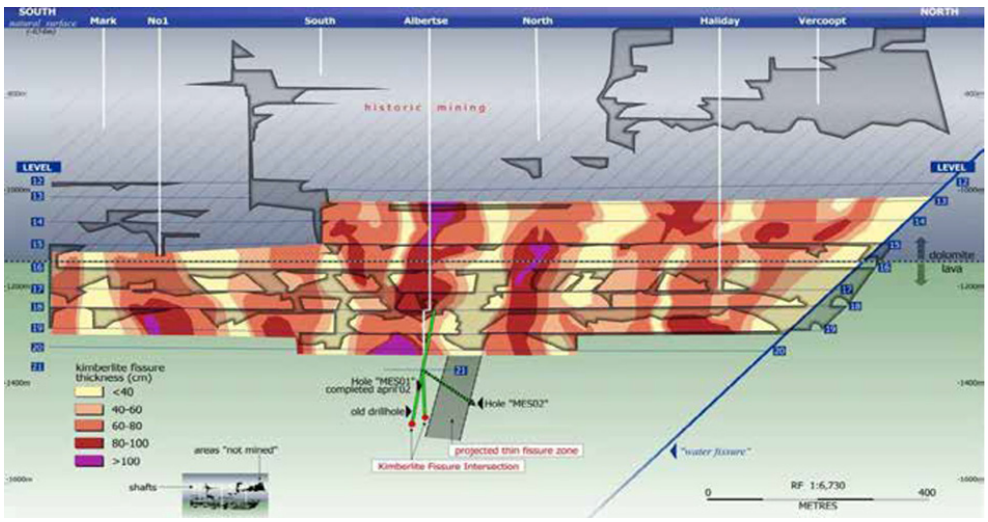

Figure 1—Sedibeng (Messina) core drilling showing locations of boreholes MES01 and MES02 and modelled kimberlite fissure widths to 20 level – Star diamonds (Retter and Snowden, 2003). Figure 7 of le Roux (2017)

Beers first evaluated, and then mined, the kimberlite. Arguably, it is this feature of the geology and mineralization that has encouraged the current operating company to steer clear of the more restrictive and prescriptive stock exchanges such as Toronto and Johannesburg.

In summary, the history of Letšeng is:

1957	Discovery by Peter Nixon
1959–1968	Artisanal diggers
1967	Famous 601 ct 'Lesotho Brown'
1968–1972	RTZ evaluation of main and satellite pipes; not viable, and option abandoned
1973–1974	De Beers one-year option
1975–1982	De Beers pre-production and production
1996	Letšeng Diamonds Company formed
2004	Letšeng Diamonds production commences
2006	Gem Diamonds buy effective 70% share in Letšeng Diamonds
2007	Gem Diamonds lists on London Stock Exchange Main Board.

In the twenty-first century, since the reopening of the mine in 2004, there has been significant new geological study of the Letšeng kimberlites, driven primarily by a need to understand more about the source and genesis of the large, high-quality Type IIA diamonds. Notwithstanding the importance of this work, the earlier interpreted distribution of the internal kimberlite phases has largely stood the test of time (*cf.* Lock, 1980 and Hetman *et al.*, 2017). This was important at the time the historical data was reviewed in 2005, and used in the Mineral Resource declaration for the Gem Diamonds prospectus when they listed on the London Stock Exchange (Stacey and Telfer, 2007).

The Main Pipe is dominated by K1 kimberlite with the later, centrally-disposed, higher grade K6 (the pipe within a pipe) and a small area of barren K4 hypabyssal kimberlite. The Satellite Pipe was mapped in 1980 with a single kimberlite phase that was petrographically similar to the Main Pipe K1.

In a confidential report to Gem Diamonds in support of their bid valuation, and in the Stacey and Telfer (2007) technical report included in the Gem Diamonds prospectus, the crucial sampling and production grade historical data was applied to understand and estimate a diamond grade for each of the kimberlite phases of economic interest. The Stacey and Telfer (2007) estimate has been declared as compliant with the SAMREC Code. In summary, the historical and current data used originated from:

1968–1972	RTZ sampling
1976–1979	De Beers sampling
1977–1981	De Beers production
2004	Letšeng Diamonds sampling
2004-2007	Letšeng Diamonds production.

While it might be contested as to whether the historical data could be used in this way, it must be further understood that the mine had been back in production for three years before the prospectus was prepared; thus the historical data and current production data could be integrated to achieve a fair and reasonable assessment as to the practical mining grade outcome to be expected in actual production. This integration was effectively duplication and validation. The De Beers production alone amounted to about 7 Mt from the Main Pipe and 2 Mt from the Satellite Pipe, producing in aggregate over 200 000 ct. Diamond production since reopening the mine in 2004 exceeded 5 Mt and 100 000 ct up till September 2006, the cut-off for the prospectus.

One should consider, in situations similar to this, the reputation of the companies involved. Stacey and Telfer (2007) stated that:

> '*The RTZ data was obtained by Letšeng Diamonds from the Department of Mines in Maseru and was also included in the information obtained by Letšeng Diamonds from De Beers. It is assumed that the sampling was accurately undertaken and that, as a consequence of strict security at all times, the results are reliable. Also, RTZ is an international mining company which was genuinely interested in exploiting the deposit for their own account. There has been no independent verification of the results.*'

Should this data have been used to estimate mineral resources? That may be a moot point, but the estimate has stood the test of time and new production.

Project examples of 'misuse'

Koidu, Sierra Leone

The history of diamond mining in the Yengema area of northeast Sierra Leone dates back to 1934 under the ownership of the Sierra Leone Selection Trust (SLST), part of the Consolidated African Selection Trust group that found the first diamonds in Botswana in 1959. The original discoveries were of alluvial diamonds but in 1948 kimberlite source rocks were also found, including Koidu Pipes 1 & 2 and Dyke Zone A. The further history of this diamondiferous kimberlite pipe is presented below:

1948	Discovery of Koidu by Sierra Leone Selection Trust (part of CAST)
1970	National Diamond Mining Company/SLST JV
1980	BP acquired SLST interest
1984	BP divested SLST interest
1986	Production decline and records incomplete
1991–2002	Civil war
1995	Branch Energy mining lease.

After the end of the civil war, attempts at rehabilitation of mining operations eventually led to ownership and production under Energem Resources Inc, a TSX company, and with Benny Steinmetz Group Resources Ltd and Magma Diamond Resources Ltd, both minority private equity partners. In 2005 a NI 43-101 report was prepared (Telfer, Clay, and Bloomer, 2005); this report provides the background for the review and comment described here, together with a 2003 report by the same consultant company (Clay, Bloomer, and Freeman, 2003) before the involvement of Energem.

The small Koidu 1 kimberlite pipe had a modest surface area of 0.45 ha but a good grade and diamond value. Open pit mining into the granite-hosted pipe was followed by a novel vertical pit mining technique that took full advantage of the competent wallrock. The open pit mining was supported by the 2003 resource report that developed an 'estimate' purely on the basis of historical estimates. Table I lists the historical estimates that were available in 2003, but not supported by the actual historical data, as had been the case with Letšeng at about the same time.

The independent consultant chose to declare Indicated and Inferred Diamond Resources to a depth of 300 m by the simple adoption of the NDMC/Outokumpu historical estimate. Clay, Bloomer, and Freeman (2003) argued that 'the mineral resource and reserve statement prepared by NDMC/Outokumpu (1988) is the most reliable. In the absence of any of the raw historical data,

Table I

Historical estimates for various resource and/or reserve categories (Clay, Bloomer, and Freeman, 2003)

Year	Company	Depth	Tons	Grade	Carats
1978	SLST	to 316 m	1.9 Mt	0.6 ct/t	1.2 million
1988	NDMC/Outokumpu	to 300 m	1.7 Mt	0.7 ct/t	1.1 million
1995	Branch Energy	to 305 m	1.9 Mt	0.7 ct/t	1.3 million
1996	WGM	to 305 m	1.7 Mt	0.6 ct/t	1.0 million

Venmyn has retained the tonnages and grades proposed but has reclassified the resources in accordance with the SAMREC Code and CIM Standards.'

A slightly different approach was adopted in 2005 when a resource diamond grade was estimated by averaging the historical (sampling grade) estimates (Telfer, Clay, and Bloomer, 2005), together with some current production data (Table II). Previously (Clay, Bloomer, and Freeman, 2003) it was admitted that little of the historical (raw) data was available except in summary form, and that no verification had been possible; there is no evidence that this changed for the 2005 report.

Although the report includes some qualifying comments on the process adopted, it is questionable whether this could be accepted as good practice or compliant with the CIM Standards under which the resources were declared. Be that as it may, the purpose here is to use these project reports to illustrate what is not acceptable or compliant under the current Code (SAMREC, 2016a). In addition to misuse of historical estimates and data, the Consultant has downplayed or ignored such basic requirements as:

- Bottom cut-off screen size
- Reasonable Prospects for Eventual Economic Extraction
- The fundamental issue of the absence of the raw historical data.

Lace, South Africa

The Lace (or Crown) Mine in the Free State of South Africa has a long history dating back to the nineteenth century. Although there was intermittent production until the global depression in 1931, closure at that time led to a period of inactivity and eventual sale to De Beers in 1939. Periodic technical review may have kept the project lukewarm, but the advent of the New South Africa encouraged De Beers to relinquish the project in a divestment programme perhaps aimed at economic empowerment. Ironically, the project (and its problems) was eventually sold to the Christian Potgieter Trust. The alluvial diamond background of that organization was not optimal for successful development. Attempts to option the property finally found DiamondCorp plc and the project was listed with that company on the AIM in London. This history is summarized thus:

1896	Discovery
1902–1907	First open pit production; sold in debt
1907–1918	Open pit operation but closure during WW1
1918–1924	Intermittent development and production
1924–1931	Open pit and underground production
1931–1939	Care and maintenance
1939	Sold to De Beers; no mining
1997	Sold to Christian Potgieter Trust.

Historical data from the early years of the mine was compiled and periodically reviewed by De Beers during the many years of quiet ownership by that company. Some attempts may have been made to use or integrate this data into our modern practices, but the reliability cannot be accepted for anything but a chronological narrative. Personal knowledge of some of the De Beers reports viewed in 1995, while a Canadian company employee, may have attributed some credence to the documents but, as a public company, disclosure of the information as anything other than history was not permitted, even before CIM Definition Standards and the SAMREC Code.

In 1997/1998, MPH Consulting of Toronto undertook geological drilling and microdiamond sampling work for Rupert Resources Ltd, a Canadian company that had optioned the property from the Potgieter Trust. It must be clearly understood that the results of this work are historical data as that phrase is defined in the SAMREC Code (2016a), because the data predates the SAMREC Code (2000). The fact that the work was commissioned by Rupert Resources, and not the current owner DiamondCorp, means that the ownership change also precludes using this data. However, there is good reason for using discretion in this latter regard as MPH Consulting has had a bridging relationship with the technical progress of the project, and was both the CP for Rupert Resources in 1998 and the DiamondCorp prospectus (Sobie, 2006).

Notwithstanding the use of discretion, this should be applied primarily for the core drilling that has provided a solid foundation for a geological model that has stood the test of time over the last decade, during which when underground access was been re-established and further drilling from surface and underground undertaken. However, the same cannot be agreed for the interpretation of microdiamond recoveries by Lennard Kleinjan,

Table II

Historical and current sampling and production data (Telfer, Clay, and Bloomer, 2005) used to estimate a weighted average grade, but excluding the two high-grade outliers

Source	Type	Tons	Carats	Grade (ct/t)
SLST (historical data)	Surface bulk sampling	23 362	14 080	0.60
	Surface bulk sampling	3 558	2 650	0.74
	K Shaft sinking	3 540	2 709	0.77
NDMC (historical data)	Surface bulk sampling	63 867	41 167	0.64
Koidu Holdings (current data)	Open pit mining	138 000	76 176	0.55
	Open pit mining (excl. grease)	76 619	34 679	0.45
Total tons and carats	**Average grade (high grades excluded)**	**301 848**	**166 102**	**0.55**

a De Beers geologist of good standing, and Luc Rombouts, an independent geologist who has contributed to progress in the mathematical background of microdiamond data interpretation.

Kleinjan's methodology, which was developed while he was employed by De Beers and working with Martinus Oosterveld, the first expert on microdiamond grade estimation, is purely visual. It depends on a comparison with microdiamond grades from known kimberlites and their macrodiamond production grades. While this was a useful first indication of potential grade (for internal use) and was developed into a semi-quantitative method by Oosterveld, it should not be relied upon for macrodiamond grade extrapolation.

Rombouts's methodology, while based on complex mathematical modelling, is in its simplest expression also a visual approach. The so-called 'extreme value statistics' is a plot of the cumulative diamond grade for samples of microdiamonds (possibly including macrodiamonds) from smallest to largest. Cumulation is for individual stones with measured weights, and the resultant plot is asymptotic to a maximum diamond grade that can be truncated to remove the stones below any chosen bottom cut-off size.

In practice this visual method is critically subject to the 'wagging tail of the dog' where the tail is the asymptote and the large-stone portion of the curve, where a single stone could seriously impact the interpretation either negatively or positively. For this reason, the technique has not been adopted across the diamond industry, whereas further development of the Kleinjan and Oosterveld method has gained ground and become standard practice. Oosterveld's work remains hidden in De Beers' internal reports but Ferreira (2013), in his thesis, has acknowledged the mentorship and knowledge legacy he benefited from in the development of current practice.

A report reviewing resources for the Lace project (Zweistra, 2012) discussed the historical estimates, the MPH-acquired microdiamond data, and results from a then-current underground bulk sampling programme. The mixed sources would always prove problematic in attempts at estimating a compliant diamond resource, but one aspect of the methodology needs highlighting to illustrate the fallibility in this specific case.

For the Main Pipe K6 tuffisitic kimberlite breccia the 2012 estimate was not able to use any of the current underground bulk sampling data for this phase because the sample was not 'representative'; it was mined from close to the pipe margin and contained excessive dilution. Fair comment. As an alternative, the historical estimates were taken and added to the grade from a tailings retreatment sampling campaign carried out between 2007 and 2009. Thus a grade of 24.4 cpht was 'guesstimated' and declared by adding a 'historically mined grade' of 16 cpht and a tailings grade of 8.43 cpht.

By 2016 a new estimate including microdiamonds properly analysed by Johannes Ferreira (Sobie *et al.*, 2016) estimated a much more robust grade for K6 of 10 cpht. Thus, the fallibility of the 2012 estimate was exposed. The only possible saving grace might be that the earlier guesstimate was stated at a bottom cut-off of 1 mm, and the later estimate at 1.25 mm, but it is doubtful this change of cut-off would have such a great impact on the grade.

For the Main Pipe K4 hypabyssal kimberlite the 2012 estimate could not be used to develop an estimation method as there were no historical estimates or current underground sampling results. An alternative method was conceived based on the historical (microdiamond) estimate, but not the Rombouts estimate, only the Kleinjan estimate. From the Kleinjan grades (Table III) for K4 and K6 a ratio of 2.33 (70/30) can be computed. Even though this ratio is meaningless, the number was used to determine a grade for K4 by multiplying the K6 grade discussed above by 2.33; thus 24.4 × 2.33 = 56.8 cpht.

No explanation was given as to why the Rombouts 'ratio' of 3.00 (90/30) was not used; clearly a higher K4 grade could have been evaluated, but a conservative approach was adopted.

Zweistra (2012) states that in their review of these estimates, Garner, Roux, and Noppe (2007) of Snowden Mining Industry Consultants note that 'the statistical validity of the two methods employed relies on knowing the relationship in the tested kimberlite between macro- and microdiamond populations, and that this is not available for Lace. Snowden therefore downgrades the validity of these estimates.'

Thus, we have an unreliable K6 diamond grade multiplied by a questionable ratio, based on low-confidence microdiamond grade estimates, used to guesstimate a K4 diamond grade.

The 2016 resource update (Sobie *et al.*, 2016) for K4, similarly to K6, again exposes the fallibility of the method with a new grade estimate of 40 cpht (*cf.* 56.8 cpht in 2012) even though the reduction is not as dramatic as for K6. The change in bottom cut-off is unlikely to explain the difference.

In addition to the fundamental issue of the reliability of the historical data and the concept of prediction *versus* estimation, the consultant has ignored the basic requirement for RPEEE.

Sobie (2006) noted that there are 'extremely positive indications of potentially mineable grades' based on the microdiamond results. That report made certain predictions but did not report any diamond resources. By 2012, using the same microdiamond data integrated with both historical estimates and current sampling data, DiamondCorp was able to proclaim diamond resources derived from a mix-and-match process of questionable industry practice.

Discussion and conclusions

Comparison of the two most familiar codes to the SAMREC Code may be useful. These are JORC (2012) in Australasia and the CIM Definition Standards (CIM, 2014) and National Instrument 43-101 (CIM, 2011) in Canada.

The Australasian JORC Code and CIM Definition Standards make no explicit reference to historical estimate.

Canadian National Instrument 43-101 contains careful and explicit guidance for the disclosure of historical estimates (subsection 2.4) that includes identifying the source and date, and commenting on the relevance and reliability, and other technical matters. The disclosure must include discussion of the work needed to bring the estimate in line with current practice, and provide appropriate qualifying statements to affirm that sufficient work to declare a Mineral Resource or Reserve has not

Table III

Microdiamond grade estimates for the 1997/1998 MPH consulting data (Table 7a, Sobie, 2006)

Kimberlite facies	Kleinjan estimate (cpht)	Rombouts estimate (cpht)
TKB – K6	30	30
HYP – K4	70	90

been undertaken and that the historical estimate is not being treated as a current Mineral Resource or Reserve.

There are also complex requirements where project ownership may have changed and there is a previous estimate that could be defined as a historical estimate. The new property owner may choose to disclose this historical estimate as an Exploration Target (subsection 2.3(2)); alternatively, if the previous owner is already an issuer, as that term is defined, the new owner may elect to disclose the estimate with the proviso that a new technical report will be filed within 180 days (subsection 4.2(7)).

In conclusion it appears that Australasia has not formally paid close attention to guidelines for historical estimates and data. Canada, however, has expressed similar guidance in its distinctive contrary manner.

While it may be desired company practice to achieve a declarable Diamond Resource as early as possible, and by whatever method that can be devised, when it comes to the use of historical estimates and data there is such a thing as good practice. As laid out in the scope of the SAMREC Code and Guidelines, it is intended to 'provide a required minimum standard for the Public Reporting of Exploration Results, Mineral Resources and Mineral Reserves'. These reports 'are prepared as information for investors or potential investors and their advisers'. With this in mind we must all be reminded of the liability we hold.

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Consideration for declaring a Mineral Reserve for TSF mining projects

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Synopsis

The mining of old tailings storage facilities (TSFs) or dams/dumps has become a common operation in South Africa. This practice has several interesting aspects that are different to normal surface mining operations. When considering the estimation of Mineral Reserves, the Competent Person must take into account the conditions of mining historical TSFs that are often situated in close proximity to communities. This paper discusses the Modifying Factors required to convert a Mineral Resource to a Mineral Reserve, such as mining sequence, dilution, mining losses, and environmental, social/community, and government factors. The paper also investigates the role that Inferred Mineral Resources may play in the life-of-mine plans for tailings dam projects.

Keywords

TSF, tailings, hydraulic mining, Mineral Reserve estimation.

Introduction

The mining of old tailings storage facilities (TSFs) or tailings dams (Figure 1) has several unusual aspects compared to the mining of most mineral deposits. The reason for this is that TSFs are not geological features but are man-made deposits similar to an alluvial deposit. However, the grade and distribution of the mineralization in the deposited tailings is generally a function of the historical metallurgical plant's design and processing efficiency, and no metallurgical plant is 100% efficient. Critical to the successful exploitation of a tailings dam is the efficiency of the modern metallurgical process to recover the mineralized material contained in the tailings, noting that modern processes are more efficient than most older processes.

Background to hydraulic mining

The use of water plus energy to mine unconsolidated material has a long history. The ancient Romans introduced the notion of large-scale mining with a technique known as hydraulic mining. This method, developed in 25 AD, required large tanks of water and aqueducts to supply the water to the tanks.

Figure 1—Typical tailings storage facility (TSF) (DRDGold IAR, 2017)

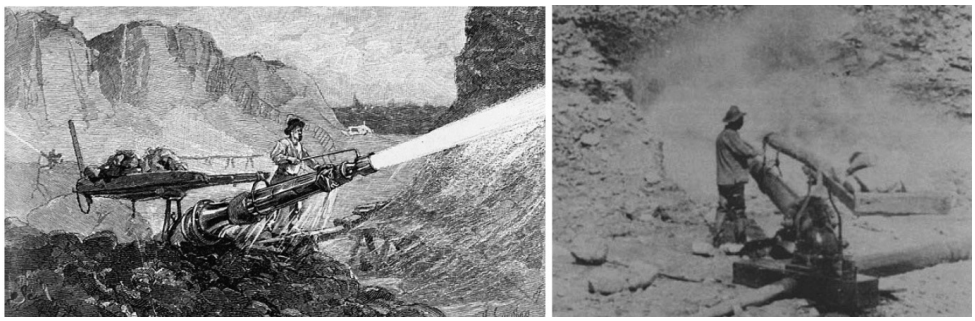

Figure 2—An example of hydraulic mining in the California gold mining operation (Anon., 2016)

When the water was released from the tanks, huge volumes of water washed away or broke the rocks, exposing the bedrock and any gold material (Anon., 2016). The best-known use of hydraulic mining is in the California Gold Rush, which was documented and provides physical evidence of the widespread and sophisticated use of hydraulic mining in the mid-19th century (Figure 2). Thousands of kilometres of ditches and flumes were constructed to gravitate water from high in the mountains at sufficient pressure to flush the alluvial gravel beds into sluices. It is reported that an 8-inch (200 mm) monitor could deliver 185 000 ft^3 (5240 m^3) of water with a velocity of 150 ft per second (45.7 m/s) (Anon., 2016).

In recent years, hydraulic mining (Figure 3) has been used for the extraction of unconsolidated mineralized materials, alluvial deposits, freshly blasted ores, and the recovery (or re-mining) of dewatered tailings dams/storage facilities. Hydraulic mining can be loosely defined as the excavation of material from its *in-situ* state using water at high pressure. A stream of water is directed at the tailings with the purpose of mechanically breaking and/or softening the mineralized material so that it can be carried away by the water flow. The application or effectiveness of the method is a function of a variety of factors, ranging from the size, velocity, and pressure of the water stream to the location, hardness, particle size, and moisture content of the material being mined.

The effectiveness of hydraulic mining is a result of decades of successful reclamation and re-deposition operations. The method entails high-pressure water delivered to the reclamation sites by a network of pipelines to the reclamation faces.

Modifying Factors

Establishing Mineral Reserves is important for mines and projects as the ability to declare a Mineral Reserve demonstrates the financial viability of a mine or project (Rupprecht, 2016a). The conversion of Mineral Resources to Mineral Reserves requires Modifying Factors to be considered. Notably, TSFs are man-made facilities used to store the residue material from metallurgical processing plants. If the contained mineralized material is found to have Reasonable Prospects for Eventual Economic Extraction (RPEEE) based on improved metallurgical recovery techniques, original sulphide material partially oxidized making recovery possible, and/or significant increases in metal prices, a Mineral Resource can be declared. Typically, TSFs are low-grade deposits that require a high volume/low-cost operation to enable viable mining. In some instances, government authorities have requested mining companies to remove old TSFs in areas deemed high risk to the general population as a means to eliminate the dangers of dam failure, dust pollution, and other socially unacceptable factors.

Mining method

Typically, the pressure used for hydraulic mining is 3000 kPa to 4000 kPa (30–40 bar) at the pump station, reducing to about 2400 kPa bar at the gun or hydraulic monitor (monitor gun). It should be noted that for well-graded, non-cohesive (sandy), coarser tailings that are often found along the tailings dam wall, the hydraulic water pressure should not be less than 3200 kPa at the monitor gun (Fraser Alexander, 2002).

Depending on production rates required, multiple monitor guns will be in operation and the high-pressure water delivered to the reclamation area will generally be distributed by 250 mm diameter pipes with water reaching the monitor guns at a pressure of approximately 3000–4000 kPa. The monitor guns concentrate and direct the high-pressure water to the reclamation

Figure 3—Examples of hydraulic mining of a TSF/tailings dam (DRDGold IAR, 2017)

Figure 4—Gully used to transport slurry (Fraser Alexander, 2012)

focus area to pulp the tailings into a slurry and maintain slurry densities between 1.40 t/m^3 and 1.46 t/m^3. Each monitor (150 mm) can excavate approximately 4000 t of tailings per day.

The slurried tailings material is broken up and mixed with water by the monitor guns, which enables the slurry to flow under gravity along channels or gullies (Figure 4). Finger screens are used in the gullies to remove coarse (+50 mm) material. The slurry or run-of-mine (RoM) material gravitates to the pond pumps or sump pumps. These pumps then pump the slurry over vibrating screens, fitted with 2–3 mm aperture polyurethane screen panels, to remove any fine debris that may impact the pumping operations (Figure 5). The slurry is then pumped to the slurry tank and from there to the processing plant. Hydraulic mining is not labour-intensive and can produce 100 000 t/month with only a few workers.

Slurried tailings tend to flow at a natural beaching angle that is generally self-correcting. If the slope is too steep, the flow velocities increase in the channels, causing erosion until the equilibrium slope is attained. The tailings material consists of fine tailings slimes with a typical particle size of 80% passing 150 µm. Relatively flat flow channels are developed, with gradients in the order of 1:100 or flatter (Figure 4).

The mining of old or defunct TSFs differs from other mining operations in that once the decision is made to mine the TSF, the mining needs to be completed in totality. As the TSFs are frequently in or close to populated areas, their removal can free up valuable land and eliminate an environmental nuisance, especially dust.

The final stage of the mining of a TSF is floor cleaning, where the hydraulic gun is on the natural ground level, washing the shallow depth of tailings away from the gun. Floor cleaning is conducted until too much stone enters the ROM feed (blocks the screens too often). The final ore is collected during final clean-up of the tailings dam floor at the end of the life of the operation, after which the tailings dam is rehabilitated.

Figure 5—Example of vibrating screen to remove debris from the slurry (Fraser Alexander, 2012)

Cut-off grades

A break-even cut-off grade should be used based on establishing the average grade of the tailings dam. Although historical TSFs are generally auger drilled and sampled, allowing the construction of a 3D block model, it is common practice to apply an average grade (pay limit). The reason for this is that TSFs are re-mined and processed in their entirety, and no selective mining is possible. Therefore, the grade of the entire tailings dam must be used to determine its economic viability. Another reason that the average mining grade is used in the cut-off estimation is that mining is conducted from several faces, making reconciliation more difficult. Grade reconciliation is further complicated when ore is sourced from a number of TSFs, thus making it difficult to accurately reconcile grades on a monthly basis. Experience based on the DRDGOLD Limited operations indicates a very close correlation between planned and actual grades on an annual basis (Labuschagne, 2019).

Mining loss and dilution

As the entire tailings dam is mined, inclusive of the rehabilitating of the original footprint, it is common practice in South Africa not to apply a mining loss or mining dilution. Accordingly, no dilution or mining loss is applied in the process of converting Mineral Resources to Mineral Reserves. Furthermore, due to the size of TSFs the potential dilution and mining losses are small and would not have a material impact on the Mineral Resource to Mineral Reserve conversion.

Geotechnical

TSFs in South Africa have been successfully exploited since 1977, and the geotechnical characteristics are well understood from practical experience. A safe bench height is dependent upon the material strength, which is influenced by the phreatic surface within the dump (The term 'phreatic surface' indicates the location where the pore water pressure is under atmospheric conditions, *i.e.* the pressure head is zero. The phreatic zone, below the phreatic surface where rock and soil is saturated with water, is the counterpart of the vadose zone, or unsaturated zone, above). As no open-pit mining is taking place, the mine design does not account for slope angle but rather the natural angle of repose from hydraulic mining. As most TSFs have been dormant for many years the phreatic surface is below the surface of the dry tailings material and therefore the risk of encountering saturated zones or phreatic surfaces that may cause slope failure is low. To ensure the competency of the wall, an angle of 45 to 35 degrees (Figure 6) is typically used. This angle is based on years of experience of working with dry tailings.

Figure 6—Mining at a 45° angle (Garling and Prentice, 2010)

Metallurgical processing and recovery

An understanding of the recovery of any metallurgical process is essential. In the case of mining mineralized TSFs, the recovery is low, typically in the order of a 40% to 50%, noting that not all tailings respond favourably to flotation. Like all mineral projects, metallurgical test work should be conducted. In the case of mining tailings dams, the project should include metallurgical test work during the evaluation drilling programme.

