SAIMM

JOURNAL OF THE SOUTHERN AFRICAN INSTITUTE OF MINING AND METALLURGY

CAPACI OCCASIO

FOUNDED 1894

VOLUME 117 NO. 3 MARCH 2017

ELBROC
MINING PRODUCTS (PTY) LTD

ARCH SUPPORT

BENEFITS

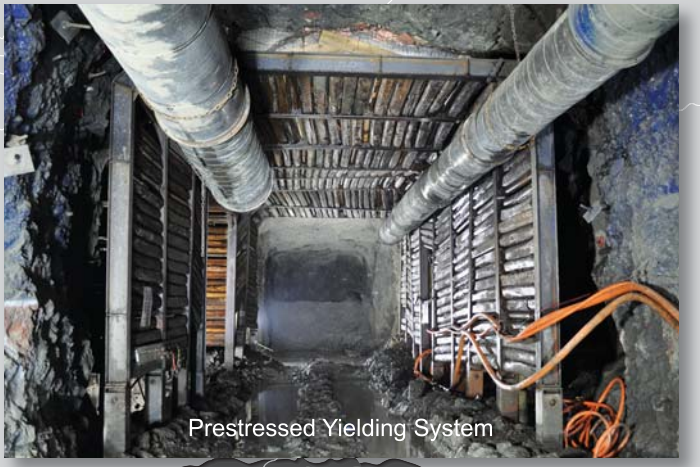
- Simple and fast installation
- High load carrying capacity
- Long operational life
- Active support by the arch set using Hydraulic prop technology
- Cost effective
- Dimensional and design flexibility
- Suitable for various excavation heights and widths
- Reduced cribbing required

APPLICATIONS

- Tunnel junctions
- Over and under ore passes
- Haulage
- Shaft portals
- Active support through bad ground conditions, dykes, faults, friable ground


TH System


Omni Sets


Prestressed Yielding System

SABS
ISO 9001

FOR OUR FULL PRODUCT RANGE
PLEASE VISIT OUR WEBPAGE

a member of the rbh royal bafokeng holdings

Tel: 011 974 8013 • Fax: 011 974 2019 • sales@elbroc.co.za

www.elbroc.co.za

mogs

PROXIMITY DETECTION AND COLLISION AVOIDANCE SYSTEMS IN MINING COLLOQUIUM 2017

Striving for zero harm from mining mobile machinery

BACKGROUND

Considerable attention is being focused through the Department of Mineral Resources, the Mine Health and Safety Council, the mining companies and the original equipment manufacturers on the development and implementation of Proximity Detection Systems (PDSs) and Collision Avoidance Systems for underground and surface mobile machinery and underground track bound machinery. These initiatives are all in support of the quest by all stakeholders for reaching "zero harm".

Systems have been developed and trialled for vehicle to person detection and warning, vehicle to beacon, and vehicle to vehicle detection.

Technical challenges have been encountered in terms of the various system reliability and pedestrian/operator acceptance, as well as the application on diesel powered and electric powered machinery, respectively.

Whilst it is the intention to regulate this area in the near future, it is important that the various issues that could be problematic to implementation be aired and solutions found.

SAIMM
THE SOUTHERN AFRICAN INSTITUTE OF MINING AND METALLURGY

20 April 2017
Emperors Palace, Hotel Casino, Convention Resort Johannesburg

Conference Co-ordinator:
Camielah Jardine • SAIMM
Tel: +27 11 834-1273/7 • Fax: +27 11 833-8156
E-mail:camielah@saimm.co.za
Website: http://www.saimm.co.za

Secretaries

The Southern African Institute of Mining and Metallurgy
Fifth Floor, Chamber of Mines Building
5 Hollard Street, Johannesburg 2001 • P.O. Box 61127, Marshalltown 2107
Telephone (011) 834-1273/7 • Fax (011) 838-5923 or (011) 833-8156
E-mail: journal@saimm.co.za

Typeset and Published by
The Southern African Institute of Mining and Metallurgy
P.O. Box 61127
Marshalltown 2107
Telephone (011) 834-1273/7
Fax (011) 838-5923
E-mail: journal@saimm.co.za

Printed by
Camera Press, Johannesburg

Advertising Representative
Barbara Spence
Avenue Advertising
Telephone (011) 463-7940
E-mail: barbara@avenue.co.za

The Secretariat
The Southern African Institute of Mining and Metallurgy

ISSN 2225-6253 (print)
ISSN 2411-9717 (online)

SAIMM
JOURNAL OF THE SOUTHERN AFRICAN INSTITUTE OF MINING AND METALLURGY
CAPACI OCCASIO FOUNDED 1894

VOLUME 117 NO. 3 MARCH 2017

Contents

Journal Comment

This issue contains just one paper from the University of South Africa (UNISA), the only South African university offering mining engineering and mine surveying by open distance learning. Lugoma explores the possibility of supplementing online course content with on-campus practical sessions. The encouraging findings have prompted him to roll out this approach to education so as to enable students to familiarize themselves with mine surveying equipment before they begin their professional careers.

It must be noted that of the 10 papers originally submitted by UNISA, seven were rejected by the refereeing process; one paper was withdrawn, and one is still being revised prior to possible publication at a later date. The referees were almost unanimous in their reasons for rejection, which were: lack of structure in the paper; the content of the paper did not correspond with the title description; and the results of the work did not produce anything that is not already well known.

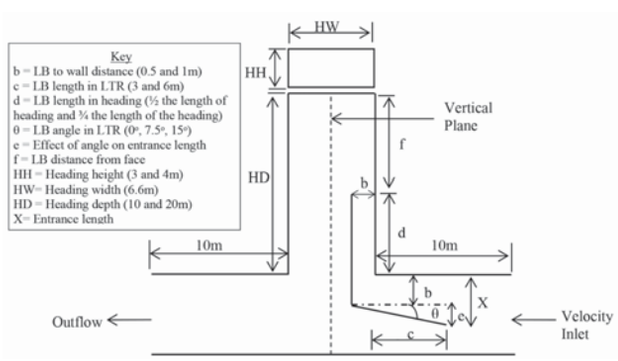
I hope that this will not deter the researchers at UNISA from submitting papers on their work in the future, and that the constructive comments by the referees will enable researchers and authors to produce papers of an acceptable standard for publication in the Journal of the SAIMM.

The diligence of the referees of the UNISA papers is commendable in that they provided meaningful feedback to the authors and at the same time contributed to maintaining the standard of the publications in our *Journal*.

D. Tudor
Editorial Consultant

Erratum

The paper in the February issue of the SAIMM *Journal,* vol. 117, no. 2. pp. 97–108 by Feroze, T. and Genc, B. entitled '*A CFD model to evaluate variables of the line brattice ventilation system in an empty heading*' contained an incorrect version of Figure 1 (p. 98). The correct figure is as follows. We apologise for any misunderstanding that may have ensued.



Advancing international collaboration through the Global Mineral Professionals Alliance (GMPA)

When Finance Minister Pravin Gordhan was delivering his mid-term budget speech in 2016, he made reference to the following Pedi quote which is relevant to one of SAIMM's strategic initiatives: '*Ditau tsa hloka seboka di shitwa ke nare e hlotsa*' (translated into English as 'Lions that fail to work as a team will struggle to bring down even a limping buffalo'). This quote cannot be any truer when one reflects on the need for collaboration for a common purpose. In the October 2015 edition of the SAIMM *Journal* our Immediate Past President, Rodney Jones, wrote about a 2011 inaugural meeting in London which ultimately resulted in the formation of the Global Mineral Professionals Alliance (GMPA). I am happy to share with you a positive development – in February 2017 the SAIMM hosted the Annual GMPA Meeting in Cape Town, where we formally signed a Memorandum of Understanding (MoU).

Some of you may wonder what the GMPA is, and why it is important that the SAIMM participates in such a collaboration.

The GMPA is currently composed of six sister institutes: the Australasian Institute of Mining and Metallurgy (AusIMM), the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), the Institute of Materials, Mining and Metallurgy (IOM3), the SAIMM, the Society for Mining, Metallurgy and Exploration Inc. (SME), and the Instituto de Ingenieros de Minas del Peru (IIMP). Each of these six professional organizations is active in its specific geographic area in order to advance and promote the professional development of its members. This common goal brings us together under the GMPA umbrella to share challenges and solutions. We are able to move towards this goal collectively through sharing institutional information and benchmarking, and sharing technical information via OneMine.org and the Global Mining Standards and Guidelines Group (GMSG). I urge you to visit the OneMine.org website to see the wealth of technical papers hosted on that platform. Through our collaborative partnership with the GMPA, we are able to enhance a mutually beneficial relationship among member associations. Standing alone, we would run the risk of starving like the independent-minded lions that fail to bring down the limping buffalo.

The SAIMM is fully committed to the GMPA initiative as its members stand to benefit from the reciprocal arrangements that exist among the member institutions. For example, any SAIMM member in good professional standing who visits any of the GMPA geographical locations for no more than 12 months qualifies for 'Visiting Member' status. This allows the member, among other things, to register for conferences, seminars, and workshops conducted by the host society at the host society's member rate, and to have access to the host society's facilities. SAIMM members can login via the SAIMM website to the OneMine.org website and view, download, and print documents at no charge. Is this not a great leap forward in benefits for our members? Is this not a global approach to dealing with global challenges? As you read this edition of the *Journal,* which is dedicated to papers from UNISA, the New Technology Conference, and other selected papers of topical interest, I would therefore like to leave you with this thought on collaboration. If a sorrow shared is a sorrow halved, then a joy shared is a joy doubled. Will challenges shared not be challenges halved, and success shared be success doubled?

C. Musingwini
President, SAIMM

PAPERS IN THIS EDITION

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

Unisa Paper

New Technology Papers

These papers will be available on the SAIMM website
http://www.saimm.co.za

PAPERS IN THIS EDITION

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

General Papers

These papers will be available on the SAIMM website
http://www.saimm.co.za

http://dx.doi.org/10.17159/2411-9717/2017/v117n3a1

On-campus mine surveying practicals: their contribution in training mining engineering students in an open distance learning context

by F.M. Lugoma*

Synopsis

The University of South Africa (UNISA) is the only South African university offering mine surveying by open distance learning (ODL). Initially intended to service mining engineering practitioners in search of formal training, the university has seen a growing enrolment of private students. Because this cohort of students has had little contact with the mining environment, there is a need to supplement the theoretical training by ODL with practical sessions on campus.

In order to investigate this concern, historical data pertaining to the performance of students over the last five years was collected. Six mine surveying modules forming part of the national diplomas in Mining Engineering and in Mine Surveying offered at UNISA were considered: Mine Survey II and III, Mine Valuation II and III (Mine Surveyors subjects), and Mine Survey and Valuation II and III (Mine Engineering subjects).

A roster was then designed for mine surveying practicals for implementation over one year, with students' attendance being optional. The students who accepted the invitation were monitored for the duration of the research.

From the initial findings, a noticeable improvement in pass rate was observed for students who attended the practical sessions compared with those who did not. This can be regarded as an incentive to make the mine surveying practicals compulsory in future while integrating them with the current learning and assessment. One consequence of this would be the training of well-rounded technicians with sought-after skills for the South African mining industry.

Keywords

mine surveying, open distance learning, on-campus practicals, mining engineering education.

Introduction

The University of South Africa (UNISA) is the only South African university offering mine surveying by open distance learning (ODL). The set-up was intended to primarily offer flexible training to mine practitioners in search of formal qualifications without affecting their professional lives. However, over time, professional students employed in or by the mining industry have been outnumbered by private students. These students are freshly graduated from high school with little to no prior industrial experience.

It is clear that the current curriculum needs to be rethought considering the changing profile of students enrolling for mining-related qualifications. That is why the Department of Mining Engineering at UNISA has introduced on-campus practical sessions. The programme is initially aimed at supplementing students' formal training with relevant mine surveying exercises.

In order to assess the impact of the programme, historical data pertaining to the performance of students over the last five years was extracted from the University database. Six mine surveying modules forming part of the National Diploma in Mining Engineering and National Diploma in Mine Surveying offered at UNISA were considered. Mine surveying subjects include Mine Survey II and III and Mine Valuation II and III; while mining engineering subjects include Mine Survey and Valuation II and III. A roster was designed for mine surveying practicals for implementation over one year, with students' attendance being optional. The students who accepted the invitation were monitored for the duration of the research. Statistical analysis centred on the hypothesis was performed to determine the change in status before and after the inception of the programme. Finally, recommendations for future work are proposed and conceivable changes to tuition policies evoked.

The National Diploma in Mining Engineering and Mine Surveying

The National Diploma in Mining Engineering and in Mine Surveying are two of the formal engineering qualifications offered by UNISA's Department of Electrical and Mining Engineering, which is based on the Science Campus in Florida, Roodepoort. The Department of Electrical and Mining Engineering forms part of the School of Engineering, which itself forms part of the College of Science, Engineering and Technology (CSET), as shown in Figure 1.

* *Department of Electrical and Mining Engineering, Univerity of South Africa, Science Campus, South Africa.*

 Paper received Jul. 2016; revised paper received Feb. 2017.

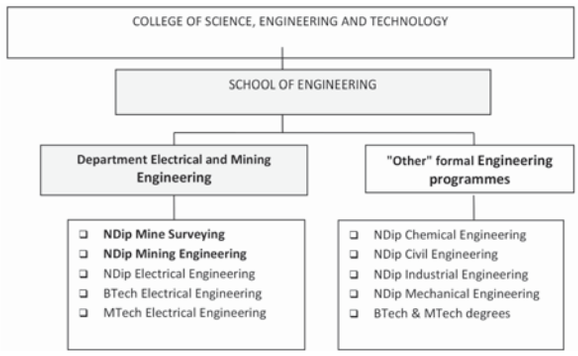


Figure 1—Engineering qualifications offered by CSET at UNISA (source: Self-Evaluation Report-National Diploma in Mine Surveying, internal unpublished document)

The National Diploma in Mining Engineering

The purpose of this qualification is to produce competent mining technicians whose responsibilities would include the selection and managing of the optimal mining process for the mineral deposit in question. A qualified person with sufficient experience will be able to register with the Engineering Council of South Africa (ECSA) as a Professional Technician in this field of engineering. This programme is a prerequisite for the BTech Mining Engineering offered by the University of Johannesburg.

The NDip Mining Engineering programme helps candidates to obtain a Mine Manager's Certificate of Competency (MMCoC) since it covers the syllabus of Part A as well as sections of Part B. Students who completed the diploma may apply to the Department of Mineral Resources (DMR) for exemption from Part A of the MMCoC.

The curriculum of the programme is presented in Table I.

The National Diploma in Mine Surveying

The main purpose of the National Diploma in Mine Surveying is to produce competent mine surveyors whose responsibilities would include taking measurements on and under the surface of the earth in a horizontal and vertical plane, showing the results in the form of a map, plan, or 3D computer model after certain calculations have been done.

Many previously disadvantaged students are enrolled for mining programmes at UNISA. Statistics collected from the Institutional Information Portal (Directorate: Information and Analysis, 'HEDA' (Unisa, 2016a)) reveal that they represent about 95% of all mining-related students over the last five years. UNISA thereby assists mining companies to meet the requirements of the Mining Charter. Health and safety in the minerals industry continues to be a priority. UNISA wants to better equip future graduates with regard to mine health and safety in particular.

The curriculum of the National Diploma in Mine Surveying is outlined in Table II. It is important to point out that UNISA and the University of Johannesburg are the only two universities in South Africa offering a diploma in mine/minerals surveying. UNISA is the only university that offers such a programme by distance learning.

ODL context of learning

Methodologies used in distance learning differ from those used in residential universities. In ODL, students are issued with a study package upon registration, consisting of study guides, tutorial letters, and course notes. Textbooks are prescribed where course notes are not provided by UNISA. The assignments are included in tutorial letters. Students do not attend classes, but limited discussion classes and lecture consultation sessions are offered in some cases. Discussion classes are not compulsory and are not offered to remote students in South Africa, Africa, and the rest of the world. To compensate for this, UNISA uses e-learning technologies to a large extent. The UNISA learning management system (LMS), called myUnisa, is the UNISA online student portal and the university's most important study tool (UNISA, 2016b). Students can download all course material, as well as additional resources. myUnisa is also called the 'virtual campus for distant learners'. It constitutes a platform where students can interact with fellow students, lecturers, and tutors.

Through self-study the students work through the study guides in their own time, complete the assignments (two or three), and submit them for assessment. Tutorial letters are used by lecturers to provide feedback on assignments and to discuss general problems regarding the subject content. Some lecturers provide such feedback individually when they mark student assignments. Lecturers, and in some cases also tutors, are appointed for each subject to offer academic support to the students via telephone, e-mail, MyUnisa (internet), and personal sessions where required.

A year mark is calculated from assignment marks. The year mark has a weight of 20% when the final mark is calculated; the examination (summative assessment) has a weight of 80%. There are three examination periods, namely January/February, May/June, and October/November. All semester modules have examinations in May/June and October/November respectively for students registered in semester 1 or semester 2. Examinations for full-year modules are held in October/November, and in January/February of the following year for students who qualified to write a supplementary examination (*i.e.* students who have failed with a minimum of 40%). The supplementary examination for semester modules is granted on the same condition but is written in the following session of examination.

At a residential institution a student will typically register for all level 1 subjects in a semester. At UNISA a student may register for subjects from levels 1, 2, and 3 simultaneously provided that the prerequisites are fulfilled. As indicated in Tables I and II, nearly all subjects/modules are offered over a period of a semester, while a few are offered over a full year. The remaining full-year modules are in the process of being semesterized. Certain subjects, for example work integrated-learning (WIL) subjects, remain full-year subjects.

The experiential learning components of the programmes, also called work-integrated learning (WIL) are divided into separate modules covering the required levels of the programmes as shown in Tables I and II. These subjects are administered as individual modules. UNISA staff monitor the experiential learning of students continuously through feedback from students and mentors, completion of assignments and logbooks, and visits to the workplace where possible.

Table I

Curriculum of the National Diploma in Mining Engineering

National Diploma: Mining Engineering			
Qualification code: NDMIN	**NQF Exit level: 6**	**Total credits: 360**	
Admission requirements: A National Senior Certificate (Diploma endorsement) with at least 50% in the language of teaching and learning and with at least 50% in Mathematics and Physical Science. Students who do not meet the additional requirements may follow Unisa's alternative pathway.			
Purpose statement: To produce competent mining engineers whose responsibilities would include the selection and managing of the optimal mining process applicable to the relevant mineral deposit to be mined. A qualified person with sufficient experience will be able to register with the Engineering Council of South Africa (ECSA) as a Professional Technician in this field of Engineering.			
Module Code	**Module name**	**Prerequisite/Co-requisite**	**Semester/Year**
First level: Group A. Compulsory			
ENN103F	English for Academic Purposes		Semester
EUP1501	Ethical Information and Communication Technologies for Development Solutions		Semester
MAT1581	Mathematics I (Engineering)		Semester
MEP171X	Mineral Exploitation I		Semester
SMI181Q	Science: Mining I		Semester
STA1510	Basic Statistics		Semester
Second level: Group A. Compulsory			
CAD161S	Computer Aided Draughting I	Co-requisite: MED161Q	Semester
ENV301E	Environmental Management		Semester
EWS121Q	Engineering Work Study I		Semester
FAC1501	Introductory Financial Accounting		Semester
MAT2691	Mathematics II (Engineering)	Prerequisite: MAT1581	Semester
MED161Q	Mechanical Engineering Drawing I		
MPR101E	Mining Engineering Practice I		Year (WIL)
MPR201E	Mining Engineering Practice II		Year (WIL)
Third level: Group A. Compulsory			
EMA2601	Engineering Management II (Module A)		Semester
EMA2602	Engineering Management II (Module B)		Semester
MBE2601	Mineral Beneficiation II	Prerequisite: MEP171X	Semester
MEN2601	Mine Engineering II	Prerequisite: SMI181Q	Semester
MGY2601	Mining Geology II	Prerequisite: MEP171X	Semester
MIN2601	Mining II	Prerequisite: MEP171X	Semester
SAV2601	Mine Survey and Valuation II	Prerequisite: MEP171X	Semester
Fourth level Group A. Compulsory			
EMA3601	Engineering Management III	Prerequisite: EMA2601 or EMA2M1E & EMA2602 or EMA2M2E	Semester
GMI3601	Geology: Mining III	Prerequisite: MGY2601 or MGY291S	Semester
MIE3601	Mine Engineering III	Prerequisite: MEN2601 or MEN251W	Semester
MIN3601	Mining III	Prerequisite: MIN2601 or MIN201E	Semester
MTS351X	Mining Technical Services III	Prerequisite: MEP171X	Semester
SAV3601	Mine Survey and Valuation III	Prerequisite: SAV2601 or SAV231Y	Semester

The need for mine surveying practical sessions on campus

Mine surveying practicals on campus were developed to support the theoretical study of mine surveying subjects, since these subjects have been found to be challenging to the majority of students. This is particularly true of private students, since they have never had the opportunity to visit or undergo training in a mine survey department.

Table III shows the pass rates and the number of students enrolled in 2012, indicating the difference between private and professional students. The challenge faced by private students transpires through their poor performance compared with professional students. For 2012 used as baseline, the average pass rate of private students is only 10%, against 26% for professional students.

Data collection

In this section, the structure of the programme for on-campus practicals is presented. Student performance is also reviewed in order to form a baseline for comparison before and after implementation of the on-campus programme.