Project infrastructure

Unlike most surface mining operations, tailings operations are dependent upon the ability to transfer mineralized material from the dams to the beneficiation plant as a slurry. Key points are the availability of servitudes and surface rights for slurry and water pipelines, including road crossings. The availability of power and water is critical to establishing the viability of the project. Both these items are significant when considering operating costs and recovery.

Capital and operating costs

Capital expenditure for general infrastructure must be considered in the process of converting a Mineral Resource to a Mineral Reserve. The availability of a suitable processing plant is critical, and this must be adequately designed and costed. For existing operations planning to re-treat tailings material, this may not be a critical issue. However, for a new entrant, the capital cost of a dedicated beneficiation plant could be very high, and in some cases the capital requirements may make the project uneconomic. As most companies make use of specialized contractors to conduct the mining operations, the mining-related infrastructure requirements are often included in the contractor's rates.

In terms of operating costs, sand dams involve a higher cost than slimes, since sand can wear the pipelines quicker than slimes. In some cases, sand can be added to slimes to improve the flow. In addition to the operational issues, a further reason for the increase in the operating cost is that sand must be milled to approximately 80% less than 75 µm while the slime, although coarser, can be treated in the carbon-in-leach tanks directly. Total mine operating costs range from R50 to R200 per ton (2019) with R85 per ton being a reasonable average. Operating costs are inclusive of the following:

- Labour
- Consumables
- Electricity
- Water
- Rehabilitation
- Corporate
- Other.

Governmental, legal, and permitting

The Competent Person (CP) must ensure that all legal aspects, mining rights, surface agreements, servitudes, and permits are in place to justify the declaration of a Mineral Reserve. The CP must also ensure that there are currently no legal actions/impediments that would prevent operations on any of the current mineral rights.

Social, environmental, or community impact

One of the biggest concerns by communities is the nuisance created by dust from the TSFs, which can impact on those living within the mining areas of influence. The re-mining of TSFs is unique in that, unlike other mining projects, the exploitation (rehabilitation) of a historical TSF can improve the local environment, as depicted in Figure 7. Many of the TSFs that are subject to re-mining are historical and due to their age, many are surrounded by towns and communities. For example, Johannesburg, Benoni, and Boksburg have expanded beyond their original city limits with communities now living adjacent to these old TSFs.

Figure 7—Sequence of rehabilitation through mining (left to right, tailings dam, re-mining site, industrial site) (DRDGold IAR, 2017)

Figure 8 – Example of a rehabilitated dam site

Importantly, the mining of the TSFs opens up large tracts of land which, once properly rehabilitated, can be used for other purposes such as industrial sites or local housing. Furthermore, as TSFs are completely mined it is important that the tailings dam footprint is rehabilitated, closure is certified, and the site released for land redevelopment purposes as soon as practically possible. The final clean-up is a time-consuming and costly activity, and because of this, many sites where mining has been completed are not fully rehabilitated. Travelling around the greater Johannesburg area, one can observe numerous sites requiring final rehabilitation and closure approval.

Figure 8 provides an example of a tailings dam site that has been rehabilitated.

Environmental management is a crucial aspect during the project planning phase of new mining sites. Key impacts from the mining of TSFs include the following.

- Air quality can be affected in that mining activities can increase dust levels in the vicinity of the operations.
- Surface water can be affected by slime spillage from pipelines and pump areas, which could impact on local water quality.
- Mining of some TSFs may promote acid mine drainage as minerals in the tailings are oxidized.
- Noise due to mining operations and associated activities may be a nuisance to local communities.

As part of the Environmental Impact Assessment, operations are required to identify and mitigate anticipated impacts. During the Mineral Resource to Mineral Reserve conversion process, the CP must ensure that environmental aspects are well managed and that the company has addressed all legal and environmental compliance requirements. The CP must identify all sensitive areas or environmental factors that could materially affect the tailings project and ensure that these have been addressed or efficiently mitigated. Finally, the CP must ensure all necessary permits are in place or that there is a reasonable basis to believe that all permits required will be obtained.

Market studies and economic considerations

As with any mineral project for which a Mineral Reserve is to be declared, the market conditions must be understood, especially the cyclical nature of commodity prices as mining of TSFs can be a lengthy (medium-to-long-term) process. Mining engineers also need to account for the fact that TSFs are marginal deposits that require sufficient Mineral Resources mined at an appropriate production rate. Most require production of the order of 300 000 t/month to establish sufficient economies of scale and lower operating costs enough to make mining economically viable. For example, many gold tailings dumps found around Johannesburg have typical grades of 0.3 g/t or lower. These dumps are sensitive to revenue drivers such as grade, recovery, and the gold price, and a material decrease in any of these revenue drivers can have a direct negative impact on the viability of the operations. Therefore, it is crucial that the projected commodity price used in the Mineral Reserve estimation be sufficiently conservative to support mining operations and account for the cyclical changes in commodity prices for the full duration of the project.

Once a tailings operation commences it is often very difficult to halt operations during periods of low commodity prices. This is due to the fact that mining operations disturb previous environmental safeguards such as grass cover, water management, *etc*. The premature stoppage of a tailings project will cause the TSF to be exposed to the natural elements, which may increase the risk of dust, acid mine drainage, and other hazards. Some partially mined dumps around the Johannesburg can attest to this, notably the partially mined sand dumps in Krugersdorp.

Risk assessment

Risk assessment is a vital factor to be considered when

Table I

Key inputs

Geochemical and physical characteristics (Steenkamp,2016)	Grain size (Steenkamp, 2016)
Homogeneity (Steenkamp,2016)	Level of oxidation (Steenkamp,2016)
Penalty elements (Steenkamp,2016)	Grade or percentage of metal
Size of Mineral Resource	Specific gravity
Metal price	Process recovery
Access to reliable power and water supplies	Infrastructure, surface rights, and servitudes
Operational costs – including the cost of electricity and cost and supply of water	Social license to operate
Governmental compliance	Environmental compliance
The proximity of communities and informal settlements	Regulatory compliance

converting Mineral Resources to Mineral Reserves, but its importance is often underestimated by mining companies. The risk assessment should include all technical specialists involved in the mining operations to identify aspects that could impact not only on the project achieving its business plan, but more importantly those aspects that may affect the estimation of the Mineral Resource and the declaration of a Mineral Reserve.

The mining of TSFs does embody risks often not usually associated with standard surface mining. TSFs are high-tonnage low-grade deposits that are revenue-sensitive (metal price, recovery, grade). Technical aspects that that may influence the declaration of Mineral Reserves are listed in Table I.

Inferred Mineral Resource

Owing to the nature of old TSFs, it is often not feasible to conduct exploration on the full footprint of the tailings dam due to external issues, such as municipal waste, dense vegetation, or water ponding. As previously discussed, the very nature of mining TSFs requires the mining of the full deposit, regardless of the classification. Leaving material or Mineral Resources that are not upgradeable to Mineral Reserves would only negate the positive attributes of the mining process. This would in fact create far worse environmental conditions as the remaining TSF would be affected by the natural environment (rain and wind) and adversely impact the local environment. Although it is best practice to limit the amount of Inferred Resources in a mine design (Rupprecht, 2016b), it may be necessary to include Inferred Mineral Resource in the mine plan. Under the SAMREC Code 'It is accepted that mine design and planning may include a proportion of Inferred Mineral Resources, albeit under certain reporting conditions'.

However, it is interesting to note that in many international reporting codes, the use of Inferred Mineral Resources in the life-of-mine plans is prohibited, and is allowed only in the reporting in Scoping Studies (Clause 38 – JORC Code, 2012) or Preliminary Economic Assessments (NI 43-101). For example, the JORC Code (2012) provides the following guidance:

'Confidence in the estimate of Inferred Mineral Resources is not sufficient to allow the results of the application of technical and economic parameters to be used for detailed planning in Pre-Feasibility or Feasibility Studies.'

In the author's opinion, the inclusion of Inferred Mineral Resources is a perfect example of where the CP should be allowed to use his/her judgement to include Inferred Mineral Resources into the mine plan, as:

- The tailings dam consists of material produced from metallurgical plants known to yield less-than-perfect recovery. Therefore, there is a high probability of continuous mineralization and homogenous grade.
- In general, the mining method does not support selective mining.
- Once mining commences the tailings dam should be removed in its entirety.

Therefore, it would not be unreasonable to consider the inclusion of Inferred Mineral Resources in a TSF re-mining project. However, one should take into account and comment on the risk associated with the inclusion of Inferred Mineral Resource in the mine plan.

Conclusion

The mining of TSFs, and therefore the declaration of a Mineral Reserve, is unlike most mineral projects in that the deposit is man-made and its characteristics depend upon the deposition characteristics of original TSF design and operation. Furthermore, unlike natural deposits, TSFs are above ground, often in close proximity to housing, communities, and other businesses, and therefore the removal of the TSF is often viewed as removing an environmental nuisance, *i.e.* dust, source of acid mine drainage, as well as opening up ground for urban development. Once mining commences, the grass cover and other environmental protections are disturbed, and the mining project must be completed. Therefore, understanding the sensitivity of the pay limit of the dumps in the life of mine and the long-term metal prices is essential, as one does not want to leave a partially mined tailings dam.

Owing to the environmental issues surrounding tailing projects, once mining commences the project should be completed so that the area can be returned to the landowner in as close to the original condition as possible.

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Measurement of air and ground vibrations produced by explosions situated on the Earth's surface

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Synopsis

Most equations used to predict the ground motion produced by explosions were developed using confined blasts that were detonated for breaking rock in mining or tunnelling. Ground motion is usually recorded by geophones or seismometers. The air blast produced by open-pit blasts and explosions on the surface can pose a significant risk, thus microphones and pressure gauges are often also used to monitor the effects of the explosion. The aim of this study is to determine whether the predictive equations developed for confined explosions can be used to predict the effects from explosions on the surface, with appropriate adjustments to the various coefficients.

Three predictive equations developed for buried explosions were tested. The study shows that the US Bureau of Mines peak particle velocity (PPV) predictive equation is the most reliable. In addition, a predictive equation that uses the secondary atmospheric shock wave phenomenon also produced good results, and uses the scaled delay time parameter, which is easier to measure. These equations may be utilized for demolition sites, where old and potentially unstable explosives and obsolete equipment are destroyed on the surface, and for assisting in forensic seismology to determine the details of an unexpected and unknown explosion.

Keywords

surface explosions, prediction, demolition, PPV, secondary shock wave.

Introduction

Military demolition sites are areas where old and potentially unstable explosives and obsolete equipment are destroyed. Typically, the ordnance is placed on the ground surface and detonated. Ideally, the area should be located far from any human activity or dwellings, and is cleared prior to use. However, some demolition sites have been encroached upon by human settlements. Consequently, modelling and monitoring of the ground vibrations and air blasts have become a priority to ensure that the blasts do not cause irritating disturbances to the local inhabitants or damage to structures, which may result in protests and legal challenges. Equations that could be utilized to predict the effects of the demolition activities on the surrounding population and structures would be useful in mitigating these risks.

Using a 'mixed' content of explosives on the surface in order to determine whether predictive equations can be utilized would not only assist with determining the effects of explosions from a military demolition range on the surrounding infrastructure, but can also assist with forensic investigations of unexpected explosions. Instrumental tests with surface explosions ranging up to 25 t have not been conducted or reported in South Africa, and thus this study aims to provide insight into this novel aspect.

The main features of the study are:

- Vibration measurements of large surface explosions (up to 25 t)
- Applicability of the USBM PPV equation to long distances (29 km), which is far beyond the usual measuring range for blasting vibrations
- Measurement of the delay between the main and secondary air shocks at equally long distances
- Possible relevance of the vibration analysis to truck bomb explosions and to forensic analysis of large unplanned explosions.

Blast monitoring

Blasts or explosions on the Earth's surface, such as at demolition ranges, create seismic waves that

travel through the ground and overpressure (air pressure waves) through the atmosphere, both of which can shake buildings and disturb people. Together, these phenomena are referred to as seismoacoustics (Ford *et al.*, 2014). The Comprehensive Nuclear Test Ban Treaty Organization measures this phenomenon with infrasound and seismic stations in their international monitoring system, in order to monitor nuclear explosions both above and below the Earth's surface.

Typically a blast is characterized by the nature of the explosive material utilized, whether there was any reflection (due to geology, topography, atmosphere, *etc.*), and the extent to which the blast was contained. Overpressure generally occurs in the case of explosions on or above the surface of the ground (Williams and Newell, 1991), and in some cases the overpressure can be recorded up to 2 seconds after the seismic waves when the explosion is approximately 1 km from the sensors.

The recorded ground motion from blasts is not simple to characterize due to the radiation pattern from the site of the explosion, which is affected by geophysical factors such as (1) geometrical spreading, (2) separation of compressional shear and surface waves, (3) reflection, refraction, diffraction and scattering, and mode conversion, and (4) frequency-dependent attenuation (New, 1991). In addition, it is influenced by aspects such as the mass of explosive and the distance from the explosion (Puri and Prakash, 1991). Similarly, different types of detonations and the prevailing weather conditions, such as temperature gradients, wind direction, and cloud cover, can affect the overpressure. These phenomena, along with the features of the structure (such as the foundation) make accurate predictions of nuisance or damage difficult.

In order to encompass the complexity of the blasts, charge weight and distance are combined to produce a parameter called 'scaled distance', which generally uses the square root of the maximum weight of the charge for predicting ground vibration and the cube root for overpressure. Thus, the scaled distance is a combination of the maximum weight of explosive used in generating the vibrations through the ground and air and the distance. The 'cube root scaled' distance is the distance from the blast to a defined position divided by the cube root of the maximum weight of explosives. Scaled distance is similarly calculated; however, the distance is divided by the square root of the maximum weight of explosives in any 8-millisecond period (Joint Committee on Administrative Rules, 1998).

Kahriman (2002) and Ozer, Kahriman, and Aksoy (2008) examined a number of predictor equations proposed in the literature, and concluded that, owing to the large number of complications related to surface blasting, a suitable generic formula has not yet been determined. In addition, Ford *et al.* (2014) found that there is less information available on seismic motions from surface explosions than on underground explosions.

Prediction equations for underground explosions

There is extensive literature describing prediction equations for the vibrations induced by blasting in open pit mines and during tunnelling, for example Aloui *et al.* (2016), Bongiovanni *et al.* (1991), Khandelwal and Singh (2007); Kahriman (2002), Ozer (2008), Ozer, Kahriman, and Aksoy (2008), Milev *et al.* (2016), and Puri and Prakash (1991). The most common approach is to plot the variation of PPV with scaled distance, especially since PPV is not as sensitive to changes in geological conditions as acceleration and displacement and provides an excellent criterion for assessing the damage (Kumar, Choudhury, and Bhargava, 2016; Puri and Prakash, 1991). Kumar, Choudhury, and Bhargava (2016) and Gupta and Hartenberger (1981) summarized the vast number of prediction equations developed since 1959, when they examined methods of creating a model that can address the rock properties of the sites when using predictive relationships. Koper *et al.* (2002), Bonner *et al.* (2013), and Yan, Jiang, and Duan (2009) noted that seismic waveforms are more complex than atmospheric acoustic waveforms due to the geological structure of the ground, and thus the scaling laws will be site-specific.

Ozer, Kahriman, and Aksoy (2008), Ozer (2008), Khandelwal and Singh (2007), and Bongiovanni *et al.* (1991) used three PPV-based predictor equations chosen from a number of equations proposed in the literature: (a) the US Bureau of Mines (USBM) equation, (b) the Ambraseys-Hendron equation (Ambraseys and Hendron, 1968), and (c) the Langefors-Kihlstrom equation (Langefors and Kihlstrom, 1963) to establish a relationship between PPV and scaled distance (R_s) (Table I). The USBM equation uses the square root scaled distance, Ambraseys-Hendron uses the cube root scaled distance, and the Langefors-Kihlstrom uses a different formula for scaled distance. Thus, these three equations represent a good sample of the typical predictive equations available.

where

PPV = peak particle velocity (mm/s)
R_s = scaled distance
K = the ground transmission coefficient
β = the specific geological constant
R = distance from explosion to station (m)
W_d = maximum charge per explosion (kg)

Prediction equations for explosions on the ground surface and in the air

When a high explosive detonates in the atmosphere, an increase in pressure, known as the blast wave, is created (Williams and Newell,1991). The blast wave moves away from the centre of the blast in a spherical wave front, the peak overpressure of which subjects an additional pressure to any building it encounters.

Table I

PPV predictive equations used by Ozer (2008), Ozer, Kahriman, and Aksoy (2008), and Puri and Prakash (1991)

	USBM	Ambraseys-Hendron	Langefors-Kihlstrom
PPV prediction equation	PPV (mm/s) = $K(R_s)^{-\beta}$ ($\log PPV = -\beta \log R_s + \log K$)	PPV (mm/s) = $K(R_{s\,A})^{-\beta}$ ($\log PPV = -\beta \log R_s + \log K$)	PPV (mm/s) = $K(R_{s\,L})^{\beta}$ ($\log PPV = \beta \log R_s + \log K$)
R_s equation	$R_s = R/(W_d)^{0.5}$	$R_{s\,A} = R/(W_d)^{0.3}$	$R_{s\,L} = (W_d/(R)^{0.6})^{0.5}$

$R_{s\,A}$ = scaled distance for Ambraseys-Hendron equation
$R_{s\,L}$ = scaled distance for Langefors-Kihlstrom equation

Different types of detonation and weather condition can affect the overpressure, which may cause complaints usually attributed to ground vibration. The energy released by a surface explosion varies with the soil conditions as well.

Bonner *et al.* (2013), Gitterman and Hofstetter (2012), and Gitterman (2013) monitored calibration explosions on the ground surface at a military range in Israel, which they recorded with high-pressure sensors, accelerometers on the surface, buried seismometers, acoustic gauges, and high-speed cameras. Using the seismometer data, they developed a unique empirical scaled relationship for the secondary shock (SS) delay for ANFO (ammonium nitrate/fuel oil, which is a widely used bulk industrial explosive) charges and various distances. The obtained relationship can be used as a new yield estimator.

The SS delay phenomenon (Figure 1) occurs when the air blast wave for an explosion source displays a number of recurrent shocks of smaller magnitudes and at different times, caused by successive implosion of rarefaction waves (the region of low relative pressure following a shock wave). It is important to note that these shock waves are 'supersonic', *i.e.* travel at a speed greater than the normal velocity of sound in air (Cullis, 2001). These air waves may produce a signal in seismometers. A higher pressure shock front travels faster, therefore the time delay between the main shock (MS) and the SS phase increases with distance, as well as with the amount of explosive. Thus longer SS delays are observed for longer distances, as well as for larger amounts of explosives. Bonner *et al.* (2013), Gitterman and Hofstetter (2012) and Gitterman (2013) noted that the ambient temperature and pressure (altitude) would lead to only minor changes in the air blast parameters and thus did not consider them in the distance and SS delay scaling relations. The empirical relationship developed for SS delay (the time between secondary shock and main shock arrivals) is dependent on the charge and distance, as are standard air blast parameters, and also on the type of explosive (velocity of detonation).

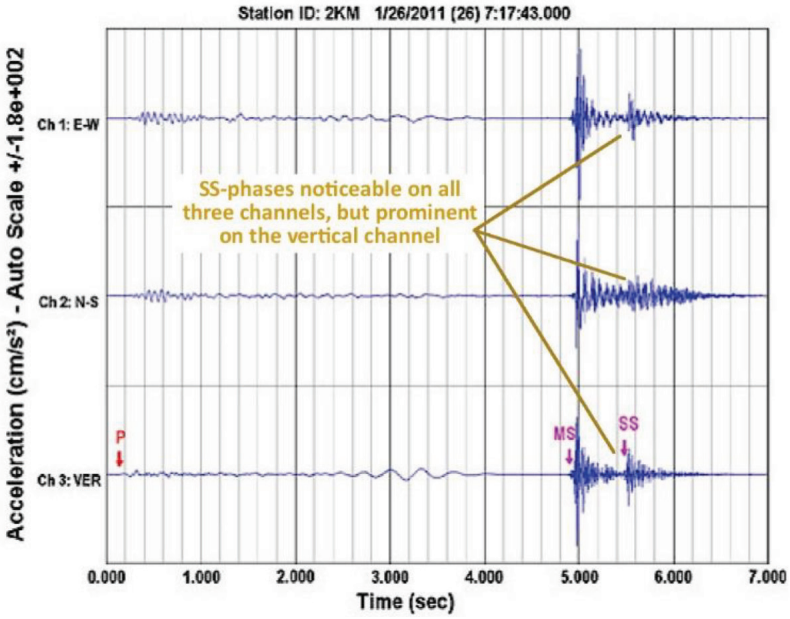

Figure 1—Waveforms recorded by a three-component accelerometer, modified from Gitterman (2013). P – primary seismic wave, MS – main shock, and SS – secondary shock

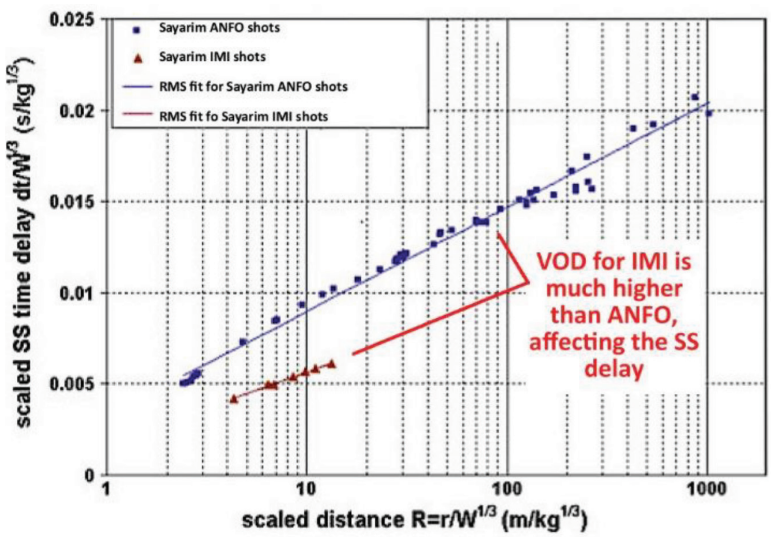

Figure 2—Comparison of ANFO and IMI explosives and the effect on the SS delay, modified from Gitterman (2013)

Gitterman (2013) noted that the SS delay increased with the amount of explosives used and with decreasing velocity of detonation (VOD) of the explosives (Figure 2). This is due to the time taken to completely detonate the finite charge (the tests compared ANFO with IMI, which is a military explosive). However, Gitterman acknowledged that there may be other explosive parameters that can affect the delay. This new method may be less accurate than that of the traditional procedures based on high pressure measurements, but it is significantly less expensive and, in the case of an accidental explosion, may be the only method available. It should work well for determining the relative sizes of two charges of the same explosive.

There is very little literature available on utilizing seismograph systems to determine prediction equations for atmospheric waves created by explosions above or on the surface of the Earth. However, the SS delay is a unique parameter that can be measured by both pressure gauges and seismograph systems and could prove very useful for prediction if records from either of the sensors are not available.

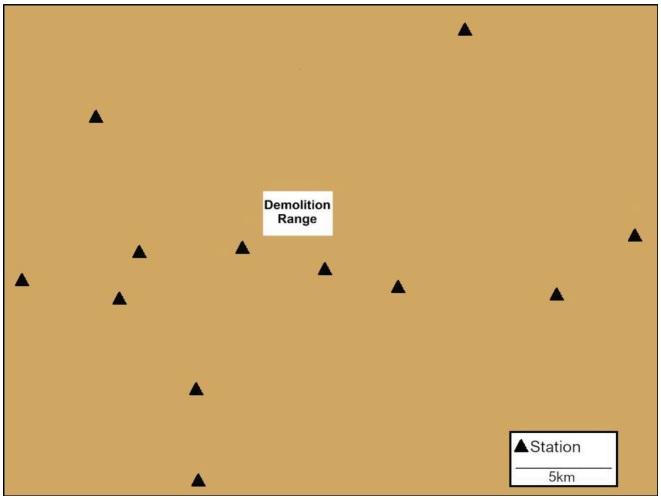

Figure 3—Schematic of a typical station distribution in the vicinity of the demolition ranges at each site

Methodology

Monitoring of seismic signals

The purpose of this study was to perform an investigation similar to those by Aloui *et al.* (2016), Bongiovanni *et al.* (1991), Khandelwal and Singh (2007), Kahriman (2002), Ozer (2008), Ozer, Kahriman, and Aksoy (2008), and Puri and Prakash (1991).

Five different military demolition ranges (the local geology is briefly described in Table II) were monitored by a mobile network of seismometers in order to develop an empirical relationship for the prediction of PPVs from on-surface blasting, which can then be used for other demolition ranges with similar properties. The results from the three different vibration prediction equations given in Table I were then compared in order to determine the most reliable equation.

The blasting was monitored using a temporary mobile network of between five and twelve stations consisting of a combination of 4.5 Hz triaxial geophones and 1 Hz triaxial seismometers installed on the ground surface (Figure 3) at varying distances from the source, with the furthest stations being 29 km distant for site 5. The number of stations used was dependent on the local site conditions and accessibility.

The amount of explosive used and the distances between the source and stations were carefully recorded. The weight of the ordinance was recorded as net explosive content (the total mass of the enclosed explosive material, without the casings/shells) that was heaped on the surface of the ground and detonated.

A linear RMS fit regression curve of the logarithm of PPV against the logarithm of R_s was obtained for each of the explosions and each of the stations. In order to find the site constants, the graph of PPV values *versus* R_s was plotted for all predictor equations, along with the 95% mean prediction interval. The coefficient of determination (R^2) was calculated as this is generally accepted as a basic measure of the quality of fit.

In Table I, the constants K and β are site constants that vary between sites and are dependent on the nature of the site geology and the type of explosive. K (the ground transmission coefficient) represents the line intercept at $R_s = 1$ on the log-log graph and

Table II

Geological descriptions of each of the sites

Site	Geology
Site 1	Situated on the Tierberg Formation, Ecca Group. The Tierberg Formation is a blue-grey to black shale with carbonate-rich concretions, subordinate siltstone, and sandstone in the upper part. Just south of the site is the contact with the younger Abrahamskraal Formation, Adelaide Subgroup, Beaufort Group. The Abrahamskraal Formation is predominantly a blue-grey silty mudstone, with subordinate brownish-red mudstone and sandstone. Dolerite is abundant and overlays much of the area. Some Tertiary to Quaternary calcrete was delineated and alluvium covers most of the drainages.
Site 2	The area is mostly underlain by volcanic and sedimentary rocks of the Ventersdorp Supergroup, shale of the Prince Albert Formation of the Karoo Supergroup, and Quaternary sand, calcrete, and alluvium. Most of the central, southern, and eastern portions are situated on a low plateau underlain by the volcanic rocks. The remaining western portion forms part of the relatively lower lying, eastern side of the Vaalharts valley, which is predominantly underlain by sand, calcrete, and shale.
Site 3	The area is underlain by Karoo Supergroup rocks developed in the Roedtan sub-basin. The rock units dip shallowly towards the west. The surface rocks are dominantly fine-grained, locally amygdaloidal basalts of the Letaba Formation. However, at this southern end of the basin, these volcanic rocks are not thick and this, together with the proximity to faulted basin margins, has resulted in areas of the underlying fine-grained reddish sandstones of the Clarens Formation being exposed. In rare instances even the shales of the Irrigasie Formation below the sandstones crop out, but their extent is very limited. There are localized occurrences of clay (attapulgite) and gypsum associated with weathered basalts.
Site 4	The site lies on Hout River Gneiss, a metamorphosed granitic rock which includes remnants of greenstone belt material. The structural trend of the gneisses and greenstones is variable, but on the scale of kilometres they can be regarded as being homogeneous. Diabase dykes are prevalent in the area, and trend in a northeasterly direction. As is typical for the region, the area is essentially sand-covered and outcrops of gneissic rock are scarce.
Site 5	The site is underlain by the Proterozoic Groblershoop Formation, the Wilgenhoutsdrif, Koras, and Nama Groups; and rocks of the Phanerozoic Karoo Supergroup. Aeolian sand of the Gordonia Formation of the Kalahari Group covers about 90% of the area. Karoo dolerite and kimberlite occur in the area and structural deformation, seen as folding, is restricted to the Proterozoic rocks.

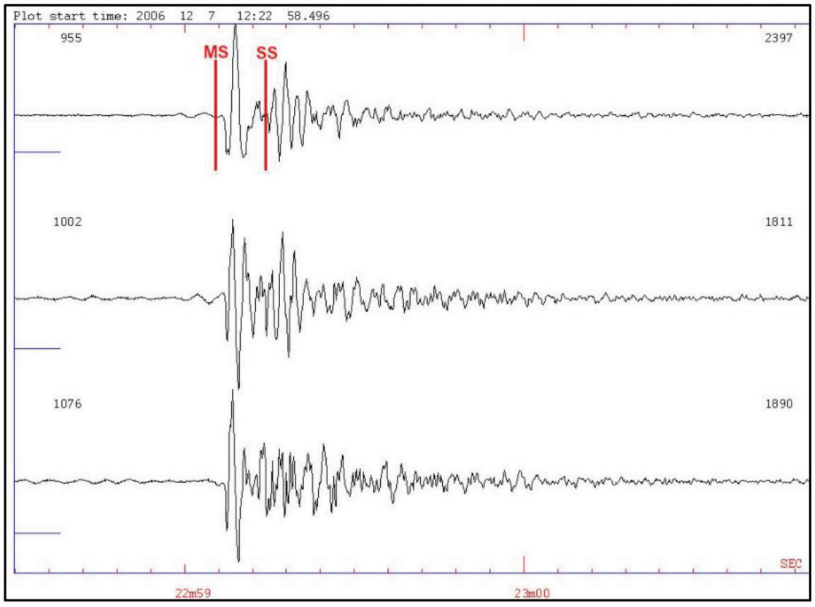

Figure 4—An example of the signal registered on all three components at a station 4.89 km from the demolition range. The P-wave arrival was not recorded but the main shock (MS) and secondary shock (SS) were recorded

is the initial energy radiated into the immediate rock and thus is reliant on the features of the medium and the explosive used. β (the specific geological constant) is a slope factor that provides an indication of the attenuation rate of the PPV caused by the geometric spreading and the specific characteristics of the rock (Aloui *et al.*, 2016; Puri and Prakash, 1991).

Monitoring of atmospheric signals

Another major aim of this study was to perform similar analyses to those by Koper *et al.* (2002), Bonner *et al.* (2013), Gitterman and Hofstetter (2012), and Gitterman (2013), but using the data obtained from the mobile network of temporary stations described above.

Bonner *et al.* (2013), Gitterman and Hofstetter (2012), and Gitterman (2013) examined data from three large-scale on-surface explosions, where the explosives were packed on dry desert alluvium. They used ANFO and military explosives (IMI) with different velocities of detonation (IMI VOD is 7130–7980 m/s and ANFO is 2400 m/s). The VODs for the explosives used in this current study ranged from 3000 m/s to 10 000 m/s, and thus fall in the same range as the military explosives used by Bonner *et al.* (2013), Gitterman and Hofstetter (2012), and Gitterman (2013). The following formulae were used to plot the measured scaled SS time delay (Dt) against the scaled distance (R_s):

$Dt = \Delta t/Wd^{0.33}$ and: $R_s = R/Wd^{0.33}$

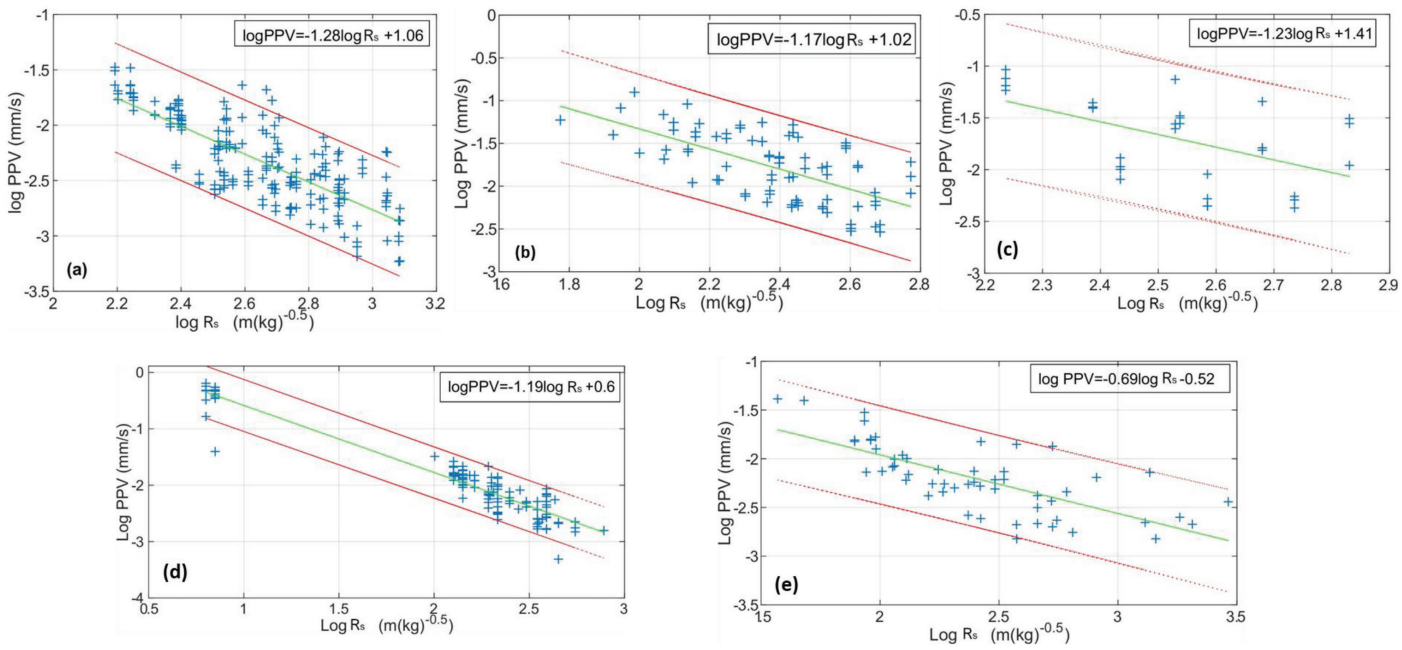

Figure 5—Relationship between PPV and R_s for all five sites using the USBM equation. The green lines represent the best fit using the least squares method, and the red lines represent the 95% confidence range. The '+'symbol represents the recorded PPV. (a) = Site 1, (b) = Site 2, (c) = Site 3, (d) = Site 4, (e) = Site 5. The gap in the data for Site 4 is due to the lack of stations within that range from the explosion

where:
Dt is the scaled SS time delay
Δt is the measured delay in seconds
Wd is the amount of explosive in kg

where:
R_s is the scaled distance
R is the distance in metres from the explosion to the station
Wd is the amount of explosive in kg

A linear RMS-fit regression curve of the scaled SS time delay against the scaled distance was then obtained for the explosions.

Figure 4 is an example of the atmospheric shock waves recorded at one of the stations. The horizontal line at the start of the plot is the DC level. The small number above each trace to the right is the maximum absolute count with the DC level subtracted, and the small number to the left above the trace is the DC level.

Results

Ground monitoring

PPV predictive equations

The largest PPV recorded was 0.485 mm/s for a 250 kg explosion at a distance of 10 m. However, this was only 2.5% of the 19 mm/s limit set out in the US Code of Federal Regulations (30 CFR part 816.67 of 1998), which deals with the control of adverse effects caused by the use of explosives. As was expected, since the readings were so low, no damage to the infrastructure in and around the stations was observed.

The relationships between log PPV and log R_s for each of the sites were examined. Figure 5 displays the results for all five sites for log PPV and log R_s using the USBM equation.

Figure 6 displays the results from each of the sites on one graph for the USBM prediction equation. The results of the regression analyses and the coefficient of determination (R^2) value for each of the sites and equations are summarized in Table III.

Generally, from Table III, the Langefors-Kihlstrom equation for each of the sites gives a lower value for the coefficient of determination (R^2) than the values obtained from the USBM and Ambraseys-Hendron equations. This result agrees with the findings of Bongiovanni *et al.* (1991); they concluded that since the data shows a poor correlation for the Langefors-Kihlstrom equation, the equation is not reliable for predicting motion produced by blasts at their site.

It is interesting to note that the highest R^2 values (greater than the generally accepted value of 0.7 (Kahriman, 2002) for all three equations were for site 4, which is located on metamorphosed granitic rock. The other R^2 values were far lower.

The results obtained from the USBM equation for site 1 (Table III) are similar to those of Gupta and Hartenberger (1981), and the geology is similar to one of the sites at which their study took place. The geological constants (β) for sites 1, 2, 3, and 4 are similar for the USBM equation, and sites 2 and 3 for the

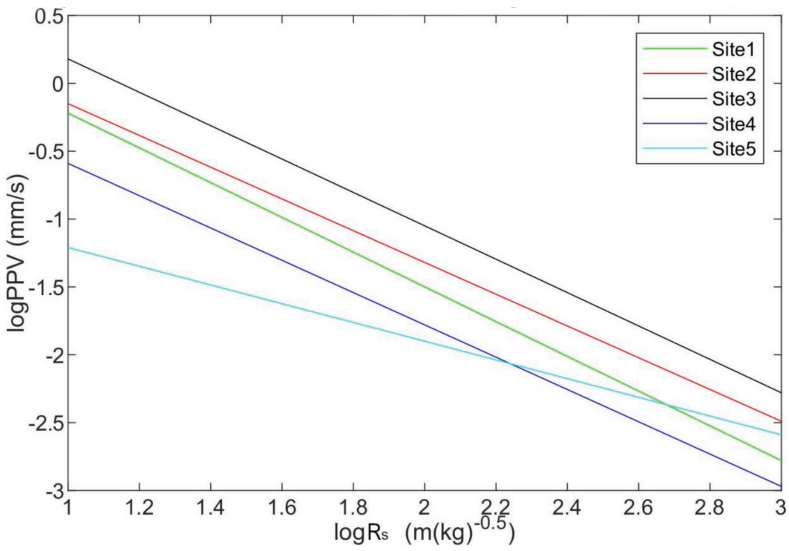

Figure 6—Results from the USBM equation for all five sites. The coefficients calculated for each site are listed in Table III

Table III

Empirical formulae for PPV obtained from the five sites

	Site 1		Site 2	Site 3	Site 4	Site 5
USBM	K	11.4	10.4	25.5	4.02	0.3
$PPV = K(R_s)^{-\beta}$	β	1.3	1.2	1.2	1.2	0.7
R^2	0. 61		0.41	0.28	0.90	0.58
Ambraseys	K	562.3	220.3	240.2	9.79	3.75
$PPV = K(R_{sA})^{-\beta}$	β	1.7	1.5	1.5	1.2	0.9
R^2	0.58		0.45	0.29	0.89	0.57
Langefors	K	0.01	0.01	0.04	0.003	0.003
$PPV = K(R_{s\ L})^{\beta}$	β	0.9	1.2	1.2	3.4	0.7
R^2	0.27		0.39	0.15	0.81	0.43

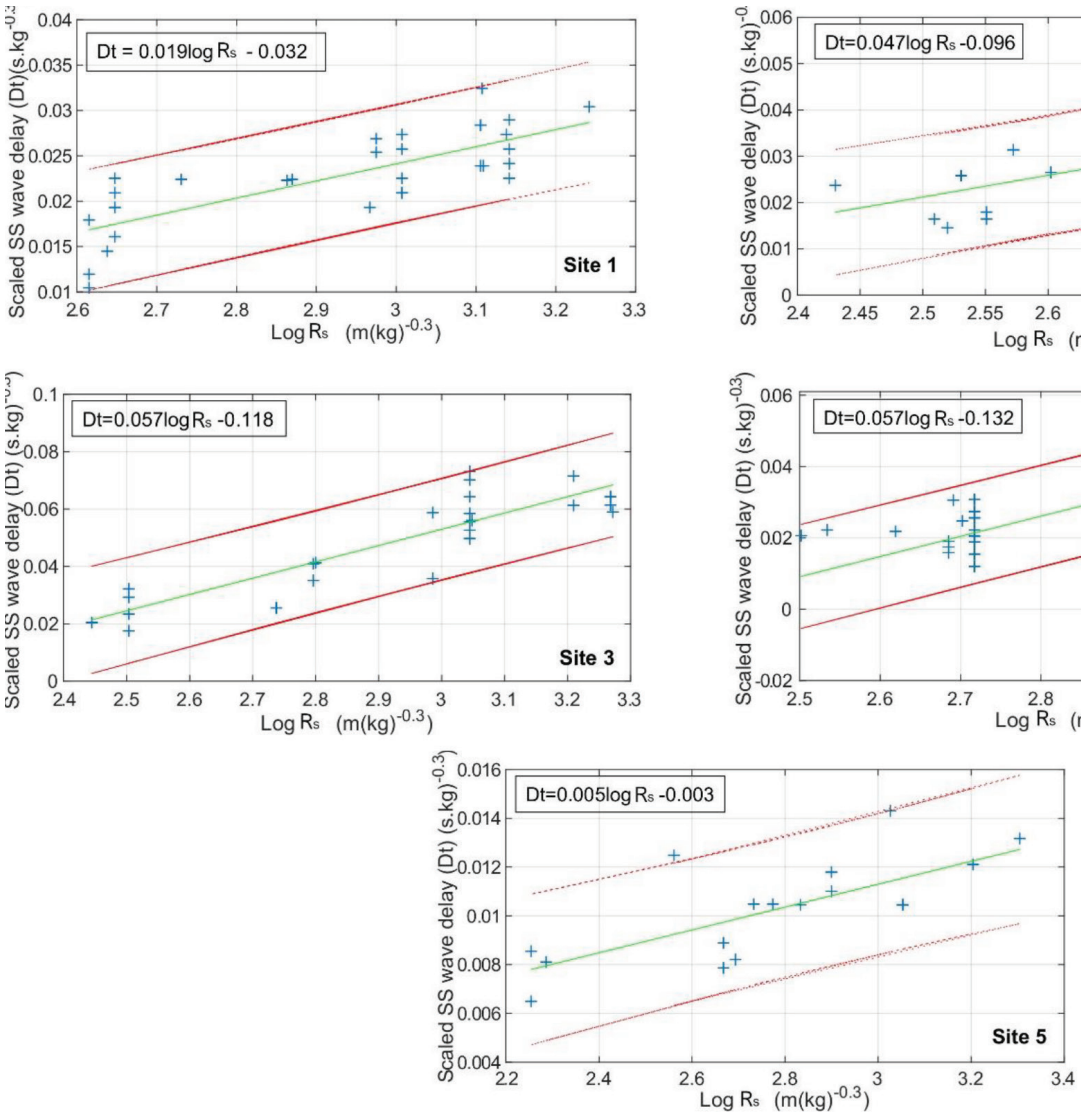

Figure 7—Relationship between the scaled SS wave delay (*Dt*) and log R_s for all the sites. The green lines represent the best fit using the least squares method and the red lines represent the 95% confidence range. The '+' symbol represents the recorded secondary shock wave delay

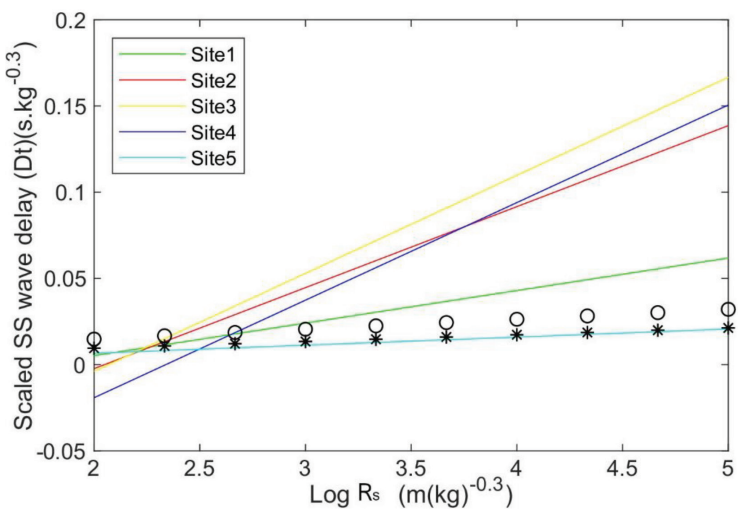

Figure 8—Relationship between scaled SS wave delay (*Dt*) and log *Rs* for all five sites. The coefficients calculated for each site are listed in Table V. The results from Gitterman (2013) are included for comparison 'o' indicates the results from the ANFO shots and '*' the results from the IMI shots as found in Figure 2

Ambraseys and Langefors equations. The ground transmission coefficient (K) for each relationship for site 5, compared to the other sites, is expected to be low due to the sandy nature of the region.

Atmospheric shock wave monitoring

Secondary shock delay

The values for the atmospheric shock waves were easier to measure than those for the ground motion because the atmospheric shock waves are more prominent on the seismograms than the ground motion. Johnston (1987) encountered a similar situation and noted that analysing the atmospheric shock waves was useful when the P and S waves are not clearly recorded.

All of the graphs of the logarithm of the scaled secondary shock delay (*Dt*) *versus* the natural log R_s (as described in Bonner *et al*., 2013; Gitterman and Hofstetter, 2012, and Gitterman, 2013) for each of the sites are found in Figure 7, together with the 95% mean prediction interval. Figure 8 depicts the results from all the sites compared to the results obtained by Gitterman (2013) and displayed in Figure 2.

Table V summarizes the results obtained from the relationship between the logarithm of the scaled secondary shock delay (*Dt*) and log R_s (as described in Bonner *et al*., 2013; Gitterman and Hofstetter, 2012; and Gitterman, 2013) for each of the sites.

From Table V and Figure 8, it is clear that sites 2, 3, and 4 display similar gradients, which may be indicative of a different VOD within the assortment of explosives used compared to sites 1 and 5. However, as shown in Figure 8, site 5 is the only site that displays a similar gradient to the results obtained by Gitterman (2013). These differences could also be related to the differing topographies of the sites, especially since site 5 is very similar to the site used in Bonner *et al*., 2013; Gitterman and Hofstetter, 2012, and Gitterman, 2013.

The results in Table V display high R^2 values (greater than the generally accepted value of 0.7 (Kahriman, 2002). In addition, it is clear that sites 2 to 4 display similar gradients while sites 1 and 5 differ.

Discussion

Ground monitoring

Although there is extensive literature describing prediction equations for the vibrations induced by blasting in open pit mines and tunnelling (*e.g.* Aloui *et al*., 2016; Bongiovanni *et al*., 1991; Khandelwal and Singh, 2007; Kahriman, 2002; Ozer, 2008; Ozer, Kahriman, and Aksoy, 2008; Puri and Prakash, 1991; Kumar, Choudhury, and Bhargava, 2016), many authors conclude that a reliable general approach or formula has not yet been established due to the complexity of issues relating to blasting.

The low R^2 values obtained for the PPV predictive equations (Table III) are comparable with those of Khandelwal and Singh (2007), who used the same three equations and obtained R^2 values of 0.2276 for the USBM equation, 0.1836 for the Ambraseys-Hendron equation, and 0.2218 for the Langefors-Kihlstrom equation. Thus, these low R^2 are not unique to the present study. However, unlike Khandelwal and Singh (2007), the explosions in this study were detonated on the surface and, as Evers *et al*. (2007) and Williams and Newell (1991) stated, very little of the energy goes into the ground.

The higher coefficients of determination (R^2) in Table III indicate that the formulae obtained using the USBM equation show better fit than for the other relationships, and a good fit for site 4. This is contrary to Ozer, Kahriman, and Aksoy (2008) and Kahriman (2002), who obtained the highest R^2 value for the Ambraseys-Hendron equation and thus recommended the use of that equation.

However, although the R^2 values for the Ambraseys-Hendron and Langefors equations are not as high as for the USBM sites, it is not necessarily true that there is something wrong with the fit. R^2 measures how useful the independent variable is in predicting values of the dependent variable, not how appropriate the polynomial model is for the data. When analysing inherently unpredictable data, a small value of R^2 indicates that the independent variable does not predict the dependent variable precisely.

This is particularly true for measurements of explosions on the ground surface, as opposed to excavation/mining explosions, which are within the bedrock. Thus, a lot of the energy from surface explosions is lost to the atmosphere and therefore an increase in distance from the explosions results in smaller P and S waves than calculated by the predictor equations, such as at site 5 where the sensors were placed at greater distances than at the other sites. Ichinose, Smith, and Anderson (1998), Johnston (1987), and Evers *et al*. (2007) noted similar findings. In addition, site 4 is a good example of an area where the geology plays an important role, as noted by Johnston (1987) and Koper *et al*. (2002), who stated that even minimal changes in the anelastic structure can affect the predictor equations, and Redmayne and Turbitt (1991) who noted that different soils can magnify or diminish the values obtained from predictor equations by a factor of 4.

Table IV

Empirical formulae for the scaled SS wave delay (*Dt*) obtained from the five sites

Site	Equation	R^2	Season
1	Dt = 0.018865 log Rs – 0.032479	1.00	Autumn
2	Dt = 0.046972 log Rs – 0.096245	0.9996	Spring
3	Dt = 0.056846 log Rs – 0.11758	0.9998	Summer
4	Dt = 0.05658 log Rs – 0.13233	0.9997	Spring
5	Dt = 0.0046869 log Rs – 0.0027627	1.00	Summer

The geological constants for regions 1, 2, 3, and 4 are similar for the USBM equation, indicating that the USBM is the more reliable PPV predictive equation for each of these sites, especially because geological constants of 1.2 are considered satisfactory (Atkinson, 2004; Atkinson and Mereu, 1992; Ford *et al.*, 2014). Redmayne and Turbitt (1991), using a simple attenuation relationship, obtained a geological constant of 2.18, which differs from the results obtained during this study. However, the use of predictive equations designed for mining and excavation must be applied with caution, because during blasting on the surface most of the energy escapes into the atmosphere. Thus, other methods should be sought to enhance these predictive equations in order to predict the ground vibrations caused by blasting on the surface.

Although the results of this study are different at the various sites, the equation which seems to produce the best fit is the USBM equation, which is in agreement with Aloui *et al.* (2016). What is also noticeable is that the equation seems to hold at distances of up to 29 km, as found for site 5, which is further than the typical distances of up to 1 km used for monitoring (Bongiovanni *et al.* 1991).

Surface/air monitoring

Although this study only used seismographs on the surface and did not utilize the sensors used by Koper *et al.* (2002), Bonner *et al.* (2013), Gitterman and Hofstetter (2012), Gitterman (2013) and Ford *et al.* (2014), it provided an opportunity to replicate their studies. The fact that the explosives used included casings/shells as well as net explosive content provided an excellent correlation with the study performed by Koper *et al.* (2002) on truck bombings, which also constitute a similar 'mixed' content of explosive and casings, as well as that by Redmayne and Turbitt (1991) on an aircraft crash. This study agreed with findings of Gitterman (2013), Johnston (1987), and Evers *et al.* (2007), who stated that the atmospheric shock waves were clearly observed at all sites, especially compared to the ground motion measurements. The seismograms showed clear and strong main shock peaks and smaller amplitude secondary peaks, with easily measured arrival times. Johnston (1987) examined the seismograms created by a missile silo explosion and noted that large arrivals were recorded over an eleven-minute interval as opposed to a scatter of less than one minute, which is expected for P-wave arrivals. The missile silo explosion was 84 km west of the station, which registered the event four minutes later. There was no trace of P- or S- waves, probably because most of the energy went into the atmosphere, but there were very small signals at the stations on hard rock.

However, the results do improve dramatically when examining the secondary shock wave phenomenon as described by Gitterman and Hofstetter (2012) and Gitterman (2013). This unique phenomenon seems to provide a better solution than the USBM, Ambraseys-Hendron, and Langefors equations for surface explosions. Although the R^2 values reported in Table V, which covers the scaled secondary shock wave delay, equate to unity, which indicates that the model explains all the variability of the response data around its mean, they are similar to the Gitterman and Hofstetter (2012) and Gitterman (2013) studies. From Figure 8 it is noticeable that the gradient for site 5 is also similar.

Sites 2 to 4 display similar gradients, which could indicate a different assortment of explosives used with different VODs to those used at sites 1 and 5. Gitterman and Hofstetter (2012) and Gitterman (2013) noted similar differences when using different explosives with varying VODs. However, what is noteworthy is the similarity of the results from site 5 and those of Gitterman and Hofstetter (2012) and Gitterman (2013). This is to be expected because the conditions are similar at both sites.

Conclusions

This study has identified a number of predictive equations that have produced acceptable results using data obtained from disposal of military ordnance that included not only the explosives but also the casings/shells, the demolitions being conducted on the surface.

In general, the values for the atmospheric shock waves were easier to determine than those of the ground motion because the atmospheric shock waves are more prominent on the seismograms than those of the ground motion, due to the fact that very little of the energy is transmitted into the ground. Therefore, utilizing atmospheric shock wave measurements may be more useful because the ground motion waves are not clearly recorded. In addition, the secondary shock wave delay increases with distance from the explosion as well as the amount of explosives used, unlike the PPV, which increases with the amount of explosives used but decreases with distance. The secondary shock wave delay equations may therefore be used for sites further from the source of the explosion, where the ground motion equations are less applicable.

Although, the USBM PPV predictive equation was identified as the most reliable equation out of the three examined, the predictive equation using the secondary shock wave phenomenon produced better results.

The usefulness of the suggested prediction equations would depend on the quantity that requires predicting. Thus, when assessing how best to mitigate the hazard posed by a military demolition range, where PPV prediction is required, the USBM equation would be the better solution. Alternatively, when assisting in forensic seismology to determine the details of an unexpected explosion (such as the amount of explosives used), the secondary shock wave has the potential to be useful, especially if the ground motion waves are not obvious in the seismograms.

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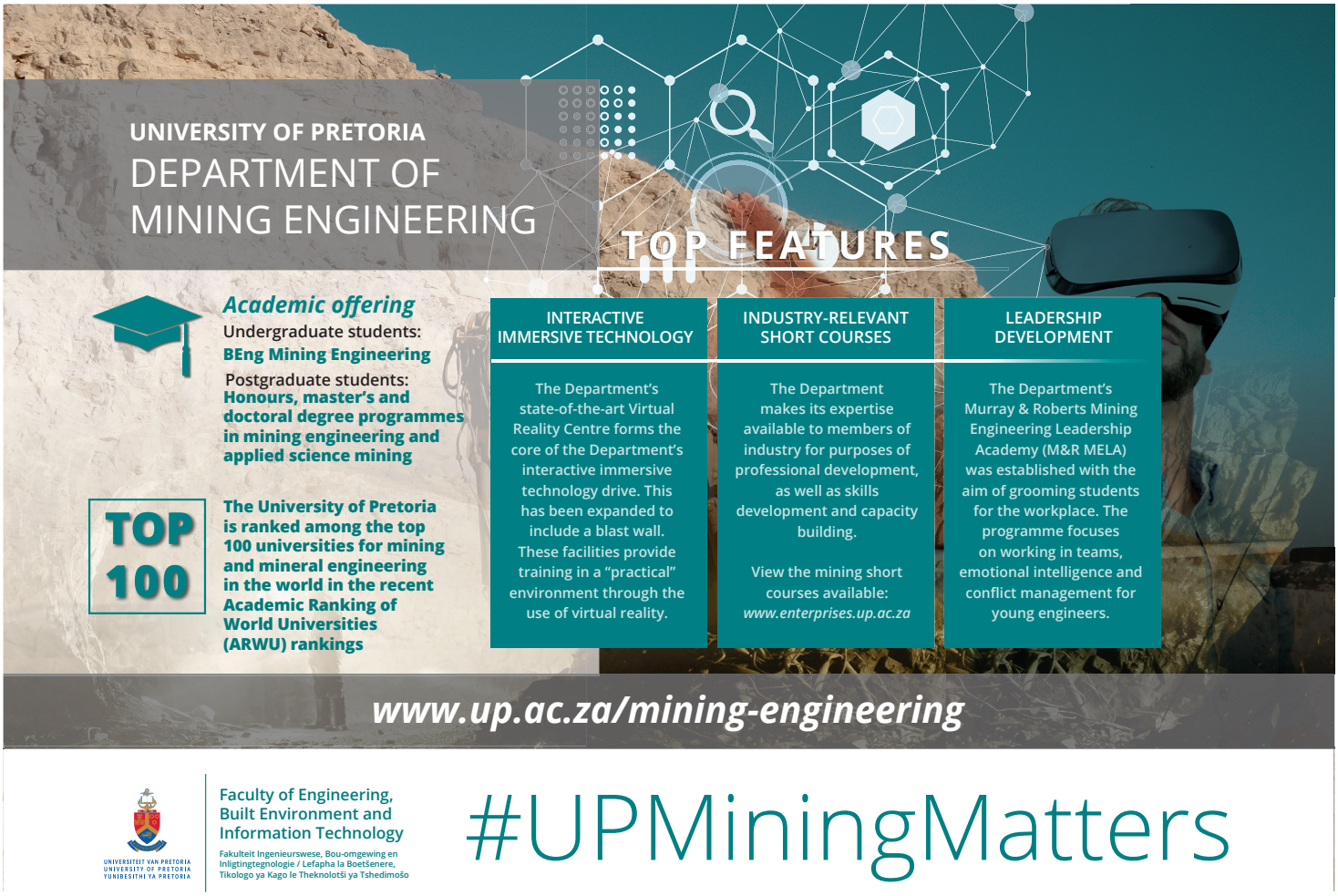

The application of coal discards for acid mine drainage neutralization

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Synopsis

The neutralization of acid mine drainage (AMD) with coal discards in percolating columns was investigated as a potential precursor to lime neutralization. The neutralizing capacity of three coal samples, A (70% ash), B (25.3% ash), and C (28.9% ash, estimated), sourced from three South African coal mines, was determined at different crush sizes (–40 mm, –12.5 mm and –6.3 mm). AMD solution obtained from another local coal mine was percolated over the coal samples packed in 1 m and 6 m columns, until the pH of the accumulated drainage solution measured approximately pH 7. Samples B and C, with an alkalinity content equivalent to 2.3% $CaCO_3$, achieved neutralizing capacities of 2112 L AMD and 929 L AMD per ton coal respectively, at a –6.3 mm crush size. Sample A, with an alkalinity content equivalent to 0.48% $CaCO_3$, neutralized only 282 L AMD per ton coal at the same crush size.