Structure of on-campus practicals

Mine surveying practicals were implemented from 22 February 2016 on the Science Campus of UNISA. The Mining Engineering laboratory and the vicinity of the campus were

Table II

Curriculum of the National Diploma in Mine Surveying

National Diploma: Mine Surveying			
Qualification code: NDMSR	**NQF Exit level: 6**	**Total credits: 360**	
Admission requirements: A National Senior Certificate (Diploma endorsement) with at least 50% in the language of teaching and learning and with at least 50% in Mathematics and Physical Science. Students who do not meet the additional requirements may follow Unisa's alternative pathway.			
Purpose statement: To produce competent mine surveyors whose responsibilities (simplified) would include taking measurements upon and under the surface of the earth either in a horizontal or vertical plane and showing the results in the form of a map or plan applicable to the relevant mineral deposit to be mined.			
First level: Group A. Compulsory			
ENN103F	English for Academic Purposes		Semester
MAT1581	Mathematics I (Engineering)		Semester
MEP171X	Mineral Exploitation I		Semester
MSP101R	Mine Survey Practice I		Year (WIL)
SMI181Q	Science: Mining I		Semester
STA1510	Basic Statistics		Semester
Second level: Group A. Compulsory			
EMA2601	Engineering Management II (Module A)		Semester
EMA2602	Engineering Management II (Module B)		Semester
EUC1501	End-User Computing I (Theory)		Semester
EUP1501	Ethical Information and Communication Technologies for Development Solutions		Semester
MAT2691	Mathematics II (Engineering)	Prerequisite: MAT1581	Semester
MGY2601	Mining Geology II	Prerequisite: MEP171X	Semester
MSG211Q	Mine Survey II	Prerequisite: MEP171X	Semester
MSP242S	Mine Survey Practice II		Year (WIL)
MVA231Q	Mine Valuation II	Prerequisite: MEP171X	Semester
Third level: Group A. Compulsory			
EMA3601	Engineering Management III	Prerequisite: EMA2601 or EMA2M1E & EMA2602 or EMA2M2E	Semester
MSG3601	Mine Survey III	Prerequisite: MSG211Q	Year
MSP352T	Mine Survey Practice III		Year (WIL)
MVA3601	Mine Valuation III	Prerequisite: MVA231Q & STA1510	Year
STG381S	Structural Geology III	Prerequisite: MGY291S	Year

Table III

Number of students enrolled in 2012 and their pass rate per subject and category

	Number of students enrolled in 2012			Pass rates per subject/category		
	Private	**Professional**	**Total**	**Private**	**Professional**	**Total**
Mine Survey II	45	12	57	12%	26%	39%
Mine Valuation II	68	19	87	13%	21%	33%
Mine Survey III	18	12	30	27%	47%	73%
Mine Valuation III	10	14	24	13%	21%	33%
Mine Survey and Valuation II	90	39	129	2%	6%	8%
Mine Survey and Valuation III	38	58	96	15%	38%	56%

used for this purpose. Student attendance was optional. The practical sessions were run on a weekly basis from Monday to Friday. The students who accepted the invitation to be part of the programme were received on a 'first-come first-served' basis in a group of 5 to 6 per session.

The first week of the practical sessions was devoted to traversing. The following week, students were exposed to levelling and grading techniques. This sequence was repeated alternately to accommodate all the students booked for training.

Performance of students

A study conducted at the University of the Witwatersrand developed methodologies for assessing the effectiveness of supplemental instruction (SI). The method involves various kinds of comparison between students who have attended a given number (or more) of SI sessions and those who have attended fewer than this number, or no sessions at all ((Martin *et al.*, 1992; Bidgood, 1992; Congos and Schoeps, 1993), cited by McCarthy, *et al.*, 1997). Practicals on campus can be assimilated to supplemental instruction. Therefore the same methodology is used in this paper for comparing the performances of two groups of students.

The key metric used to quantify students' performance is the pass rate calculated after each examination period (January/February, May/June, and October/November). For

simplicity, these periods are labelled as January, June, and October respectively. The official examination results are kept in the database of the academic department for each subject. These are filed as historical data for auditing purpose and statistical analysis. They can also be obtained on request from the Examination Administration Division. These two sources of information were used and consulted to compile the pass rates over five years (2011 to 2015) for six mine surveying modules forming part of the National Diploma in Mining Engineering and National Diploma in Mine Surveying offered at UNISA during the period under consideration: Mine Survey II and III, Mine Valuation II and III, and Mine Survey and Valuation II and III.

Mine Survey II and Mine Valuation II were offered during the period under investigation as semester modules with two sitting examinations June and October. Mine Survey and Valuation II and III were offered as full-year modules until 2013, with one examination in October followed by a supplementary in January. Therefore, the results for the January session are combined with those of the October session from the previous year to obtain the overall results for each year. However, these modules were 'semesterized' from 2014 and now have two examination sittings – one in June and one in October. For semester modules, the June and October results are combined to reflect the results for each year. Students who fail in June and qualify for supplementary examination re-write in October of the same year, or in general during the next semester session of examination.

The principle of combining October and January results applies also to Mine Survey III and Mine Valuation III, which were full-year modules during the entire period of investigation.

The pass rate is calculated as the number of students who passed as a percentage of the number of students who wrote the examination in each period. However, it should be noted that all students admitted for examination do not necessarily write, due to the drop-out rate.

The full names of modules are presented in Tables I and II. However, for the analysis of the results, each subject is further described by its code as listed in Table IV.

Student attendance at mine survey practicals

A roster was designed for mine surveying practicals for implementation over a year, split in the two semesters. Students had the choice to opt into two of the six sessions scheduled per semester on the roster, shown in Table V. Three groups were formed; labelled A, B, and C.

Students' attendance at practicals was recorded and is summarized in Table VI.

It appears that the attendance was very low, at 6%, for the first semester. This can be explained in part by the fact that the practicals were optional. The attendees were monitored for the duration of this research. It will be shown later, when discussing the results, that not all students registered were admitted to sitting for the May/June examination period. Some of students cancelled their registration for financial reasons, which is one of the causes of drop-out. Some did not meet the minimum requirement of submitting at least one assignment in order to gain access to examination.

Results and findings

As stated prior to the analysis, the average pass rate for all the subjects over the last five years is very low (28%), and varies significantly from one year to another as well between

Table IV

Mine surveying modules offered in the two qualifications NDMIN and NDMSR

Subjects	Module codes
Mine Survey II	MSG211Q
Mine Valuation II	MVA231Q
Mine Survey and Valuation II	SAV2601
Mine Survey III	MSG3601
Mine Valuation III	MVA3601
Mine Survey and Valuation III	SAV3601

Table V

Roster of survey practicals

	Group A	Group B	Group C
Semester 1			
Traversing	22/02–26/02	07/03–11/03	04/04–08/04
Levelling /grading	29/02–04/03	14/03–18/03	11/04–15/04
Semester 2			
Traversing	18/07–22/07	15/08–19/08	29/08–02/09
Levelling /grading	25/07–29/07	22/08–26/08	05/09–09/09

Table VI

Number of students who attended practicals *vs* number of students enrolled in 2016

	Number of students enrolled in 2016 (Semester 1 and year)	Number of students attended practicals	Attendance (%)
Mine Survey II	61	7	11%
Mine Valuation II	40	2	5%
Mine Survey III	29	2	7%
Mine Valuation III	27	1	4%
Mine Survey and Valuation II	170	13	8%
Mine Survey and Valuation III	163	6	4%
Total	490	31	6%

the different modules. The low pass rate can be attributed to the lack of practical experience among students enrolling directly from high school.

Figure 2 shows the pass rates for all the modules from 2011 to 2015.

In order to assess the trend in pass rates over the period under investigation, separate charts with trend-lines are provided for each module in Figures 3 and 4. Similar trends can be observed for the other mine surveying modules as shown in Figure 2.

It can be noted that the pass rate is decreasing for all the modules except for MVA3601. This is alarming and indicates the need to find solutions that can help to improve the situation.

The pass rates for all the modules over five years from 2011 to 2015 are summarized in Table VII.

The number of students enrolled in 2016 was reported in Table VI. As stated previously, not all students who registered wrote in the June examination period, which will be used to assess the impact of practicals on the results of semester's mine survey modules MSG211Q, MVA231Q, SAV2601, and SAV3601. Table VIII shows the number of students who wrote in the June examination period and therefore indicates the number of students who dropped out for any of the reasons mentioned.

The ratio of students writing the examination to those enrolled is 82%, indicating a significant rate of absence (18%) at the examination. Dropping out remains a serious concern for engineering studies. A study conducted at the Faculty of Engineering of the University of KwaZulu-Natal reports a similar academic drop-out rate of 14% (Pocock, 2012). An analysis of, and interviews with, a sample of the students who left showed that financial reasons played a significant role, with 49% of non-academically excluded students having financial difficulties, and that a significant proportion of students continue their studies at universities of technology.

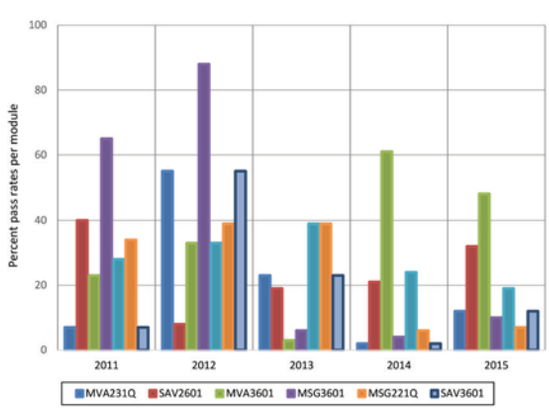


Figure 2—Historical records of pass rates per mine surveying module over the period 2011–2015 at UNISA

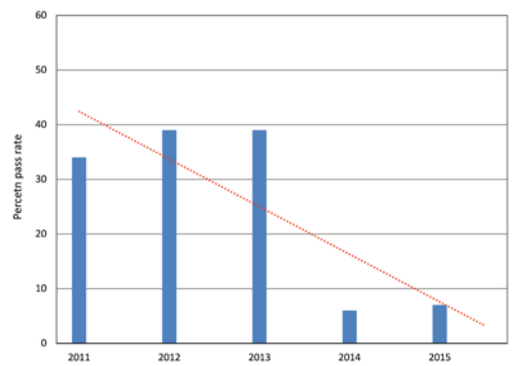


Figure 3—Percentage pass rates for Mine Survey II (MSG211Q)

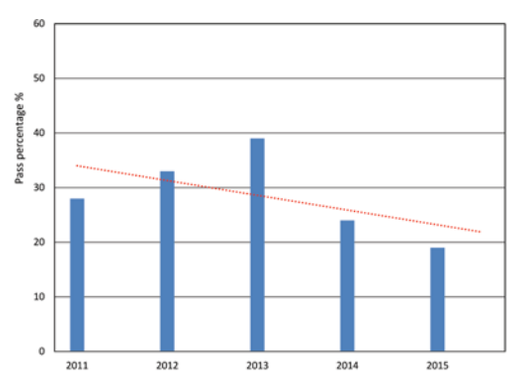


Figure 4—Percentage pass rates for Mine Valuation II (MVA231Q)

Table VII

Pass rates per module per year (%)

	2011	2012	2013	2014	2015
Mine Survey II	34	39	39	6	7
Mine Valuation II	28	33	39	24	19
Mine Survey III	65	73	6	4	10
Mine Valuation III	23	33	3	61	48
Mine Survey and Valuation II	40	8	19	21	32
Mine Survey and Valuation III	7	55	23	2	12

Table VIII

Number of students writing examinations *vs* number enrolled and drop-outs

	Number of students enrolled in 2016 (S1)	Number of students written in 2016 (S1)	Number of students absent (not admitted and drop-outs)	Students writing examination (%)
Mine Survey II	61	50	11	82%
Mine Valuation II	40	31	9	78%
Mine Survey and Valuation II	170	145	25	85%
Mine Survey and Valuation III	163	128	35	79%
Total	434	354	80	82%

After the inception of the practicals, it was noted that a low proportion of students attended compared to the total number of students enrolled for the year and the first semester. From Table VI, a total of 31 students, representing 6% of the total enrolled, attended. A further investigation shows that out the 31 students, only 19 students wrote the June examination, which represents 5% of the cohort examined, as illustrated in Table IX.

To assess the impact of practicals, the pass rates of students who attended are compared to those who did not attend. The average pass rate of students who attended is 46%, while it is 39% for the other group. A single factor analysis of variance (ANOVA) was performed using Microsoft Excel to test the hypothesis that the means of the two groups are equal. The result of the test is summarized in Table X.

The terms from Table X have the following meanings in the test as performed using the data analysis tool in Excel: SS = Sum of Squares, df = Degree of freedom, MS = Mean square, F = Test statistic, and F crit = the critical value of the test. In concluding the test, F < F crit means that we fail to reject the hypothesis that the two means are equal. In other words, the sample evidence is not strong enough to warrant rejection of the null hypothesis.

The individual pass rates for the modules written in the June examination are shown in Table XI. A noticeable improvement is evident compared to the previous examination period.

Discussion

The improved pass rate among students who attended the practical sessions can be regarded as an incentive to make the mine surveying practicals compulsory in future while integrating them to the current learning and assessment.

Is the overall and partial pass rate increase compared to five previous years attributable to practicals? Considering the amount of information available at this stage, it would be premature to give a definite answer. Therefore it is recommended that the scope of the investigation be extended beyond one year and to cover various cohorts of students.

Conclusions

The results of this study constitute an incentive to make the mine surveying practicals compulsory in future while integrating them with the current learning and assessment. One consequence of this would be the training of well-rounded technicians with sought-after skills for the South African mining industry.

Table XI

Comparison of pass rates in June examination to the previous sessions

	Pass rates	
	2011–2015	2016 (S1)
Mine Survey II	24	26
Mine Valuation II	29	68
Mine Survey and Valuation II	25	50
Mine Survey and Valuation III	20	22
Average	25	36

Table IX

Summary of the results per module

	Number of students examined 2016 (S1)	Number of students attended practicals	Attendance (%)	Examination pass rate	Relative pass rate increase
Mine Survey II	50	4	8%	26%	8%
Mine Valuation II	31	0	0%	68%	134%
Mine Survey and Valuation II	145	10	7%	50%	100%
Mine Survey and Valuation III	128	5	4%	22%	10%
Total	354	19	5%	36%	44%

Table X

Analysis of variance

Anova: single factor

Summary

Groups	Count	Sum	Average	Variance
Students who attended	19	864.08	45.478	428.06
Students who did not attend	335	13211	39.437	326.3

ANOVA

Source of variation	SS	df	MS	F	P-value	F crit
Between groups	656.24	1	656.24	1.9796	0.1603	3.86801
Within groups	116689	352	331.5			
Total	117345	353				

The findings in this paper are limited to UNISA. However, it would be interesting to investigate and compare the performance of students in surveying disciplines at other institutions in South Africa and internationally. In this regards, attempts were made to collect statistical information from neighbouring institutions (in South Africa), but data could be not be obtained before this paper was submitted for publication. The same challenge may arise when dealing with institutions abroad.

In the opinion of the author, a more detailed paper can presented in the future that can shed light on what is happening elsewhere with teaching and learning of geomatics.

Acknowledgements

The author would like to thank the Examination Department of the University of South Africa for providing the statistics used in this work.

Special thanks to the Department of Electrical and Mining Engineering for support throughout the execution of the research work.

References

McCarthy, A., Smuts, B., and Cosser, M. 1997. Assessing the effectiveness of supplemental instruction: A critique and a case study. *Studies in Higher Education 22*, no. 2: 221–31.

Pocock, J. 2012. Leaving rates and reasons for leaving in an engineering faculty in South Africa: A case study. *South African Journal of Science*, vol. 108, no. 3–4. pp. 60–67.

Triola, M.F. 2005. Essentials of Statistics. Pearson Addison Wesley, Boston, MA.

UNISA. 2016a. Directorate: Information and Analysis (DIA), Information and Institutional Analysis Portal, Unisa, http://heda.unisa.ac.za/indicator-dashboard/default.aspx [Accessed 5 July 2016]

UNISA. 2016b. Learning Management System (LMS)-myUnisa,http://www.unisa.ac.za/sites/default/

UNISA. 2016c. MyRegistration@Unisa. College of Science, Engineering and Technology (online brochure). https://registration.unisa.ac.za/info/ [Accessed 25 April 2016].

UNISA. 2016d. XMO - examination marks online. (Intranet). http://www2.unisa.ac.za/AOL/XMO/

UNISA. 2014. Self-evaluation portfolio: National Diploma in Mine Surveying, 2014 (unpublished). ◆

http://dx.doi.org/10.17159/2411-9717/2017/v117n3a2

Factors and challenges affecting coal recovery by opencast pillar mining in the Witbank coalfield

by P.L. Ngwenyama*, W.W. de Graaf*, and E.P. Preis*

Synopsis

The depletion of coal reserves in the Witbank coalfield in Mpumalanga Province has resulted in mining companies exploring the possibilities of extracting coal pillars. These are pillars that were left behind for hangingwall support during underground bord-and-pillar operations. Recent studies of *in situ* pillar mining have found the extraction of the pillars to be feasible during opencast mining due to the high extraction rates of coal, relatively low stripping ratio, safety of the operation, and general environmental requirements. The geological model of an opencast pillar mining operation within the Witbank coalfield has indicated that some 30% of the coal in the no. 2 seam remains in pillars. The no. 4 and no. 1 seams are yet to be mined. Opencast pillar mining requires maximizing coal recovery in order to be competitive in the market, since a portion of the resource has already been extracted. Exposure and recovery of the coal are crucial in reducing coal losses and dilution due to the coal pillars and voids, and challenges experienced during the mining of pillars from surface. The reconciliation process evaluated the overall flow processes, from *in situ* coal to the mined-out coal. The similarities between opencast pillar mining and conventional opencast mining were studied in terms of the mining sequence, pit layout, and operations. A correlation between the SAMREC Resource and Reserve definitions was conducted through an investigation of coal losses and contamination during mining. The various types of coal losses affecting production volumes were investigated. The dilution of coal was found to be higher in the no. 2 seam due to blasted material filling the voids in the bords. The presence of bord voids is one of the factors that increases the risk of spontaneous combustion. This in turn affects the productivity of the operation, with buffer blasting management and cladding techniques used to reduce the risk of spontaneous combustion.

Keywords

coal recovery, pillar extraction, opencast mining, coal losses.

Introduction

The Witbank coalfield in Mpumalanga Province holds one of the largest coal resources in South Africa and the rest of the world . Both export and domestic coal are produced from various mines in the coalfield. The study was based on one of the mines, which produces both export coal and domestic coal which is supplied to a nearby Eskom power station. The mine is currently mining pillars from a previous underground bord-and-pillar operation as well as a virgin coal seam. The pillars are extracted using a surface mining method referred to as opencast pillar mining, with an aim of maximizing coal recovery from the pillars. The selection and design process of the opencast pillar mining method was also influenced by the coal production rates and demand from the power station. Three seams are being mined; the no. 4 lower seam (S4L), no. 2 seam (S2), and the no. 1 seam (S1). The S4L and S1 are virgin coal seams while the S2 resources consist of pillar coal remaining from a previous bord-and-pillar operation that reached its end of life in the 1980s. The underground mine was operated using the conventional drill-and-blast, bord-and-pillar mining method. The pillars were left intact when the underground operation was shut down.

In 1995, a project to extract the remaining pillars was started as part of an initiative to extend the life of the mine. The extraction of the pillars was found to be economically feasible due to the good quality coal remaining in the pillars (Table I). Overburden removal began in 2000 using a truck-and-shovel fleet. This was an initial and experimental approach with the intention of exploring the risks associated with the pillar mining method, which was uncommon at that time. The truck-and-shovel operation included cleaning around the pillars, which resulted in extensive coal losses. Due to the size of the equipment it was possible to dig the pillars without the need for blasting. It was then decided to implement a dragline operation together with the truck-and-shovel fleet to overcome challenges with stability and spontaneous combustion on the midburden bench above the mined-out pillars. The first dragline was introduced in 2004. The use of the dragline was not part of the original design, and it was introduced only after it was identified as a lower risk operation compared to the truck-and-shovel fleet on the midburden.

* *Department of Mining Engineering, University of Pretoria, South Africa.*

 This paper was first presented at the New technology and innovation in the Minerals Industry Colloquium', *9–10 June 2016, Emperors Palace, Johannesburg, South Africa.*

Table I

Average coal properties for the three mined seams

Seam	Thickness (m)	Calorific value (MJ/kg)	Ash content (%)
No. 4 seam	2.3	24.6	22.7
No. 2 seam (S2RPB)	4.0	24.9	22.1
No. 1 seam	2.3	23.9	25.9

The S2RPB seam comprises three horizons: the S2R (no. 2 seam roof coal); S2P (no. 2 seam pillar coal); and S2B (no. 2 seam bottom coal). The S1 seam is separated from the S2RPB seam by parting material that varies between 0.1 m and 28.0 m in thickness across the Witbank coalfield, and is approximately 7.0 m thick on average. The S1 seam is extracted together with the S2RPB seam where the parting is very thin. The extraction of the pillars was found to be feasible through the use of opencast mining methods due to the low stripping ratio, high recovery and extraction rate, and environmental and safety factors. This seam split was a function of the different products, with the pillar coal destined to produce export-type coal products and the roof and floor coal used in the adjacent power station after beneficiation in a separate plant. The pillars of the S2 seam in the ground are depicted in Figure 1.

The S2RPB seam, simply referred to as the S2RP seam, was of major concern due to the potentially disturbed and deteriorated conditions of the pillars since the underground workings stopped some years ago. The geological polygon of the mine indicated that some 30% of the coal remains in the pillars and as remnant coal in the roof and floor of the previous underground workings. The amount of coal remaining in the pillars was determined from the areal extraction ratio and the average mining height (seam thickness). The average mining height was taken into account due to the inconsistencies in the seam thickness. In some areas, roof and floor coal had been recovered, but this was not always indicated on the survey plans. This discovery was based on recent exploration activities preparatory to the exploitation of the pillars. Investigations were conducted mainly to establish wash plant parameters in the initial design, while the volumes of the floor and roof remnants were calculated from survey plans of the previous operation. As physical access to the workings was not possible, all information had to be obtained from survey plans, which are not always accurate, and from earlier exploration boreholes, which were not subjected to wash analysis as is done currently. The pillars and remnant coal of the S2RP seam situation are illustrated in Figure 2.