An economic analysis was performed to compare neutralization with waste coal against lime neutralization in tanks. The analysis was based on a rate of AMD generation of 1750 m^3/d, a neutralizing capacity of 1.4 m^3 AMD per ton coal, with capital costs estimated at R18 million for lime neutralization and R27.6 million for coal neutralization. Operating costs were estimated at R24 million for lime neutralization and R9 million for coal neutralization. AMD neutralization with suitable waste coal may therefore be less expensive than neutralization with lime.

Keywords

acid mine drainage, neutralization, waste coal.

Introduction

Globally, approximately 6.9 billion tons of coal is extracted annually and used for electricity generation, steel manufacture, cement manufacture, and conversion to liquid fuel (World Coal Association, 2017). Waste coal stockpiles are a source of air and surface pollution, generating dust emissions and acid mine drainage (AMD), which may release heavy metals and toxic elements into the environment. From a South African perspective, the production of coal wastes currently stands at approximately 6 Mt/a (Cornish, 2016). In addition, approximately 50–62 ML/d of AMD is decanted from active and obsolete coal mines in the Mpumalanga Province alone (Hobbs, Oelofse, and Rascher, 2008).

The use of lime for the treatment of acidic mine water was implemented during the 1980s and is the method of choice to date (Mey and van Niekerk, 2009). However, with the cost of lime increasing annually and the liming process producing a low-density sludge that is difficult to filter (Mey and van Niekerk, 2009; Department of Mineral Resources, 2010), it is necessary to search for alternative low-cost options for AMD treatment. Acid-consuming minerals in waste coal could potentially be utilized to counteract sulphuric acid (H_2SO_4) generation, *e.g.* from the oxidation of pyrite (FeS_2) if present in the coal. Examples of acid-consuming minerals include carbonates, such as calcite ($CaCO_3$) and dolomite ($CaMg(CO_3)_2$); and also clay minerals such as kaolinite ($Al_2Si_2O_5(OH)_4$) and other phyllosilicates. The neutralization of AMD with alkaline-rich waste coal is therefore being investigated as an alternative low-cost process route that may be implemented to complement, or even substitute for, lime neutralization (Perry and Brady, 1995; Watten, Sibrell, and Schwartz, 2005; Fu and Wang, 2011).

Test work was performed to determine the neutralizing capacity of coal samples obtained from three South African mines: sample A (from a mine in Limpopo), sample B (from a mine in Mpumalanga), and sample C (from another mine in Mpumalanga). The coal samples were loaded into 1 m and 6 m columns, and irrigated with AMD solution (pH 2.5–2.8). The neutralizing capacity was quantified as the volume of solution that can be neutralized to approximately pH 7 per ton of coal, and was also expressed as an equivalent lime consumption in kilograms of $Ca(OH)_2$ per ton coal.

An economic analysis was also performed to compare the costs of AMD neutralization with waste coal and the traditional lime neutralization route. Three proposed flow sheets were developed:

- Partial neutralization of AMD over waste coal heaps with only 75% of the neutralization requirement provided by coal (*i.e.* the ratio of AMD to coal was 1.9 m^3 per ton instead of 1.4 m^3 per ton), followed by neutralization with lime in stirred tanks to pH 7.
- Neutralization with lime only to pH 7.
- Neutralization with waste coal only to pH 7.

Design criteria, reagent consumptions, and reagent costs based on experimental and published data were used to calculate capital and operating costs for each process option.

Experimental

Sample preparation and analysis

Sample A was obtained from a coal mine in Limpopo. Approximately 1 ton of sample was transferred from a discard stockpile into bulk bags by front-end loader, and transported to Mintek. Sample B originated from a coal mine in Mpumalanga. The sample was already on-site at Mintek, and constituted material remaining from a prior sorting programme. Approximately 1 ton of sample C was sourced, in the same fashion as for sample A, from another coal mine in Mpumalanga.

Samples A, B, and C were blended separately and split into three batches each. The individual batches were then stagewise crushed (to reduce excessive fines generation) in a laboratory jaw crusher with screening between stages to –40 mm, –12.5 mm, and –6.3 mm. Only the –6.3 mm material from sample C was used in this study since the –40 mm and –12.5 mm material had been used in an earlier programme. Representative sub-samples were split out in 25 kg charges for chemical analysis and column neutralization test work. The particle size distributions (PSDs) were determined by dry screening at the following screen sizes: 40 mm, 19 mm, 12.5 mm, 9.5 mm, 6.3 mm, 3.35 mm, 1.18 mm, 500 µm, and 150 µm.

Splits of the representative sub-samples were pulverized and analysed by inductively coupled plasma-optical emission spectroscopy (ICP-OES) for: Al, Ca, Cu, Co, Cr, Fe, Mg, Mn, Ni, Pb, Si, Ti, V, and Zn. Sulphide sulphur (S^{2-}) was analysed on a LECO instrument after pre-leaching with trichloroethylene to remove the other sulphur species. K and Na were analysed by atomic absorption spectroscopy (AAS). Alkalinity, expressed as calcium carbonate ($CaCO_3$) equivalent, was analysed by an acid-base accounting (ABA) method (Lawrence and Wang, 1997).

In addition, the moisture content was determined from the mass loss upon heating for 2 hours in an oven at 110°C. The content of volatile material was determined from the mass loss upon ignition in a muffle furnace at 900°C for 7 minutes in the absence of air. The ash content was determined by ignition in air in a muffle furnace at 820°C for 90 minutes. The fixed carbon content was calculated by difference; in other words, by subtraction of the sum of the ash, moisture, and volatiles mass percentages from the total (100%).

The AMD and accumulated drainage solutions were analysed for: Al, As, Ca, Cd, Co, Cr, Cu, Fe, Li, Mg, Mn, Mo, Ni, Pb, S, Si, Ti, V, and Zn (ICP-OES), as well as for K and Na (AAS).

Column neutralization test work

Tests were conducted in 1 m and 6 m, water-jacketed polypropylene columns (160 mm ID) connected to a chiller or geyser for temperature control. The columns were irrigated from the top through a single dripper point from a tight-fitting 3 mm, plastic tube inserted through a hole in the centre of a flanged lid. The irrigated solution drained down through the packed coal bed by gravity (with lateral flow by conduction), and was collected at the column base through an enclosed perforated plate, with a drainage hole (fitted with a 'pigtailed' tube), for daily collection of drained solution.

A schematic representation of the experimental set-up for the neutralization test work is shown in Figure 1a. The facilities for the test work are shown in Figure 1b.

The experimental matrix is summarized in Table I.

The coal samples were agglomerated in a rotating drum mixer (for the 6 m columns) or on a plastic sheet (for the 1 m columns), with 5% agglomeration moisture (AMD) added. Agglomeration is a technique used to bind fines to coarse particles. This allows for even permeability of solution through the packed bed. The columns were then charged with the wet agglomerates and percolated with AMD solution by means of Watson Marlow 120 S peristaltic pumps (Figure 1a). The AMD solution was placed in 50 L feed tanks and daily measurements of mass, volume, specific gravity (SG), redox potential (*vs.* Ag/AgCl; 3 M KCl), pH, and temperature were taken. The daily drainage solutions were collected from the base of the columns (Figure 1a) and the following parameters were measured: mass, volume, SG, redox potential, pH, and temperature. The daily drainage solutions were then accumulated in plastic tanks and the pH of the accumulated solutions measured on a daily basis. Once the daily drainage

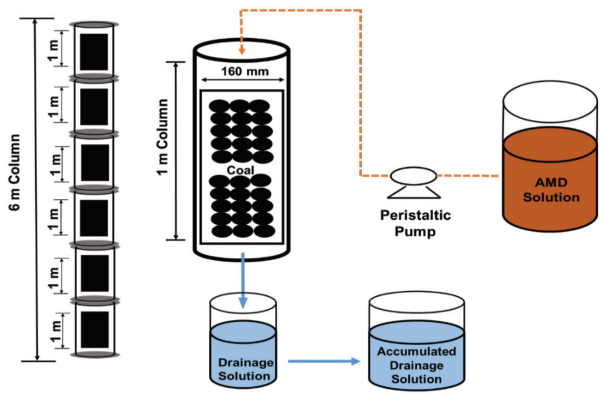

Figure 1a—Schematic diagram of experimental set-up

Figure 1b—Test work facilities

Table I

Experimental test matrix

Sample	Column		Crush size (mm)	Moisture content (%)	Temp. (°C)	Irrigation rate (L/m²/h)	Duration (days)
	Height (m)	Diameter (mm)					
A	1	160	–40	5	25	1.6	18
A	1	160	–12.5	5	25	1.6	18
A	1	160	–6.3	5	25	1.6	11
A	6	160	–40	5	25	1.6	36
A	6	160	–12.5	5	25	1.6	39
A	6	160	–6.3	5	25	1.6	54
B	1	160	–40	5	25	1.6	18
B	1	160	–12.5	5	25	1.6	18
B	1	160	–6.3	5	25	1.6	51
B	6	160	–40	5	25	1.6	121
B	6	160	–12.5	5	25	1.6	204
C	1	160	–6.3	5	25	1.6	23

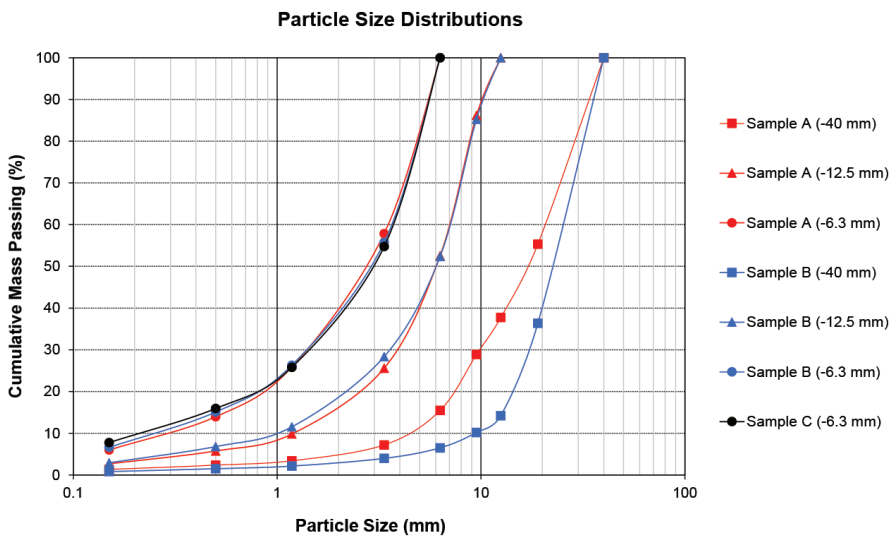

Figure 2—Particle size distributions of the crushed coal samples

pH levels dropped below pH 6, irrigation was stopped and the columns were allowed to drain. The final accumulated drainage solutions were analysed by ICP-OES and AAS.

Results and discussion

Particle size distribution

The particle size distributions of the three coal samples are presented in Figure 2. The fines (–1 mm) content increased from less than 5% in the coarser crush size (–40 mm) to above 20% in the finer (–6.3 mm) crush size. As the exposed surface area required for chemical reaction increases with increasing fines content, it was expected that the finer crush sizes would have improved neutralizing capacity. In all cases, a fines content of less than 10% passing 150 µm was achieved, which is below the maximum limit of 10% to 14% passing 150 µm typically reported to prevent permeability restraints in percolation leaching (Scheffel, 2017).

Chemical analysis

The chemical analyses of the AMD solution and samples A, B, and C are presented in Table II.

The coal samples contained high contents of Al (3.3% to 9.9%) and Si (5.6% to 21.6%), which are common characteristics for coal wastes (Modarres and Ayar, 2014; Vegas *et al.*, 2015). Sample A contained 70% ash and sample B contained 25.3% ash as determined by combustion method. Semi-quantitative X-ray diffraction (XRD) performed on samples A and B indicated kaolinite ($Al_2Si_2O_5(OH)_4$) and quartz (SiO_2) to be the predominant Al- and Si-containing minerals. On the assumption that Al occurs only in kaolinite and Si in both kaolinite and quartz, the combined contents for these two minerals can be calculated as 71.5% (sample A), 22% (sample B), and 28.9% (sample C), with the former two estimations in reasonable agreement with the corresponding ash contents determined by combustion. The ash content in South African coal discards is typically greater than

Table II

AMD solution and coal sample assays

Constituent	AMD[1] (mg/L)	Coal sample (%)		
		A	B	C
Al	424	9.88	3.89	3.32
As	<2	NA	NA	NA
Ca	492	0.33	1.00	1.21
Cd	<2	NA	NA	NA
Co	2.1	<0.05	<0.05	<0.05
Cr	<2	<0.05	<0.05	<0.05
Cu	10.9	<0.05	<0.05	<0.05
Fe	6325	1.78	0.98	0.83
K	10.6	0.50	0.02	0.35
Li	<2	NA	NA	NA
Mg	494	0.17	0.31	0.18
Mn	103	<0.05	<0.05	<0.05
Mo	<2	NA	NA	NA
Na	79.9	0.08	0.001	0.07
Ni	<2	<0.05	<0.05	<0.05
Pb	<2	<0.05	<0.05	<0.05
S^{TOT}	6455	NA	NA	NA
S^{2-}	NA	1.27	1.13	0.56
Si	90.6	21.6	5.64	9.52
Ti	<2	0.47	0.25	0.20
V	<2	<0.05	<0.05	<0.05
Zn	12.5	<0.05	<0.05	<0.05
$CaCO_3$[2]	NA	0.48	2.32	2.33
SiO_2[3]	NA	24.2	3.40	13.0
$Al_2Si_2O_5(OH)_4$[3]	NA	47.3	18.6	15.9
Ash	NA	70.0	25.3	NA
Fixed carbon	NA	14.4	48.0	NA
Moisture	NA	1.47	3.20	NA
Volatiles	NA	14.5	23.5	NA

Notes

1. Initial conditions of the AMD solution: pH 2.5 to pH 2.8 and 392 mV at 15°C
2. Alkalinity expressed as $CaCO_3$ equivalent from acid-base accounting (ABA) method
3. Calculated from Al and Si contents on the assumption that Al occurs only in kaolinite ($Al_2Si_2O_5(OH)_4$) and Si in both kaolinite and quartz (SiO_2). NA: not assayed

40% (Lloyd, 2000; Coal Resources, 2001; North, Engelbrecht, and Oboirien, 2015). Sample A can therefore be classified as waste coal. Samples B and C had ash contents of less than 30% and can therefore be classified as power station coal (Cornish, 2016).

Samples B and C had a higher degree of alkalinity expressed as $CaCO_3$ (2.3%) compared with sample A (0.48%), which suggests that these two samples should show greater neutralizing capacities.

The AMD solution contained elevated concentrations of Fe (6.3 g/L) and (sulphate) sulphur (6.4 g/L), which is in agreement with AMD characteristics (Johnson and Hallberg, 2005).

Column neutralization test work

The accumulated drainage pH profiles of the 1 m columns are presented in Figure 3. Sample A at the coarser crush sizes (–40 mm and –12.5 mm) did not achieve a drainage pH above 7, whereas the –6.3 mm crush size achieved a drainage pH above 7 for 11 days. Sample B at the coarse (–40 mm) crush size never achieved a drainage pH above 7, whereas the finer crush sizes achieved a drainage pH above 7 for 18 days and 51 days for the –12.5 mm and –6.3 mm crush sizes, respectively. Sample C (–6.3 mm) maintained a pH above 7 for 23 days.

The accumulated drainage pH profiles of the 6 m columns are presented in Figure 4. The columns containing sample A maintained an accumulated drainage pH above 7 for 36 days (–40 mm), 39 days (–12.5 mm), and 54 days (–6.3 mm), respectively. The test on sample B –40 mm crush size showed an accumulated drainage pH above 7 for 121 days, whereas the –12.5 mm crush size test continued to maintain an accumulated drainage pH above 7 for 204 days.

The neutralizing capacities are summarized in Table III. Sample A neutralized more than 150 L AMD per ton coal at the coarser crush sizes (–40 mm and –12.5 mm) in the 6 m columns, and 258 L AMD per ton coal (6 m columns) to 282 L AMD per ton coal (1 m columns) at the finer crush size (–6.3 mm). For sample B, the capacity increased from 804 L AMD per ton coal at the –40 mm crush size (6 m columns) to 1410 L AMD per ton coal at the –12.5 mm crush size (6 m columns), and as high as 2112 L AMD per ton coal at the –6.3 mm crush size (1 m columns). Lime equivalents ranged from 2.28 kg $Ca(OH)_2$ to 31.7 kg $Ca(OH)_2$ per ton coal.

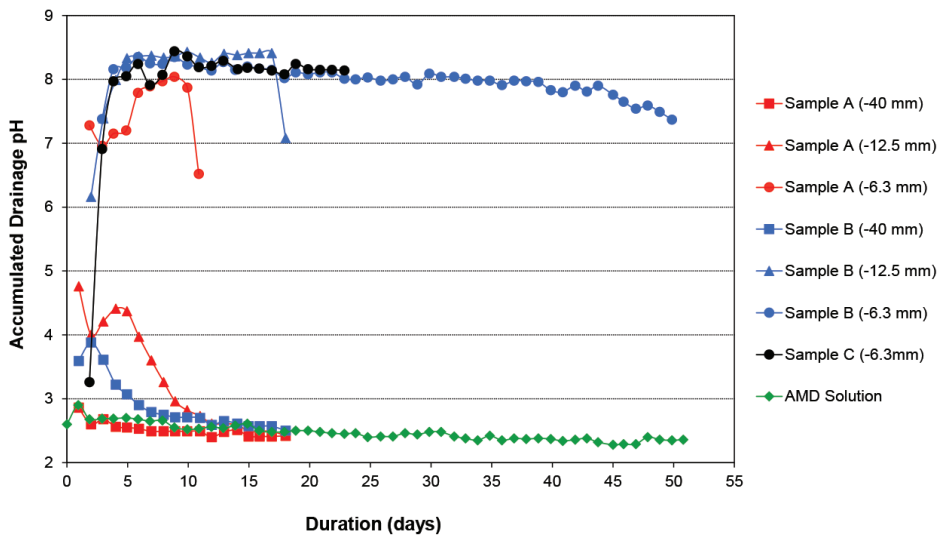

Figure 3—Accumulated drainage pH profiles, 1 m columns

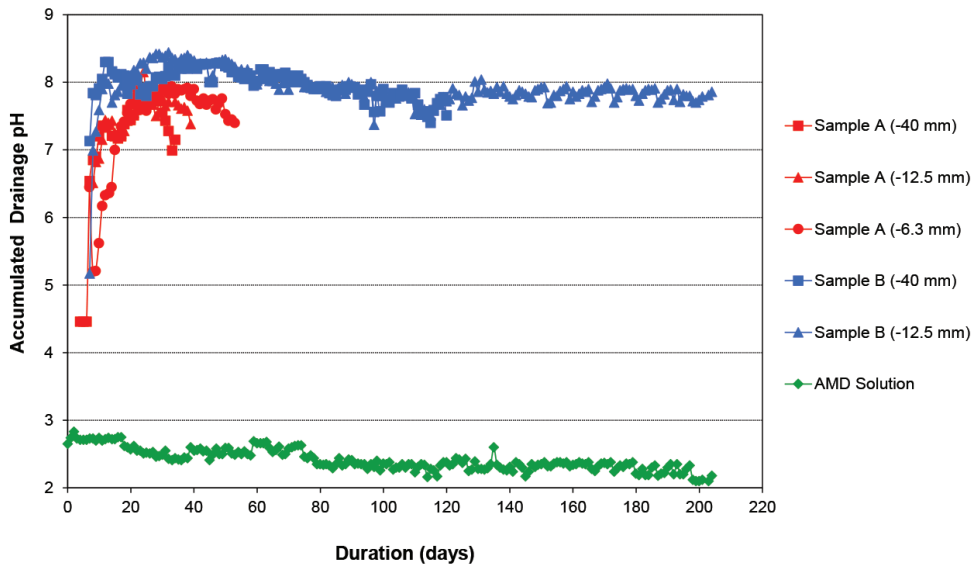

Figure 4—Accumulated drainage pH profiles, 6 m columns

Table III

Neutralizing capacities and lime equivalents

Sample	Column height (m)	Crush size (mm)	Neutralizing capacity (L AMD per t coal)	Lime equivalent (kg $Ca(OH)_2$ per t coal)[1]
A	6	–40	152	2.28
B	6	–40	804	12.1
A	6	–12.5	158	2.37
B	1	–12.5	705	10.6
B	6	–12.5	1410	21.2
A	1	–6.3	282	4.23
A	6	–6.3	258	3.87
B	1	–6.3	2112	31.7
C	1	–6.3	929	13.9

Note

1. Calculated from an experimentally determined lime consumption of 15 kg $Ca(OH)_2$ per m^3 AMD to treat the AMD solution to pH 7.

In general, decreased crush size resulted in an increase in neutralizing capacity (see samples A and B). Also, as expected from the alkalinities, the capacities for samples B and C (2.3% $CaCO_3$) are markedly higher than for sample A (0.48% $CaCO_3$). For example, at 1 m and –6.3 mm crush size the capacities are 282 L AMD per ton coal (sample A) *versus* 2112 L AMD per ton coal (sample B) and 929 L AMD per ton coal (sample C). However, it is interesting to note that despite the same alkalinity of 2.3% $CaCO_3$ (determined on pulverized material) and PSDs (Figure 2), sample B achieved a higher neutralizing capacity than sample C. This may be the result of sample C's alkalinity being weighted to the finer material. Another possibility is that sample B may be more penetrable to the percolating solution than sample C.

The chemical assays of the AMD and accumulated drainage solutions are presented in Table IV. The sulphate values were calculated from the total sulphur ICP-OES assays. The solution assays indicate high removal efficiencies of Al, Cu, Fe, Si, and Zn, whereas K and Mg, on the other hand, increase in the accumulated solutions due to the leaching of acid-consuming minerals. The Na concentration of the drainage solution for sample C also increased. The results show total sulphate removals of between 63% and 74% based on ICP-OES assays. These values are in agreement with those calculated from sulphate concentrations obtained for cation / sulphate balances for the AMD and accumulated drainage solutions.

Figure 5 shows the AMD solution percolated through the column and the neutralized drainage solution after contact with the coal. The red-coloured AMD solution is dominated by iron, whereas the drainage solution is clear due to iron removal. This is supported by the solution assays data, which indicates an iron removal efficiency of 100% (Table IV).

Figure 6 presents photographs of offloaded coal particles with yellow/orange and reddish brown precipitation visible on parts of their surfaces. This suggests that species that were removed from the AMD solution can deport to the coal surface as precipitation products.

The AMD solution contained 6.325 g/L Fe at pH 2.5 to 2.8, 392 mV (*vs.* Ag/AgCl; 3 M KCl) and 15°C. Using an appropriate

Table IV

Solution assays

Description	Unit	Al	Ca	Cu	Fe	K	Mg	Mn	Na	S^{TOT}	Si	Zn	SO_4^{2-} (2)	SO_4^{2-} (3)
Sample A (6 m, -40 mm)														
AMD Solution	g/L	0.424	0.492	0.011	6.325	0.011	0.494	0.103	0.080	6.455	0.091	0.013	19.3	18.1
Drainage Solution	g/L	0.002	0.552	0.002	0.002	0.055	0.933	0.092	0.060	1.800	0.005	0.002	5.39	5.40
Removal Efficiency (1)	%	100	-12	>82	100	-400	-89	11	25	72	95	85	72	70
Sample A (6 m, -12.5 mm)														
AMD Solution	g/L	0.424	0.492	0.011	6.325	0.011	0.494	0.103	0.080	6.455	0.091	0.013	19.3	18.1
Drainage Solution	g/L	0.002	0.488	0.002	0.002	0.050	0.937	0.073	0.082	1.663	0.011	0.002	4.98	5.32
Removal Efficiency (1)	%	100	1	>82	100	-355	-90	30	-3	74	88	85	74	71
Sample A (1 m, -6.3 mm)														
AMD Solution	g/L	0.424	0.492	0.011	6.325	0.011	0.494	0.103	0.080	6.455	0.091	0.013	19.3	18.1
Drainage Solution	g/L	0.002	0.507	0.002	0.002	0.062	1.355	0.166	0.181	2.393	0.002	0.010	7.17	7.34
Removal Efficiency (1)	%	100	-3	>82	100	-464	-174	-60	-127	63	>98	23	63	59
Sample A (6 m, -6.3 mm)														
AMD Solution	g/L	0.424	0.492	0.011	6.325	0.011	0.494	0.103	0.080	6.455	0.091	0.013	19.3	18.1
Drainage Solution	g/L	0.008	0.473	0.002	0.002	0.054	1.390	0.055	0.056	2.230	0.011	0.002	6.67	7.01
Removal Efficiency (1)	%	98	4	>82	100	-391	-182	47	30	65	88	85	65	61
Sample B (1 m, -12.5 mm)														
AMD Solution	g/L	0.424	0.492	0.011	6.325	0.011	0.494	0.103	0.080	6.455	0.091	0.013	19.3	18.1
Drainage Solution	g/L	0.004	0.512	0.002	0.003	0.028	1.360	0.070	0.062	2.325	0.002	0.002	6.97	6.91
Removal Efficiency (1)	%	99	-4	>82	100	-155	-175	32	23	64	>98	85	64	62
Sample B (6 m, -40 mm)														
AMD Solution	g/L	0.424	0.492	0.011	6.325	0.011	0.494	0.103	0.080	6.455	0.091	0.013	19.3	18.1
Drainage Solution	g/L	0.002	0.468	0.002	0.002	0.030	1.255	0.015	0.081	1.945	0.002	0.002	5.83	6.31
Removal Efficiency (1)	%	100	5	>82	100	-173	-154	86	-1	70	>98	85	70	65
Sample B (6 m, -12.5 mm)														
AMD Solution	g/L	0.424	0.492	0.011	6.325	0.011	0.494	0.103	0.080	6.455	0.091	0.013	19.3	18.1
Drainage Solution	g/L	0.008	0.405	0.002	0.002	0.027	1.373	0.020	0.072	2.163	0.002	0.002	6.49	6.65
Removal Efficiency (1)	%	98	18	>82	100	-145	-178	81	10	66	98	85	66	63
Sample B (1 m, -6.3 mm)														
AMD Solution	g/L	0.424	0.492	0.011	6.325	0.011	0.494	0.103	0.080	6.455	0.091	0.013	19.3	18.1
Drainage Solution	g/L	0.010	0.476	0.002	0.002	0.069	1.265	0.091	0.068	2.145	0.005	0.002	6.43	6.61
Removal Efficiency (1)	%	98	3	>82	100	-527	-156	12	16	67	94	85	67	63
Sample C (1 m, -6.3 mm)														
AMD Solution	g/L	0.424	0.492	0.011	6.325	0.011	0.494	0.103	0.080	6.455	0.091	0.013	19.3	18.1
Drainage Solution	g/L	0.002	0.466	0.002	0.002	0.110	1.247	0.052	0.395	2.343	0.006	0.002	7.02	7.14
Removal Efficiency (1)	%	100	5	>82	100	-900	-152	49	-394	64	94	85	64	61

Notes

1. Removal efficiency = (concentration in AMD – concentration in drainage) / (concentration in AMD) × 100
2. Calculated from assays for AMD and accumulated drainage solutions.
3. Calculated from sulphate concentrations obtained from cation / sulphate balances for AMD and accumulated drainage solutions.
4. As, Cd, Co, Cr, Li, Mo, Ni, Pb, Ti, and V for both AMD and accumulated drainage solutions assayed below or near the detection limit of 2 mg/L.