When bord-and-pillar mines reach their end of life, coal pillars are inevitably left unmined. A forecast can be made as to the feasibility of effective, efficient, and responsible utilization of the reserves remaining in previously mined-out areas (Schalekamp, 2006). The underground conditions may have changed with time, and this makes it difficult to predict the underground conditions. Jeffrey (2002) investigated the geotechnical factors that influence secondary extraction in previously mined-out coal seams. The current underground conditions of opencast pillar mining operations are expected to be as illustrated in Figure 2, but cannot be defined accurately, because underground access was sealed off; however, the old survey plans have proven to be accurate enough for volume calculations. Understanding the coal volumes is crucial for maximizing coal recovery. This was done by evaluating the variations between theoretical and actual mined volumes.

The global coal industry loses about US$480 million (R5.5 billion) annually in revenue (Thompson, 2005) due to coal losses in the mining process and contamination. The coal price in 2005 was about US$45 per ton (InfoMine, 2015), which means that approximately 10.7 Mt of coal was lost. Coal losses are therefore a major concern for pillar mining operations, since only 30% of the original coal reserves remain underground as old pillars in the no. 2 seam. Coal losses affect operational revenue and are included in the current operating costs. It is thus critical for operations to strive for maximum recovery of reserves (Johnston and Kelleher, 2005). The extraction of pillars through normal opencast mining methods has been found to be a fundamental factor leading to high dilution. The high dilution is due to the large bord voids between the pillars, and this is a major concern in terms of the operating costs, especially the processing plant operating costs. The main aim of collapsing the bords, which incurs high dilution, was to maximize coal recovery, while the washing plant would remove waste from the ROM coal. The mining of pillars from surface causes air to penetrate the old workings, resulting in spontaneous combustion, which is one of the big challenges in opencast pillar mining operations. Spontaneous combustion causes *in situ* coal to burn, and this decreases coal quality and quantity, and leads to the emission of noxious gases. The continuous burning of coal in the pit also produces large

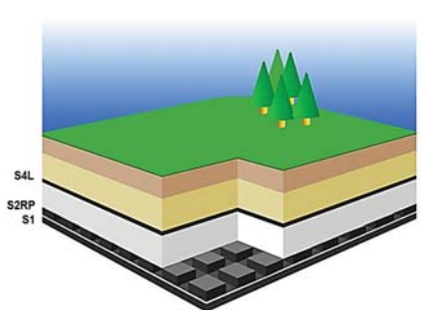


Figure 1—Pillars remaining after previous bord-and-pillar operation

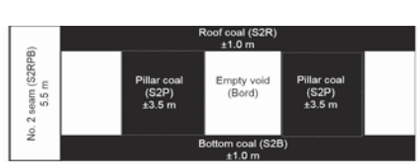


Figure 2—Sectional view of the S2RP seam composition

volumes of dust, dark smoke, and harmful gases during waste removal, which has a deleterious impact on productivity and the environment. A large portion of the dust is created during dumping on the low-wall side.

Opencast pillar mining method

Opencast pillar mining follows a similar layout and stripping sequence to conventional opencast mining. The similarities of the two methods are mainly due to the dragline operation. Equipment selection was influenced by the risks of opencast pillar mining, particularly spontaneous combustion. The dragline is allowed to mine only one bench at a time before moving to the next strip. Exposure of multi-seams at a double bench pass is risky in pillar mining. Hence the dragline on a multi-pass single bench method is preferred in pillar mining operations, while the dragline on a multi-pass double bench is applicable in conventional opencast operations. There are minor differences in the layout of the two methods; for example, opencast pillar mining makes use of a 30.0 m wide blasted buffer to prevent spontaneous combustion and sinkhole formation. Buffer blasting was, and still is, the most effective technique for containing spontaneous combustion. It may not prevent spontaneous combustion completely, but will significantly retard the process.

The buffer is indicated by the dashed blue outline in Figure 3. This is carried over to the next strip of extraction. Cladding is provided by cast-blasting the overburden to cover the edge of the highwall, thus filling the bords and creating the buffer. The mining sequence is similar to the stripping sequence in conventional opencast:

- Removal and stockpiling of topsoil material
- Drilling, charging, and blasting of overburden
- Push over and pre-stripping of overburden using truck and shovel operation
- Stripping of blasted overburden with dragline
- Extraction of S4L coal seam
- Drilling, charging, and blasting of mid-burden
- Stripping of midburden with dragline
- Free digging of S2RP seam coal and blasting of hard pillars
- Drilling, charging, and blasting of S1 parting
- Extraction of S1 coal seam.

Buffer management and cladding techniques were implemented to minimize or prevent the continuous burning of coal in the ground. The old pillars are only in the S2RP seam, and are approximately 7.0 × 7.0 m, with a bord width of 6.1 m between the pillars. There are larger pillars in certain blocks, referred to as barrier pillars during the bord-and-pillar operations. The orientation of the pillars was designed to cut across the strips at an angle, and not parallel to the strips. Thompson (2005) mentioned that this was aimed at increasing the stability of the highwall side or edge. When the pillars are at an angle, the highwall edge will always be supported by pillars, thus preventing the highwall from hanging unsupported above the bords. This also prevented large open spaces being left between two pillars. The risk of flooding from water accumulation in the panels also had an influence on the initial orientation direction of the pillars. A plan view of the pillar orientations relative to strip direction is shown in Figure 4.

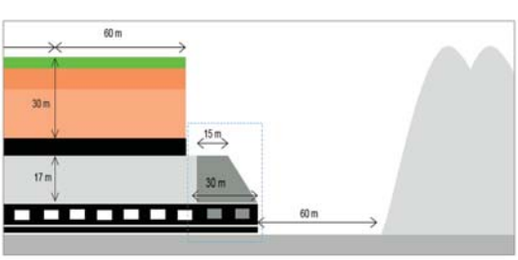


Figure 3—Opencast pillar mining method layout

Resource and Reserve utilization

Reserve models are built and scheduled from a Resource model to determine the profitability of a mine by accounting for realistic factors that may affect the operation, such as coal losses and dilution. The Resource and Reserve models are based on theoretical calculations. The theoretical calculations are put in place to accommodate wide ranges in coal qualities. These differences are corrected or modified by survey volumes that show the real volumes. Losses, dilution, and contamination are critical parameters as they play a major role in the Resource to Reserve conversion process. These factors act as intermediary factors between Resources and Reserves, as well as in moving from mineable *in situ* and run-of-mine to saleable Reserves.

There are three main parameters that should be considered, according to the SAMREC Code:

- *Mineable in situ Coal Reserve*—The tonnages and quality of the coal at a particular moisture content contained in the *in situ* coal seam. This portion of the Mineable Reserve is used in conceptual and detailed mine planning. The mining method and planned losses are determined at this stage
- *Run of Mine Reserve (ROM)*—Run of Mine Reserves are based on the Mineable *in situ* Reserves. This is the actual amount of coal expected to be extracted and delivered to the plant, usually over a particular time period. The Run of Mine Reserves are the tonnages and coal qualities remaining once the following factors have been accounted for:
  - Coal losses during mining
  - Dilution and contamination
  - Moisture content of coal
  - Others, such as geological losses
- *Saleable Coal Reserve*—The actual tonnages with a particular quality that will be available for market sales. This is calculated from the ROM tonnages after washing and beneficiation to the required product quality.

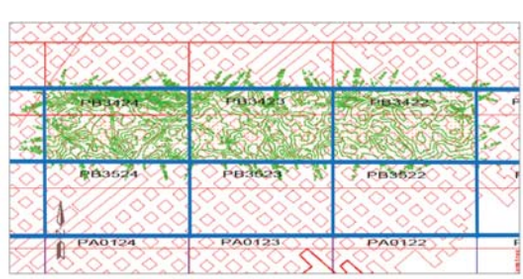


Figure 4—Plan view of intact pillars in the ground

According to the SAMREC Code, the estimation of Reserves and Resources in previously mined workings is very complex compared to virgin coal grounds. This is due to the necessity of considering and accounting for the previously extracted voids. The currently unknown conditions of the standing pillars underground makes it difficult to build the Reserve model by predicting the applied losses. Schalekamp (2006) explained that a proper and accurate reconciliation between planned and actual mineable tonnages requires an extensive survey. However, he noted that it is difficult to conduct physical surveys due to the risks of working directly above blasted and collapsed old workings. The presence of spontaneous heating also makes the risk of surveying more complex in those areas, and semi-collapsed and uncollapsed voids pose further risks. The major challenge is the estimation of the top of coal losses, while highwall and edge losses are easier to estimate.

A review of the Resource (geological) *versus* Reserve model of the mine was conducted to perform a coal mass balance (Figures 5–7). This was done to ascertain whether coal losses and dilution are correctly utilized, especially on the S2RP seam. The no. 2 seam contains voids that become filled with waste from midburden blasting. The dilution was then calculated, theoretically and from plant data, to be approximately 20%. This is a theoretically calculated value that could not be physically measured. The data was based on portions of strips that are currently being mined, and those that are scheduled to be mined in the next two to three years. The Reserve was built from the geological model data, which was obtained from the GEMCOM Minex Software used to create the Resource models, by applying some assumptions as modifying factors for the conversion process (Table II). The modifying factors were applied for the three seams being mined.

Figure 6 and Figure 7 indicate that there is no constant correlation or relationship between the Resources and Reserves of the S2RP and S1 seams as compared to the S4L (Figure 5). Figure 6 shows the Resource of the remaining pillar and remnant coal *versus* the Reserve, taking into account the expected dilution from collapsing the bords. The Reserve is greater than the Resource because the dilution will be higher than the potential losses. The dilution factor may be reduced by the presence of large barrier pillars, while the losses remain constant. This causes the interchanging of the Resource and Reserve tonnages on the S2RP seam. The interchange between the Reserve and Resource tonnages in Figure 7 is influenced by the fact that the S1 seam is sometimes mined together with the parting. The parting is blasted together with the S1 seam where its thickness is less than 0.5 m, and separately where the thickness is greater than 0.5 m. The relationship between the Reserve and Resource is depends mainly on the dilution and coal loss factors:

$$\text{Reserve} = \text{Resource} + \text{Dilution} - \text{Losses} \pm \text{Other (Minor)} \quad [1]$$

There are other factors considered in the Reserve model, such as inherent moisture content. These other factors were regarded as minor because they do not affect the relationship between the Resource and Reserve. For example, the average moisture content was estimated to be approximately 2.5% for all seams. Total moisture content is known to have an effect on coal recovery. The study was limited to losses and dilution; these are the two factors that contribute the most in the Reserve model, and can be estimated based on experience or benchmarks, and measured. These two factors can be measured through survey reports and from plant belt weightometer measurements, but both methods are open to errors. In principle, the Reserve tonnages should be consistently more or less than the Resource tonnages. An increase in the dilution factor resulted in an increase of Reserve tonnages, while losses decreased the Reserve tonnages.

Figure 5—Resource and Reserve utilization mass balance - S4L PA02

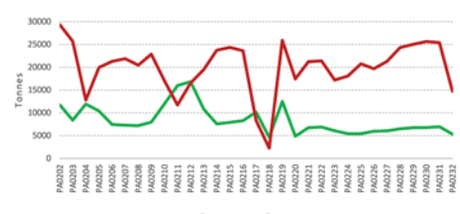


Figure 6—Resource and Reserve utilization mass balance - S2P PA02

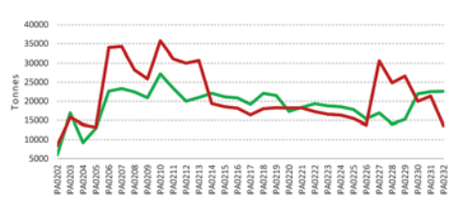


Figure 7—Resource and Reserve utilization mass balance - S1 PA02

Table II

Assumptions used in converting Resources to Reserves

	Losses (%)				Dilution (%)				
Seam	Roof	Floor	Edge	Other	Roof	Floor	Edge	Bord collapse	Other
S4L	5	5	0	7	5	5	0	0	7
S2RP	2	0	2	5	2	0	2	20	5
S1	5	5	2	7	5	5	2	0	7

However, the dilution factor has to be kept within allowable levels. If the dilution is too high the following ensues:

- Tonnages to be mined and processed increase, hence operating costs increase
- Quality of coal decreases
- Time of operating increases.

Dilution during opencast pillar mining

In general, dilution is defined as the percentage of non-coal material that is unintentionally blasted and loaded together with coal and delivered to the plant. The selected mining method uses a blasting technique called bord collapse, which intentionally allows the occurrence of high dilution by collapsing the bords. One of the main reasons for collapsing the bords is to maximize coal recovery. Baruya (2012) explained that dilution affects the quality of coal in the mining process. Reserve models take into account the amount of dilution that is expected during mining activities.

The quantification of dilution requires that waste handling is controlled and reduced by understanding its root causes. The factors that affect dilution are as follows according to Ebrahimi (2013):

- Characteristics of the deposit
- Method and scale of operation
- Types and size of equipment
- Operator skills and overall mine geometry.

It is necessary to quantify dilution in order to improve the design of a mine and evaluate project economics (Ebrahimi, 2013). According to the Reserve model, that there will be 5% roof and floor dilution for the virgin coal seams and 20% for the S2RP seam due to the bords. The high dilution in the S2RP seam is a result of the waste material that falls into and fills up the bords after blasting the midburden.

Figure 8 shows the variation in dilution in a pillar mining operation. It should, however, be noted that this data is for a blend of the three seams. The run of mine (ROM) volumes entering the plant are divided into three products, namely fines material, which is sent to slimes dams; reject material, which includes coal material that is below the cut-off grade; and product, which is the tonnages on the weightometer. The variation in data between the pit and plant has been attributed to the manner in which dilution is measured and calculated.

Figure 8 represents the percentage of discard material rejected by the processing plant from monthly ROM tonnages. The majority of the dilution material is expected to be generated from collapsing the bords in the S2RP, while the rest is expected from the top of the seam, bottom of the seam, and edge of the seams during operations in the Reserve model. However, it was noted that the data from the plant may include a blend of the three seams, losses from fines going into slimes dams, reject coal, and coal below quality cut-off. The dilution of the S1 and S4L seams was relatively easy to estimate, while the S2RP seam dilution could not be easily defined. The following factors and assumptions were taken into consideration in theoretically determining the dilution factor for the S2RP seam:

- The pillars remain intact after blasting the midburden
- The roof coal above the bords is semi-collapsed after blasting the midburden and falls directly into the voids (see Figure 9)
- Scaling of the pillars remains unchanged or the fall-outs accumulate on the floor with the tonnages remaining the same
- The blasted material increases in volume by 60%.

Measuring and estimating dilution can be attributed by a variation in data. Determining the exact amount of dilution would require continuous measuring from the plant data, pit data, as well as over an extensive area.

The blasted midburden is allowed to fall above the pillars and thus fractures the roof coal, which is approximately 1.0 m thick. The midburden is drilled to the top of the roof coal to limit the air entering the old workings. It was initially predicted that the roof coal would fully collapse directly into the voids, while the pillars remained intact. It was then noticed that the roof coal is actually semi-collapsed by the midburden material after blasting. This can be seen in Figure 9, which illustrates the two collapse scenarios for the roof coal.

The ideal scenario depicts the manner in which the roof coal and pillars were expected to collapse. It was assumed that the blasting of the midburden would fully collapse the roof coal above the bords, allowing it to fall directly in the bord void. In this scenario the roof coal above the bord is pushed to the floor and is fractured. The bord voids are completely filled with midburden material.

The actual scenario depicts the situation whereby the roof coal above the bords is semi-collapsed as the midburden is blasted, leaving some unfilled voids. The roof coal becomes mixed with the blasted material during the collapse. The bords are always collapsed, but in some cases the roof coal in the bords may not be fully or perfectly collapsed. This situation remains unchanged even when the draglines works above the pillars. The voids caused by the semi-collapse of the roof coal allow air to penetrate the underground workings, resulting in spontaneous combustion.

$$\text{Dilution} = \frac{\text{Waste material (Tons)}}{\text{Total ROM material (Tons)}} \quad [2]$$

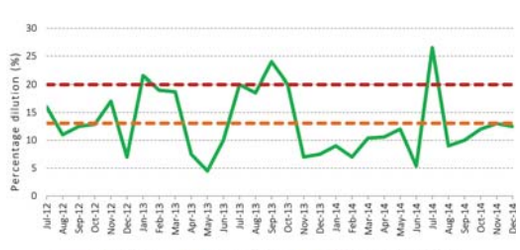


Figure 8—Dilution factor according to the processing plant

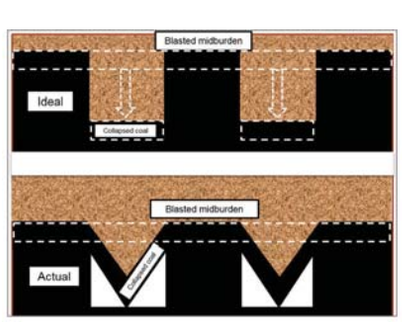


Figure 9—Roof coal collapse scenarios

The actual dilution in the S2RP seam is approximately 21.0%, which is 1.0% higher than estimated in the Reserve model. The main aim of building the Reserve model was to estimate conditions after blasting, exposure, and extraction. The dilution decreases to 19% in areas surrounded by barrier pillars. The amount of dilution in the actual scenario is reduced because of the unfilled voids. Dilution increases the ROM coal quantity while effectively reducing the calorific value and quality of the coal (Baruya, 2012). Mines will tend to increase dilution and encounter greater challenges of dilution when seeking to maximize coal recovery from the coal Reserve. This does not apply in cases where the plant is used to remove contamination, but only in cases where the coal is not beneficiated.

Coal losses and reconciliation

Mining operations expend huge sums in capital and operating costs for the extraction of resources. Several processes are carried out before coal extraction begins. A Reserve model is built to accommodate any potential losses and dilution during extraction.

Figure 10 shows the variance between the tonnages of coal in Reserves, which are converted from the theoretical Resources, and the actual amount of coal exposed and recovered per mining block as measured by the surveyor. This data was obtained over various individual mining blocks for which survey reports had indicated significant losses.

Figure 10 shows that a significant amount of coal was not recovered during the mining processes. In principle, the amount of coal recovered (actually mined) and the Reserve coal should be roughly the same. This is because the Reserve model is constructed to accommodate these potential Resource losses and dilution. These variances indicate that some valuable coal is lost during the extraction process. Field investigations were carried out to determine the types of coal losses that occur in the pit. The types of losses and average tonnage loss per mining block for the three seams are shown in Figure 12. A proposed coal process flow model that shows the stages through which the coal proceeds is shown in Figure 11.

Figure 11 illustrates the random occurrence of different types of coal losses in individual mining blocks for the three seams. The diagram shows the worst cases of recorded coal losses that were encountered in certain individual and random mining blocks. The losses were found to have very low occurrence rates.

Geological and geotechnical losses occur naturally and randomly across all seams and cannot be physically prevented. The geological losses are dominant on the S1 seam due to mining blocks being partially mined or left unmined as a result of the parting being above cut-off thickness. The parting material is situated above the S1 seam and makes it difficult or uneconomical to extract the seam in certain blocks.

Highwall losses occur as a result of underbreak on the highwall. There is potential for recovering these losses on the next strip if the exposure is dedicated and prepared for the recovery as buffer blasting is also in place. However, there are risks involved in doing so because this can result in greater losses and damage. The Reserve model does not take highwall losses on the S4 into account because of the implementation of pre-split blasting on the upper 4 seam bench. These losses are higher on the S2RP and S1 than the S4L because the midburden is not pre-split blasted. The rock properties have an influence on the effectiveness of pre-split blasting. Pre-split blasting is better suited for hard material than for oxidized or weathered overburden material.

Figure 13 illustrates the occurrence of highwall losses. Highwall blasting with respect to pre-split blasting should be reviewed and monitored frequently in order to reduce highwall losses. The implementation of accurate pre-split blasting procedures is the key to the minimization of highwall losses. The pre-split is blasted only on the overburden (S4L seam), and not on the midburden which sits above the pillars.

Edge of seam/low-wall losses occur due to a collapse of the spoil. The spoil material begins to roll down and starts covering the edge of the seam on the low-wall side. This is due to increased volume of stripped material. Edge losses increase with depth from the S4L to the S1, thus S4L edge losses << S2RP edge losses << S1 edge losses. This results from the increase in the volume of the stripped material. This is noted in the vicinity of ramps or corners in the pit where there is limited space for spoil material.

Attempts to recover edge of seam losses are strictly prohibited due to the risks associated with disturbing spoil, which can collapse, resulting in increased edge of seam losses. Correct parameters, such as the amount of material to

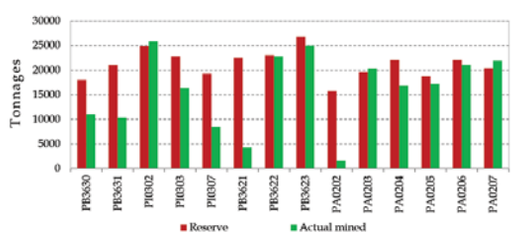


Figure 10—Random and individual mining blocks affected by losses and dilution

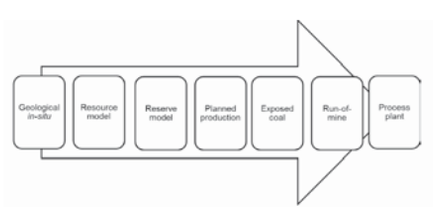


Figure 11—Coal flow process model

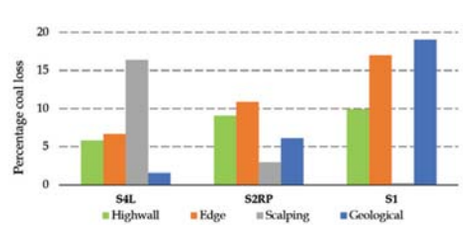


Figure 12—Coal losses for individual mining blocks in the three seams

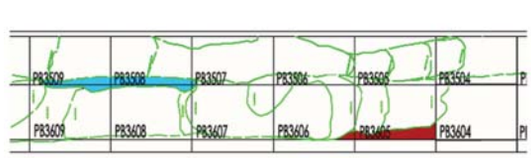


Figure 13—Highwall losses (red shading)

be exposed, positioning of draglines (total distance of reach of dragline), and positions of access ramps can prevent edge of seam coal losses. Another method is to create a 5.0 m wide void at the edge of the seam as a safety measure for occasionally collapsing spoil. The void can alternatively be used in areas where there are high volumes of material to be exposed. For localities with limited spoil area, such as ramps, the void can be deepened where necessary. A review on the pre-stripping of the overburden can be conducted to reduce and eliminate the occurrence of edge of seam coal losses.

Scalping or top of coal seam losses. The dragline bucket starts scalping the top of the coal seam during exposure, which results in losses. This can be quite significant during the night shift as the visibility of the interface between waste material and coal is affected. The hardness of the coal determines the rate of this loss. The S4L seam is generally softer, hence is subject to higher scalping losses, whereas the hardness of the S2RP and S1 prevents large scalping losses. The scalping losses in this case decrease with depth, thus S4L scalping losses >> S2RP scalping losses >> S1 scalping losses.

The top of coal losses are indicated by the purples lines in Figure 15, where green represents some geological losses. Top of coal losses during exposure can be effectively managed through careful dragline operation. With the functional capabilities of draglines and the known depths of the seams, a digging limit with reference to the seam depth should be implemented on the draglines to prevent scalping. The S4L is still virgin coal and thus scalping losses in pillar mining and conventional mining should not be different.

Floor losses result when coal is left on the floor. Floor losses were found to be minimal across all seams. Credit can also be given to the efficiency of operators in the extraction crew. The occurrence of floor losses depends on the ability of the loader or excavator bucket to penetrate the bottom of the seam. Water accumulation affects the visibility of the bottom of the seam, and the loader operator may struggle to see coal remaining on floor. However, floor losses are often marginal.

Figure 16 displays a sectional view summarizing the different coal losses as described above.

Preventative strategies need to be implemented in order to minimize coal losses. These could include non-technical strategies, which influence the overall productivity of workers. Operators are working directly with the coal in the pit and can play a major role in the prevention of coal losses. Operators should be familiarized with the effects of coal losses and dilution. Refresher courses where operators are taught about the influences of coal losses in the mining business can be implemented. The courses should also be aimed at training operators, such as dragline and loaders operators, to distinguish the types of material by colour, hardness, or other properties.

The stages that the coal goes through (from *in situ* to the plant) are illustrated in the proposed coal recovery model in Figure 17. The factors associated with each stage or processes are also highlighted. Some coal losses occur through blast damage to the coal seams, when the coal becomes locked up with the overburden material during the blast. Guidelines for the drilling and blasting cycle should be provided, and compliance with working procedures should be well supervised. The workers and operators should be acquainted with the relevant working procedures. For example, the blasting crew should understand the reasons for backfilling over-drilled blast-holes, and the drill operators should understand the reasons for re-drilling an under-drilled blast-hole.

Scott and Wedmaier (1995) investigated the sources of coal losses and dilution. Coal losses have a direct relationship to dilution. It was concluded that the roof of the seam experiences more losses than dilution, while there is more dilution than losses at the bottom of the seam (Baruya, 2012). The following were identified by Baruya (2012) as the major factors influencing the occurrence of coal damage, losses, and dilution:

- Blast damage: during blasting operations of the overburden, mid-burden, and parting material at the top of the coal seam
- Type of equipment used (draglines *versus* truck and shovel) for coal exposure
- Extraction method and equipment: loader and truck extraction method and size of the equipment
- Equipment movement on top of coal
- Geometry of the top of coal and floor conditions: the deposition of the coal seams relative to the horizontality of the surface
- Coal properties: hardness and colour of the coal seam
- Operator experience and visibility

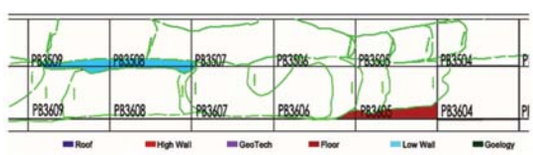


Figure 14—Edge of seam losses (blue shading)

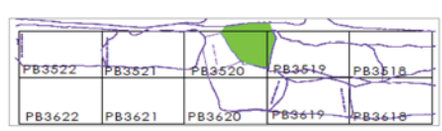


Figure 15—Scalping losses

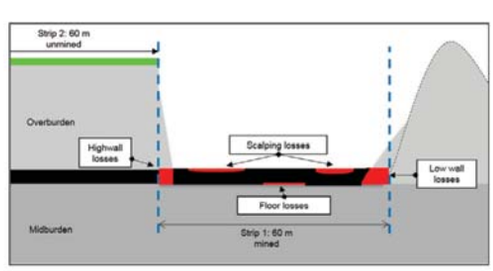


Figure 16—Sectional view illustrating the different coal losses

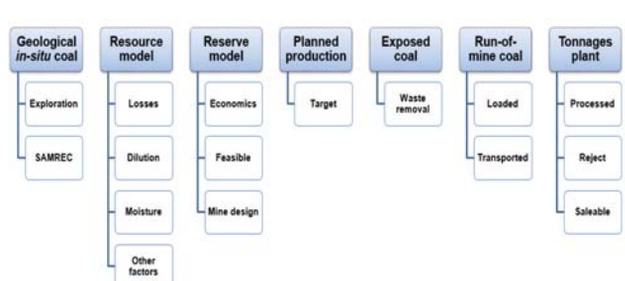


Figure 17—Coal recovery model – flow of coal

- Presence of water in the coal exposure and loading activities.

Effects of spontaneous combustion

Opencast pillar mining can be widely affected by spontaneous combustion. Spontaneous combustion in coal mines is simply the oxidation of self-heated coal, resulting in burning of *in situ* coal (Eroglu and Moolman, 2003; Phillips *et al.*, 2011). Spontaneous combustion affects the calorific value of coal, leads to the emission of noxious gases, and contributes to coal losses both directly and indirectly. The direct losses occur as coal continues to burn in the ground, and the value and quantity of the coal is decreased. The indirect losses are a result of the effect of burning coal on productivity.due to:

- Relatively complex hot-hole blasting procedures
- Excessive dust generation during exposure activities, which affects visibility during dragline operations
- Delays (such as treating hot holes and extinguishing open fires) due to spontaneous combustion that affect productivity and operating costs
- Reduced stockpiling as burning coal is sent directly to the plant
- Risk of people and machines working close to burning areas.

The basic principle in controlling and preventing spontaneous combustion is to eliminate one of the requisite components – fuel, heat, and oxygen. Fuel (coal in this case) and heat from underground self-heating cannot be eliminated in practice. While there are various ways in which oxygen can ingress the underground self-heated coal, oxygen seems to be the key issue in dealing with spontaneous combustion (Eroglu and Moolman, 2003). Spontaneous combustion will further contribute to coal losses if air continuously penetrates the old workings. The quality and quantity of coal is affected by spontaneous combustion. Spontaneous combustion necessitates the need for specialized techniques such as hot-hole blasting, buffering, and cladding. Excessive dust is encountered during dragline exposure activities as a result of burning coal and this affects the productivity of the operation as a whole. The presence of dust increases dragline cycle times, hence lowering productivity.

The occurrence of spontaneous combustion can be effectively prevented and well managed through the use of a buffer and cladding blasting. These two techniques were initially used separately, but there were still signs of spontaneous combustion. It was then decided to combine the two techniques for the same purpose, as shown in Figure 18.

The cladding technique is used to minimize the entry of oxygen into the bord areas. The blasted muck-pile covers the no. 2 seam pillar coal with oxidized overburden material. A dozer may be used to push over more material to make sure that the area is sealed off completely. The buffer management technique ensures that the voids are completely filled with non-combustible material to limit entry of oxygen into the bord voids. These techniques have proven to be successful in the elimination and prevention of spontaneous combustion, and have the following benefits:

- Reduction of direct coal losses
- Improved productivity
- Reduced hot areas and hence a minimization of the use of hot-hole blasting procedures
- Control of the risks to people and equipment associated with loading of burning coal
- Healthier and safer working conditions.

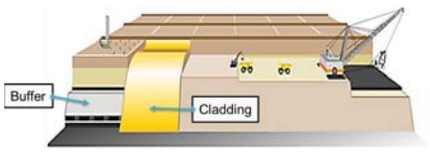


Figure 18—Typical cladding and buffer management techniques

Conclusion

Opencast pillar mining is fairly similar to conventional opencast mining in principle; however, it entails various unique challenges. Opencast pillar mining is faced with the risks of spontaneous combustion, coal losses, high dilution due to the bords, imperfectly known conditions, and inaccurate plans of the old workings. Mining of old pillars is, however, viable due to the good quality coal in the pillars. The consequences of dilution and coal loss in pillar mining methods are more severe than in conventional opencast mining. This paper has shown that coal recovery can be improved if the process plant is capable of handling large amounts of dilution material. If this is not the case, the operation should attempt to mine accurately to avoid dilution, but this may result in coal losses. The flow of the coal process was investigated from *in situ* geological tonnages up to the point that the coal reaches the process plant. This procedure indicated errors and inaccuracies in the data measuring practices. Various types of coal losses were investigated, and certain types of losses were found to be influenced by the mining of old pillars, such as coal lost due to spontaneous combustion and flooding of the old workings. The impact of the factors and challenges of opencast pillar mining on coal recovery can be managed to prevent adverse effects on productivity, minimize coal losses, and control dilution.

References

Baruya, P. 2012. Losses in the coal supply chain. IEA Clean Coal Centre, London, UK.

Ebrahimi, A. 2013. The importance of dilution factor for open pit mining projects. *Proceedings of the 23rd World Mining Congress*, Montreal. CIM, Montreal.

Eroglu, N. and Moolman, C. 2003. Development methods to prevent and control spontaneous combustion associated with mining and subsidence. *Coaltech 2020*. Division of Mining Technology, CSIR.

InfoMine. 2015. http://www.infomine.com/investment/metal-prices/coal/all/

Jeffrey, L.S. 2002. Geotechnical factors associated with previously mined areas of coal and their impact on subsequent extraction. MSc thesis, University of the Witwatersrand.

Johnston, S.N. and Kelleher, M.L. 2005. Keep the cream - reconciling coal recovery at BMA Goonyella Riverside. *Proceedings of Coal 2005: Coal Operators' Conference*. Aziz, N. (ed.). University of Wollongong and the Australasian Institute of Mining and Metallurgy. pp. 161–168.

Mine X Colliery. 2015. Survey plans, Resource and Reserve models, Process plant data, and Internal reports and presentations.

Mine X Colliery personnel. 2015. Personal communications.

Phillips, H., Uludag, S., and Chabedi, K. 2003. Prevention and control of spontaneous combustion. Best practice guidelines for surface coal mines in South Africa. *Coaltech 2020*. Division of Mining Technology, CSIR.

SAMREC. 2009. South African Mineral Resource Committee. The South African Code for Reporting of Exploration Results, Mineral Resources and Mineral Reserves (the SAMREC Code). 2007 Edition as amended July 2009. http://www.samcode.co.za/downloads/SAMREC2009.pdf

Schalekamp, E.E. 2006. The financial viability of coal reserves within previously mined areas of the Witbank coalfield. Masters thesis, University of Pretoria.

Scott, A. and Wedmaier, R. 1995. The assessment and control of coal damage and loss. Final report C3017. Australian Coal Association Research Program (ACARP), Brisbane, Australia. 93 pp.

Thompson, R.J. 2005. Surface Strip Coal Mining Handbook. South African Colliery Mine Managers Association. ◆

http://dx.doi.org/10.17159/2411-9717/2017/v117n3a3

Weathering the 'perfect storm' facing the mining sector

by N. Singh*

Synopsis

The South African mining industry has made a significant contribution to the country's economy for more than a century. Changes in legislation for mining licences, stricter health and safety targets, and stronger focus on reducing the impact on the environment coupled with increasing labour and electricity costs are some of the factors that significantly changed the landscape in which the mining industry now operates. Further to this, low commodity prices force South African mines to seek new, more technically advanced, cost-effective ways of increasing production without compromising occupational health and safety.

Research, development, and innovation (RD&I) in the mining sector is therefore required to find solutions that are cheaper, safer, and more efficient. Alternative programmes must be developed and put in place to transform South Africa's comparative mineral endowment advantages into more sustainable and globally competitive strengths.

This paper discusses the recently developed and accepted South African Mining Extraction Research, Development & Innovation (SAMERDI) strategy, which is strongly focused on productivity-related issues in order to ensure that mineral resources can be converted into mineable reserves economically, safely, and with minimal impact on the environment. Furthermore the paper will discuss the merits of revitalizing the mining RD&I environment through the strenghtening and consolidation of efforts in a manner that is collaborative and ultimately sustainable.

Keywords

South African mining, competitiveness, productivity, innovation, research and development.

Introduction

The underground coal mining industry underwent a resurgence when blasting was replaced by the use of continuous miners. Not only were there significant improvements in productivity and efficiency; the safety record improved dramatically. Hard rock underground operations have not made this major transition, despite significant efforts to develop similar technology for narrow-reef, tabular metalliferous mines. Metalliferous underground mining in South Africa industry is carried out using cyclic drilling and blasting operations. Although there have been improvements over time, these have been small and incremental.

- Ore is still broken from the host rock by the use of explosives, which requires blast-holes to be drilled into the rock mass
- Detonation of the explosives is now done in a much more controlled and coordinated manner, by the use of centralized blasting and electronic detonators rather than the manual lighting of each fuse by the miner as in the early days of mining
- Scrapers and winches for cleaning, which were introduced in the 1920s, are still in use today
- Support systems, to this day, consist of combinations of *in situ* pillars, backfill, wood, and steel.

Mining in South Africa

South Africa's mineral resources endowment is the foremost in the world, not only as regards gold and platinum group metals (PGMs) but also manganese and chromium, to name but a few. The minerals industry contributes significantly to South Africa's internal energy requirements, trade balance, internal investment, domestic savings, foreign capital, and direct and indirect employment creation. Mining has made a significant contribution to the South African economy for more than a century. Many of the challenges facing the domestic minerals sector over the medium to long-term parallel those at the global level.

- South Africa faces increased competition in terms of attracting foreign direct investment in the minerals sector
- Mining horizons within existing orebodies are becoming deeper and more difficult to access and extract, necessitating the adoption of mechanized extraction methods, optimization of

* *CSIR, South Africa.*

 This paper was first presented at the New technology and innovation in the Minerals Industry Colloquium', *9–10 June 2016, Emperors Palace, Johannesburg, South Africa.*

existing process flow sheets, and the development of new and improved methods to increase efficiency and improve recoveries
- There is an increasing push for the development of technologies capable of maximizing energy and water efficiency and ensuring environmental integrity
- Furthermore, as prices across all the mining commodities continue in a downward trend, marginal mines face an ever-increasing risk of closure and thus the post-mining rehabilitation and regeneration of mine land is another critical issue that needs to be addressed now.

The components of a perfect storm

A 'perfect storm' is *a detrimental or calamitous situation or event arising from the powerful combined effects of a unique set of circumstances*. In relation to the South African mining industry the issues can be summarized as follows.

- The ubiquitous need to ensure that the target of zero harm is achieved with regard to occupational health and safety
- The South African gold industry is characterized by decreasing production, increasing depth of operations, and harder-to-access payable material
- Labour and other costs have increased significantly and have adversely affected the profitability and sustainability of gold mining in South Africa
- South Africa dominates global production of PGMs. Labour costs and rising steel and electricity costs all result in profitability being marginal in terms of operational costs
- In the iron ore industry, transport infrastructure and transport costs are major problems. The low commodity price as well as the decrease in the demand for steel has had a negative impact on this sector
- The need to ensure that mine rehabilitation requirements are well resourced and funded so that the environmental impact is minimized both during mining as well as once mining activities cease
- The skills shortage from a technical as well as operational perspective
- Socio-economic issues, especially in communities where mining operations are located.

Figure 1 is a graphical representation of the challenges facing the mining industry. As per the definition, it can be postulated that the South African mining sector is facing a *perfect storm*.

The severity of the challenges facing the South African mining industry is summarized in the following points that were raised during various mining stakeholder forums. The Chamber of Mines (Baxter, 2016) reported that:

- 59 407 jobs were lost between 2012 and 2015
- Productivity has declined in the gold and platinum sectors
- Mining's contribution to the GDP declined from 21.9% in 1970 to just 5.1% in 2014
- Having led the world in gold production in 2006, South Africa now ranks 7th
- Easy accessible resources, especially in gold, are becoming depleted
- The industry has left a legacy of environmental problems, including acid mine drainage.

All hands on deck

Following the Malaysian model for quick tangible results, the South African government-led Phakisa process (Department of Planning, Monitoring and Evaluation, n.d.) was applied in areas that would help foster growth and the attainment of national development goals. Phakisa is essentially a collaborative process, convened by government but involving a range of key stakeholders, to plan and oversee the implementation of initiatives that will have a positive impact on the economy and society. Mining is one such area (South African Government News Agency, 2016).

The mining cluster is facing deep-seated economic and socio-economic historical, structural, and immediate challenges. Phakisa is tasked with developing collaborative interventions that will have a meaningful impact on the short- and medium-term challenges faced by the mining cluster, and putting in place institutional mechanisms that will entrench collaboration. The broad aim of the Mining Phakisa is to galvanize growth, transformation, investment, and employment creation along the entire mining value chain, in relevant input sectors and in mining-related communities.

Stakeholders from the mining industry, organized labour, government departments, and research organizations agreed to endeavour to '*extend the life of Platinum and Gold mines in South Africa beyond 2025 and establish global leadership in narrow-reef, hard rock mining systems*'. Furthermore, '*this is enabled through partnerships in research and development, skills and competitive local manufacturing capability that will focus on the current and future mining operations through next generation mining systems. To achieve this, a just transition must be at the core*' (South African Government News Agency. 2016).

Many leaders of mining companies have spoken of the need to modernize mining in the country. However, modernization is often confused with mechanization. Modernization may involve mechanization, but it is more than just mechanization.

Modernization of mines through mechanization and automation, and ultimately fully autonomous operations, is the envisaged path that will bring change to processes, technologies, skill-sets, and social and environmental impacts associated with current mining practices. This was echoed by

Figure 1—Challenges facing the South Africa mining industry

Improving mining...
a systems approach is needed...
Modernisation is the first step...

Anglo Platinum CEO, Chris Griffith '*Modernisation is not just about innovation, new technologies, mechanisation and automated processes. We will only attain our vision of a modern mine if we work in partnership with our employees, government, unions and NGOs*' (Griffiths, 2015).

A vibrant, strongly capacitated, and adequately resourced local mining R&D community is needed to ensure that the solutions are designed for South African conditions from a socio-economic as well as technical perspective, due to the nature of South African mining conditions and the complexity of addressing the social imbalances of the past.

A strategy for mining

In November 2014, under the leadership of the Deputy Minister of Mineral Resources, Mr Godfrey Oliphant, stakeholders discussed R&D within the mining sector. During the discussions, the CSIR was tasked with developing a consolidated mining strategy utilizing the draft documents of the Department of Science and Technology (DST) (Craven *et al.*, 2014) and the Department of Mineral Resources (DMR) (Vogt and McGill, 2014). The resulting document was entitled the *South African Mining Extraction Research, Development and Innovation (SAMERDI) Strategy*.

At the core of the SAMERDI strategy, aligned to the visions, are the following objectives:

i. To improve the competitiveness of the sector and create new opportunities for South African based companies to operate in areas along the entire value chain from the innovations stemming from the RD&I areas
ii. To reposition South Africa as the world leader in mining RD&I by building human resources skills, capabilities, and capacities across industry, academia, and the science councils to ensure its sustainability.

The solution design

SAMERDI served as an input document into the Mining Phakisa that was held in November 2015. Although the Mining Phakisa outcomes have not been formally announced by the Office of the Presidency, there have been significant developments during the process.

- Firstly, stakeholders agreed that the SAMERDI strategy would be the consolidated strategy for mining extraction after consolidation with the inputs of the Chamber of Mines
- Secondly, there is a collective agreement by stakeholders in that there is a need to strengthen mining extraction research capabilities and capacity in South Africa as the sector moves towards modernization. Stakeholders agreed that consolidation of the various resources and research offerings in one common space is necessary
- Finally, there was agreement on the need to develop systems that will migrate current mining from conventional, labour-intensive, highly diluted methods to the utilization of mechanized drilling and blasting equipment, and ultimately to a continuous mining method that is independent of the use of explosives.

To this end, two focus areas were developed.

1. *Mining R&D*—An intensively collaborative research and development model with a holistic systems approach, that is substantially funded and which will focus on core technologies, patents, and commercialization opportunities and ultimately position South Africa as a global leader in developing solutions for narrow-reef hard-rock mining
2. *Mining manufacturing equipment cluster*— A mining manufacturing equipment cluster that is embedded with other existing clusters and initiatives, which will ensure that development requirements are translated into coherent R&D programmes, enabling local partnerships to develop and manufacture equipment for next-generation mining systems.

VISION:

To maximise the returns of the South Africa's mineral wealth through collaborative, sustainable RESEARCH, DEVELOPMENT & INNOVATION of mining technologies in a socially, environmentally & financially acceptable manner that is rooted in the wellbeing of local communities and the national economy.