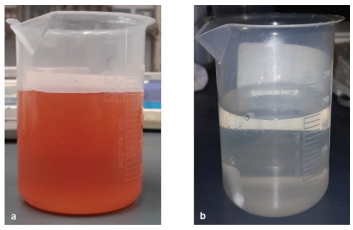

Figure 5—Photographs of (a) AMD solution and (b) neutralized drainage solution

Figure 6—Precipitation on (a) wet –40 mm sample A, (b) dry –40 mm sample B, and (c) dry –12.5 mm sample B

formal potential (E^{Φ}) of 0.678 V (*vs.* SHE), the Fe(III) and Fe(II) concentrations can be estimated as 0.332 g/L and 5.993 g/L respectively from the following equations:

$$E = E^{\Phi} - (RT/nF)\ln([\mathrm{Fe(II)}]/[\mathrm{Fe(III)}]) \quad [1]$$

and

$$[\mathrm{Fe}] = [\mathrm{Fe(III)}] + [\mathrm{Fe(II)}] \quad [2]$$

where

E Solution potential (V *vs.* SHE)

E^{Φ} Formal potential for the Fe(III)/Fe(II) redox couple (V *vs.* SHE)

F Faraday's constant (96 487 C/mol)

Fe Total iron concentration (mol/L)

Fe(III) Ferric concentration (mol/L)

Fe(II) Ferrous concentration (mol/L)

n Number of electrons (mol)

R Universal gas constant (8.314 J/mol/K)

T Temperature (K)

Acid in the AMD solution reacts with acid-consuming minerals present in the coal when the solution percolates through the coal bed. This causes a decrease in the acidity of the solution as reflected by the increasing solution pH; most notably for samples A (–6.3 mm), B (–12.5 mm and –6.3 mm), and C (–6.3 mm) for the 1 m columns (Figure 3), and samples A and B for the 6 m columns (Figure 4). The alkalinity provided by the

coal could be due to the presence of acid-consuming minerals that include carbonates such as calcite and dolomite; and also clay minerals such as kaolinite and other phyllosilicates. For example, in the case of calcite:

$$CaCO_3 + H_2SO_4 \rightarrow CaSO_4 + CO_2 + H_2O \quad [3]$$

As the solution pH increases above pH 4 it is expected that Fe(III) and Al(III) would precipitate from solution as metal hydroxides. By way of example, in the case of Fe(III):

$$Fe(III) + 3H_2O \rightarrow Fe(OH)_3 + 3H^+ \quad [4]$$

However, it should be noted that depending on pH and solution composition, a variety of iron(III) precipitates are possible such as jarosite (H_3O, K, Na, $NH_4)Fe_3(OH)_6(SO_4)_2$, haematite (α-Fe_2O_3), schwertmannite ($Fe_8O_8(OH)_6(SO_4)$), goethite (α-FeOOH), and ferrihydrite ($Fe_5(OH)_8{\cdot}4H_2O$); and in the case of aluminium(III) precipitates, jurbanite ($Al(SO_4)OH$), alunite ($KAl_3(SO_4)_2(OH)_6$) and gibbsite ($Al(OH)_3$) (Petrik, 2004). The above precipitation reactions generate acid, which can be consumed again by acid-consuming minerals present in the coal.

The AMD solution's Fe(III) concentration forms a much smaller part of the total iron concentration (about 5%) than the Fe(II) concentration (about 95%). However, it has been shown that Fe(II) can be removed by oxidative precipitation where Fe(II) is oxidized by dissolved oxygen to Fe(III). For example, Stumm and Lee (1961) studied the kinetics of the reaction in bicarbonate (HCO_3^-) solutions over the range pH 6 to pH 7.5, and found that their experimental measurements were in agreement with the stoichiometric relationship:

$$4Fe(II) + O_{2\,(aq)} + 8OH^- + 2H_2O \rightarrow 4Fe(OH)_3 \quad [5]$$

They found that the rate of Fe(II) to Fe(III) oxidation was first order in Fe(II) concentration, first order in dissolved oxygen concentration, and second order in hydroxyl (OH^-) concentration, with the rate of the reaction increasing 100-fold for an increase in one pH unit.

The reaction presented in Equation [5] may well have been responsible for the effective removal of Fe(II) from the AMD solution in the columns where the drainage solution pH increased to as high as pH 8 to pH 8.4. These results are consistent with the hydroxide precipitation diagram by Monhemius (1977), which shows that Fe(II) will precipitate over the range pH 6.3 to pH 8.5.

The 1 m and 6 m columns were not aerated during operation. This would mean that if the reaction in Equation [5] was responsible for Fe(II) removal, the oxygen supply from dissolved oxygen in the irrigating AMD solution and from air in the voidage of the packed coal bed was sufficient to sustain the reaction. Ambient air ingress into the coal bed was unlikely since the 'pigtailed' drainage tube always contained some solution, as well as the bottom of the bed and column base, thus forming a natural seal.

Economic analysis

An economic analysis was performed to compare the cost of neutralizing AMD with waste coal (in heaps) *versus* lime (in agitated vessels).

The stacking methodology for the coal heaps comprises an on-off pad, whereby a permanent pad is constructed and the ore is stacked (using either trucks or a mechanical stacker) in a single 6 m lift. AMD is pumped over the heaps and dripper or sprinkler irrigation is used. Solution draining from the base of the heaps is collected in drainage pipes and solution ponds. Spent coal is removed with a front-end loader. The lime neutralization plant consists of a series of agitated tanks with overhead motors, a clarifier, and auxiliaries such as feed pumps and holding tanks.

The capital cost includes ground preparation, construction of a permanent pad of sufficient area to allow the stacking, equipping, percolation, and drainage, and de-equipping and removal operations. Adequate sustaining capital or alternative operating cost provision must be allowed for replacement of the gradually depleted drainage layer and blocked or damaged drainage piping. The operating costs include labour, pumping power, reagents, and maintenance materials.

Economic analysis methodology

The economic analysis employs a method of factoring costs from best available benchmark data, by the following sequence of steps.

- Basic production criteria are specified (*e.g.* rate of AMD generation, coal neutralizing capacity, coal bulk density, heap stacking height, lime tank residence time, number of tanks, and lime neutralizing capacity).
- Mass balance data is calculated (*e.g.* coal stacking rate, solution application rate, mass under percolation, area under percolation, pond volumes, tank volumes, pump sizes, and lime addition rates).
- Direct capital costs are calculated based on the mass balance and selected multipliers. Appropriate benchmark factors are selected for the calculation of the heap capital costs (*e.g.* cost per m^2 of pad area constructed, cost per m^3 of pond volume excavated, and cost per m^2 of plastic liner).
- Uninstalled costs of capital items such as tanks and pumps are calculated from the Mintek equipment cost database (Ruhmer, 1996). Installed capital costs are calculated from the uninstalled capital costs by multiplying with an appropriate factor, *e.g.* civils, piping, instrumentation, and electrical costs.
- Indirect capital costs such as engineering, procurement, and construction management (EPCM), owner's cost, *etc.* are calculated from the total installed capital cost by multiplying with an appropriate factor.
- Reagent consumptions are specified from experimental and published data. Reagent costs from published journals are used.
- Total operating costs are calculated taking into account labour, power, reagents, and maintenance.

Economic analysis data

The economic analysis was used to compare the costs of three proposed flow sheet options, as illustrated in Figure 7:

- Option 1: partial neutralization with coal followed by lime
- Option 2: neutralization with lime only
- Option 3: neutralization with coal only.

Table V provides a summary of the design criteria from the three flow sheet options. The analysis is based on a rate of AMD generation of 1750 m^3/d (Maree *et al.*, 2013). A neutralizing capacity of 1.4 m^3 AMD per ton coal, a solids bulk density of 1 t/m^3, and a heap height of 6 m were specified from the experimental results from the neutralization test for sample B (6 m column; –12.5 mm; Table III). A lime cost of R2000 per ton and a lime neutralizing capacity of 15 kg $Ca(OH)_2$ per m^3 AMD were used (Table III).

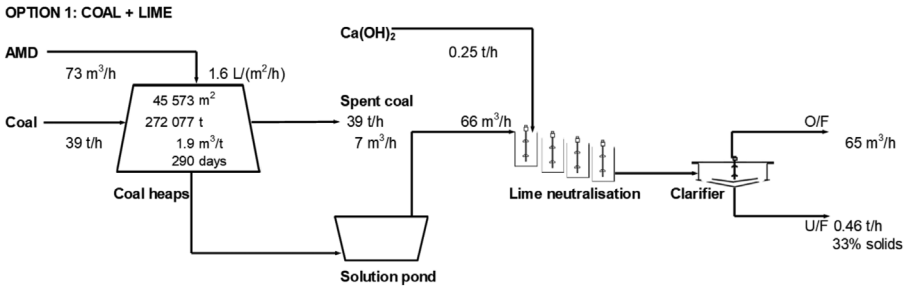

Figure 7a—Flow sheet – neutralization with coal and lime

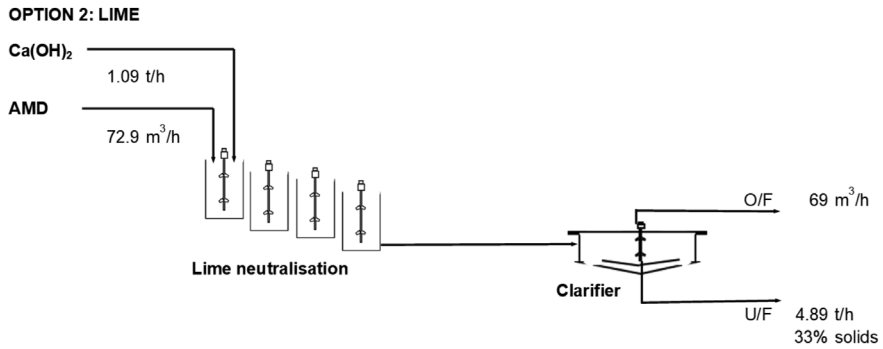

Figure 7b—Flow sheet – lime neutralization

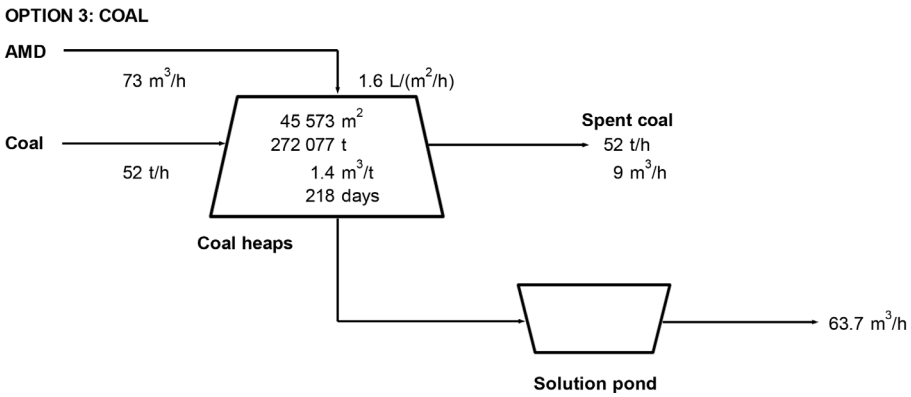

Figure 7c—Flow sheet – coal neutralization

A summary of the capital costs is provided in Figure 8, and a summary of operating costs in Figure 9. Amortized capital is included in the operating costs, based on a 10% annual interest and a 10-year payback period. A South African electricity cost of 86 cents per kWh was used (Deloitte, 2017). Costs of crushing and coal transportation are not included in the calculation of capital and operating costs. The analysis also does not include costs of a tailings facility for storage of the lime slurry.

Capital cost for lime neutralization was estimated at R18 million *versus* R27.6 million for coal neutralization, and R44 million for partial coal and partial lime neutralization (Figure 8). Operating costs (including amortized capital) were approximately R24 million for lime neutralization, followed by R17.4 million for coal and lime neutralization, and R9 million for coal neutralization (Figure 9).

Conclusions

- The concept of acid mine drainage treatment by means of percolation (or trickle) neutralization, through a packed bed of coal discards has been demonstrated in 1 m and 6 m high columns.
- Two of the three coal samples tested, *viz.* samples B and C, were more effective as neutralizing agents and may contribute to lime savings, as these showed significantly higher neutralizing capacities than sample A (<300 L AMD per ton coal). It is anticipated that samples with too low capacities would require to be used in excessive amounts, with accompanied costs. Therefore, the neutralizing capacity at which a waste coal sample would be deemed suitable needs to be determined by an overall economic assessment.
- Samples B and C, which yielded higher neutralizing capacities, also contained higher levels of alkalinity (expressed as $CaCO_3$ equivalent), *i.e.* 2.3% $CaCO_3$ compared with 0.48% $CaCO_3$ for sample A. However, Sample C only achieved 44% of sample B's neutralizing capacity at –6.3 mm crush size and virtually the same particle size distribution. The reason for this is unknown, but it is possible that the minerals responsible for

Table V

Summary of design criteria

PLANT CAPACITY	Unit	Option 1: Coal + lime	Option 2: Lime	Option 3: Coal
Lime neutralizing capacity	kg $Ca(OH)_2$ / m^3 AMD	15	15	15
Waste coal neutralizing capacity	L AMD / t coal	1400	1400	1400
AMD drainage	m^3/a	638 750	638 750	638 750
Coal (solid) feed	t/a	342 188	0	456 250
Percentage of neutralization performed with coal	%	75	0	100
HEAPS	**Unit**	**Option 1: Coal + lime**	**Option 2: Lime**	**Option 3: Coal**
Temperature	°C	Ambient	–	Ambient
Solution feed rate	m^3/h	73	–	73
Solution irrigation rate	$L/m^2/h$	1.6	–	1.6
Solids feed rate	t/h	39	–	52
Stacked bulk density	t/m^3	1.0	–	1.0
Area under neutralization	m^2	45 573	–	45 573
Mass under neutralization	t	272 077	–	272 077
Lift height	m	6	–	6
Residence time	days	290	–	218
LIME NEUTRALIZATION	**Unit**	**Option 1: Coal + lime**	**Option 2: Lime**	**Option 3: Coal**
Temperature	°C	Ambient	Ambient	–
Temperature pH	pH	7–8	7–8	–
Feed tank volume	m^3	132	146	–
Feed flowrate	m^3/h	66	73	–
Number of stages		4	4	–
Total residence time	h	5	5	–
Residence time per stage	h	1.3	1.3	–
Reaction tank volume	m^3	83	91	–
Power input per tank	kW	7.5	7.5	–
Clarifier area	m^2	0.0007	0.0030	–
Clarifier diameter	m	0.029	0.062	–
O/F flowrate	m^3/h	65	69	–
U/F solids	m^3/h	1.11	4.89	–
U/F solids concentration	%	33	33	–

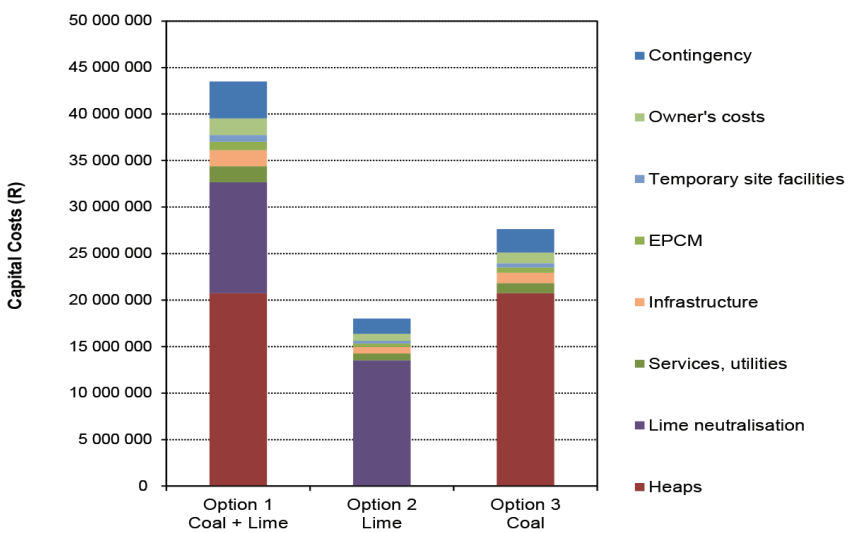

Figure 8—Summary of capital costs

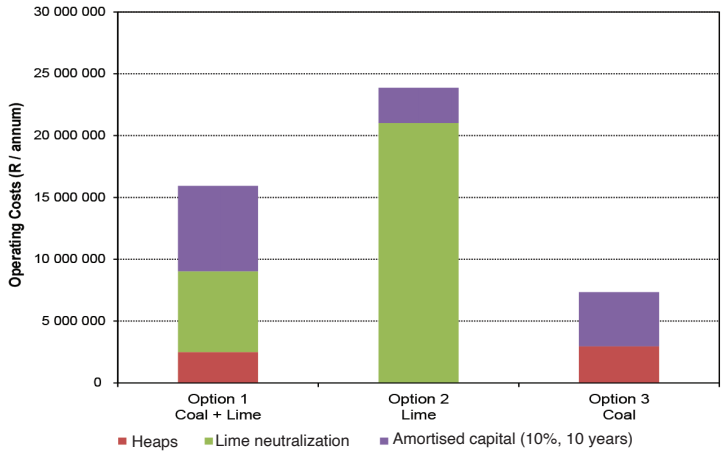

Figure 9—Summary of operating costs

neutralization in sample B were more liberated, and hence more exposed to the percolating solution. Quantitative, size class modal analyses from a mineralogical investigation, on both head and residues for these two samples, could shed more light on this.

- Neutralizing capacities increased with decreasing crush size, with sample B achieving capacities of between 804 L AMD per ton coal and 2112 L AMD per ton coal at –40 mm and –6.3 mm, respectively. Sample C neutralized 929 L AMD per ton coal at –6.3 mm crush size.
- High removal efficiencies for Al, Cu, Fe, Si, and Zn were achieved, with total sulphate removals of between 63% and 74%.
- Offloaded coal particles showed yellow/orange and reddish brown precipitation on parts of their surfaces. This suggests that species that were removed from the AMD solution deport to the coal surface as precipitation products.
- Capital costs were estimated at R18 million for lime neutralization, R27.6 million for coal neutralization, and R44 million for partial coal and partial lime neutralization. Operating costs (including amortized capital at 10% over 10 years) were estimated at R24 million for lime neutralization, followed by R17.4 million for coal and lime neutralization and R9 million for coal neutralization. Costs do not include crushing and transport of coal or tailings treatment of lime slurry. Hence it may be concluded that the coal neutralization route may constitute a lower cost alternative to lime neutralization provided suitable discard coal is freely available at the site and no additional capital for crushing is required.

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A novel economic-filter for evaluating sub-Saharan diamondiferous kimberlites

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Synopsis

Because of their high degree of geological complexity, kimberlite-hosted diamond deposits are exceedingly difficult to evaluate for economic viability. Accordingly, standard mineral asset evaluation protocols (*e.g.*, the Cost-, Market-, and Income Approaches defined in the SAMREC Code) may not hold sufficient predictive abilities for these deposit types, especially at the early stages of exploration. Here we present a novel tool, a cost filter approach towards preliminary evaluation of economic viability of southern African kimberlite-hosted diamond deposits, using the AK6 and BK11 diamond deposits from the Orapa diamond field as case studies. The development of this cost filter is underpinned by elements of both the Market Approach (*i.e.*, comparisons to similar deposits) and the Income Approach (*i.e.*, use of net present value (NPV) calculations) for mineral asset evaluation. Importantly, the cost filter is constrained through modification of only two primary variables (the average diamond value and the diamond grade) and thus differs significantly from other cost filters that rely on estimation and assumptions for every parameter input into an NPV calculation. The cost filter correctly predicts the sub-economic status of the BK11 diamond pipe, and is thus presented as a useful geo-economic tool for early stage kimberlite evaluation within the local southern African context. The approach and its theoretical underpinning foreseeably hold vast potential for use in the economic evaluation of other ore commodities, particularly where socio-economic and political risk factors can be negated by employing a geographic constraint.

Keywords

diamond, economic viability, kimberlites, southern Africa, cost models filter.

Introduction

Sub-Saharan Africa has an established diamond resource industry which began with the first alluvial diamond discovery in the Gariep River (South Africa) in 1866, followed shortly thereafter by the discovery of the first kimberlitic diamond on the farm Koffiefontein and then the Jagersontein and Dutoitspan pipes near Kimberley, all in 1870 (Davenport, 2013). In the next 40 years, significant diamond deposits were discovered in Zimbabwe, Namibia, Angola, and the Belgian Congo (now the Democratic Republic of Congo); and later in Lesotho and Botswana in the 1950s and 1960s (McKechnie, 2019 and references therein). In more than 150 years of mining history, southern Africa has consistently been the dominant producer of the world's diamond supply (McKechnie, 2019), producing some of the world's largest and most famous stones, including the 3106 ct Cullinan diamond (Cullinan Mine, South Africa, 1905), the 1109 ct Lesedi La Rona diamond (Karowe mine, Botswana, 2015), and the 972 ct Excelsior diamond (Jagersfontein Mine, South Africa, 1893). In the last two decades, southern Africa has continued to contribute vastly towards the world's total diamond output, both in terms of value and volume, producing around 40–50% of global supply (Zimnisky, 2017; Kimberley Process Statistics 2004–2016), with the remainder produced predominantly by Russia, Australia, and Canada.

Despite both the historical and present day importance of southern African diamond supply, parts of the region are regarded as being mature mineral provinces with diminished exploration potential (*e.g.*, South Africa), while other parts are hampered by various geopolitical risk factors (Campbell, 2019), and many of the region's tier-1 diamond operations are approaching the end of their production lifespans. Open pit operations at Orapa and Jwaneng are predicted to end in 2030 and 2035 respectively, while Venetia is in the process of moving from an open pit to underground operation (Table I). These local supply factors, compounded by the predicted 1–4% annual increase in natural diamond demand (Linde *et al.*, 2016), emphasize the need for continued exploration efforts to identify diamondiferous kimberlite pipes in sub-Saharan Africa that can be developed at lowered financial risk. To this end, the current

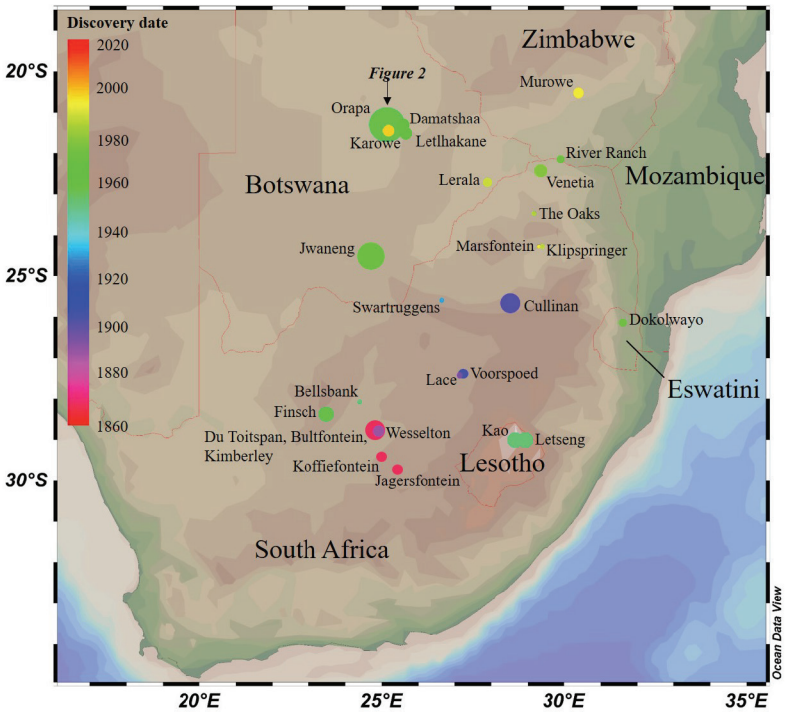

Figure 1—Distribution, size, and discovery date of selected southern African diamondiferous kimberlites. The location circles are scaled according to the original kimberlite pipe surface area and coloured according to the discovery date. All data extracted and redrawn from McKechnie (2019), and references listed therein

study evaluates the geological and financial data reported for 12 southern African kimberlite diamond mines, using a critical comparison between the economic Karowe deposit, hereafter referred to by its former name AK6 (Campbell and Jooste, 2017), and the sub-economic BK11 mineral occurrence as a useful point of departure. A key outcome of the work is the development of a novel economic filter which may be utilized as a geo-economic tool for early assessment of diamond projects, to ultimately improve future diamond mining opportunities in sub-Saharan Africa.

Use of economic filters in mineral asset valuation

Although less precise than full feasibility studies, simple economic filters (or cost models) have found extensive use as a tool for broadly discriminating between economic and non-economic mineral assets (*e.g.*, Long and Singer, 2001, and references therein). Much of this work has been pioneered by the US Bureau of Mines (Camm, 1991; Smith, 1992) and the US Geological Survey (Singer, Menzie, and Long, 1998; Long and Singer, 2001; Robinson and Menzie, 2014). The basic premise underpinning these works is that empirical data from existing mines can be used to predict operating costs and capital expenditure for yet-unmined mineral deposits (or, more correctly, mineral occurrences). This predicted data in turn can be used to model net present value (NPV) calculations to discern whether or not a mineral asset is likely to be economically viable under the current economic conditions. The results from these theoretical economic filters can then be plotted on a grade-tonnage graph to test whether the predictions of economic viability hold true against empirical data from known mines/deposits. This approach has been applied successfully to open-pit gold-silver deposits (Singer, Menzie, and Long, 1998), underground massive sulphide deposits (Singer, Menzie, and Long, 2000), and porphyry copper deposits (Robinson and Menzie, 2014).

Economic filter development: A case study using AK6 and BK11 as examples

Our approach towards developing a cost filter for southern African diamondiferous kimberlites differs fundamentally from the approach described in the preceding section. Instead of using predicted expenditure cash flows to model a theoretical NPV for an unknown deposit, we effectively 'reverse engineer' the process by modifying the input parameters for an NPV calculation of a known economic kimberlite deposit until this deposit becomes uneconomic. To limit the effects of external factors on the results (*e.g.*, socio-political risks, differences in currency strength, labour costs, *etc.*), our study considers two spatially associated open-pit kimberlite deposits; AK6 (economic in 2017) and BK11 (sub-economic in 2017). It should be noted that this geographically constrained approach assumes broadly similar capital costs, operating cost structures, and fiscal regimes for any sub-Saharan kimberlite mining operation. It is recognized that there will always be local differences driven by scale, mining differences, processing differences, and other cost differences, but the goal of the filter is to provide a tool for initial assessment. The tool will also be valid only if statistically relevant sampling for diamond grade and average diamond value has been conducted.