Figure 2—Vision of SAMERDI strategy

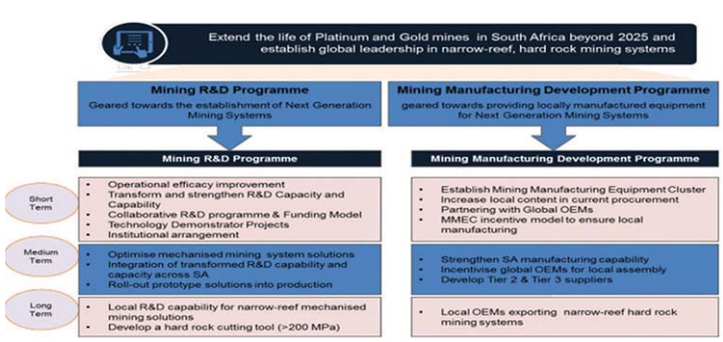


Figure 3—Advancing the mining cluster

Collaborative R&D approach – '*all hands on deck*'

The South African mining R&D environment is very fragmented compared to other countries. South Africa does not currently have any single, central entity to guide and coordinate innovation in the mining industry. Instead, there are number of mining-related collaborative initiatives such as:

a) Centre for Sustainable Mining Industries (CSMI) – University of the Witwatersrand
b) Centre for Mechanised Mining Systems (CMMS) – University of the Witwatersrand
c) Wits Mining Institute (WRI) – University of the Witwatersrand
d) Mining Resilience Research Centre (MRRC) – University of Pretoria
e) Centre of Excellence on Occupational Health and Safety (CoE) – Mine Health and Safety Council
f) Mining Research Impact Area (RIA) – CSIR.

Efforts to spread financial support across these various institutions in South Africa have led to a situation where there is insufficient critical mass in any one field, resulting in research capability and capacity, and thus offerings, deteriorating. There is a dire need for a vibrant, strongly capacitated, and adequately resourced local mining RD&I community to ensure that the solutions for the South African mining industry address local socio-economic as well as technical challenges as identified in the SAMERDI strategy. Until now, research providers have had to compete for limited funding. The end result of this process saw the erosion of research capability and capacity, which led to a loss of confidence in the ability to develop new solutions and further reductions in R&D investments due to lack of capability. The SAMERDI strategy aims to counter the erosion of research capability by striving for true collaboration as follows:

1. To improve the competitiveness of the sector and create new opportunities for South African based companies to operate in areas along the entire value chain from the innovations stemming from the RD&I areas
2. To reposition South Africa as the world leader in mining RD&I by building human resources skills, capabilities, and capacities across industry, academia and the science councils, ensuring its sustainability.

Enhancing the competiveness of mining sector

Since the completion of the Mining Phakisa in November 2015, significant work has been done under the leadership of the Department of Science and Technology (DST) through the CSIR and the Chamber of Mines in developing programmes of work for the implementation of the SAMERDI strategy, *viz.*:

(i) *Modernization of current mining operations*—The focus is on increasing the efficiency of extraction, improving occupational health and safety, and reducing costs
(ii) *Mechanized drill and blasting*—The development fully mechanized mining systems that will allow remote drilling and blasting in narrow-reef hard rock mines (in particular the gold and platinum mines)
(iii) *24/7 non-explosive rockbreaking*—The development of complete mining systems for extraction that are entirely independent of the use of explosives.
(iv) *Advanced orebody knowledge*—Mechanization and modernization of mining requires better knowledge of the orebody ahead of the mining face. This project aims to 'make the rock transparent', so that instead of mining blind, an accurate 3D real-time model can be used for safety and planning
(v) *Real-time information management systems*—The inability to accurately monitor issues in real time poses a significant challenge across the entire mining process, even more so when using mechanized mining methods. Production-related issues have a direct impact on the efficiency of the mines. Using real-time information for monitoring and control allows proactive intervention that can correct deviations and unsafe conditions
(vi) *Human factors*—Modernization via automation and mechanization of South African mining processes will have significant implications for the number of people employed in the industry, as well as the required skills levels. This will also require attention as regards change management issues. The requirements for the upstream and downstream processes associated with mechanization will have to be understood.

For each of the above areas, research questions were developed in the areas identified by members of the Chamber of Mines Innovation Task Team (Table I).

Rebuilding and repositioning South Africa as the world leader in mining RD&I

The rebuilding of mining R&D capability requires a two-pronged approach:

1. *The establishment of a forum or community of researchers*—The hub-and-spoke model was adopted since it provides the opportunity to leverage the existing capabilities of various research organizations while utilizing the administrative strengths of the host organization. A central core management team will serve as the 'hub' and coordinate the work programmes. The researchers from mining-focused research institutes (universities and science councils) will serve as research providers and thus act as the 'spokes'. Figure 4 provides an overview of the process listed below:
   (i) Researchers from the identified research organizations (universities and science councils) will form teams to collectively develop solutions
   (ii) Representatives from the various research organizations will participate in a series of workshops and meetings to form teams based on the mix of skills and capabilities
   (iii) The research teams will develop proposals and submit them to a committee made up of appropriate subject-matter experts from industry for comment
   (iv) Project proposals must indicate the level of collaboration. If a proposal is submitted that does not have all four research providers' inputs, the parties not included must provide reasons for their non-participation.

Table I

Research questions per thematic area

Thematic area	Research question
Modernizing current mines	• How can we improve the safety and health of employees as well as the profitability of existing operations?
Mechanized mining for gold and platinum	• What are the system requirements to ensure the successful application of mechanized mining (including partial, hybrid, and full mechanization)?
Non-explosive rockbreaking	• What are the critical rock engineering requirements to ensure the stability of the rock mass so that that mining can be undertaken? • What rockbreaking mechanism is most appropriate for both gold and platinum operations?
Advanced orebody knowledge	• What techniques and technologies could be used to 'see' 30 m through rock underground, and to model the orebody and surrounding rock in 3D, in multiple characteristics?
Real-time information management systems Human factors•	• What are the appropriate means and systems to enable real-time communication between the operating face of the mine and the surface? • How can resistance to change be overcome, (covering all levels in the organization, and extending across the mining value chain)? Studies to be undertaken across all mining companies and OEMs to research and consolidate these failures, so as to design practical solutions and strategies

Figure 4—Model for collaboration

Facilitation of collaboration - a Mining Precinct – 'Minds for Mines'

The nature of the mining R&D environment does not necessarily imply that the already established centres, as mentioned above, are ineffective or not essential. However it does imply that the model for collaborative working needs to be further investigated and revisited. The CSIR premises at Carlow Road in Auckland Park, Johannesburg has been revitalized to form the Mining Precinct.

The Mining Precinct will serve as a nerve centre for R&D activities that could be spread across many geographical boundaries. The Mining Precinct will provide an environment for collaboration utilizing modern means of communication. Figure 5 shows the Carlow Road facility and the various buildings on the site that provide the following facilities.

- *Building 1*—This will be the main office complex and will house (either temporarily or permanently) the researchers and the administrative functions of the Mining Precinct. The details of the occupants in the building will be dealt with later in the document
- *Building 2*—This is planned as a learning centre, which will include an auditorium for hosting conferences and symposia. This facility will attract mining stakeholders to the Mining Precinct and thus provide opportunities for further engagement with the researchers
- *Building 3*—This houses the existing testing facilities for rock engineering support systems and ventilation services. Considering the critical functions of both these disciplines in mining sector, it is envisaged that the facilities could be revamped and upgraded. Furthermore, the other buildings and workshops can be revitalized to function as mining laboratories and test facilities, such as a backfill laboratory. Other state-of-the-art laboratory facilities may be established for mine design, rock drilling and cutting, mechatronics and robotics, multiple-use sensors, geophysics, geometallurgy, and mine digitization
- *Warehousing and workshop space*—The large warehouse at the back of the premises would be ideal for building a mock stoping and development end that can be used to simulate mining conditions, allowing for the development of concepts and testing of research hypothesis before embarking on full-scale in-situ testing.

Enhancing manufacturing capability - '*South African OEMs to the rescue*'

The other significant aspect of *Advancing the Cluster* is the need to develop a manufacturing capability for mechanized mining equipment designed to operate in narrow tabular orebodies. The development of technologically advanced equipment can be described with reference to various stages, or 'technology readiness levels' (TRLs) as developed by NASA in the 1970s (Mankins, 1995). Figure 6 is conceptual diagram showing the quantum of funding required for the development of technology solutions and the relationship between state funding and industry funding. For R&D purposes, only those initiatives at TRL 6 and below will be

Figure 5—The Mining Precinct

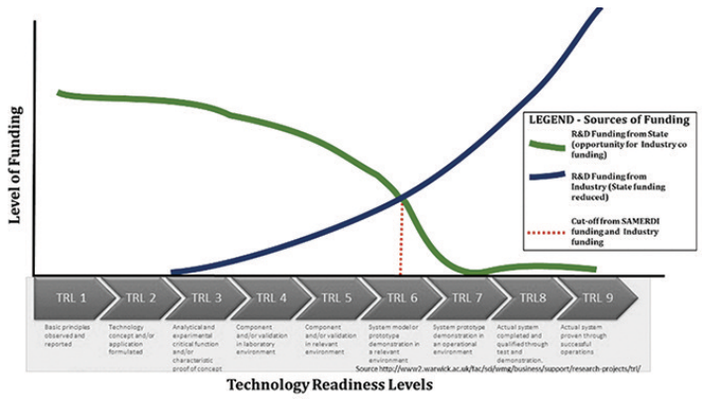


Figure 6—Quantum of funding required and sources of funding according to TRL (Alastair MacFarlane, Chamber of Mines)

considered for funding. Technologies above TRL 6 will be allocated to the mining equipment manufacturers for development into commercialization opportunities. There is opportunity for the mining industry to provide funding in both areas (with the state and with the OEMs).

South African solutions for South African problems

Current labour-intensive methods of mining tabular gold deposits are increasingly expensive, and not conducive to the objective of zero harm. It is therefore imperative to design and develop an integrated mining system that allows the mining of deeper and lower-grade reefs, cost-effectively and safely, for the sustainability of the industry.

The first and foremost aim of developing mechanized mining solutions is to improve safety, health, and the effectiveness of underground narrow-reef mining operations, which are still based on cyclical drill-and-blast operations. The objective is to remove the operators from the working faces, thus reducing their exposure to the 'danger area', while employing technology that is rugged, capable of negotiating mining dips and narrow (less than 1 m) widths, while ensuring accuracy and efficiency. In addition, the project aims to deliver increasing ore reserves and production.

With the call for modernization of South African mines and the nature of the orebodies (depth, reef width, and the dip of the reef), the solutions that need to be sought will be uniquely South African. Mechanization is not a new concept in the South African mining sector and has been used with varying degrees of success for different mineral commodities, particularly in the coal mining sector and most recently in some platinum operations. Gold mines have had the least successes of all in deploying mechanized (trackless) mining, except for two mines that exploit massive orebodies. Previous attempts to mechanize mines, gold mines in particular, have not been sustainable, either because of cost, or because of management intervention to return to conventional methods. The depth of mining, the steep angle of the orebodies, as well as the narrow width of the reef package will require significant investment in RD&I to understand the various challenges such as rock engineering requirements, the mechanical design of the equipment, and related ventilation flow into the mines. To this end, a detailed programme has been developed for the mechanization of gold and platinum mines. The approach to mechanization is seen as two-phased:

- Mechanization that is still based on a cyclic drill and blast approach
- Full mechanization that is independent of the use of explosives and employs some mechanical form of rockbreaking.

Figure 7 shows some of the mechanized mining solutions being considered by South African mining operations.

Irrespective of the manner of mining using mechanization, the solution that is sought is a full systems approach that covers all the mining activities associated with either stoping or development (inclusive of rockbreaking, support installation, as well as tramming and hauling). The development of the Mining Equipment Manufacturing Cluster was supported by the Department of Trade and industry (DTI) and led to the formation of MEMSA. It is envisaged that MEMSA will help position South African mining capital goods manufacturers as a world-class, competitive, innovative, dynamic and transformative industrial cluster, through which South African OEMs will be able to develop and export mechanized narrow-reef hard-rock mining equipment.

Just transition

While modernization and technological innovation are paramount to the future of the mining sector, the human consequences of introducing change also need to be considered. Modernization will have a direct impact on the workforce, and inevitably on the number of people employed in mining. This will have a follow-on effect on the communities dependent on mining. Therefore as well as the technical challenges, such as the development of suitable technologies, and mining and geotechnical constraints, critical aspects such as change management and leadership, re-skilling, training requirements, work planning and operation of the mine, and relationships with supporting industries need to be considered.

Conclusions

The need to access and process low-grade mineral deposits that lie at greater depths, under more complex geotechnical constraints, in areas with sparse infrastructure, coupled with increasing calls for maximum socio-economic returns from

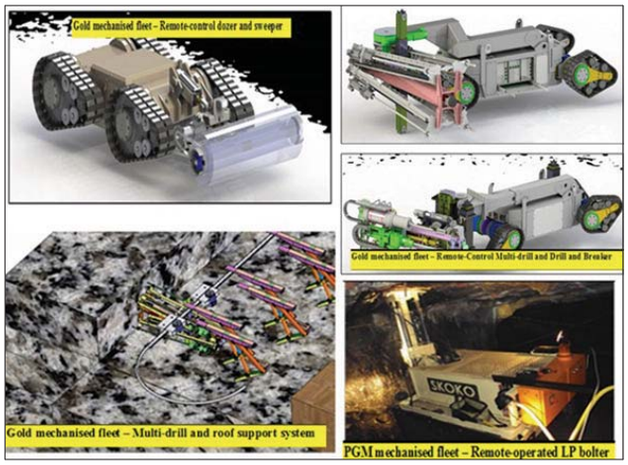


Figure 7—Some examples of potential mechanized mining solutions under consideration

mining, and increasing costs of energy, water, and labour, constitutes the *perfect storm*.

All stakeholders in the industry agree that ensuring the longevity of the South African mining sector will require a coherent, collaborative approach to the current and future challenges.

The plateauing of production, set against the constant increase in costs, the ubiquitous need to improve safety, and a depressed global market with commodity prices forecast to remain low for years to come, necessitates the revitalization of mining R&D capabilities in South Africa.

The South African Mining Extraction Research, Development & Innovation (SAMERDI) strategy provides a roadmap on working collectively towards technological solutions that will increase safety and productivity, reduce costs, and ultimately extend the life of mines.

Thus the implementation of SAMERDI strategy through collaboration and support from researchers (science councils and academia), government departments, mining houses (individually and collectively), and organized labour unions can be the solution to weather the *perfect storm* that besets the mining sector.

Acknowledgments

The CSIR is thanked for allowing the time to publish this paper. The work was done partly in conjunction with the Chamber of Mines, and the contributions of Mr Alastair MacFarlane and Mr Katlego Letsoalo are appreciated and acknowledged.

Further acknowledgments are extended to Mr Imraan Patel, the DDG of the DST; Mr Beeuwen Gerryts, Chief Director of the DST; Mr Llandley Simpson, Director of Advanced Minerals and Deputy Director; and Ms Candice Willard for their efforts and leadership in the DST programme '*Development of a South African Research Agenda for Mining And Geosciences*' that led to the Rock Innovation Programme.

The members of the DMR, and in particular those involved in the Mining Industry Growth Development and Employment Task Team (MIGDETT), are thanked for their foresight in undertaking the DMR project '*A Technology Innovation Roadmap for the South African Minerals Industry*'.

It is therefore with full gratitude that the following people are recognized for their individual and collective efforts in the drafting of the initial documents for the DST and DMR:

- Peter Craven, Mintek
- Stewart Foya, Council for Geosciences
- Marian Lydall, Mintek
- Declan Vogt, University of the Witwatersrand
- Jeannette McGill, Anglo Platinum
- Alan McKenzie, Mintek

A special acknowledgement to the Deputy Minister of Mineral Resources, Mr Godfrey Oliphant, for his leadership in driving the initial processes of collaboration and coordination towards a consolidated mining RD&I strategy.

Finally, the members from the various stakeholder groupings involved in the RD&I workshops on mining as well as the Mining Phakisa are acknowledged for their inputs and comments.

References

Baxter, R. 2016. Mining in South Africa the challenges and opportunities. http://www.chamberofmines.org.za/industry-news/publications/presentations/send/7-2015/269-mining-in-south-africa-the-challenges-and-opportunities

Craven, P., Lydall, M., Vogt, D., Foya, S., and McGill, J. 2014. A technology innovation roadmap for the South African minerals industry. Department of Mineral Resources, Pretoria.

Department of Planning, Monitoring and Evaluation. Not dated. Operations Phakisa. Office of the Presidency. http://www.operationphakisa.gov.za/Pages/Home.aspx

Griffiths, C. 2015. Modernisation – building a sustainable mining industry in South Africa. Address at the Mining Indaba. http://www.angloamerican.com/media/our-stories/modernisation-building-mining-in-sa

MacFarlane, A. 2017. Chamber of Mines. Personnel communications

Mankins, J.C. 1995. Technology readiness levels. White Paper, NASA. http://www.hq.nasa.gov/office/codeq/trl/trl.pdf

South African Government News Agency. 2016. Mining Operation Phakisa to start work in August. http://www.sanews.gov.za/south-africa/mining-operation-phakisa-start-work-august

Singh, N. 2015. South African Mining Extraction Research, Development & Innovation Strategy. Department of Science and Technology. (Unpublished)

Vogt, D. and McGill, J. 2012. Development of a South African research agenda for mining and geosciences- part C: the agenda. Department of Science and Technology, Pretoria. (Unpublished).

Zuma, J. 2016. State of the Nation Address. http://www.gov.za/speeches/president-jacob-zuma-state-nation-address-2016-11-feb-2016-0000 ◆

SAMREC
THE SOUTH AFRICAN CODE FOR THE REPORTING OF EXPLORATION RESULTS, MINERAL RESOURCES AND MINERAL RESERVES

SAMCODES
THE SOUTH AFRICAN MINERAL REPORTING CODES

SAMVAL
THE SOUTH AFRICAN CODE FOR THE REPORTING OF MINERAL ASSET VALUATIONS

THE SAMREC and SAMVAL CODES

Advanced Workshop: Can you face your peers?

15–16 August 2017

Emperors Palace, Hotel Casino Convention Resort, Johannesburg

The object of the interactive workshop is to address the more complex application of the codes through case studies. It will be assumed that the participants are fully conversant with the codes and are able to discuss their perspective on aspects of the case studies. The emphasis will be on being able to face one's peers and include both compliance and best practice aspects of the codes.

The workshop will take the form of group discussions of various case studies to facilitate discussion. The four topics selected are Precious Metals, Coal, Diamonds and Valuation covering Exploration Results, Mineral Resources, Mineral Reserves and Valuations.

Participants will be supplied with material to review prior to the workshop. They will be placed in groups to discuss and dissect the material – 2 or 3 groups will be asked to present their findings at the end of each session. Each case study is designed for a 3 hour morning or afternoon session.

The workshop is intended to allow mining industry professionals to improve their knowledge and application of the advanced aspects of the SAMREC and SAMVAL Codes.

PROGRAMME

Wednesday: 3 May 2017

07:00–08:00 Registration
08:00–08:30 Introduction to the Course

Case Study 1: Precious and Base Metals

08:30–10:00 Session 1—Precious and Base Metals
10:00–10:30 **Tea**
10:30–12:00 Session 2—Precious and Base Metals
12:00–12:45 **Lunch**

Case Study 2: Coal

12:45–14:15 Session 1—Coal
14:15–14:45 **Tea**
14:45–16:15 Session 2—Coal

Thursday: 4 May 2017

07:00–08:30 Registration

Case Study 3: Diamonds

08:30–10:00 Session 1—Diamonds
10:00–10:30 **Tea**
10:30–12:00 Session 2—Diamonds
12:00–12:45 **Lunch**

Case Study 4: Valuation

12:45–14:15 Session 1—Valuation
14:15–14:45 **Tea**
14:45–16:15 Session 2—Valuation

SAIMM
THE SOUTHERN AFRICAN INSTITUTE OF MINING AND METALLURGY

SPONSORSHIP

Sponsorship opportunities are available. Companies wishing to sponsor should contact the Conference Co-ordinator.

For further information contact:
Camielah Jardine Conference Co-ordinator
Saimm, P O Box 61127, Marshalltown 2107, Tel: +27 (0) 11 834-1273/7
E-mail: camielah@saimm.co.za, Website: http://www.saimm.co.za

GSSA
Geological Society of South Africa

http://dx.doi.org/10.17159/2411-9717/2017/v117n3a4

Adapting oil and gas drilling techniques for the mining industry with dewatering well placement technology

by A. Rowland*, M. Bester†, M. Boland‡, C. Cintolesi§, and J. Dowling**

Synopsis

Although increasing R&D spent to develop original technologies will benefit the mining industry, adaptation of appropriate existing technologies from other industries can be a more cost-effective alternative. Schlumberger Water Services, now WSP|Parsons Brinckerhoff, has undertaken a 6-year programme assessing the adaptation of oil and gas (O&G) drilling and geophysical characterization techniques to a range of mining applications, including dewatering. Conventional dewatering systems for open pit mines generally use vertical boreholes that target hydraulically productive zones within an orebody. The drilling and completion of vertical dewatering boreholes can be complicated by mine planning constraints, where optimum hydrogeological targets are not accessible from the available drilling locations. As these boreholes are often located within the operating open pit, they can interfere with the mining operation and the ability to carry out significant dewatering ahead of mining is limited. Dewatering Well Placement Technology (DWPt) is WSP|Parsons Brinckerhoff's next-generation mine dewatering solution aimed at addressing the limitations of conventional dewatering systems through placement of permanent, high-performance dewatering wells in optimum orientations beneath an open pit using large-diameter directional drilling technology commonly used in O&G. Ideally, well collars are located outside of the mine operating areas, resulting in improved compatibility between the dewatering system and mine plan. Recently drilled and constructed pilot directional dewatering wells in hard rock mining environment in the USA and Mexico have demonstrated that DWPt offers significant benefits for groundwater inflow control and value to mining operations compared to conventional open pit mining dewatering practices.

Keywords

open pit mines, dewatering, well placement, directional drilling.