Geological background

The AK6 and BK11 kimberlites are located in east-central Botswana and form part of the Orapa kimberlite field (Figure 2). The Orapa kimberlite field comprises 86 kimberlites that were emplaced between 111 and 85 Ma (Preston *et al.*, 2012). These kimberlites are found towards the north-western margin of the Kalahari Craton where a deep, stable, and relatively cool lithospheric keel provided ideal conditions for diamond formation and, ultimately, subsequent entrainment (*e.g.*, Stachel *et al.*, 2003; Deines and Harris, 2004). The kimberlites crosscut the Archean

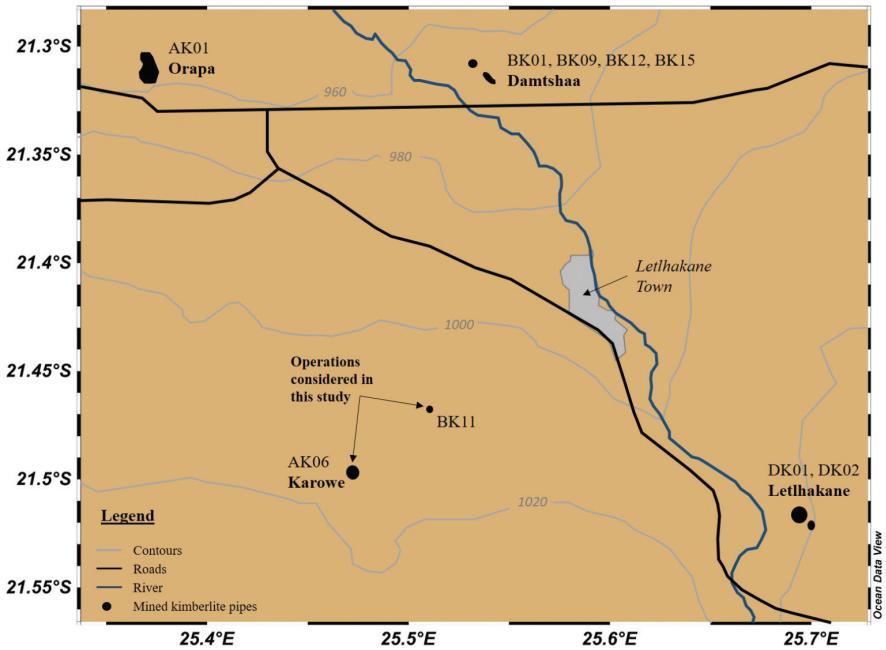

Figure 2—Kimberlite mining operations in the Orapa diamond field. Contours and approximate kimberlite pit sizes are redrawn from Google Earth imagery

basement lithologies and intrude into the 200–250 m thick Karoo Supergroup sediments and overlying basalt of the Stormberg Group (approx. 130 m thick). The upper host rocks units have been highly calcretized due to extended periods of weathering during the late Cenozoic (Lucara Diamond Corp., 2014), and are veneered by a thin layer of Quaternary-aged Kalahari sands.

The AK6 kimberlite comprises three distinct lobes that have been divided into 23 different facies assemblages (Lucara Diamond Corp., 2014). Current diamond production is focused on the South Lobe, which is relatively homogeneous in its facies distribution and hosts the majority of AK6's large and exceptional Type-II diamonds (Lucara Diamond Corp., 2014). In comparison, the BK11 kimberlite is a single, broad pipe that has been divided into nine different kimberlite facies, of which the fragmental upper crater facies (RVK-K1) is the most diamondiferous (Senlis Consultancy Limited, 2015). Mining of BK11 commenced in 2010, but ceased in 2012 on account of production not meeting forecast performance levels. Both pipes are endowed with Type-I and Type-IIa diamonds (Lucara Diamond Corp., 2014; Senlis Consultancy Limited, 2015).

Table I

Economic geology of the southern African diamond mining operations considered in this study (after De Wit *et al.*, 2016; Petra Diamonds, 2017; Paul Ziminsky's Diamond Analytics; De Beers Analyst Seminar)

Tier	Mine	Age (Ma)	Surface expression (ha)	Resource (Mt)	Grade (cpht)	Contained diamonds (Mct)	Cut-off size (mm)	LoM (years)	Value US$/ct
1	Jwaneng (Botswana)	240	54	320	117	343	1.47	20	228
	Orapa (Botswana)	93	118	505	93	364	1.65	21	95
	Venetia (RSA)	519	23.3	238	103	193	1.00	27	122
	Cullinan (RSA)	1115	32	429	47	193	1.00	50	126
2	AK6 (Botswana)	88, 93	10	69	13	11	1.25	15	696
	Finsch (RSA)	118	18	85	60	51	1.00	16	101
	Voorspoed (RSA)	131	12	33	22	7	1.47	10	125
	Lethlakane (Botswana)	93	15	19	18	21	1.65	4	97
3	Letseng (Lesotho)	95	20.6	210	2	5	2.00	30	2229
	Mothae (Lesotho)		9	39	3	1	2.00	12	1063
	Koffiefontein (RSA)	90	11	155	4	7	2.50	11	506
	BK11 (Botswana)	93	9	20	4	1	1.60	7	230

Comparison with other southern African kimberlite diamond deposits

Despite the geological similarities between AK6 and BK11, only the AK6 pipe is considered economic under the current market conditions. Table I reports pertinent data (grade, tonnage, average diamond value) for AK6, BK11, and a selection of other southern African kimberlite diamond mining operations. The data has been collected from the respective mines' 2016/2017 online published company reports and various other sources that are available in the public domain (*e.g.*, De Wit *et al.*, 2016; Petra Diamonds, 2017; Paul Zimnisky's Diamond Analytics; Anglo American plc, 2017). The selected non-alluvial diamond deposits have been classified, on the basis of million carats produced per annum (Mct/a), into the categories defined by De Wit *et al.* (2016); *viz.* tier 1: > 200 Mt and 0.4 Mct/a; tier 2: > 0.4 Mct/a; tier 3: 0.05–0.4 Mct/a. According to this classification scheme, the AK6 pipe is regarded as a tier 2 kimberlite diamond deposit, whereas BK11 is classified as tier 3. The latter deposit is further characterized by the lowest grade (measured in carats per hundred tons, cpht) and the smallest number of contained diamonds (measured in Mct). The reported diamond value (measured in US dollars per carat, however, is greater than the median of the diamond values (at 2016/2017 US dollar prices) reported for the other operations recorded in this data-set.

An economic filter constrained using net present value (NPV) calculations

Figure 3 plots the grade and value data from Table I for the kimberlite operations considered in this study. Although economic deposits can be discerned qualitatively or empirically from sub-economic or uneconomic deposits using such a two-parameter plot (*e.g.*, Kjarsgaard 2007; Kjarsgaard, Januszczak, and Stiefenhofer, 2019), we employ a novel theoretical underpinning towards constraining the absolute position of the economic filter in grade-value space (shown as the dashed line in Figure 3). This was achieved by considering a series of NPV calculations for the AK6 kimberlite; a pipe which is profitable (*i.e.*, has a strongly positive NPV) under current economic conditions.

The basic or simplified financial modelling of the AK6 mineral asset was achieved using data updated from Lucara's NI 43-101 Technical Report (2014) and the standard NPV calculation (Equation [1]). The discount rate used in this equation was set at 8%, which is the standard for the diamond industry, and the calculation was run over 15 years (equal to the life of mine, LoM). Lucara's predicted revenues, cash flows, and capital expenditures for AK6 (Lucara, 2014) were updated for the LoM on the basis of operational statistics between 2013 and 2017, which saw a significant increase in the recovery of high-value Type-IIa diamonds.

$$NPV = PV \times 1/(1+i)^{n} \qquad [1]$$

where, i = discount rate, PV = present value of net cash flow, and n = number of years.

The revenue stream in this model includes two key variables, *viz.* the diamond grade (how many carats of diamond are extracted per ton of kimberlite material mined) and the average diamond value (the average value of each extracted stone in US dollars per carat). A decrease in the value of either of these two variables will thus have a strong negative impact on the NPV calculated for the AK6 mineral asset. Based on these relationships, a more quantitative positioning of the economic filter was derived by simulating iterative 10% reductions in either the grade (cpht) or the value (US dollars per carat) until the financial model reflected a negative NPV, *i.e.*, the operation was no longer deemed profitable (black dots in Figure 3). The resulting economic filter is therefore represented by an exponential regression line (Equation [2]) that has been 'forced' through these simulated points of negative NPV while still maintaining a 'best-fit' to the 12 empirical data-points representing the kimberlite operations considered in this study.

$$y = 1725.9x^{-0.991} \qquad [2]$$

where y is the average stone value per carat and x is the average grade of the mineral asset.

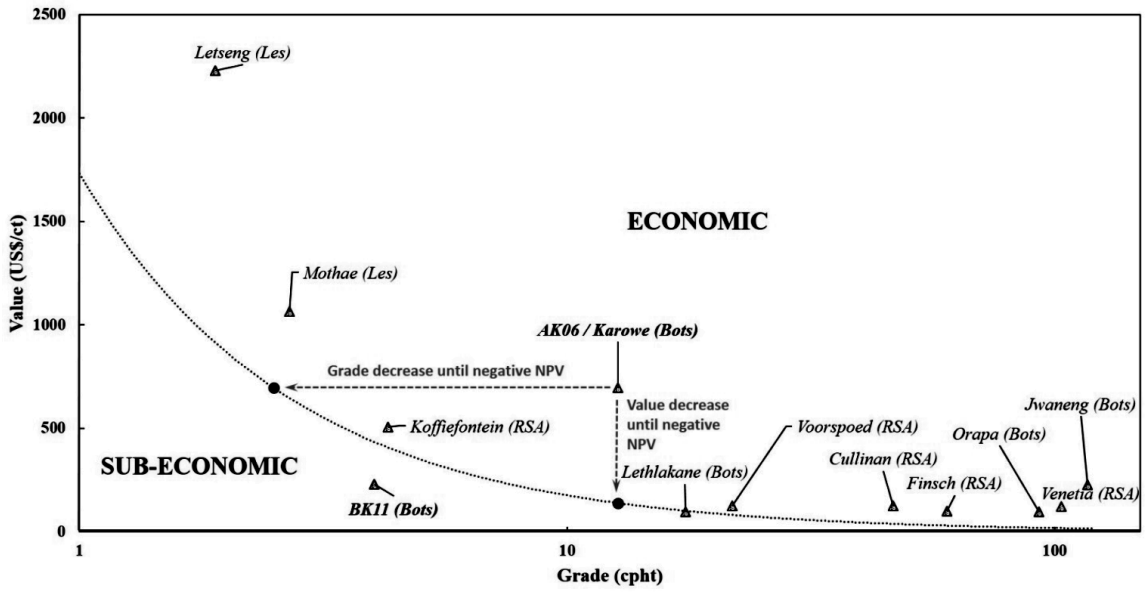

Figure 3—Novel economic filter for early assessment of kimberlite mining viability. Absolute position of the filter line is given by Equation [2] (see main text) and was constrained by iteratively decreasing the grade and value parameters of a simplied NPV calculation for the Karowe mining operation (AK6 kimberlite) until the NPV became negative (*i.e.* economically unfeasible)

Mineral assets that fall above this line in grade-value space are deemed economic, whereas those that fall below this nominal cut-off will be either too low in grade (relative to the average diamond value) or have insufficient high-value stones (relative to grade), and are thus deemed sub-economic or uneconomic.

Conclusion

The successful exploitation of a diamond deposit depends on a diverse array of geological, mining, metallurgical, financial, political, and social parameters (Kjarsgaard, Januszczak, and Stiefenhofer, 2019). This high level of complexity poses a significant degree of difficulty when it comes to mineral asset valuation. To this end, several different approaches have been developed, *viz.* the Income Approach, the Cost Approach, and the Market Approach (SAMVAL, 2018), each with variable applicability to the different stages of mineral asset development (*e.g.*, Davis, 2002; Njowa, Musingwini, and Clay, 2010). Our work introduces a novel methodology of developing an economic filter to assist with mineral asset valuation, specifically for use in comparing diamondiferous kimberlites discovered in southern Africa. Uniquely, the presented methodology combines elements of the established Market Approach (comparison among existing kimberlite operations) with elements of the Income Approach (NPV calculations for an existing operation used to constrain the absolute position of the economic filter in grade-value space). Such a combined approach effectively simplifies a preliminary economic evaluation into two parameters, while inherently accounting for operation-scale considerations (NPV constraints) under assumed broad regional similarities (regional socio-political conditions/ risk factors). We foresee that the developed economic filter will be useful in preliminary economic consideration of new, or known but yet unmined, kimberlite targets in sub-Saharan Africa; although cognisance must be taken of the fact that this model was developed from known production values between 2012 and 2017 (and diamond prices at the time). Crucially, application of the economic model to new deposits can be successful only if representative and statistically relevant sampling of that deposit has been undertaken to ensure an accurate knowledge its diamond grade and average diamond value. Finally, and importantly, we foresee that the novel approach holds high potential for adaptation and use in valuation of a range of commodity types (*e.g.*, coal, Witwatersrand gold, *etc.*) for which the deposit style and the local socio-economic parameters (*e.g.*, fiscal regimes and operating cost structures) are comparable.

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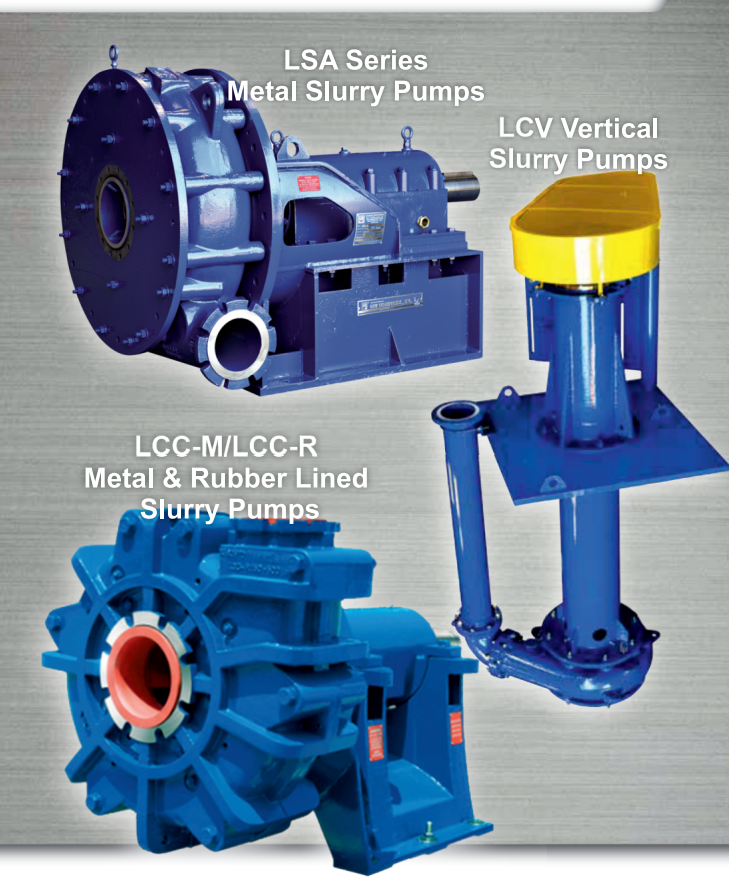

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Assessment of whole-body vibration exposure of mining truck drivers

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Synopsis

Whole-body vibration (WBV) exposure measurements taken from 105 truck drivers employed in 19 mines and other workplaces were evaluated with the criteria prescribed in EU 2002/44/EC directive, BS 6841 (1987), ISO 2631-1 (1997). and ISO 2631-5 (2004) standards. The highest vibration acceleration was measured on the vertical Z-axis. The highest WBV exposure occurred in the RETURN, HAUL, and SPOT phases while the lowest exposure took place in the LOAD and WAIT phases. Crest factors on all axes were generally greater than nine, yet strong correlation coefficients were achieved in VDV–eVDV analyses. Driver seats generally dampened the vibration along the Z-axis but exacerbated it along X and Y axes. The dominant frequency for the X and Y-axes rose up to 40 Hz while it ranged between 1 Hz and 2.5 Hz along the Z-axis. While the probability of an adverse health effect was higher with BS 6841 (1987) and ISO 2631-1 (1997) standards, it was low according to EU 2002/44/EC and ISO 2631-5 (2004). The 91 t, 100 t, and 170 t capacity trucks produced lower vibration magnitudes. Drivers were exposed to approximately equivalent levels of WBV acceleration and dose in contractor-type trucks and mining trucks. Rear-dump trucks exposed their drivers to a slightly higher level of vibration than bottom-dump trucks. Underground trucks exposed their drivers to a significantly higher level of vibration than mining trucks. Both driver age and driver experience were inversely proportional to vibration acceleration and dose. Conversely, there was a positive relationship between the truck service years and the WBV acceleration and dose to which drivers were exposed to. Loads of blocky material exposed drivers to higher vibration acceleration and dose levels than non-blocky material.

Keywords
whole-body vibration, mining truck, A(8), BS 6841, EU 2002/EC/44, ISO 2631-1, ISO 2631-5, VDV(8).

Introduction

Operators of trucks, buses, locomotives, helicopters, heavy machinery, and farm equipment as well as workers using vibrating machines are exposed to occupational whole-body vibration (WBV). Continuous exposure to mechanical vibration can cause irreversible physical damage, depending on the intensity and frequency of the vibration. The International Labour Organization (ILO, 1977) described vibration as a professional hazard, emphasising that 'measures should be taken to protect workers from vibration' and that the responsible authorities should establish criteria to determine the hazard.

There is consensus among researchers about the vibration exposure of machinery operators and the long-term adverse health effects of WBV (Miyashita *et al.*, 1992; Mandal and Srivastava, 2010; Aye and Heyns, 2011). Risk factors include muscle fatigue, decreased stability, vestibular dysfunction, and impairement to the female reproductive system (Seidel and Heide, 1986; Bongers *et al.*, 1988; Griffin, 1998). Frequencies between 1 Hz and 20 Hz can cause damage to the body in the form of back pain, spinal degeneration, stomach problems, headache, and sleep problems (Thalheimer, 1996; Okunribido, Magnusson, and Pope. 2006; Eger *et al.* 2008).

Factors affecting WBV exposure such as operating conditions, tonnages, service lives of machinery, and properties of the material handled were evaluated individually or collectively. Mandal *et al.* (2006) stated that 18% of the workers in the Indian mining industry were subjected to occupational vibration. Noorloos *et al.* (2008) reported that the vibration magnitude caused by a vehicle depends on many factors such as conditions of the site, maintenance status, operator's driving style and speed, type of propelling mechanism, type of seat, characteristics of material handled, and operator experience. In

contrast, Darby, Heaton, and Mole (2010) stated that the exact cause of back pain is unclear, but back pain is more common in those working over long distances or on bad ground conditions. They also commented that shocks and sudden jolts during driving would increase pain. Working for prolonged durations in an incorrectly adjusted seat position, sitting for a long time without changing the posture, and exposure to shocks and sudden jolts during operation are among the causes of back pain, the most reported effect of WBV exposure. Waters *et al.* (2008) emphasized the possibility of a causal relationship between working as a heavy equipment operator and lower back pain. However, according to Burgess-Limerick (2012) it is difficult to clearly demonstrate the link between WBV and back pain, as there are many possible causes of back pain.

In a survey conducted by Palmer *et al.* (2000) it was found that the BS 6841 (1987) limit of 15 $m \cdot s^{-1.75}$ for the vibration dose value (VDV) during daily operations was exceeded for truck drivers, farmers, agricultural workers, and forklift drivers in Great Britain. Cann, Salmoni, and Eger (2004) studied the effects of driver experience, truck service years, road conditions, truck type, truck mileage, and seat type on WBV exposure. They concluded that the drivers were not subjected to negative health effects caused by daily exposure in accordance with the ISO 2631-1 (1997) standard. In an evaluation of dumper operators' WBV exposure in India, Mandal and Srivastava (2010) found that the RMS acceleration values along the dominant Z-axis ranged from 0.644 m/s^{-2} to 1.820 m/s^{-2}. When evaluated together with an average daily exposure of 5 hours, it was found out that all dumpers caused high health risks according to the ISO 2631-1 (1997) standard.

Previous studies indicate that vehicles of different models affect WBV exposure. Village, Morrison, and Leong (1989) studied mining trucks and reported that vehicle size plays an important role in WBV exposure. Maeda and Morioka (1998) found that heavy road conditions caused an increase in WBV exposure of garbage truck drivers. In contrast, Noorloos *et al.* (2008) conducted surveys of 467 participants using various types of vehicles, including automobiles, minibuses, trucks, wheel loaders, dumpers, excavators, bulldozers, steamrollers, mobile cranes, and asphalt machines. There was no significant relationship between WBV exposure and low back pain, which was attributed to the small number of samples. Lundström and Holmlund (1998) reported that the WBV acceleration recorded in the Z direction was much more detrimental than vibration in the X and Y directions, supporting other studies and the ISO 2631-1 (1997) standard.

Nishiyama, Taoda, and Kitahara (1998) conducted a study on a small number of samples in order to evaluate drivers' back pain and determined that old model trucks caused higher WBV exposure. Kumar (2001) recorded the vibration measurements to which dumper operators in open-pit mines were subjected along three orthogonal axes. Vibration-induced effects were detected in operators' third lumbar and seventh neck vertebrae. The data obtained from new and old trucks of different makes and carrying capacities revealed that the exposure threshold of ISO 2631-1 (1997) has been exceeded. In another study, Kumar (2004) commented that the driver's gender and truck capacity have no significant impact on vibration; however, the body mass of the driver, the vehicle segment, and the measurement region showed significant differences in vibration. He concluded that heavy mining trucks pose a significant health hazard to operators in extreme mining conditions and generate WBV accelerations that often exceed ISO standards. Smets, Eger, and Grenier (2010) evaluated the vibration exposure of heavy truck drivers against the ISO 2631-1 (1997) and ISO 2631-5 (2004) standards. According to the ISO 2631-1 (1997) criteria, drivers were exposed to vibration levels above the daily exposure limit, but there was a low probability of adverse health effects in accordance with ISO 2631-5 (2004). The authors emphasized the discrepancy between the two standards.

Mayton, Jobes, and Miller (2008) and Mayton *et al.* (2018) found that old trucks gave higher vibration values, with the vibration acceleration in the Z-axis being dominant. An evaluation of the WBV and GPS position data revealed that most of the shocks occurred during loading and unloading, and from the potholes in the road. In another study by Mayton, Jobes, and Gallagher (2014) if was found that roughly half of the dominant vibration was either in the Z-axis or Y-axis. Frimpong, Galecki, and Chang (2011) stated that significant improvements in production and economy have been achieved with the use of trucks of larger than 360 t capacity and electric excavators of >90 t per cycle. However, in terms of operator health, significant negative side effects of continuous work and long working hours were also revealed.

In this study, WBV exposure measurements taken from the driver's seat in 105 trucks of different types, brands, and models, which were deployed in several open pits and one underground mining operation in Turkey, were analysed. The assessment of truck drivers' WBV exposure was based on four criteria; BS 6841 (1987), ISO 2631-1 (1997), European Directive 2002/44/EC (2002), and ISO 2631-5 (2004). The potential health risk categories described in each criterion are given in Table I.

Materials and method

Data acquisition

The seat accelerometer, which was mounted in a polyurethane housing, was placed and secured on the seat pan to accommodate the driver's ischial tuberosities. A floor-type accelerometer was used to determine the seat effective amplitude transmissibility (SEAT) factor. It was placed on the floor in an appropriate position, very close to where the seat foot joins the floor. In order to prevent any shock-induced damage caused by the operator to the highly sensitive accelerometer, a steel container was built and attached to the floor with strong magnets.

A SV106 model six-channel vibration analyser (serial no. 34613) manufactured by Svantek Ltd (2013) was used to record and process data from the accelerometer's $^1/_3$-octave (with the centre frequencies from 2.50 kHz down to 0.40 Hz) digital passband filters in real time. The sampling rate was 6000 Hz with 16-bit resolution. Measurement results were stored on a micro-SD flash card and downloaded to a PC using a USB interface, and processed on the environmental monitoring module of the SvanPC++ software (Svantek, 2015).

A SV38V triaxial seat-type disk accelerometer (serial no. 32980) was used to determine drivers' WBV exposure at the seat pan. This required a supply voltage of 5.2 V DC and had a sensitivity of 50 mV/(m/s^2) at 15.915 Hz. Another SV151 triaxial accelerometer (serial no 31359) was used to determine the vibration level on the cabin floor. It required a supply voltage of 3.3 V DC and had a sensitivity of 5.81 mV/(m/s^2) at 15.915 Hz. The instrumentation set-up is shown in Figure 1.

Figure 1—The WBV exposure recording instrumentation set-up

Vibration measurement

A haul truck cycle is composed of five consecutive phases: approaching the loader or dumping point at reduced speed (SPOT), being loaded when it is immobile (LOAD), travelling with a full load (HAUL), offloading when it is immobile (DUMP), and travelling with no load (RETURN). Apart from the DUMP and SPOT phases, the truck may come to a complete halt with the engine idling in a specific phase. In this situation, that part in the acceleration-time log is regarded as a separate phase (WAIT). Lastly, the overall WBV exposure of the driver in a typical truck cycle was recorded in (RECORD). It should be noted that although effort was made to monitor at least a complete truck cycle, a few records did not include all phases. For example, a driver moved to park the truck instead of travelling to the loading point after dumping, as he thought the shift had finished.

The WBV exposure was recorded from six channels, three of which were from the seat ($Seat_X$, $Seat_Y$, $Seat_Z$) and three from the cabin floor ($Floor_X$, $Floor_Y$, $Floor_Z$) in accordance with TS EN 1032+A1 (TSE, 2011), which refers to ISO 2631-1 (1997). The frequency-weighted acceleration (a_w), vibration dose (VDV), maximum transient vibration value (MTVV), peak vibration acceleration (PEAK), and peak-to-peak vibration acceleration (P-P) were recorded on both the seat pan and the cabin floor. The recording times varied from 00:04:46 hours to 00:52:37 hours.

A researcher travelled with the driver to observe the hauling operation during the recording process. A precision stopwatch was started simultaneously with the vibration analyser, to mark the start and finish times of the phases of a cycle. For all cases, the first 10 seconds were removed from the RECORD as it was observed that the driver's involuntary movements while he was trying to sit down on or get up from the seat could result in very high vibration accelerations.

Parameters related to WBV in all phases were calculated in accordance with the ISO 2631-1 (1997) and ISO 2631-5 (2004) standards.

BS 6841 (1987) uses the w_b frequency weighting filter for the vertical accelerations on the seat cushion, while ISO 2631-1 (1997) utilizes the w_k frequency weighting filter. As there are minor differences between the two frequency weighting filters, in this study, data recorded with the w_k filter was used while assessing the probability of adverse health risk in accordance with both standards.

Contrary to ISO 2631-1 (1997), the vibration dose for the seated person is calculated using different multiplying factors (unity for all axes) according to the BS 6841 (1987) standard. For this reason, the daily vibration dose values were calculated according to both standards and the driver exposures were evaluated by considering the health risk criteria of the relevant standards given in Table I.

Study domain

Out of 105 trucks 33 were of 30 t nominal load capacity, 48 of 77 t, two of 91 t, three of 100 t, and 19 of 170 t. The 30 t units were typical contractor trucks while the remaining 72 were mining trucks. All contractor trucks and 69 out of the 72 mining trucks were dumpers, while the three trucks of 100 t capacity were bottom-dump machines.

All mines, trucks, and drivers were coded. The test sites included coal, gold, and iron mines; aggregate, marble, gypsum, clay, and limestone quarries, road construction sites, and ore preparation and scrap iron plants. The types of trucks and operations from which WBV exposure measurements were taken are illustrated in Figure 2. Distribution of trucks among workplaces is given in Table II.

Results and discussion

RECORD phase

Evaluation by vibration acceleration

The equivalent vibration acceleration normalized to an 8-hour shift, A(8) on the driver's seat of the trucks for the worst-case axis (WCA) ranged from 0.319 m/s² to 1.172 m/s², with mean and standard deviation of 0.707 ± 0.174 m/s². On the other hand, $A(8)_{xyz}$ for the vector sum (VS), as recommended in BS 6841 (1987), ranged from 0.427 m/s² to 1.444 m/s² with mean and standard deviation 0.949 ± 0.198 m/s².

The WCA with the highest vibration was the Y-axis in 9 trucks, and the Z-axis in 96 trucks. This result is corroborated by previous studies (Özkaya, Goldsheyder, and Willems, 1997; Newell, Mansfield, and Notini, 2006). The vibration acceleration measured by Cann *et al.* (2005) was 0.79–0.83 m/s² on the X-axis, 0.81–0.97 m/s² on the Y-axis, and 1.08–1.36 m/s² on the Z-axis. In a study on 18 trucks, Mandal and Srivastava

Table I

Potential health risk zones for WBV (Eger and Godwin, 2014)

	ISO 2631-1(1997)		EU 2002/44/EC (2002)		BS 6841 (1987)	ISO 2631-5 (2004)	
Health risk	**A(8)v (m/s²)**	**VDVc (m/s$^{-1.75}$)**	**A(8) (m/s²)**	**VDV (m/s$^{-1.75}$)**	**VDV (m/s$^{-1.75}$)**	**Sed (MPa)**	**R**
Low	<0.45	<8.5	<0.5	<9.1	<0.50	<0.80	
Moderate	0.45–0.90	8.5–17.0	0.5–1.15	9.1–21.0	–	0.5–0.80	0.80–w1.20
High	>0.90	>17.0	>1.15	>21.0	>15.0	>0.80	>1.20

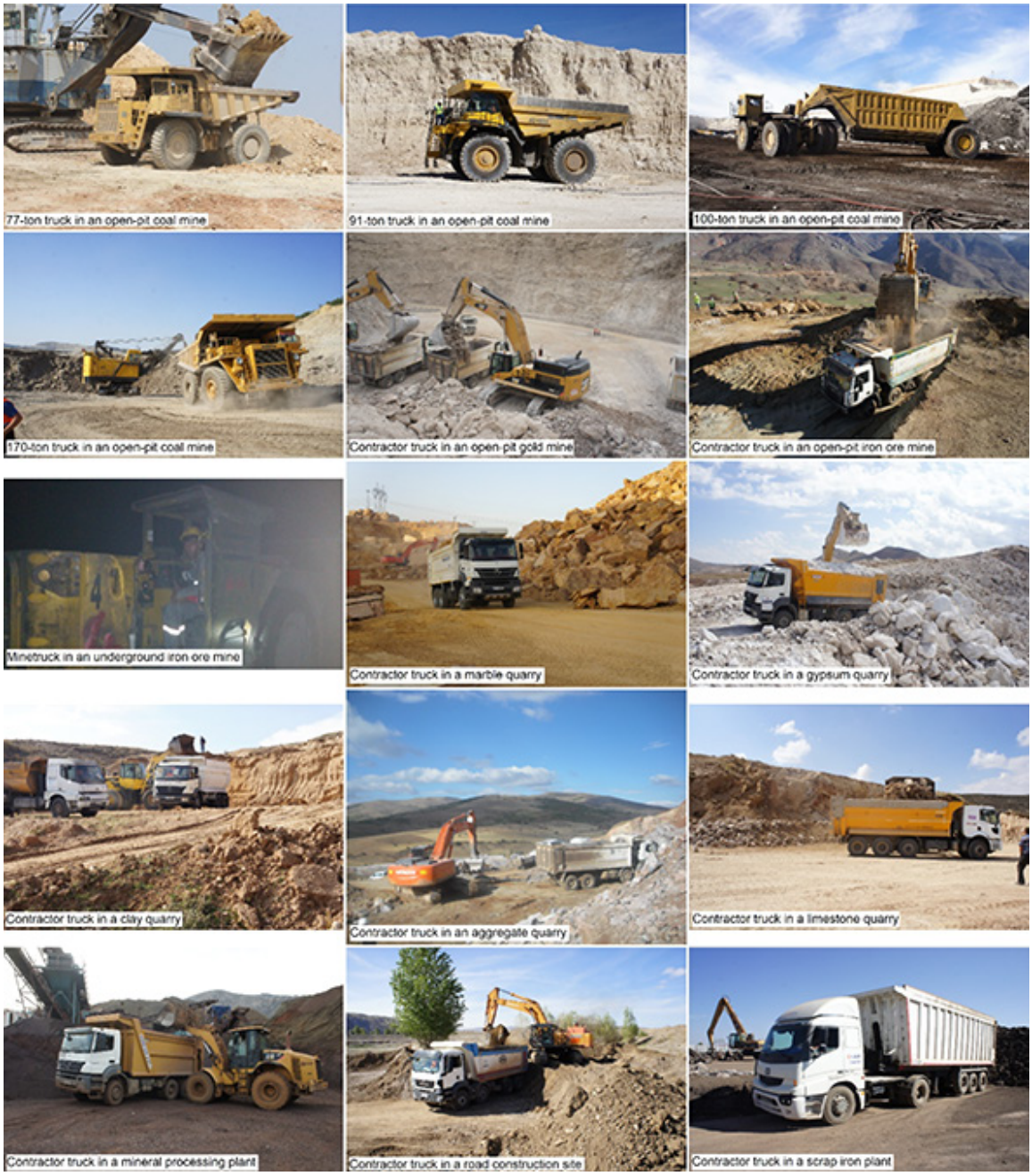

Figure 2—Types of truck and operation

(2010) recorded the highest vibration on the Z-axis. Measured values ranged from 0.64 m/s^2 to 1.82 m/s^2 and the average was 1.10 m/s^2. Eger *et al.* (2011) recorded vibration values in the dominant Z-axis between 0.89 m/s^2 and 1.18 m/s^2. Chaudhary, Bhattacherjee, and Patra (2015) measured the highest vibration value in the Z-axis as 1.61 m/s^2. In the study by Burström *et al.* (2016), which included 95 machines of different models and capacities, the daily average vibration exposure was between 1.9 and 6.7 hours, and the average A(8) value ranged from 0.2 m/s^2 to 1.0 m/s^2.

Table III shows that according to the EU 2002/44/EC (2002) directive, the vast majority of the trucks are within the health guidance caution zone (HGCZ) when the WCA and VS criteria are taken into consideration. Considering the WCA criterion according to ISO 2631-1 (1997), which is more conservative, the majority of the trucks again remain within the HGCZ, while the greater part of the trucks are above the daily ELV limit as per the VS criterion as the averaged A(8) and $A(8)_{xyz}$ are just below and above the 0.90 m·s^{-2} limit respectively.

Considering the WCA criterion, the time required to reach the daily exposure action value ($EAV_{TT(RMS)}$) of truck drivers ranged from 01:27:22 hours to 19:37:45 hours with mean and standard deviation of 04:53:00 ± 2:50:52 hours. The time required to reach the daily exposure limit value ($ELV_{TT(RMS)}$), however, ranged between 07:42:09 and 103:50:20 hours with mean and standard deviation 25:50:00 ± 15:03:52 hours. In this case, 95 out of 105 trucks exposed their drivers to enough vibration to reach the daily exposure action value (EAV) before the end of the 8-hour shift, while only one truck reached the daily exposure limit (EL) before the end of the shift.

The crest factor shows the sudden load and shocks to which the drivers are exposed. In cases where this is greater than nine, exposure assessment should be done by considering the vibration acceleration and vibration dose values together in accordance with ISO 2631-1 (1997). In the RECORD phase, the crest factor was greater than nine for 51 trucks in the X-axis, 36 trucks in the Y-axis, and 70 trucks in the Z-axis (Table IV). Yet, in another study on nine trucks by Mandal *et al.* (2006) the crest factor was between 4.4 and 8.2.

Table II

The number of trucks in the workplaces

Mine type[a]	Workplace	Capacity class 30 t	77 t	91 t	100 t	170 t	Σ	ΣΣ
①	Mine 1 Mine 3 Mine 4 Mine 5		2 15 22 9	2	3	4 6 9	11 21 31 9	72
②	Mine 1	7					7	7
③	Mine 1	2					2	2
④	Mine 1	2					2	2
⑤	Mine 2 Mine 3	1 1					1 1	2
⑥	Mine 1 Mine 2	3 1					3 1	4
⑦	Mine 1	3					3	3
⑧	Mine 1 Mine 2	2 1					2 1	3
⑨	Mine 1	3					3	3
⑩	Plant 1	1					1	1
⑪	Plant 1 Plant 4	3 1					3 1	4
⑫	Plant 1	2					2	2
Σ		33	48	2	3	19	105	105

a
① Open-pit coal mine
② Open-pit gold mine
③ Open-pit iron ore mine
④ Underground iron ore mine
⑤ Marble quarry
⑥ Gypsum quarry
⑦ Clay quarry
⑧ Aggregate quarry
⑨ Limestone quarry
⑩ Mineral processing plant
⑪ Road construction site
⑫ Scrap iron plant

Table IV

Crest factors for vibration acceleration

Crest factor	CF_x	CF_y	CF_z
Minimum	6.359	5.263	5.982
Maximum	24.975	29.223	46.167
Mean	9.779	8.792	12.155
Standard deviation	3.364	2.806	6.019
≥9	51	36	70
<9	54	69	35

The frequency spectrum of WBV acceleration was evaluated with $^1/_3$ – octave band distribution. The dominant frequency ranged between 10 and 40 Hz on the X-axis, 20–40 Hz on the Y-axis, and 1–2.5 Hz on the Z-axis. Village, Morrison, and Leong (1989) found that the dominant frequencies were between 1.6 and 2 Hz in the X- and Y-axes and 3.15 Hz in the Z direction. Sherwin *et al.* (2004) recorded the highest vibration value on the Z-axis with 3.2 Hz as the dominant frequency. Mansfield, Newell, and Notoni (2009) found that less than 1% of the vibration energy in the Z-axis was below 1 Hz. According to Smets, Eger, and Grenier (2010) the dominant frequencies were in the range of 1 Hz to 1.25 Hz.

The SEAT value, which shows the vibration isolation efficiency of the driver's seat, ranged from 0.494 to 1.702 for the X-axis, from 0.732 to 1.600 for the Y-axis, and from 0.589 to 1.286 for the Z-axis. The vibration was attenuated in 21 trucks in the X-axis, 12 in the Y-axis, and 75 trucks in the Z-axis. Driver seats attenuated vibration in the Z-axis while amplifying it in the other axes.

Evaluation by vibration dose

The equivalent vibration dose normalized to an 8-hour shift, VDV(8), on the driver's seat of trucks ranged from 7.684 m·s$^{-1.75}$

Table III

Comparison of health risk severity of truck drivers

A(8)						Human response			
ISO 2631-1 (1997)				EU 2002/44/EC (2002)			ISO 2631-5 (2004)		
WCA[a]		VS[a]		WCA[b]		VS[b]	Sed[c]		R[d]	
5	Low	1	Low	10	Low	1	Low	70	Low	100	Low
87	Moderate	45	Moderate	94	Moderate	89	Moderate	23	Moderate	4	Moderate
13	High	59	High	1	High	15	High	12	High	1	High
VDV(8)											
ISO 2631-1 (1997)				BS 6841 (1987)			EU 2002/44/EC (2002)			
WCA[e]		VS[e]		WCA[f]		VS[f]	WCA[g]		VS[g]	
2	Low	1	Low	62	Low	55	Low	2	Low	1	Low
74	Moderate	66	Moderate	—	Moderate	—	Moderate	92	Moderate	90	Moderate
29	High	38	High	43	High	50	High	11	High	14	High

a The frequency-weighted acceleration values corresponding to the EAV and ELV limits for 8-hour exposure are 0.45 m/s^2 and 0.90 m/s^2
b The frequency-weighted acceleration values corresponding to the EAV and ELV limits for 8-hour exposure are 0.50 m/s^2 and 1.15 m/s^2
c While the negative health effect of S_{ed} value below 0.5 MPa is low, the Sed value above 0.8 MPa has a high negative health effect
d While the negative health effect of R value below 0.8 is low, an R value above 1.2 has a high negative health effect
e Vibration dose values corresponding to EAV and ELV limits for 8-hour exposure are 8.5 m/s$^{-1.75}$ and 17.0 m/s$^{-1.75}$
f Vibration dose value corresponding to hazard limit for 8-hour exposure is 15.0 m/s$^{-1.75}$
g Vibration dose values corresponding to EAV and ELV limits for 8-hour exposure are 9.1 m/s$^{-1.75}$ and 21.0 m/s$^{-1.75}$

to 32.914 m·s$^{-1.75}$ with a mean and standard deviation of 15.479 ± 4.414 m·s$^{-1.75}$. VDV(8)$_{xyz}$ ranged from 8.366 m·s$^{-1.75}$ to 33.754 m·s$^{-1.75}$ with a mean and standard deviation of 16.820 ± 4.130 m·s$^{-1.75}$. According to the measurements taken by Mandal *et al.* (2006) on nine trucks, VDV values ranged from 7.71 m·s$^{-1.75}$ to 13.0 m·s$^{-1.75}$. Vanerkar *et al.* (2008) reported that the average vibration dose in 60 dumpers in an open pit iron operation was 10.81 ± 3.44 m·s$^{-1.75}$, and the average for a bauxite mine was 9.27 ± 3.98 m·s$^{-1.75}$. In a study on 18 trucks, VDV(8) ranged from 6.05 m·s$^{-1.75}$ to 25.13 m·s$^{-1.75}$ (Mandal and Srivastava, 2010). Burström *et al.* (2016) reported VDV(8) values between 7 m·s$^{-1.75}$ and 17 m·s$^{-1.75}$.

A comparison of the VDV(8) and VDV(8)$_{xyz}$ values of all trucks with the HGCZ limits according to BS 6841 (1987), ISO 2631-1 (1997) and EU 2002/44/EC (2002) is presented in Table III. When the exposures are examined with reference to ISO 2631-1 (1997) and EU 2002/44/EC (2002), the majority of trucks fall within the HGCZ when the WCA and VS criteria are taken into consideration.

Similarly, considering the WCA criterion according to BS 6841 (1987) standard the majority of trucks fall below the hazard limit of 15 m·s$^{-1.75}$. According to the VS criteria, more than half of the trucks remained in the area below the hazard limit. The smooth transition of truck units between criteria could be attributed to the fact that average VDV(8) values fluctuate around the 15 m·s$^{-1.75}$ limit.

The estimated vibration dose value (eVDV, m·s$^{-1.75}$) is the cumulative measure of vibration received by a person and is calculated using the frequency-weighted RMS value. As the eVDV is not suitable for shocks, repeated shocks, and intermittent vibration and will give correct results when the crest factor is less than six, it is recommended that the vibration dose value be determined directly from the entire acceleration-time log for vibration recordings where the crest factor is greater than about six (BS 6841, 1987). Of the 105 trucks sampled in the study, the crest factor was greater than six in 105, 101, and 104 trucks in the X-, Y-, and Z axes, respectively. The eVDV – VDV relationship for the trucks is presented in Figure 3. The coefficients of determination of linear regression analyses ranged from 80.57% to 86.90%, indicating a strong correlation although almost all crest factors were greater than six.

In terms of the WCA criterion, the time required to reach the daily exposure action value (EAVTT$_{(VDV)}$) of truck drivers ranged from 00:02:48 to 15:44:15 hours with a mean and standard deviation of 01:52:01 ± 02:22:29 hours. The time required to reach the daily exposure limit value (ELVTT$_{(VDV)}$) of truck drivers, however, ranged between 01:19:32 and 446:19:14 hours with mean and standard deviation 52:56:38 ± 67:20:54 hours. 103 out of 105 trucks exposed their drivers to enough vibration to reach the EAV before the end of the shift, while 11 trucks reach the ELV before the end of the shift.

Evaluation by vibration containing multiple shocks

According to ISO 2631-5 (2004), a S_{ed} value less than 0.5 MPa for lifetime exposure indicates a low probability of an adverse health impact caused by vibration, and a value greater than 0.8 MPa a higher probability. Likewise, a daily risk factor (R) below 0.8 for a certain number of working days per year indicates a low probability of an adverse health effect, while values greater than 1.2 pose a high probability of adverse health effects.

S_{ed} values, which were calculated using acceleration values measured at the seat pad (asx, asy, asz) ranged between 0.141 MPa and 1.749 MPa with mean and standard deviation of 0.466 ± 0.240 MPa. The R(IOP) factor ranged between 0.109 and 1.351 with mean and standard deviation 0.401 ± 0.201, and the R(TOP) factor between 0.177 to 2.187, with mean and standard deviation 0.583 ± 0.300.

When the WBV exposure is evaluated with S_{ed}, 70 drivers are exposed to a pressure of less than 0.5 MPa and the probability of adverse health effects due to vibration is low, 23 are exposed to between 0.5 MPa and 0.8 MPa with moderate probability of adverse health effects, and 12 are exposed to greater than 0.8 MPa with high probability of adverse health effects (Table III). Evaluation with the R(IOP) factor reveals that 100 drivers have an R factor less than 0.8 MPa and the probability of an adverse health effect due to vibration is low, four have an R factor

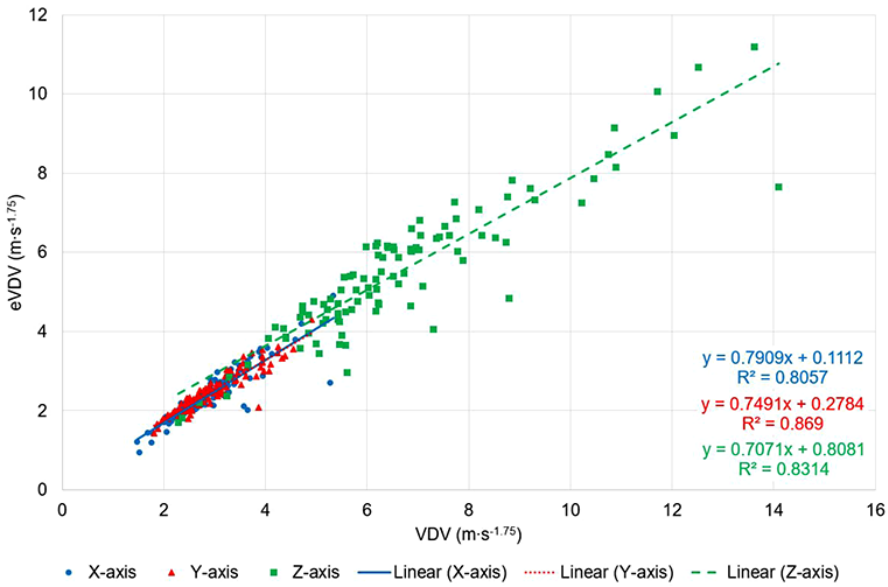

Figure 3—eVDV – VDV relationship for the RECORD phase

between 0.8 and 1.2 with moderate probability of an adverse health effect, and only one has an R factor greater than 1.2 with a high probability of an adverse health effect.

According to the ISO 2631-5 (2004) standard, the majority of drivers fall in the low health risk category. More specifically, according to the S_{ed} criterion, the majority of drivers are unlikely to suffer adverse health effects. According to the R factor, the probability of an adverse health effect for almost all drivers is low. With the ISO 2631-5 (2004) standard, the adverse health risk due to WBV exposure is prominently low compared to the BS 6841 (1987) and ISO 2631-1 (1997) standards and the sEU 2002/44/EC (2002) directive.

Evaluation of truck cycle phases

Evaluation of truck cycle phases by vibration acceleration

Parameters related to the WBV acceleration to which truck drivers were exposed are evaluated separately for the phases of the truck cycle. Assuming that the working shift is composed of a specific phase, averaged A(8), $A(8)_{xyz}$, $EAV_{TT(RMS)}$, and $ELV_{TT(RMS)}$ values, along with the number of units sampled, are presented in Table V for all phases. For the WCA, the highest WBV exposure occurs during the RETURN phase followed by the HAUL and SPOT phases, while the lowest WBV exposure was recorded in the WAIT phase. The same pattern is observed in the VS criterion.

The health risk severity of the WCA and VS criteria according to the EU 2002/44/EC (2002) directive and ISO 2631-1 (1997) standard in truck cycle phases is presented in Table VI. As far as the WCA criterion is concerned, all drivers were exposed to a vibration level below the HGCZ, indicating a low probability of adverse health risk in the WAIT phase. LOAD is another passive phase, the vibration transmitted to the driver increases only when the loader dumps the bucket into the truck body, while the remaining parts of the phase are quiet in terms of vibration. A similar pattern is observed for the DUMP phase. However, since the VS criterion produces a greater WBV acceleration, more trucks pass into the HGCZ of moderate health risk. In terms of the WCA criterion, the SPOT, HAUL, and RETURN phases are located within the HGCZ. However, according to the more conservative VS criterion, more than half of the trucks are above the HGCZ where an adverse health risk is likely.

Table VII shows the WCA with the highest vibration recorded for all phases. In the RETURN and HAUL phases, where the drivers are subjected to high levels of vibration, and to a lesser degree in the DUMP phase, this is the Z-axis. On the relatively undulating ground characterized by higher rolling resistances than main haul roads, where the truck is manoeuvering to approach the loader or the dump point, the WCA in the SPOT phase was Y, indicating excessive lateral movement. In other words, the WBV delivered to the driver along the vertical Z-axis by the slow-moving trucks in the SPOT phase was less than that along the horizontal Y-axis, where a shaking movement was predominant due to rough ground conditions. A similar pattern is observed in the LOAD phase. The vibrations caused by the material being dumped into the body of the stationary truck jolt it mostly along the sideways (lateral) Y-axis. On the other hand, during the WAIT phase, while the truck was stationary and the engine was running at low speed, the WCA was the Z-axis. In general terms, the WCA was the Z-axis in the phases where the truck was at a higher speed, and the Y-axis in the phases at lower speeds.

Averaged crest factors in all axes in the SPOT, HAUL, DUMP, and RETURN phases are less than the critical threshold of nine. These phases consist of stable sections in terms of vibration, where the driver is less exposed to shocks. In the WAIT phase, where 50 trucks were recorded, the crest factors of 24, 15, and 24 trucks were greater than nine on the X-, Y-, and worst-case Z-axis, respectively. In this phase, where the vehicle is stationary, and therefore the vibration exposure is very low, the crest factor could easily exceed nine, as even small jolts could cause high amplitudes in terms of WBV acceleration. In the LOAD phase, where some 99 trucks were recorded, the crest factor in the driver's seat was greater than nine for 62 trucks on the X-axis, 32 in the Y-axis (the WCA), and 70 trucks on the Z-axis. Shocks on the X and Z axes are greater than the Y-axis, where the highest vibration acceleration values are recorded. This indicates that the driver's seat is subjected to a steadily high acceleration along the Y-axis, and that the shocks on the X and Y axes caused by dumping the bucket are returned to the stationary state more quickly.

Analysis of the SEAT factor in the truck cycle phases revealed that vibration on the cabin floor is attenuated in some of trucks but exacerbated in others in all three axes. Driver seats exacerbated vibration in the X and Y axes in all cycle phases. More clearly, the driver's seat had almost no vibration dampening capability in the Y-axis. Although it exhibited a

Table V

Descriptives of vibration acceleration in cycle phases in accordance with ISO 2631-1

Phase	n	Mean	A(8) (m/s²) standard deviation	Min.	Max.	Mean	$A(8)_{xyz}$ (m/s²) standard deviation	Min.	Max.
Spot	89	0.628	0.184	0.267	1.159	0.904	0.241	0.306	1.521
Load	99	0.243	0.182	0.064	1.421	0.335	0.224	0.099	1.766
Haul	104	0.762	0.171	0.430	1.404	0.988	0.180	0.593	1.609
Dump	99	0.423	0.153	0.171	1.226	0.588	0.198	0.273	1.469
Return	102	0.862	0.251	0.398	1.552	1.147	0.271	0.546	1.914
Wait	50	0.116	0.079	0.008	0.344	0.154	0.103	0.012	0.392
		Mean	$EAV_{TT(RMS)}$ (hh:mm:ss) standard deviation	Min.	Max.	Mean	$ELV_{TT(RMS)}$ (hh:mm:ss) standard deviation	Min.	Max.
Spot		06:48:41	05:01:42	01:29:20	>24	>24	>24	07:52:34	>24
Load		>24	>24	00:59:26	>24	>24	>24	05:14:23	>24
Haul		03:57:31	01:43:43	01:00:53	10:49:36	20:56:28	09:08:40	05:22:02	>24
Dump		15:34:44	11:30:25	01:19:47	>24	>24	>24	07:02:04	>24
Return		03:31:43	02:20:49	00:49:49	12:39:05	18:39:59	12:24:58	04:23:33	>24
Wait		>24	>24	16:51:42	>24	>24	>24	>24	>24

Table VI

Decomposition of health risk severity of drivers in truck cycle phases

Phases	A(8)								Human response			
	ISO 2631–1 (1997)				EU 2002/44/EC (2002)				ISO 2631–5 (2004)			
	WCA[a]		VSa		WCA[b]		VS[b]		Sed[c]		R[d]	
Spot	17	Low	3	Low	23	Low	4	Low	80	Low	89	Low
	64	Moderate	37	Moderate	65	Moderate	71	Moderate	9	Moderate	–	Moderate
	8	High	49	High	1	High	14	High	–	High	–	High
Load	92	Low	80	Low	95	Low	88	Low	94	Low	97	Low
	5	Moderate	17	Moderate	3	Moderate	9	Moderate	4	Moderate	2	Moderate
	2	High	2	High	1	High	2	High	1	High	–	High
Haul	1	Low	–	Low	3	Low	0	Low	73	Low	99	Low
	88	Moderate	38	Moderate	98	Moderate	89	Moderate	25	Moderate	5	Moderate
	15	High	66	High	3	High	15	High	6	High	–	High
Dump	64	Low	24	Low	76	Low	36	Low	93	Low	97	Low
	34	Moderate	69	Moderate	22	Moderate	61	Moderate	4	Moderate	2	Moderate
	1	High	6	High	1	High	2	High	2	High	–	High
Return	2	Low	–	Low	7	Low	0	Low	71	Low	92	Low
	58	Moderate	19	Moderate	82	Moderate	50	Moderate	17	Moderate	9	Moderate
	42	High	83	High	13	High	52	High	14	High	1	High
Wait	50	Low	50	Low	50	Low	50	Low	49	Low	50	Low
	–	Moderate	–	Moderate	–	Moderate	0	Moderate	1	Moderate	–	Moderate
	–	High	–	High	0	High	0	High	–	High	–	High
	VDV(8)											
	ISO 2631–1 (1997)				BS 6841 (1987)				EU 2002/44/EC (2002)			
	WCA[e]		VS[e]		WCA[f]		VS[f]		WCA[g]		VS[g]	
Spot	10	Low	5	Low	79	Low	69	Low	18	Low	7	Low
	66	Moderate	60	Moderate	–	Moderate	–	Moderate	68	Moderate	79	Moderate
	13	High	24	High	10	High	20	High	3	High	3	High
Load	74	Low	68	Low	96	Low	96	Low	77	Low	72	Low
	23	Moderate	29	Moderate	–	Moderate	–	Moderate	21	Moderate	25	Moderate
	2	High	2	High	3	High	3	High	1	High	2	High
Haul	1	Low	0	Low	57	Low	54	Low	1	Low	0	Low
	78	Moderate	71	Moderate	–	Moderate	–	Moderate	96	Moderate	95	Moderate
	25	High	33	High	47	High	50	High	7	High	9	High
Dump	31	Low	24	Low	92	Low	92	Low	28	Low	29	Low
	65	Moderate	71	Moderate	–	Moderate	–	Moderate	58	Moderate	67	Moderate
	3	High	4	High	7	High	7	High	3	High	3	High
Return	1	Low	0	Low	42	Low	34	Low	2	Low	1	Low
	59	Moderate	45	Moderate	–	Moderate	–	Moderate	75	Moderate	70	Moderate
	42	High	57	High	60	High	68	High	25	High	31	High
Wait	46	Low	45	Low	49	Low	45	Low	48	Low	47	Low
	4	Moderate	5	Moderate	–	Moderate	–	Moderate	2	Moderate	3	Moderate
	–	High	–	High	1	High	5	High	–	High	–	High

Table VII

WCA distribution for A(8) in truck cycle phases

Phase	Number of trucks		
	X	Y	Z
Spot	19	48	22
Load	15	55	29
Haul	1	8	95
Dump	24	21	54
Return	–	6	96
Wait	9	15	26

similar behaviour on the X-axis, the vibration dampening ability here was somewhat better. Yet, the seat dampened the vibration in the Z-axis except for SPOT phase, where the dampening and aggravation numbers were close.

Evaluation of truck cycle phases by vibration dose

Assuming that the working shift is composed of a specific phase, vibration dose values measured in the driver's seat were converted to VDV(8) along the WCA and to $VDV(8)_{xyz}$ for the VS for all phases. The highest level of vibration dose that drivers are exposed to in the seat was in the RETURN phase, followed by the HAUL, SPOT, and DUMP phases. The lowest vibration dose occurs during LOAD and WAIT (Table VIII). Similarly, if the shift only consisted of a certain phase the drivers would, for example, reach the daily exposure action value ($EAV_{TT(VDV)}$) after 1 hour 39 minutes and 1 hour 49 minutes of work at the RETURN and HAUL phases, respectively. On the contrary, a driver would never reach the daily exposure action value in 24 hours of work if the shift consisted of WAIT or LOAD phases.