Introduction

The management of groundwater inevitably becomes an integral part of open pit operations as they eventually encounter groundwater. Depending on the conditions, this management may take the form of proactive dewatering through the use of boreholes. In such situations, a robust and effective dewatering system can be vital for maintaining slope stability and safety, and enabling the minimization of stripping ratios. Furthermore, effective dewatering removes groundwater from the operating areas in the base of a pit and helps to reduce wear- and tear-related costs on mining equipment, along with reduced haulage costs owing to the reduced haulage of wet ore and waste (Dowling and Rhys-Evans, 2015). The impacts on the mining operations from poor dewatering often include:

- Wet drilling and blasting, requiring more expensive blasting agents and reduced fragmentation efficiency
- Wet working benches, which increase equipment wear and introduces additional safety risk factors
- Inundation of the pit floor and slow mine advancement
- Reduced geomechanical performance of pit slopes that, in some cases, leads to the design of more conservative slope angles resulting in higher strip ratios and deferral or loss of ore (Figure 1).

Although conditions vary widely, where wall stability and reducing groundwater inflow are the main dewatering objectives, conventional dewatering is normally implemented with a combination of: (1) in-pit vertical pumping wells to remove groundwater from the formation and lower groundwater levels within the ultimate pit shell, (2) pit perimeter wells to intercept and remove groundwater moving toward the mine, and (3) a series of sumps and surface pumps to remove standing and near surface groundwater (Dowling and Rhys-Evans, 2015) (Figure 2). This tends to be effective for ore deposits in moderate to high permeability settings where productive zones are present and within the reach of conventional drilling techniques.

Adapting innovation

The petroleum industry has faced many of the same challenges as those currently facing the mining industry, especially in terms of declining grades and a precipitous decrease in large, easily accessible discoveries (McCartney and Anderson, 2015). However, in order to address this, the O&G industry has maintained significant investment in research and development (R&D) (McCartney and

* *Piteau Associates, South Africa.*

† *Kumba Iron Ore, South Africa.*

‡ *WSP | Parsons Brinckerhoff, United Kingdom.*

§ *Anglo American, Chile.*

** *Piteau Associates, USA.*

 This paper was first presented at the New technology and innovation in the Minerals Industry Colloquium', *9–10 June 2016, Emperors Palace, Johannesburg, South Africa.*

Figure 1—Many large open pit operations are eventually impacted by the inflow of groundwater, requiring proactive dewatering systems

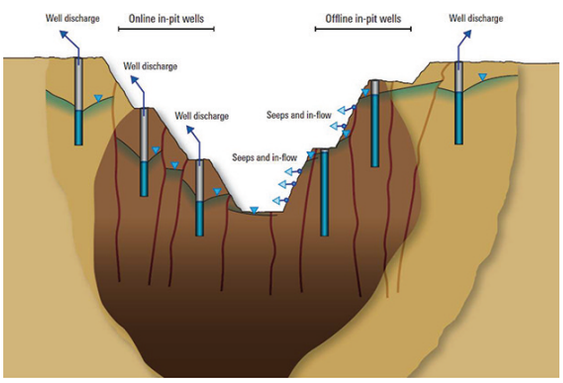


Figure 2—A conceptual layout of a conventional dewatering system in a hard rock, fractured environment. Some or all of the various aspects illustrated may be included in any given system, and often a combination is deployed based on local requirements (Dowling and Rhys-Evans, 2015)

Anderson, 2015). R&D spend as a percentage of revenue averages around 0.5% for the major oil companies, and is higher than 2% for the main service companies (McCartney and Anderson, 2015). In contrast, R&D spend in the mining industry has previously been pegged in the media at as little as one-tenth of that in the O&G industry.

Directional drilling as a whole is considered a mature technology with widespread acceptance and commonplace use in the O&G, utilities, and infrastructure industries. However, directional well placement in hard rock mining has a very limited track record and was previously untested for dewatering applications in an open pit mine. The geological and geomechanical environments, size and scale of equipment, flow and production pumping regime, and the associated well design requirements are significantly different, requiring substantial adaptation and modification. However, WSP|Parsons Brinckerhoff and Freeport-McMoRan recognized that the principal benefits of directionally drilled dewatering wells are highly applicable to open pit mine dewatering (Dowling and Rhys-Evans, 2015). Crucially, the use of directional drilling enables:

- Enhancement of hydraulic contact between multiple fractures zones and the production well or wells
- Access to permeable water-bearing zones unreachable with vertical drilling
- Positioning of the well-heads permanently outside of the planned mine operating areas.

The combined impact of the benefits listed above has been shown to result in a step-change improvement in dewatering well efficiency, performance, and overall effectiveness of the mine dewatering programme, resulting in significant cost and risk reductions. Improvements that have been demonstrated by pilot programmes in the USA and Mexico are:

- Increased well yield due to the design trajectory, interception of sub-vertical structure, and enhanced hydraulic contact.
- Improved well runtimes with the well-heads located outside of operating areas, thereby avoiding interference between dewatering infrastructure and mine operations
- High well yield and improved runtimes leading to a step-change increase in long-term volumes of groundwater produced from the dewatering programme
- Reduced number of well-head installations with associated burdens of procurement, implementation, and in-pit operation interactions.

Recognizing the limitations of conventional dewatering practices and the potential value of improved dewatering, WSP|Parsons Brinckerhoff and Freeport-McMoRan have collaborated to develop, test, and implement a new generation of high-performance mine dewatering well systems, combining mine hydrogeology and dewatering expertise with crossover technology of O&G directional well placement (Dowling and Rhys-Evans, 2015) (Figure 3). Subsequently, the success of various pilot programmes at Morenci with Freeport McMoRan led Kumba Iron Ore (KIO) to approach WSP|Parsons Brinckerhoff to conduct a technical feasibility study (TFS) for the evaluation of directional well placement to improve dewatering effectiveness at their Sishen operation in South Africa.

Adaptation of existing technology requires a deep understanding of the goals of the adaptation, as well as the technology to be adapted. For example, the directional placement of a well to a pre-planned trajectory involves complex interactive consideration of multiple factors, including the ore deposit geology, geological structure and geomechanical environment, the ranges of performance for directional tools, downhole surveying, and the ability to control and steer the well to the target (Dowling and Rhys-Evans, 2015). The trade-offs between alternative options, risk factors, the final dewatering goal, and overall value to the mine operation are integral to the matrix of planning, design, and implementation decisions. By adapting these techniques from the O&G industry, WSP|Parsons Brinckerhoff and Freeport-McMoRan developed a mine dewatering project integration matrix, which was subsequently implemented at the Freeport-McMoRan Morenci copper mine in Arizona, with the previously mentioned TFS having been done for KIO's Sishen operation.

Proof of concept

To date, two directionally placed dewatering wells have been successfully implemented at Morenci as part of the Garfield open pit mine dewatering programme. An initial well was constructed on a 'proof-of-concept' basis and commissioned in early April 2013. The well site was located on the west wall of the open pit, outside of planned mining limits. The borehole was steered underneath the centre of the planned pit on a pre-planned directional trajectory. Attaining a measured depth of approximately 700 m, the well intercepted hydrogeological targets associated with major northeast-trending geological structures and hydrogeological compartments. After completion, the well was equipped with an oilfield-style high-lift, slim-hole electrical submersible pump system designed to minimize well drilling and construction hole diameters while permitting high production pumping rates for variable head pressure conditions (Figure 4). The well initially produced between 150 m^3/h and 160 m^3/h, which

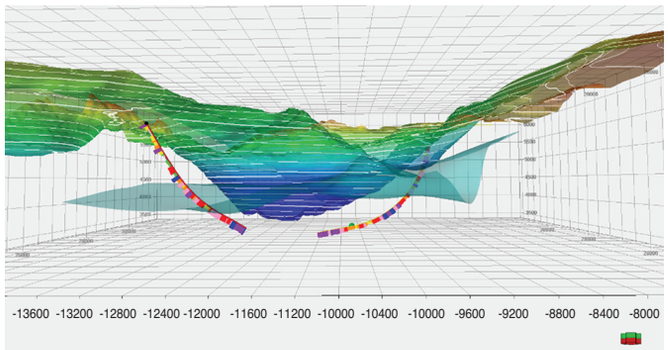


Figure 3—3D visualizations of directional well placement trajectories beneath the Garfield pit at the FreePort McMoRan Morenci Copper Mine, USA with January 2014 phreatic surface shown. The trajectory on the left (Well C) is the proof-of-concept well completed in April 2013, while the trajectory on the right (Mammoth Well) was completed in January 2015. The cross-section runs from NNW to SSE (Dowling and Rhys-Evans, 2015)

Figure 4—The collar of the first directionally placed well at Morenci, permanently plumbed and operating outside the footprint of the Garfield Pit

was at the high end of the planned production, and is five to ten times greater than the previously installed conventional, vertical in-pit wells. The well was immediately commissioned into the active dewatering programme and during the first year operated at 96% availability. Due to the combination of high production rate and high availability, it effectively produced up to two orders of magnitude more groundwater than any of the pre-existing in-pit vertical wells and exceeded the combined groundwater production from the rest of the dewatering system, comprised of six vertical production wells (Dowling and Rhys-Evans, 2015).

Following the success of the first well at Morenci and another successful well in Mexico, Freeport-McMoRan in partnership with WSP|Parsons Brinckerhoff commissioned an additional programme to construct a second well collared on the pit perimeter of the east high wall. The target for the second well was a set of northeast-trending geological structures and lower permeability compartments. With a more aggressive drill bit trajectory, design modifications were made during implementation to control risk while attaining the planned hydrogeological target. Results from early stages of operation indicate the well to be a high-performance high-value dewatering asset (Dowling and Rhys-Evans, 2015).

Since initiation of the programme at Morenci, monitoring data has shown a distinct acceleration in the rate of the groundwater level reduction in the open pit. A number of piezometers located within and around the edges of the pit have shown a 200–250 foot (60–90 m) decline in static water elevation (Dowling and Rhys-Evans, 2015).

South African example

The Sishen iron ore mine, Kumba Iron Ore's flagship operation, is currently the largest open pit iron ore mine in Africa, and one of the largest open pits mines in the world at almost 14 km in length (Kumba Iron Ore, 2016a). Total annual production at Sishen is approximately 35 Mt, with the most recent values being 5 842 kt in the first quarter of 2016 (Kumba Iron Ore, 2016a). Additionally, up to 190 Mt of waste is removed annually from the open pits (Kumba Iron Ore, 2016b). This large-scale mining operation targets a high-grade haematite orebody with grades of significantly more than 60% Fe and a sought-after lump content that commands higher prices on the global steel market (Astrup, Hammerbeck, and van den Berg, 1998).

Sishen currently consists of four operating areas that have been excavated to near or below the natural groundwater surface (Schlumberger Water Services, 2014). As a result of the relatively shallow pre-mining groundwater level, dewatering activities have been ongoing since the beginning of mining operations, with a series of vertical in-pit and perimeter pumping wells targeting productive geological formations within the mine area (Schlumberger Water Services, 2014). The groundwater regime and rate of mining require that dewatering pumping operates on a continuous basis. The abstraction rate of the overall mine dewatering system is approximately 1 800 m^3/h as of November 2015, from a total installed capacity of approximately 2 130 m^3/h (Nel and White, 2015). Of the four operating pits, the GR35 pit is currently the deepest and is mining at 950 m above sea level (masl) from an original surface elevation of approximately 1 200 masl.

Leading up to early 2014, dewatering operations at GR35 pit faced significant challenges related to lithology, structure, and operations, including:

- Typically 2 to 4 'dry' wells are drilled before a high-yielding water strike is intersected at the required depth
- The drill rigs at site could not drill beyond water strikes of 250 m^3/h. Dewatering boreholes were therefore constructed below target abstraction as a result of their limited depth of penetration
- Due to highly fractured formations, exploration boreholes are not reamed and production wells are drilled a few metres away. During production drilling the risk of missing the water-bearing structure encountered in the exploration well close by is significant due to the vertical orientation of structures
- The presence within the pit of dewatering, exploration, and production boreholes and the associated

dewatering infrastructure interferes with the active mining front and pit operations
- Interruptions to dewatering operations from mining activities resulted in significantly reduced utilization of the production boreholes and a consequent quick rebound of the groundwater system due to high connectivity and hydraulic conductivity.

In order to address the challenges raised above, WSP|Parsons Brinckerhoff was approached to conduct a pre-feasibility study (PFS), and subsequently a full technical feasibility study (TFS) on the use of directional well placement to replace or augment the existing system in GR35 pit. The PFS concluded that directional wells placed outside of the final pit shell, targeting chert (CH), banded iron formation (BIF), and sub-vertical structures would intercept fracturing that would yield significant amounts of groundwater and lead to effective dewatering of the CH, BIF, and haematite (HEM) rock mass. Based on this and considering pumping systems, drilling, and completion diameter, a yield of about 360 m^3/h from a single directionally placed well was determined to be an achievable abstraction target (Schlumberger Water Services, 2014). The collar location for the drill pad was proposed by Sishen based on the following criteria (Figure 5):

- Location outside the planned final GR35 pit shell
- Easy access for the drill rig and ancillary services
- Outside areas identified for construction of future waste dumps and therefore the pad could be used for future directional wells if required.

During the TFS, 14 different directional well configurations were assessed, with the Plan 11 and Plan 14 trajectories selected for detailed engineering design and costing. Detailed engineering design work was carried out in order to define the feasibility of drilling the Plan 11 and Plan 14 wells using engineering inputs from a number of groups within Schlumberger, including:

- Smith Bits: drill-bit selection, rate of penetration (ROP) calculations
- Drilling and Measurements: directional drilling plan, bottom hole assembly (BHA) and casing design
- MI Swaco: drilling fluid plan for hole cleaning
- Drilling Tools and Remedial: turbine and mud-motor selection
- Artificial Lift: submersible pump design.

Plan 11 provided a base case as it involved assessment of drilling of all of the main lithologies present at GR35, at a range of drilling diameters from 24 inches to 8½ inches, with a significant directional component, and completion of a long horizontal production zone within the well that would involve significant challenges and risks. Apart from assessing the technical feasibility of drilling the well, this proposed trajectory also allowed the cost implications of focusing the drilling on the dolomite units, as opposed to the shallower chert and BIF, to be assessed. On the other hand, Plan 14 (Figure 6) presented a simpler well trajectory, which would require less directional drilling to achieve its target placement beneath the GR35 pit. This was assessed to have a higher probability of success, especially considering that this would be the world's first directionally placed dewatering well drilled in iron ore formations. This plan allowed the lower risk associated with the directional drilling component to be assessed against the cost and risk implications of drilling predominantly within the harder and more fractured chert and BIF units. Eventually, the Plan 14 trajectory was selected as the preferred option

A cost-benefit analysis was carried out to compare the current approach to dewatering at Sishen, based on the use of vertical in-pit wells, and the cost associated with developing a dewatering programme based on DWPt. In addition, a number of intangible benefits associated with the DWPt approach were identified, and although the cost benefits of these were not assessed their value was to be considered in assessing the DWPt approach. These intangible benefits, which are applicable for other Sishen open pits and nearby mines (Kolomela), included:

- Improved in-pit safety environment due to reduced personnel movements in the pit related to dewatering activities, and reduced in-pit infrastructure associated with dewatering
- Simplified mine planning due to removal of the need to incorporate dewatering infrastructure and maintenance in-pit
- Improved dewatering leading to:
  - More efficient and cheaper blasting. The reduced block size resulting from more effective blasting will in turn reduce the need for crushing, grinding, and potentially drying of material and double-hauling
  - Reduced mining equipment maintenance due to lower humidity and acid rock drainage (ARD) generation at the mining front
  - Improved ore transportation efficiency due to the reduction in the volume of water carried in ore
  - Dewatering infrastructure that will remain in use after backfilling of the GR35 pit, thus supporting ongoing site-wide dewatering supporting GR80 pit.

Overall, multiple conventional vertical well dewatering scenarios were evaluated against the chosen DWP plan. The values for lost revenue were calculated on the basis that benches that could not be mined as a result of high groundwater levels and/or changes to the mine plan as a result of insufficient dewatering would be 'lost'. The value of ore contained in the various benches as per the mine plan would as a result be defined as 'lost revenue'. These were described as follows:

- *Scenario 1*: 'Most Likely' – conventional dewatering continues as planned and two benches in the base of the pit are lost
- *Scenario 2*: 'Least Likely' – a best-case scenario where conventional dewatering continues as planned and no benches are lost
- *Scenario 3*: 'Worst Case' – a worst-case scenario where conventional dewatering continues as planned but four levels in the base of the pit are lost
- *Scenario 3*: 'DWP' – the planned DWP is executed successfully and all levels in the pit are mined.

The results of the trade-off analysis indicated that although the estimated capex for continued vertical well dewatering was less than that of the proposed DWPt plan, the cost differential was offset by the estimated cost saving related to more efficient dewatering, and the resulting reduction in wet mining and water haulage. Additionally, in the event that the GR35 mine plan could not be met as a result of constraints related to the conventional dewatering system (such as evaluated in Scenarios 1 and 3), high-grade ore representing up to US$80 million in revenue could have been lost. Figure 7 shows the relative costs calculated for the various scenarios, clearly indicating that the greatest economic risk lies with potential lost revenue associated with lost benches as a result of the inadequacy of the conventional dewatering system. The lost revenue values used in the cost-benefit analysis were calculated at iron ore prices of US$50 to US$56 per ton, depending on the accessible volumes of fine and lump ore remaining for each bench in the GR35 pit according to the mine plan at the time.

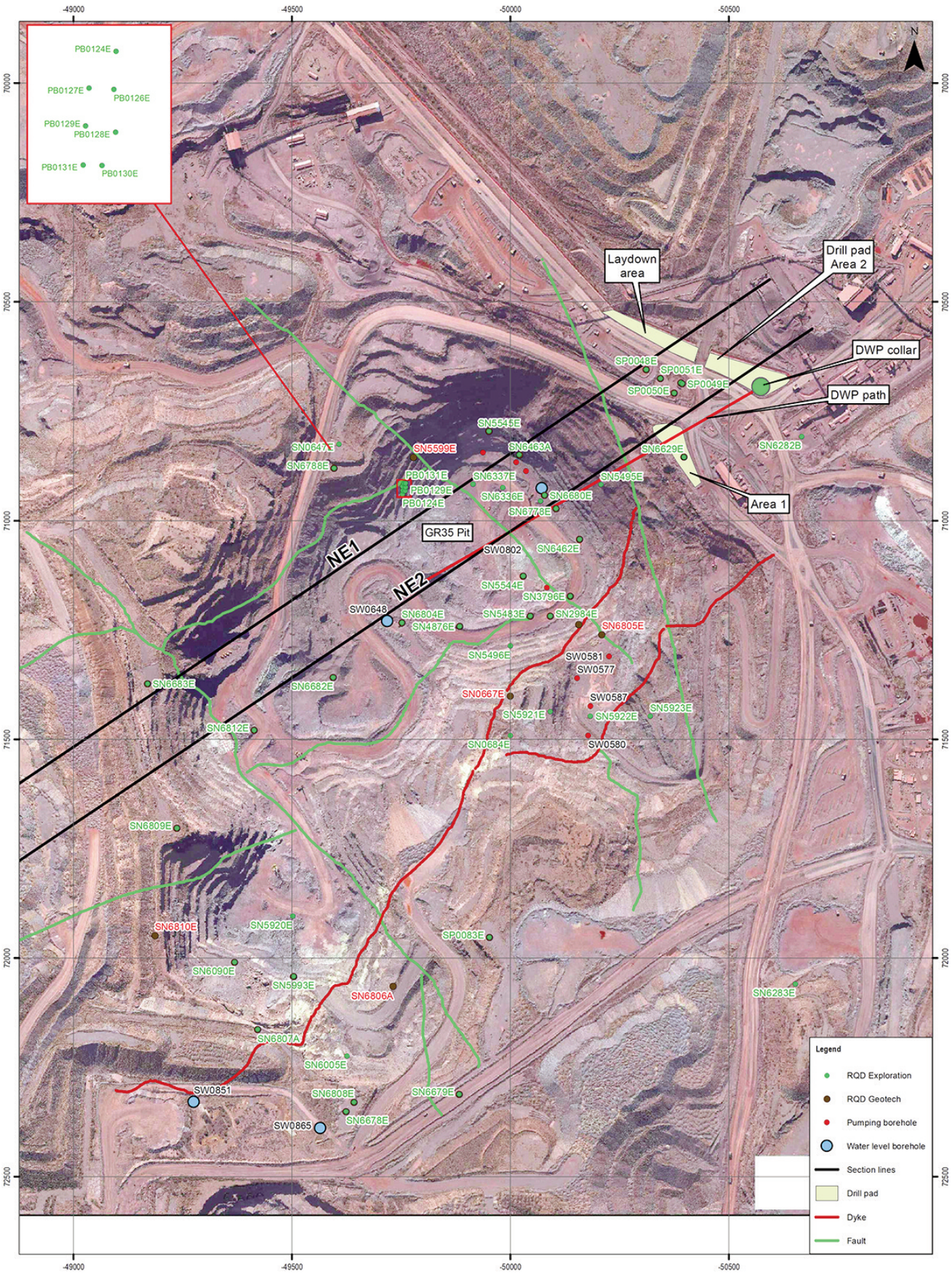


Figure 5—GR35 pit with the location of the DWP collar and drill path shown relative to critical structures and surface infrastructure