The vibration dose values are examined in Table VI for HGCZ limits in accordance with BS 6841 (1987), ISO 2631-1 (1997),

Table VIII

Descriptives of vibration dose in cycle phases in accordance with ISO 2631-1

Phase	VDV(8) ($m \cdot s^{-1.75}$)				VDV(8)xyz ($m \cdot s^{-1.75}$)			
	Mean	Standard deviation	Min.	Max.	Mean	Standard deviation	Min.	Max.
Spot	12.949	3.766	6.865	23.781	14.596	3.963	7.248	25.774
Load	6.721	4.194	1.616	29.466	7.438	4.344	1.883	30.123
Haul	15.241	4.423	7.816	35.785	16.287	4.199	10.141	36.566
Dump	10.450	3.728	3.988	26.547	11.359	3.678	5.087	27.065
Return	17.252	5.742	7.897	35.391	18.645	5.504	8.651	36.397
Wait	3.389	3.005	0.490	15.596	3.605	3.105	0.544	15.840
Phase	**$EAV_{TT(VDV)(sa)}$**				**$ELV_{TT(VDV)(sa)}$**			
	Mean	Standard deviation	Min.	Max.	Mean	Standard deviation	Min.	Max.
Spot	04:22:38	05:09:17	00:10:17	>24	>24	>24	04:51:52	>24
Load	>24	>24	00:04:22	>24	>24	>24	02:03:50	>24
Haul	01:49:14	01:53:19	00:02:00	14:42:05	>24	>24	00:56:56	>24
Dump	14:15:02	>24	00:06:38	>24	>24	>24	03:07:57	>24
Return	01:39:11	02:16:30	00:02:06	14:06:14	>24	>24	00:59:30	>24
Wait	>24	>24	00:55:38	>24	>24	>24	26:11:34	>24

and the EU 2002/44/EC directive (2002). In terms of the WCA criterion, most of the trucks fall within the HGCZ in the RETURN, HAUL, SPOT, and DUMP phases. LOAD and WAIT phases are below the HGCZ. When evaluated according to the VS criterion, there are no significant transitions between the HGCZ and the region above the HGCZ according to the ISO 2631-1 (1997) standard and EU 2002/44/EC directive (2002) as the phase averages fall in the HGCZ. Most drivers were exposed to vibration levels below the BS 6841 (1987) hazard limit of 15 $m \cdot s^{-1.75}$ in phases other than RETURN for both WCA and VS criteria. If the 8-hour shift consisted only of the RETURN phase, most drivers would have been exposed to a vibration dose over the hazard limit before the shift ended, according to both WCA and VS criteria.

Similar to vibration acceleration, the WCA along which drivers were exposed to the highest level of vibration dose is the Z-axis for RETURN, HAUL, DUMP, and WAIT phases (Table IX). Again, the WCA is the Y-axis for the SPOT and LOAD phases in which trucks are stationary or moving very slowly on the undulating ground.

The estimated VDV (eVDV, $m \cdot s^{-1.75}$) that truck drivers are exposed to was calculated in accordance with the BS 6841 (1987) standard and the eVDV–VDV relationship for all stages is given in Figure 4. Linear regression analyses for all three axes produced coefficients of determination (R^2) that varied between 71% and 96%. A strong correlation between the VDV and eVDV relationship is indicated for phases including those with a crest factor greater than six. It should be noted that for the vast majority of trucks, the crest factor is less than six on the X and Z axes only for the SPOT phase. Again, it is less than six on the SPOT, DUMP, and RETURN phases on the Y-axis. In all other cases, it is greater than six.

Table IX

WCA distribution for VDV(8) in truck cycle phases

Phase	Number of trucks		
	X	Y	Z
Spot	20	41	28
Load	21	51	27
Haul	–	8	96
Dump	27	17	55
Return	1	7	94
Wait	9	17	24

Evaluation of truck cycle phases by vibration containing multiple shocks

Averaged equivalent static compression dose value (S_{ed}), risk factor for the individual operator R(IOP), and typical operator R(TOP) values of the drivers are presented in Table X for all phases. In general terms, drivers are in the low adverse health risk zone in all phases. If the phases are examined individually, the highest S_{ed} and R factors belong to RETURN and HAUL. The negative health risk assessment of all truck drivers according

Table X

S_{ed} and R factor values of drivers based on cycle phases

Phase	S_{ed} (MPa)				R(IOP)				R(TOP)			
	Mean	SD	Min.	Max.	Mean	SD	Min.	Max.	Mean	SD	Min.	Max.
Spot	0.300	0.140	0.086	0.761	0.259	0.133	0.082	0.765	0.375	0.175	0.107	0.952
Load	0.201	0.155	0.041	1.024	0.176	0.142	0.028	0.853	0.251	0.194	0.052	1.280
Haul	0.426	0.216	0.142	1.294	0.369	0.192	0.118	1.000	0.533	0.270	0.178	1.619
Dump	0.297	0.159	0.098	1.122	0.260	0.144	0.084	0.981	0.372	0.198	0.123	1.403
Return	0.482	0.289	0.134	2.088	0.413	0.235	0.103	1.613	0.602	0.362	0.168	2.612
Wait	0.100	0.128	0.012	0.793	0.088	0.110	0.008	0.580	0.126	0.161	0.015	0.992

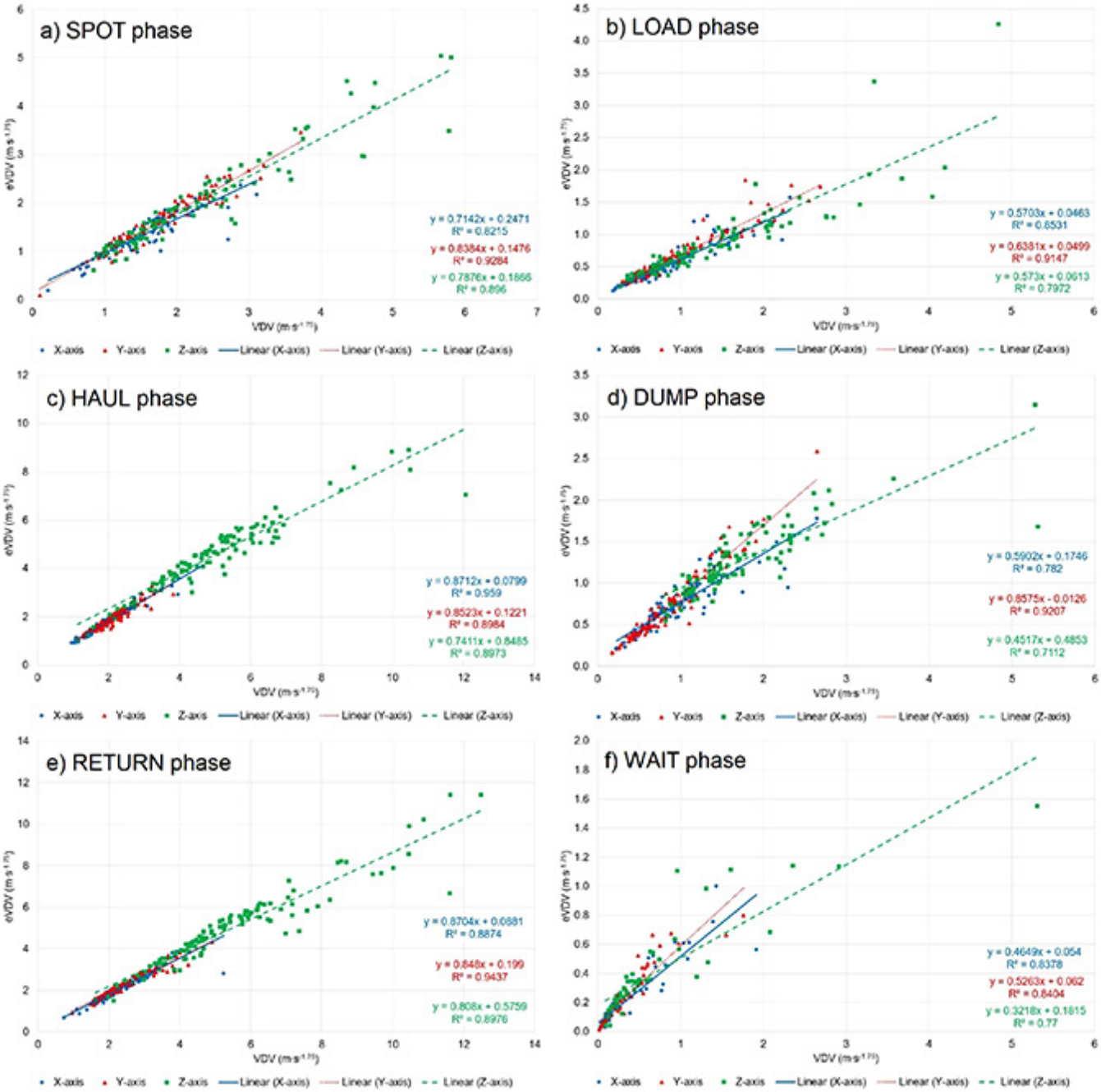

Figure 4—eVDV–VDV relationship for truck cycle phases

to the S_{ed} and R factor criteria can be seen in Table VI. In the RETURN phase, 14 drivers are exposed to compression greater than 0.8 MPa, and the potential for adverse health effects due to vibration is high. Twenty-five drivers in the HAUL phase and 17 in the RETURN phase are subjected to a pressure in the range of 0.5 MPa to 0.8 MPa, and the probability of adverse health effects due to vibration is moderate. In other phases, almost all drivers are exposed to a compression level of less than 0.5 MPa and the probability of adverse health effects due to vibration is low. When the drivers are evaluated according to the risk factor (R) criterion, the health risk for almost all drivers is low except for one driver in the RETURN phase.

Statistical analyses

Whole-body vibration data collected in this study was analysed using the SPSS statistics package (George and Mallery, 2010). The effect of differences in truck cycle phases, truck capacity, truck type (contractor truck/mining truck and regular/underground), unloading mechanism, driver age and experience, truck service life, and material hauled were evaluated by hyphotesis testing. Normality of data was tested prior to analysis by the Shapiro-Wilk test and skewness-kurtosis values. The confidence interval (CI) was chosen as 95%.

The one-way analysis of variance was used to determine whether there are any statistically significant differences between the means of A(8) and VDV(8) levels of truck cycle phases. The difference between the means of phases constituting the truck cycle was examined by the Tamhane test in multiple comparison tables. As far as the vibration acceleration is concerned, there was significant difference between the means of all cycle phases

at 95% CI. For the vibration dose, there was no significant difference between the HAUL, RETURN, and SPOT phases, but there was a statistically significant difference between all other phases at the 0.05 level.

In order to analyse the effect of hauling capacity on vibration level, trucks were grouped based on hauling capacity; 30 t (33 trucks), 77 t (48 trucks), 100 t (two 91 t and three 100 t), and 170 t (19 trucks). The difference between group means was examined by the Tamhane test in multiple comparison tables. Both for the vibration acceleration and vibration dose, there was a statistically significant difference between 30 t and 170 t trucks and between 77 t and 170 t trucks at the 0.05 level.

The independent sample test was used to determine whether WBV acceleration and dose levels changed significantly with truck type. There were 33 contractor trucks and 72 mining trucks. There was no significant difference between the group means. Thus, 30 t contractor trucks and ≥77 t mining trucks expose drivers to equivalent levels of vibration acceleration and dose.

The independent sample test was used to determine whether WBV acceleration and dose levels changed significantly with unloading method. There were three bottom-dump trucks and 102 rear-dump trucks (Figure 5). There was no significant difference between group means, revealing that bottom-dump and rear-dump trucks exposed drivers to equivalent levels of vibration acceleration and dose.

To investigate the effect of driver age on vibration exposure, drivers were grouped based on age; 20–30 years (11 drivers), 31–40 years (46 drivers), 41–50 years (41 drivers), and ≥51 years (7 drivers). The difference between group means was examined by the Tukey test in multiple comparison tables. Both for the vibration acceleration and vibration dose, there was not a statistically significant difference between age groups at the 0.05 level.

Any likely effect of driver experience on vibration exposure was examined. Drivers were grouped based on experience; 1–5 years (27 drivers), 6–10 years (37 drivers), 11–15 years (20 drivers), 16–20 years (13 drivers), 21–25 years (3 drivers) and ≥26 years (5 drivers). The difference between group means was examined by the Tukey test in multiple comparison tables. For the vibration dose, there was not a statistically significant difference between age groups at the 0.05 level. For the vibration acceleration, there was a statistically significant difference between 1–5 years experience group and 16–20 years experience group at the 0.05 level.

To study the impact of truck service life on vibration exposure, trucks were grouped: 1–10 years (31 trucks), 11–20 years (22 trucks), 21–30 years (6 trucks) and 31–40 years (46 trucks). The difference between group means was examined by the Tamhane test for vibration acceleration and the Tukey test for vibration dose in multiple comparison tables. Both for the vibration acceleration and vibration dose, there was a statistically significant difference between the 11–20 years of service life group and 31–40 years of service life group at the 0.05 level.

The independent sample test was used to determine whether WBV acceleration and dose levels changed significantly with working environment. There were 103 units allocated to surface operations and two in underground operations. A significant difference between group means showed that surface and underground working environments exposed drivers to different levels of vibration acceleration and dose.

The independent sample test was used to determine whether WBV acceleration and dose levels changed significantly with material hauled. While marble, gypsum, and limestone quarries were characterized by blocky material, the clay quarry, road construction site, mineral processing plant, and scrap iron plant provided representative examples of non-blocky material. Figure 6 depicts typical scenes of handling blocky material in the upper row and non-blocky material in the lower row. Forty-seven trucks hauled blocky material and 58 hauled non-blocky material. There was no significant difference between the group means, indicating that different materials hauled exposed drivers to equivalent levels of vibration acceleration and dose.

The independent sample test was used to determine whether WBV acceleration and dose levels differed significantly with cycle phase. Trucks were grouped based on cycle phases; laden (HAUL, 104 units) and unladen (RETURN, 102 units). A significant difference between group means illustrated that change in cycle phase exposed drivers to different levels of vibration acceleration and dose.

The independent sample test was used to determine whether WBV acceleration and dose levels changed significantly with vehicle speed. HAUL, with 104 units, was accepted as a speedy phase and SPOT, with 89 units, as slow. For the vibration dose, there was no significant difference between HAUL and SPOT, but for the vibration acceleration there was a statistically significant difference at the 0.05 level. Thus, different vehicle speeds exposed drivers to different levels of vibration acceleration.

In a similar effort, trucks were grouped based on cycle phases; speedy (RETURN, 102 units) and slow (SPOT, 89 units). The independent sample test was used to determine whether WBV acceleration and dose levels changed significantly with vehicle speed. For vibration acceleration and vibration dose, there was a significant difference between RETURN and SPOT at the 0.05 level.

Figure 5—Rear-dump and bottom-dump trucks

Figure 6—Blocky (top row) and non-blocky materials

Conclusions

Whole-body vibration exposure measurements were taken from 105 trucks of different make, model, and capacity used in open-pit and underground mines. The measurements were evaluated through the work cycle phases of the machines by using a vibration analysis package. Thus, the entire and phase-based vibration exposure of all operators could be determined separately. Operator exposures were evaluated on the worst-case axis and vector sum approaches by taking into account the vibration acceleration and the vibration dose parameters in accordance with the EU 2002/44/EC (2002) directive, ISO 2631-1 (1997), and BS 6841 (1987) standards and by the daily equivalent pressure dose and the risk factor parameters in accordance with the ISO 2631-5 (2004) standard.

When the operator exposures are evaluated with the worst-case axis and vector sum approaches, taking into account the vibration acceleration (A(8)) and the vibration dose (VDV(8)), vibration exposures are within the HGCZ, which indicates a moderate level of adverse health risk. When the evaluation is made in accordance with ISO 2631-1 (1997), trucks fall in the HGCZ, pointing to a moderate level of adverse health risk. Based on the BS 6841 (1987) standard, which recommends evaluation over the vector sum criterion with more conservative limits, vibration exposures are within the region below the prescribed hazard limit. According to the ISO 2631-5 (2004) standard, when the daily equivalent pressure dose (S_{ed}) parameter is examined with the WCA criterion, trucks are placed in the moderate health risk category. When the evaluation is made according to the vector sum criterion, trucks are located in the low health risk zone. When the R factor is examined according to WCA and VS criteria, vibration exposures are within the low health risk zone.

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NATIONAL & INTERNATIONAL ACTIVITIES

2021

18–22 April 2021 — **IMPC2020**
XXX International Mineral Processing Congress
Cape Town, South Africa
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9–11 June 2021 — **Diamonds – Source To Use — 2021 Conference**
'Innovation And Technology'
The Birchwood Hotel & OR Tambo Conference Centre, Johannesburg
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13–16 July 2021 — **Copper Cobalt Africa**
Incorporating
The 10th Southern African Base Metals Conference
Avani Victoria Falls Resort, Livingstone, Zambia
Contact: Camielah Jardine
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29–30 July 2021 — **5th Mineral Project Valuation Colloquium**
Glenhove Events Hub, Melrose Estate, Johannesburg
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16–17 August 2021 — **Worldgold Conference 2021**
Misty Hills Conference Centre, Muldersdrift, Johannesburg, South Africa
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29 August–2 September 2021 — **APCOM 2021 Minerals Industry Conference**
'The next digital transformation in mining'
Misty Hills Conference Centre, Muldersdrift, Johannesburg, South Africa
Contact: Camielah Jardine
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21–22 September 2021 — **5th Young Professionals Conference 2021**
'A Showcase of Emerging Research and Innovation in the Minerals Industry'
The Canvas, Riversands, Fourways, South Africa
Contact: Camielah Jardine
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27–30 September 2021 — **8th Sulphur and Sulphuric Acid Conference 2021**
The Vineyard Hotel, Newlands, Cape Town, South Africa
Contact: Camielah Jardine
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Fax: +27 11 838-5923/833-8156
E-mail: camielah@saimm.co.za
Website: http://www.saimm.co.za

4–6 October 2021 — **8th International PGM Conference 2021**
'PGMs – Enabling a cleaner world'
South Africa
Contact: Camielah Jardine
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Fax: +27 11 838-5923/833-8156
E-mail: camielah@saimm.co.za
Website: http://www.saimm.co.za

18–20 October 2021 — **South African Rare Earths International Conference 2021**
'Driving the future of high-tech industries'
Swakopmund Hotel And Entertainment Centre, Swakopmund, Namibia
Contact: Camielah Jardine
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Website: http://www.saimm.co.za

26–27 October 2021 — **SAMCODES Conference 2021**
'Good Practice and Lessons Learnt'
Industry Reporting Standards
Glenhove Events Hub, Melrose Estate, Johannesburg, South Africa
Contact: Camielah Jardine
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8–10 November 2021 — **Global Tailings Standards and Opportunities Conference 2021**
Sun City Resort, Rustenburg, South Africa
Contact: Gugu Charlie
Tel: +27 11 834-1273/7
Fax: +27 11 838-5923/833-8156
E-mail: gugu@saimm.co.za
Website: http://www.saimm.co.za

9–11 December 2021— **Massmin 2020 Eight International Conference & Exhibition on Mass Mining Virtual Conference 2020**
Santiago, Chile, Contact: J.O. Gutiérrez
Tel: (56-2) 2978 4476
Website: www.massmin2020.com

Owing to the current COVID-19 pandemic our 2020 conferences have been postponed until further notice. We will confirm new dates in due course.

Company affiliates

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

3M South Africa (Pty) Limited
AECOM SA (Pty) Ltd
AEL Mining Services Limited
African Pegmatite (Pty) Ltd
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Alexander Proudfoot Africa (Pty) Ltd
AMEC Foster Wheeler
AMIRA International Africa (Pty) Ltd
ANDRITZ Delkor(Pty) Ltd
Anglo Operations Proprietary Limited
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Atlas Copco Holdings South Africa (Pty) Limited
Aurecon South Africa (Pty) Ltd
Aveng Engineering
Aveng Mining Shafts and Underground
Axiom Chemlab Supplies (Pty) Ltd
Axis House (Pty) Ltd
Bafokeng Rasimone Platinum Mine
Barloworld Equipment -Mining
BASF Holdings SA (Pty) Ltd
BCL Limited
Becker Mining (Pty) Ltd
BedRock Mining Support (Pty) Ltd
BHP Billiton Energy Coal SA Ltd
Blue Cube Systems (Pty) Ltd
Bluhm Burton Engineering (Pty) Ltd
Bond Equipment (Pty) Ltd
Bouygues Travaux Publics
Castle Lead Works
CDM Group
CGG Services SA
Coalmin Process Technologies CC
Concor Opencast Mining
Concor Technicrete
Council for Geoscience Library
CRONIMET Mining Processing SA (Pty) Ltd
CSIR Natural Resources and the Environment (NRE)
Data Mine SA
Digby Wells and Associates
DRA Mineral Projects (Pty) Ltd
DTP Mining - Bouygues Construction
Duraset
Elbroc Mining Products (Pty) Ltd
eThekwini Municipality
Ex Mente Technologies (Pty) Ltd
Ex Mente Technologies (Pty) Ltd
Expectra 2004 (Pty) Ltd
Exxaro Coal (Pty) Ltd
Exxaro Resources Limited
Filtaquip (Pty) Ltd
FLSmidth Minerals (Pty) Ltd
Fluor Daniel SA (Pty) Ltd
Franki Africa (Pty) Ltd-JHB
Fraser Alexander (Pty) Ltd
G H H Mining Machines (Pty) Ltd
Geobrugg Southern Africa (Pty) Ltd
Glencore
Hall Core Drilling (Pty) Ltd
Hatch (Pty) Ltd
Herrenknecht AG
HPE Hydro Power Equipment (Pty) Ltd
Immersive Technologies
IMS Engineering (Pty) Ltd
Ingwenya Mineral Processing (Pty) Ltd
Ivanhoe Mines SA
Joy Global Inc.(Africa)
Kudumane Manganese Resources
Leco Africa (Pty) Limited
Leica Geosystems (Pty) Ltd
Longyear South Africa (Pty) Ltd
Lull Storm Trading (Pty) Ltd
Maccaferri SA (Pty) Ltd
Magnetech (Pty) Ltd
Magotteaux (Pty) Ltd
Malvern Panalytical (Pty) Ltd
Maptek (Pty) Ltd
Maxam Dantex (Pty) Ltd
MBE Minerals SA Pty Ltd
MCC Contracts (Pty) Ltd
MD Mineral Technologies SA (Pty) Ltd
MDM Technical Africa (Pty) Ltd
Metalock Engineering RSA (Pty)Ltd
Metorex Limited
Metso Minerals (South Africa) Pty Ltd
Micromine Africa (Pty) Ltd
MineARC South Africa (Pty) Ltd
Minerals Council of South Africa
Minerals Operations Executive (Pty) Ltd
MineRP Holding (Pty) Ltd
Mining Projections Concepts
Mintek
MIP Process Technologies (Pty) Limited
MLB Investment CC
Modular Mining Systems Africa (Pty) Ltd
MSA Group (Pty) Ltd
Multotec (Pty) Ltd
Murray and Roberts Cementation
Nalco Africa (Pty) Ltd
Namakwa Sands(Pty) Ltd
Ncamiso Trading (Pty) Ltd
New Concept Mining (Pty) Limited
Northam Platinum Ltd - Zondereinde
Opermin Operational Excellence
Optron (Pty) Ltd
Paterson & Cooke Consulting Engineers (Pty) Ltd
Perkinelmer
Polysius a Division of Thyssenkrupp Industrial Sol
Precious Metals Refiners
Ramika Projects (Pty) Ltd
Rams Mining Technologies
Rand Refinery Limited
Redpath Mining (South Africa) (Pty) Ltd
Rocbolt Technologies
Rosond (Pty) Ltd
Royal Bafokeng Platinum
Roytec Global (Pty) Ltd
RungePincockMinarco Limited
Rustenburg Platinum Mines Limited
Salene Mining (Pty) Ltd
Sandvik Mining and Construction Delmas (Pty) Ltd
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Sebilo Resources (Pty) Ltd
Senet (Pty) Ltd
Senmin International (Pty) Ltd
Smec South Africa
Sound Mining Solution (Pty) Ltd
SRK Consulting SA (Pty) Ltd
Time Mining and Processing (Pty) Ltd
Timrite Pty Ltd
Tomra (Pty) Ltd
Ukwazi Mining Solutions (Pty) Ltd
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Webber Wentzel
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Diamonds – Source to Use – 2021 Hybrid Conference

Innovation and Technology

9–10 June 2021 Conference

The Birchwood Hotel & OR Tambo Conference Centre, Johannesburg

BACKGROUND

The *Diamonds – Source to Use* conference series targets the full spectrum of the diamond pipeline, from exploration through to sales and marketing. The 2021 conference, the eighth in the series, will focus on advances in the mining and metallurgical aspects as well as many of the downstream and related industries.

KEYNOTE SPEAKER
L. Hockaday, *Mintek – Renewable Energy Technology*

OBJECTIVE

The objective of the conference is to provide a forum for the dissemination of information relating to the latest tools and techniques applicable to all stages of the diamond industry, from exploration through mine design, processing, to cutting, marketing, and sales.

EVENT FORMAT

At this point in time, the event is planned as a hybrid conference with international participation opportunities for delegates and speakers. It is also planned to hold technical visits to nearby facilities if possible. However, as we are still in Lock down as a result of COVID-19, this will be constantly reviewed, and if it appears that the effects of the pandemic are still such as to pose a threat to the health and safety of delegates, this will be changed to a digital event.

Please continue to submit your abstracts and check www.saimm.co.za regularly for updates.

WHO SHOULD ATTEND

- Geologists
- Mineral (diamond) resource managers
- Mining engineers
- Process engineers
- Consultants
- Suppliers
- Sales/marketing
- Diamantaires
- Mine managers
- Mining companies
- Students

TOPICS

- Geology and exploration
- Mine expansion projects
- Mining, metallurgy, and processing technology
- Rough diamond sales and marketing
- Cutting, polishing, and retail
- Synthetic diamonds
- Financial services and industry analysis
- Industry governance, beneficiation, and legislation
- Mine-specific case-studies

2 ECSA CPD points, 20 GSSA CPD points and 2.5 SACNASP CPD points will be allocated to all attending delegates

Conference Announcement

For further information contact:
Camielah Jardine • Head of Conferencing · SAIMM
P O Box 61127 • Marshalltown 2107
E-mail: camielah@saimm.co.za • Website: http://www.saimm.co.za

PGM THE 8TH INTERNATIONAL CONFERENCE

4-5 OCTOBER 2021 - CONFERENCE
6 OCTOBER 2021 - TECHNICAL VISIT

VENUE - SUN CITY, RUSTENBURG, SOUTH AFRICA

PGMs - Enabling a cleaner world

The Platinum conference series has covered a range of themes since inception in 2004, and traditionally addresses the opportunities and challenges facing the platinum Industry.

This prestigious event attracts key role players and industry leaders through:

- High quality technical papers and presentations
- Facilitating industry networking
- Having large, knowledgeable audiences
- Global participation, and
- Comprehensive support from industry role players.

The 8th International PGM Conference will, under the guidance of the organising committee, structure a programme which covers critical aspects of this continually evolving and exciting industry.

The success and relevance of this event to the industry really depends on your participation and support.

You can participate in this event as an organising committee member, author/presenter, delegate or sponsor.

FORMAT OF THE EVENT

At this point in time, the event is planned as a full contact conference with international participation through web links. It is also planned to hold technical visits to nearby facilities.

However, as we are still in Stage 2 Lock down as a result of COVID-19, this will be constantly reviewed, and if it appears that the effects of the pandemic are still such as to pose a threat to the health and safety of delegates, this will be changed to a digital event.

WHO SHOULD ATTEND

- Academics
- Business development managers
- Concentrator managers
- Consultants
- Engineers
- Exploration managers
- Explosives engineers
- Fund managers
- Geologists
- Hydrogeologists
- Innovation managers
- Investment managers
- Market researchers and surveyors
- Marketing managers
- Mechanical engineers
- Metallurgical managers
- Metallurgical consultants
- Metallurgists
- Mine managers
- Mining engineers
- New business development managers
- Planning managers
- Process engineers
- Product developers
- Production managers
- Project managers
- Pyrometallurgists
- Researchers
- Rock engineers
- Scientists
- Strategy analysts
- Ventilation managers

ABOUT THE VENUE

The Sun City resort lies in a tranquil basin of an extinct volcano in the Pilanesberg adjacent to South Africa's rich platinum belt. It boasts four hotels, an award-winning golf course and many other attractions.

Nestled in the rolling hills of the Pilanesberg, one of South Africa's most scenic locations, Sun City is a world unto itself and has earned its reputation as Africa's Kingdom of Pleasure.

Finally re-discovered and now part of Sun City, the Lost City and the Valley of Waves, fabled to be the Ruins of a glorious ancient civilisation,celebrate and bring to life the legends of this mystical city.

FOR FURTHER INFORMATION, CONTACT:

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