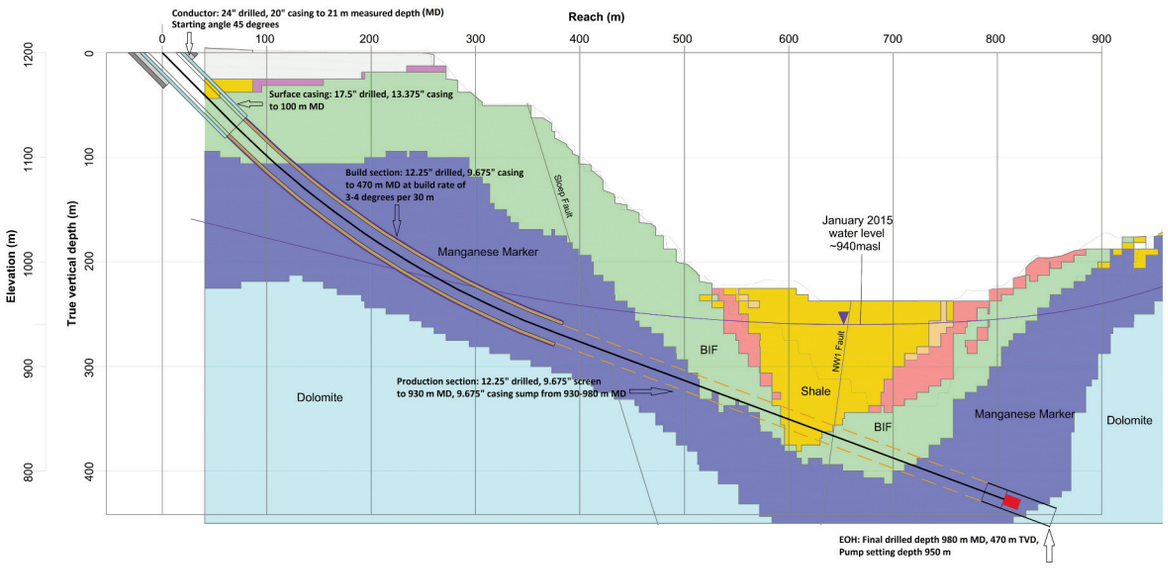


Figure 6—The final Plan 14 directional well trajectory and proposed construction

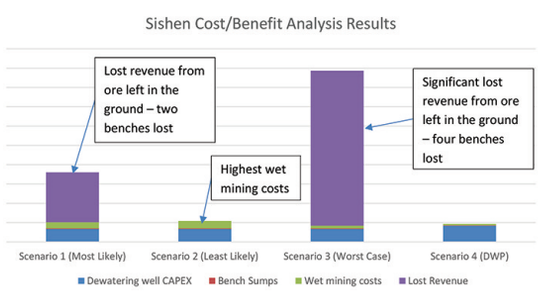


Figure 7—Chart showing the results of the cost-benefit analysis conducted for the Sishen GR35 DWP project TFS. The graph clearly shows that the costs associated with DWP are lower than all other scenarios, including the 'Least Likely', but especially relative to the 'Most Likely' and 'Worst Case' scenarios

Considering that the cost-benefit analysis was conducted using depressed iron ore prices in the same range as the current iron ore price, the trade-off analysis demonstrates the clear economic benefits of deploying DWPt for the GR35 pit at Sishen. However, the significant decline in iron ore prices and iron ore market fundamentals in 2014/15 resulted in a marked slowing of the rate of vertical advance in GR35 pit and significant changes to the mine plan. This enabled dewatering of the pit at a slower rate using the existing conventional dewatering system and resulted in DWPt being indefinitely delayed for the GR35 pit. However, since KIO remains committed to deploying the latest technologies at their Sishen and Kolomela operations in an effort to increase productivity and efficiency (Mining Review Africa, 2016), DWPt as a concept has been retained to address the potential dewatering requirements for other deep pits at Sishen and Kolomela should conditions warrant it.

Conclusion

As the mining industry is driven to exploiting ore resources that are deeper and in more inaccessible areas than ever before, systems previously thought to be robust solutions are increasingly being exposed as sub-optimal as mines get deeper and larger. The use of vertical dewatering wells is such an example, as the best hydrogeological targets cannot always be reached from the available drilling locations. Additionally, a drive to increase efficiency and productivity throughout the industry further pushes operations to reduce in-pit dewatering infrastructure and to improve the overall effectiveness of dewatering systems in the most cost-effective manner possible. As shown in this paper, the adaptation of mature directional drilling technologies has brought significant benefits to some of the largest open pit operations through the deployment of directional well placement technology. It is expected that as pits continue to get larger and deeper, and less in-pit space is available, the use of this technology will become more common as it is accepted as a cost-effective method for dewatering.

References

Astrup, J., Hammerbeck, E.C.I., and van den Berg, H. 1998. Iron. *The Mineral Resources of South Africa*. Wilson, M.G.C. and Annhaeusser, C.R. (eds.). *Handbook* 16. Council for Geoscience, Pretoria, South Africa. pp. 402-416.

Dowling, J. and Rhys-Evans, G. 2015. Oilfield directional well placement technology used for mine dewatering. *Mining Magazine*. May 2015. p. 28.

Kumba Iron Ore. 2016a. Operations. http://www.angloamericankumba.com/our-business/operations.aspx [Accessed 21 April 2016].

Kumba Iron Ore. 2016b. Kumba Iron Ore Limited production and sales report for the quarter ended 31 March 2016. http://www.angloamericankumba.com/media/press-releases/2016/21-04-2016.aspx [Accessed 21 April 2016].

McCartney, J. and Anderson, M. 2015. Mining innovation – why start from scratch. *Mining Magazine*. December 2015. p. 48.

Mining Review Africa. 2016. Kumba says technology a company game-changer. http://www.miningreview.com/news/kumba-says-technology-a-company-game-changer/ [Accessed 11 May 2016].

Nel, E. and White, T. 2015. Groundwater report, November 2015. Kumba Iron Ore Limited, Kathu, South Africa. 21 pp.

Schlumberger Water Services. 2015. Technical feasibility study for dewatering well placement technology at the GR35 Pit, Sishen Mine. *Report no*. 53810R2v11. Johannesburg, South Africa. 129 pp.

Schlumberger Water Services. 2014. Pre-feasibility assessment of the use of dewatering well placement technology at the GR35 Pit, Sishen Mine. *Report no*. 53810R1v1. Johannesburg, South Africa. 62 pp. ◆

http://dx.doi.org/10.17159/2411-9717/2017/v117n3a5

Controlled foam injection: a new and innovative non-explosive rockbreaking technology

by R.G.B. Pickering* and C. Young†

Synopsis

Controlled foam injection, or CFI, is a highly effective, novel non-explosive rockbreaking technology that with appropriate implementation and application can replace traditional drilling and blasting methods with drilling and non-explosive breaking. The CFI technology is safer, more productive, environmentally friendly, and fully developed; and it can operate with the same flexibility as more traditional small-hole drilling and blasting. To date, CFI has been regarded as an interesting rockbreaking process that could be used only in a few specialized applications. However, in extensive trials it has successfully broken every rock type encountered and in reality it could be used in place of all mining and civil engineering rockbreaking processes that utilize explosives in short and small-diameter blast-holes. The narrow-reef hard-rock mines, typical of the southern African gold, platinum, and chrome sectors are under severe pressure to mechanize and preferably operate on a 24/7 basis, and this could be achieved through the application of CFI. What is still outstanding is a suitable machine to fit a defined application, such as tunnel development or stoping.

Keywords

non-explosive rockbreaking, controlled foam injection, mining cyble, continuous operatios.

Introduction

How do we increase productivity using technology as the driver for change? From studies in mining tunnelling it is obvious that developments in drilling technology, as well as increased drilling rates, have not led to a significant increase in the rate of tunnel development. Drilling rates with mechanized drills have increased fivefold in the last 50 years, while tunnel development rates in mechanized mining operations have reduced to about one-third of those 50 years ago.

The biggest obstacle to lower cost and safer mining in narrow-reef orebodies, typical of the gold, platinum, and chrome mines found in southern Africa, is the blasting cycle, with its inherent poor utilization of the mining assets. In conventional mining operations, the total face working time is typically less than eight hours per day and less than seven days a week. In reality only 25% of the totally available time is utilized to generate revenue (Fenn, 2016). What is required is a technology that allows for continuous mining, thereby maximizing the use of the asset. By implementing some form of non-explosive mining the rate of face advance can be increased and the length of face being worked can be reduced, which leads to more concentrated mining. Concentrated mining means less service infrastructure, less ventilation, and more effective management. In soft-rock mining this has been achieved with the introduction of pick-cutting machines, such as road-headers, shearers, and continuous miners. The introduction of pick-cutting in the coal mining industry in the 1950s led the move away from blasting and by the mid-1990s fatalities, measured in deaths per million tons produced, had reduced to only 1.6% of the 1947 figure and productivity at British Coal, measured in tons per employee, had risen by more than 20 times compared to the blasting era (Pickering, 2004). Replacing blasting with cutting also reduces the blast damage to the surrounding rock and increases safety. Operating 24/7 results in much higher rates of face advance in development ends, leading to shorter lead times to production and better NPV returns Thus, we see that the move to non-explosive mining had a major impact on safety, productivity, operating costs,, and return on invested capital.

The only rock-cutting machines that have been operated successfully in hard-rock mining are raise-boring machines and, in very limited applications, tunnel-boring machines (TBMs). All these machines use discs or buttons to break the rock, primarily in compression; and to overcome the high

* *Rod Pickering & Associates, Port Alfred, Eastern Cape, Cape Town, South Africa.*

† *CFI Technologies, Steamboat Springs, Colorado.*

 This paper was first presented at the New technology and innovation in the Minerals Industry Colloquium', *9–10 June 2016, Emperors Palace, Johannesburg, South Africa.*

compressive strength of the rock requires very high forces, massive machines to supply these forces, high power, and consequently high capital cost. This is exacerbated by the high running cost of these machines due to the exceedingly high abrasivity of the rock. To make matters worse, these machines are difficult to manoeuvre, which makes following the reef very difficult, if not impossible, and furthermore they only cut circular openings.

Original equipment manufacturers (OEMs) have conducted extensive work, often funded by mining companies, to develop rock-cutting machines for hard-rock mining. Because of the inherent inefficiencies of cutting hard rock in compression, some OEMs have developed rock-cutting machines that attack the rock in an undercutting mode to break the rock in tension, as the tensile strength of these hard rocks is 5–10% of the compressive strength. Some examples are:

- The Mobile Tunnel Miner (MTM6), first developed by Wirth in the 1990s for HDRK and recently given a new lease of life by Rio Tinto when a new machine for tunnel development was commissioned (Tunnelling Journal, 2010). Within a few months of going underground in 2012 the project was cancelled. It is not known if this was due to technology challenges or a change in mine ownership (Delabio, 2015)
- The Sandvik MN220 Reef Miner, previously known as the ARM 1100, was first developed in 2001 and extensively developed over the next four years (Pickering *et al.*, 2006). It recommenced trials in 2014. Despite achieving defined key performance indicators during both trial periods it still has not been implemented as a mining machine, and the required associated mining system is currently being optimized (Janicijevic and Valicek, 2015).

To date, no-one has been able to develop a hard-rock cutting machine for everyday commercial use in the mining industry, other than for the specialized application of raise-boring circular shafts and a few specialized TBM-driven mine access projects.

The above story is interesting and has been told many times to justify the introduction of new technology. However, the recent work carried out by the Chamber of Mines sheds a new light on the possible future of the gold and platinum mines in South Africa. This work was presented at the New Technology and Innovation in Mining Colloquium organised by the SAIMM and is part of the Mining Phakisa initiative.

Figure 1 shows predicted production from the gold and platinum mines in South Africa. In each graph the block on the left is production from conventional mining and indicates the predicted closure of the gold and platinum mines by 2026; unless the mines mechanize. The central block indicates an increased lifespan if the mines are mechanized with drill-and-blast operations, and the block on the far right shows how the life of the mines could be extended until the middle of this century through the introduction of non-explosive mechanized mining operating 24/7. It has been stated that without this drive to modernize mining it will not be possible to unlock the potential to:

- Achieve zero harm and get closer to the goal of eliminating fatalities
- Mine South Africa's deep-level orebodies profitably.

Consequently, without modernization, 200 000 jobs could be lost by 2030 (Macfarlane, 2016).

In today's environment of high energy costs, the energy efficiency of the rockbreaking process is also of paramount importance. The graphs in Figure 2 clearly show how the specific energy consumption per cubic metre of rock broken is directly proportional, on this log-log graph, to the fragmen-

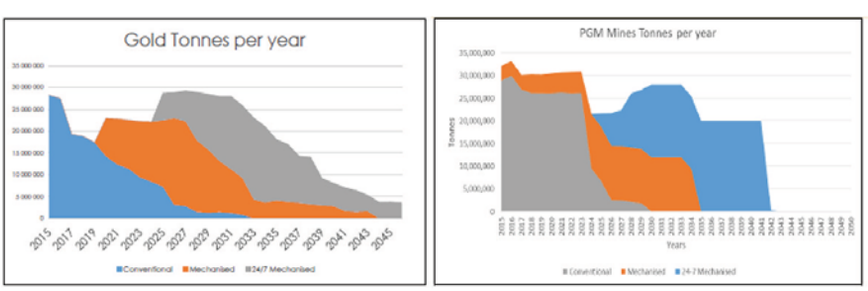


Figure 1—Predicted mine production from gold and platinum mines. (Turner, 2016)

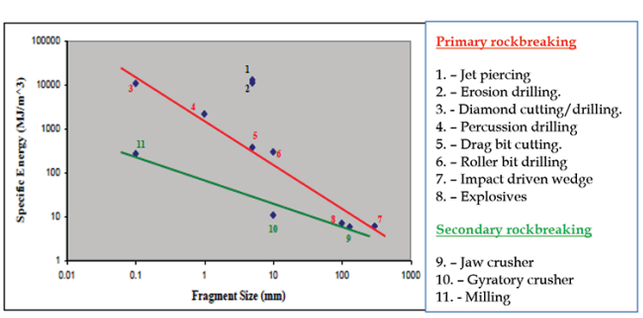


Figure 2—Specific energies of rockbreaking processes and resulting average fragmentation (after Cook and Joughin, 1970)

tation size produced by the rockbreaking process. The top line represents primary rockbreaking from a solid face, and the second line secondary rockbreaking associated with crushing and milling processes; two very different activities, best demonstrated by considering milling at 200 MJ/m³ and diamond drilling at 10 000 MJ/m³ for the same particle size. With some of the more exotic rockbreaking processes proposed by researchers, such as the use of lasers, energy consumption is often ignored. It can be seen that with blasting the average fragmentation size is 100 mm and the specific energy about 6 MJ/m³, whereas in roller-bit drilling the average fragmentation size is 10 mm and the specific energy consumption 300 MJ/m³. With explosives, this energy comes from a chemical process, but in other rockbreaking processes this energy arrives at the face in the form of electrical energy. A typical example is the Wirth MTM6, which has an installed power of 1870 kW and was designed to drive a 5 m × 5 m tunnel at a rate greater than 12 m/d.

What is really required for a continuous mining system is to eliminate the explosives, which generate fumes, dust, and violently break the rock; and instead use a simple flexible technology that relies on drilling a hole and breaking the rock in tension and that can be incorporated into a mining system that continuously removes the broken rock and installs appropriate support. Preferably, all this should be delivered with reasonable energy consumption.

Controlled foam injection

Controlled foam injection (CFI) is based on extensive research that has been funded by governments, mining companies, and OEMs. CFI breaks the rock in tension by pressurizing the bottom of a drilled hole with foam. Typical operating pressures of the foam are less than 50 MPa, as it exploits the tensile strength of rock, which is much lower than the compressive strength. The successful features of CFI are the development of a cheap and easily installed seal at the bottom of the hole and the use of foam as the pressurizing fluid. The foam is chemically inert and environmentally safe; with one exception, all the components of the foam are used in commercial food products. If water alone were used then, after the rock fractures and the pressurized volume increases, the water pressure would drop dramatically, as water at these pressures is substantially incompressible. The rock would be fractured but not liberated and removed from the breaking zone. If gas alone was the driving force, then the stored energy in the gas would propagate the fractures and violently displace the broken rock. CFI has the ability to pressurize a controlled fracture (or system of fractures) in such a manner that the pressures required to adequately propagate the fractures (without over-pressurizing them) can be maintained and it is possible to break the rock, propagate the fractures, and liberate the rock in a controlled fashion.

Foam, which is a two-phase mixture of liquid and gas, can be made to have a viscosity several orders of magnitude higher than those of gas or water, thus foam enters into a developing fracture system much more slowly than gas or water. With a much slower penetration of the fracture system, the pressures required to initiate, extend, and develop the desired fractures are much lower than if a gas is used. The use of water alone is not sufficient because this relatively incompressible liquid rapidly loses pressure as the fracture volume increases with fracture growth. The fracturing process will usually proceed so rapidly that the needed fluid pressure in a water-based system cannot be maintained by injecting additional liquid down the injection tube or barrel. In contrast, foam can maintain the pressures for efficient fracturing, due to the expansion of the gaseous phase of the fluid. Foam thus has the ability to provide the pressures for efficient controlled fracturing without requiring the excessively high pressures associated with explosives, propellants, water cannons, or electrical discharge.

Foam suitable for fracturing hard competent materials by penetrating foam injection may be made from any combination of liquid and gas; the most obvious liquid and gas to use are water and air. The surface tension properties of water alone are such that water/air foam would rapidly separate into its two components. This separation may be slowed or almost eliminated by using any of numerous commercially available surfactant materials, such as conventional soaps and detergents, or specific surfactant compounds such as lauryl sulphate (sodium dodecyl sulphate). The stability and viscosity of foam may be increased by adding a gel such as guar gum or hydroxyl-propyl guar. By varying the ratios of water, air, surfactant, and gel, foams with a very broad range of viscosities and stored energies can be manufactured. The foam may be generated externally to the actual controlled fracturing device in a conventional high-pressure reservoir using a variety of mixing and blending means.

The general features of a CFI device for rock excavation or concrete demolition are illustrated in Figure 3. The foam injection tube or barrel is inserted into a pre-drilled hole. The successful sealing of this tube into the hole, as indicated in Figure 3, is needed for the proper operation of the CFI process; this seal differentiates CFI from all the other propellant or water pressurization devices.

Once the device reservoir is charged with foam at the desired pressure, the foam is released into the pre-drilled hole by means of a rapid-acting reverse-firing poppet valve. A reverse-acting poppet (RAP) valve, as indicated in Figure 3 and illustrated in more detail in Figure 4, is attractive for controlling high-pressure foam injection because the valve has only one moving part, the poppet, which will open very rapidly when the pressure is dropped in the RAP control tube

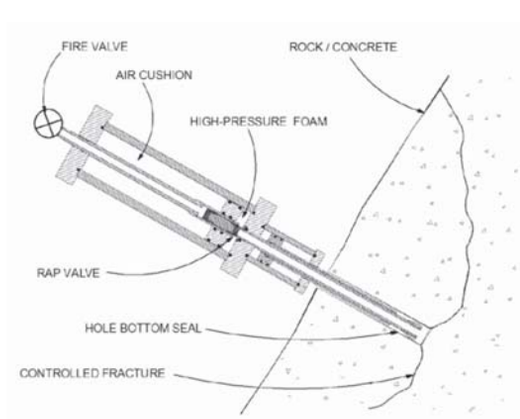


Figure 3—Basic hardware and geometry for controlled foam injection (CFI) fracture of rock

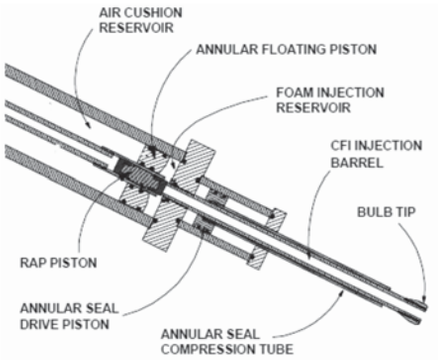


Figure 4—Details of reverse-acting poppet (RAP) valve and of annular floating piston in CFI breaker

behind the poppet. As soon as the poppet moves, the reservoir foam pressure will act on the full sealing face of the poppet, causing it to retract or open quite rapidly. The high-pressure foam will then be rapidly delivered to the bottom of the hole and effect a controlled fracturing of the rock. Rapid opening is important so that the bottom of the pre-drilled hole may be brought to a high enough pressure rapidly enough to induce the desired combination of hole-bottom fracturing and radial fracturing needed to achieve complete fragmentation.

It is desirable to avoid injecting more foam than is needed to achieve the required fracturing. Excess foam injection represents a waste of energy and would result in some increase in the (albeit low) air blast and fly-rock associated with CFI fracturing. An effective means of controlling the amount of foam injected is by incorporating an annular floating piston into the foam reservoir as shown in Figure 4. This piston separates the rear portion of the reservoir, which need only contain a high-pressure air cushion, from the forward portion which contains the foam. The accumulator effect of this piston allows for the displacement and injection of a desired foam charge into the barrel and hole bottom without a significant drop in the accumulator pressure. This feature further increases the energy efficiency of the CFI rockbreaking process.

The sealing method for CFI fracture utilizes an injection tube with a bulb enlargement at its tip and an annular hydraulic piston acting around the smaller barrel of the tube, as illustrated in Figures 3 and 4. Sealing is effected by crushing an annulus of deformable material between the bulb tip and the annular piston. The crushing is effected with the annular sleeve on the CFI barrel driven by an annular piston. The crushing of this material along the axis of the hole causes it to expand radially and seal against the hole wall near the bottom of the hole. Application of high-pressure foam will cause the barrel and bulb tip to retract and further jam the material against the hole wall. With the proper selection of bulb tip angle and deformable material, the recoil will further jam the material against the hole wall and maintain a very effective seal. Any deformable material may be used, but granular materials such as sand, fine gravel, or a cementitious mix have been found to work well. Tests to date with a variety of cementitious materials have given excellent sealing, with negligible gas/foam leakage occurring around the barrel when breaking hard granite at pressures as high as 83 MPa.

An example of the seal effectiveness is shown in Figure 5. The air reservoir trace is taken from behind the annular floating piston (see Figure 4), and the other trace is from the foam reservoir. The records in Figure 5 show the initial foam reservoir pressure of nearly 55 MPa (8 000 psi) prior to opening of the valve and the rapid drop of both pressures to 48 MPa (7 000 psi) as the hole is charged. As indicated in Figure 5, the 48 MPa foam pressure was not able to immediately fracture the rock and there was a 7.4 second delay before the rock fractured, with the seal holding firmly at this pressure. Once fracture was initiated the fracturing process was completed quite rapidly, with nearly 0.5 t of rock being excavated in a fraction of a second. Once fracture was initiated the foam pressure rapidly dropped to zero and the air-cushion pressure dropped 45 MPa.

There are other significant benefits derived from the unique viscous properties of foams. The viscosity of foam depends strongly upon foam quality, defined as the volume fraction of gas. Foams of gas volume fraction less than 50% typically have viscosities only slightly higher than that of the liquid phase. As the gas volume fraction increases from 50% up to about 90%, foam viscosity increases markedly and can be more than an order of magnitude higher than that of the liquid phase. As the foam gas volume fraction increases above 95%, the foam breaks down into a mist and the viscosity drops rapidly to approach that of the gas phase. In a CFI fracturing operation the foam might be generated initially with a gas volume fraction below 50%, albeit at very high pressure. As the foam expands into the developing fracture system, the gas volume fraction will increase with a concordant increase in viscosity until the foam has expanded to a 95% or greater quality. This variation of effective viscosity with expansion serves to improve the efficiency of the CFI process in two ways. While the higher pressure foam is being generated, delivered to the injection device, and injected via the barrel into the hole, viscosity will be low, as desired. Once the rock or concrete begins to fracture, the foam expands and viscosity increases, preventing the premature escape of the pressurizing medium before breakage is complete. Once breakage is complete the foam expands further, and as a gas volume fraction over 95% is realized, the viscosity drops, allowing the foam (now a gas mist) to escape more rapidly and thus reducing the time during which high-pressure foam can accelerate fragments of

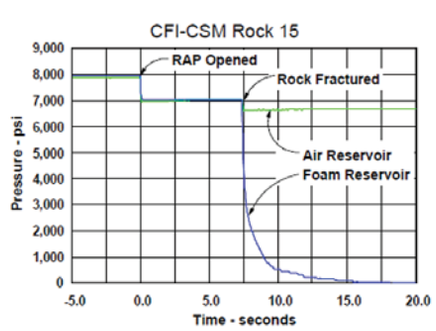


Figure 5—Pressure records from a CFI rockbreaking test showing delayed rock fracturing and seal effectiveness

the broken material. By appropriate design of the foam, a sequence of viscous behaviours optimally tailored to the foam injection/material-breakage process can be achieved. This modification of foam properties is a simple process that can be carried out by the operator.

The rockbreaking characteristic of CFI dictate that the initial fracture is propagated from the bottom of the drilled hole at an angle of about 45° before curving around to the surface, as shown in Figure 2. This initial fracture breaks a slab of rock that is roughly three times greater in diameter than the depth of the hole. The structure of the rock dictates the final fragmentation. Further than the initial penetration of the fracture below the drilled hole there is no additional damage to the sidewall or hanging of the excavation. In blasting, the shock wave generated by the blast can cause extensive damage to the surrounding rock, and work by Pusch and Stanfors (1992) has shown that in a typical mechanized mining operation where the perimeter holes are drilled with a 43 mm bit and charged with pneumatically loaded ANFO, the depth of induced fracturing will be between 2.5 m and 3 m (Figure 6).

During fracturing by blasting, excessive dust is released from the zone of crushed rock around the blast-hole. Even when propellants are used to break the rock, and there is no zone of crushed rock, dust is generated during the rock fracturing process. No measurements of dust generated during the rockbreaking process using CFI have been made; however, after the break has been completed, the surface of the exposed rock of the primary fracture is covered in a smear of foam, and it is considered that this effectively reduces the amount of dust generated. Subsequent fracturing will generate dust from the fracture planes. When CFI is incorporated into a mining machine it may be necessary to introduce further dust alleviation methods such as flooding the breaking zone with a water mist, as in coal cutting.

Field tests with the 51 mm diameter device were carried out at the Colorado School of Mines (CSM) test mine located in Idaho Springs, Colorado. The rocks in this mine range from highly fractured gneiss to massive and competent schist and gneiss. Tests in the mine environment also allowed for the effects of developing face geometry and successive breakage interactions to be observed. For example, the influence of rock anisotropy and jointing upon breakage was evaluated. Similarly, the sensitivity of breakage to foam properties as controlled by surfactant, additives, and gas concentration was studied.

The tests conducted at the CSM experimental mine provided confirmation on scaling of the CFI process. Existing data on rock breakage by a variety of methods indicates that the efficiency of breakage improves with the scale of breakage. Thus, the energy required to excavate a unit volume of rock decreases with increasing scale of breakage. This effect is primarily due to the interaction of larger and weaker rock defects (fractures, joints, parting planes, etc.) with the process at larger scales. For propellant, gas-driven fracture the scale enhancement is approximately 50% for each doubling of the linear scale. That is, a doubling of the hole diameter and an equivalent increase in the device dimensions should yield eight times the material, but in reality it results in the breakage or removal of twelve times as much material. The data for CFI breakage in the CSM test mine, along with data for penetrating cone fracture (PCF) breakage in the same mine, is shown in Figure 7. Both PCF and CFI breakage follow a nonlinear scaling. This nonlinear scaling would be manifested both in a reduction in the pressure required to break rock, and in an increased breakage efficiency at larger scales.

During the course of efforts to evaluate the ability of the CFI method to break rock in the CSM mine, considerable experimental data on the process was obtained. This data served to both confirm the proper functioning of the prototype CFI equipment and to provide guidance for the measurements that might be utilized for process control in more commercial applications of the method. It was determined that control of the CFI process, including control of foam quality, could be achieved with simple measurements of gas pressure in the air-cushion reservoir, and foam pressures in the foam generator and subsequently the foam reservoir. Other aspects of a commercial CFI process, such as hole drilling, indexing for CFI injector placement, and hole-seal crushing, could all be monitored and controlled by existing hardware and techniques utilized in commercial automated machinery, including remote radio control.

The big question that still needs to be answered is what the mining rate will be in a specific application. To date it is estimated that the mining rate from one CFI breaking tool, in solid competent rock, will be between 3 m³/h and 6 m³/h.

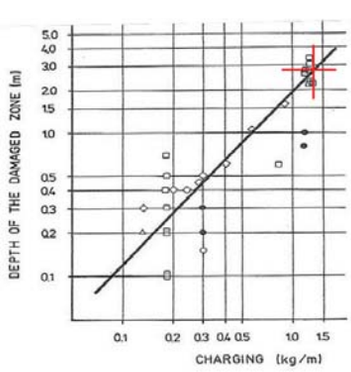


Figure 6—Effect of explosive charge weight on depth of damage zone during blasting (Pusch and Stanfors, 1992)

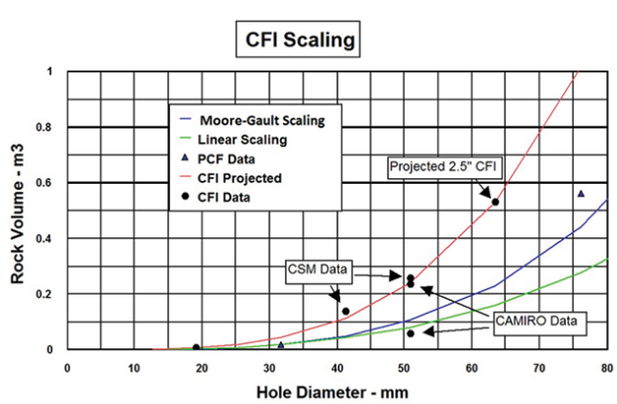


Figure 7—Graph showing nonlinear scaling of CFI and PCF rock breakage

This is based on past experience of the time to complete a cycle and experience in breaking from smaller drill-holes. The science based on Moore-Gault scaling and experience with PCF and CFI production suggests that with a 63 mm diameter drill-hole, CFI will break 0.5 m^3 per event. To put this in perspective, with a typical 600 mm staggered drilling pattern used in gold mining stopes, with a stope width of 1.1 m and an advance of 0.9 m, each drill-hole produces 0.3 m^3 of rock from the blast. Drilling intensity in platinum mines is higher due to the tougher rock conditions, and consequently production per drill-hole is even lower.

CFI has been extensively tested in a variety of different rock types and thus far all of the rock types have been successfully broken. The component parts required to compress the air and water and to generate and store the foam have been engineered and proven. Consequently, it is argued that the CFI technology is already developed – what is still lacking is a mining machine to accurately demonstrate the mining rate and operating cost in a specific application. The advantages of an integrated CFI mining machine will be a safer mining environment (as was demonstrated when coal mining moved from blasting to non-explosive excavation), low energy consumption, a relatively low capital cost combined with a low operating cost, and it will enable the mine to operate 24/7.

The exciting thing about CFI is that it can find application wherever small-diameter and short blast-holes have been used by the mining and construction industries. It can replace explosives in these applications and the mining and construction industries can have a genuine hard-rock, non-explosive rockbreaking tool. In southern Africa, a different machine could be designed and manufactured using CFI as the rockbreaking technology, and applied for tunnel development and stoping in the entire narrow-reef, hard-rock mining operations typical of the gold, platinum, and chrome mining sectors.

The following are some of the obvious applications for a tunnel development type of machine:

- In all mining, stoping, and development of narrow-reef tabular orebodies currently mined using a room-and-pillar layout and by mechanized trackless equipment. This is because in room and pillar mining the basic mining operation is tunnel development
- Globally, in all tunnel development for existing and new mines, whether they are mined conventionally or as mechanized operations.

With time, other specialized machines could be developed for:

- Narrow-reef stoping at flat and steep dips
- Narrow-vein mining practiced all over the world for a variety of minerals
- Breaking through dykes and rock too hard to mine with continuous miners in mechanized coal mining
- Secondary breaking in all cave and longhole open stoping operations.

Other applications could include the following:

- Many civil rockbreaking activities take place in built-up and congested environments. CFI could solve problems associated with noise, vibration, and fly-rock
- TBMs have found extensive application in civil rock excavation, but in hard rock it is often necessary to excavate rock not cut by the TBM, such as interconnections between two drives or the corners of a round TBM excavation. CFI could be used as it is a gentle rockbreaking process and the lack of violent fly-rock would minimize damage to other installed infrastructure in the already developed and equipped TBM drive.

Proposed project

How do we go about implementing this novel and developed rockbreaking technology in mining machines? CFI has been installed on eight different platforms; in all cases the primary concern was to develop and demonstrate the CFI technology. Machines have been tracked and wheel-mounted; the common denominator of these vehicles is that they have all used low amounts of power, with a rating of less than 60 kW. The disadvantage of these demonstration units is that none have been built for a specific underground application that takes advantage of the way CFI breaks the rock. The immediate requirement is to clearly demonstrate the rockbreaking rate and operating cost by building a rockbreaking machine based on CFI, which will be the basis for a mining machine designed to operate in a specific application such as tunnelling or stoping. In such a machine the various CFI parameters of pressure, hole size, and foam composition would be varied to determine the optimum rockbreaking performance in the defined application. At a later stage a mining machine that concurrently breaks the rock, removes the broken rock from the working face, and installs support would be designed and manufactured. At this stage it is considered that the application that should be focused on is tunnel development.

The machine that comes closest to meeting this requirement was developed by Ripamonti and used to break out safety niches when the Fréjus rail tunnel was recently enlarged and refurbished. This machine is shown in Figure 8.

Conclusion

Economic and societal pressures make it essential that the safety and productivity of the narrow-reef hard-rock mining industry improves. It is suggested that this can be achieved by replacing drilling and blasting with a novel non-explosive rockbreaking technology. The alternative technology is referred to as controlled foam injection (CFI). To date, CFI has been regarded as an interesting rockbreaking process for

Figure 8—Eagle 500 with CFI developed by Ripamonti and fitted with a 36 kW hydraulic power pack

specialized operations; however, it can operate with the same flexibility and with higher productivity than more traditional small-hole drill-and-blast operations. Just as in blasting, where the explosive charge weight and composition are varied to achieve a specific performance, the CFI parameters of pressure, hole size, and foam composition can be varied to achieve different rockbreaking performance. Changes in CFI parameters are simple to make and as rock conditions in a tunnel change so CFI parameters can be adjusted. The major benefits of CFI can be summarized as follows:

- CFI is *safer* as it does virtually no damage to the rock surrounding the excavation due to the controlled manner in which the rock is fractured. The rockbreaking process is sufficiently gentle to allow workers in close proximity to mining. There have been no measurements of dust generated during a CFI event; however, on observing a rockbreaking event it can be seen that the newly fractured surface is covered in white foam. Experience with the change from blasting to non-explosive cutting demonstrated a massive improvement in safety in coal mining; CFI can have the same impact on narrow-reef hard-rock mining
- CFI is a very *productive and efficient* method of breaking rock as it generates more broken rock per metre drilled than conventional drilling and blasting. Over a ten-year period of experimentation CFI has successfully broken every rock type in which it has been tested. It is flexible in operation, with the CFI parameters easy to modify for different rock conditions. Installed machine power is low, and the specific energy of rock-breaking is very low when compared to hard-rock cutting. The installed power for the Wirth MTM6 is just under 2 MW; the power for a CFI machine would be more like 60–90 kW, as the stored energy in the foam is accumulated over a period of time. The fracture is generated from the bottom of the hole and thus maximizes the use of the drilled hole. The production of broken rock per metre drilled is higher than that in conventional drilling and blasting. A machine utilizing CFI will have to drill a hole and then break the rock, and it is envisaged that such a machine will be similar to a mechanized drill rig and consequently the capital cost of equipment should be similar to electro-hydraulic drill rigs. Additionally, because of the low power demand and low forces the equipment can be light and hence low cost in comparison to a MTM6 or TBM. Operating costs should also be similar to electro-hydraulic drill rigs, with the main costs being drilling consumables and general operation of the machine
- The CFI breaking process is *environmentally friendly* as it yields no fumes, no dust (except that released from existing rock fractures), very low noise levels, and very limited fly-rock, so people can work relatively close to the breaking process. The chemicals used in the foam are totally inert and thus environmentally safe
- The CFI technology is *well developed*, having been tried and tested for over 15 years. The hardware used to provide the high-pressure foam has proven to function reliably over long periods of service. The hardware for generating and delivering high-pressure foam to the bottom of the drilled hole has been proven in prototype rigs. The primary function of these rigs was to develop CFI and not to develop a mining machine. Finally, and very importantly, the sand seal is a very effective and very low-cost seal. This high-performance seal allows the injected foam to maintain the pressures to initiate and propagate the unique hole-bottom fracture.

CFI could be used in place of all mining and civil engineering rockbreaking processes where the rock is broken by explosives in short, small-diameter blast-holes. The narrow-reef, hard-rock mines typical of the southern African gold, platinum, and chrome sectors would then be able to operate continuously on a 24/7 basis. The next step will be to manufacture a narrow-reef mining machine for tunnel development or stoping and prove that this non-explosive rockbreaking technology can be integrated into a machine that will deliver safer and more productive mining operations.

References

Cook, N.G.W. and Joughin, N.C. 1970. Rock fragmentation by mechanical, chemical and thermal methods. *Proceedings of the Sixth International Mining Congress*, Madrid.

Delabbio. 2015. Personal communication.

Fenn, A.G. 2016. Unlocking value by reorganizing the operational value chain through modernization. *Proceedings of the New Technology and Innovation in the Minerals Industry Colloquium*. Emperors Palace, Johannesburg, 9–10 June 2016. Southern African Institute of Mining and Metallurgy, Johannesburg. pp. 35–50.

Janicijevic, D. and Valicek, P. 2015. A review of hard-rock cutting equipment technology development at Anglo American and Anglo Platinum. *Proceedings of MPES 2015, Mine Planning and Equipment Selection 2015, 'Smart Innovation in Mining'*, Sandton Convention Centre, Johannesburg, South Africa. Southern Institute of Mining and Metallurgy, Johannesburg. pp. 455–468

Macfarlane, A. 2016. Moving to next generation mining systems: a Chamber of Mines view. *Proceedings of the New Technology and Innovation in the Minerals Industry Colloquium*. Emperors Palace, Johannesburg, 9-10 June 2016. Southern African Institute of Mining and Metallurgy, Johannesburg.

Moore, H.J., Gault, D.E., and Heitowit, E.D. 1965. Change of effective target strength with increasing size of hypervelocity impact craters. *Proceedings of the Seventh Hypervelocity Impact Symposium*, Tampa, FL, February 1965. Vol. IV (Theory). Martin Co.

Pickering, R.G.B, Smit, A., and Moxham, K. 2006. Mining by cutting in narrow reefs. *Proceedings of the International Platinum Conference – 'Platinum Surges Ahead'*, Sun City, South Africa, 8-12 October 2006. *Symposium Series S45*. Southern African Institute of Mining and Metallurgy, Johannesburg. pp. 221–230.

Pickering, R.G.B. 2004. The optimization of mining method and equipment. *Proceedings of the First International Platinum Conference, 'Platinum Adding Value'*, Sun City, South Africa, 3–7 October 2004. *Symposium Series S38*. Southern African Institute of Mining and Metallurgy, Johannesburg. pp. 111–116.

Pusch, R. and Stanfors, R. 1992. The zone of disturbance around blasted tunnels at depth. *International Journal of Rock Mechanics and Mining Science and Geomechanics Abstracts*. vol. 29, no. 5. pp. 447–456.

Tunnelling Journal. 2011. www.tunnellingjournal.com Dec 2010/Jan2011

Turner, P. 2016. Value creation through modernisation of South Africa's mining industry. *Proceedings of the New Technology and Innovation in the Minerals Industry Colloquium*. Emperors Palace, Johannesburg, 9–10 June 2016. Southern African Institute of Mining and Metallurgy, Johannesburg.

Young, C. 1999. Controlled-foam injection for hard rock excavation. *Proceedings of the 37th US Rock Mechanics Symposium*, Vail, CO, 6-9 June 1999. American Rock Mechanics Association. ◆

6th Sulphur and Sulphuric Acid 2017 Conference

9 May 2017—WORKSHOP
10–11 May 2017—CONFERENCE
12 May 2017—TECHNICAL VISIT

Southern Sun Cape Sun, Cape Town

BACKGROUND

The production of SO_2 and Sulphuric acid remains a pertinent topic in the Southern African mining, minerals and metallurgical industry. Due to significant growth in acid and SO_2 production as a fatal product, as well as increased requirement for acid and SO_2 to process Copper, Cobalt and Uranium, the Sub Saharan region has seen a dramatic increase in the number of new plants. The design capacity of each of the new plants is in excess of 1000 tons per day.

In light of the current state of the industry and the global metal commodity prices the optimisation of sulphuric acid plants, new technologies and recapture and recycle of streams is even more of a priority and focus. The 2017 Sulphuric Acid Conference will create an opportunity to be exposed to industry thought leaders and peers, international suppliers, other producers and experts.

To ensure that you stay abreast of developments in the industry, The Southern African Institute of Mining and Metallurgy, invites you to participate in a conference on the production, utilization and conversion of sulphur, sulphuric acid and SO_2 abatement in metallurgical and other processes to be held in May 2017 in Cape Town.

OBJECTIVES

- Expose SAIMM members to issues relating to the generation and handling of sulphur, sulphuric acid and SO_2 abatement in the metallurgical and other industries.
- Provide opportunity to producers and consumers of sulphur and sulphuric acid and related products to be exposed to new technologies and equipment in the field.
- Enable participants to share information and experience with application of such technologies.
- Provide opportunity to role players in the industry to discuss common problems and their solutions.

WHO SHOULD ATTEND

The Conference will be of value to:

- Metallurgical and chemical engineers working in the minerals and metals processing and chemical industries
- Metallurgical/Chemical/Plant Management
- Project Managers
- Research and development personnel
- Academic personnel and students
- Technology providers and engineering firms for engineering solutions
- Equipment and system providers
- Relevant legislators
- Environmentalists
- Consultants

EXHIBITION/SPONSORSHIP

There are a number of sponsorship opportunities available. Companies wishing to sponsor or exhibit should contact the Conference Co-ordinator.

Sponsors:

AWS, BASF, Chemical Initiatives, Clark solutions, Chemetics, DuPont, MECS, Haldor Topsoe, efc, JL Goslar, Outotec, Weir Minerals Lewis Pumps

For further information contact:
Conference Co-ordinator
Camielah Jardine, SAIMM
P O Box 61127, Marshalltown 2107
Tel: (011) 834-1273/7
Fax: (011) 833-8156 or (011) 838-5923
E-mail: camielah@saimm.co.za
Website: http://www.saimm.co.za

SAIMM
THE SOUTHERN AFRICAN INSTITUTE OF MINING AND METALLURGY

Conference Announcement

http://dx.doi.org/10.17159/2411-9717/2017/v117n3a6

An investigation of failure modes and failure criteria of rock in complex stress states

by Z. Li*, J. Shi*, and A. Tang*

Synopsis

Rock in engineering and geological environments is usually in complex stress states. Based on many experimental results for rock under different loading conditions, and combined with failure modes found in previous studies, stress triaxiality is introduced to describe different stress states, and the relationship between failure modes and stress triaxiality is analysed in detail. For a given kind of rock, with decreasing stress triaxiality the failure mechanism changes from tension fracture to local shear failure and general shear failure. Two demarcation points of stress triaxiality exist: between tension fracture and local shear failure, and between local shear failure and general shear failure. The controlling parameters that dominate tension fracture, local shear failure, and general shear failure are different. Using a reasonable assumption based on the experimental results, a failure criterion corresponding to three distinct failure modes is presented. This failure criterion is defined by seven material parameters. The accuracy and applicability of the proposed failure criterion are examined using experimental data derived for sandstone in this study and published experimental data for rock salt and marble under conventional triaxial compression tests. The data used covers a wide range of stress triaxiality and various failure mechanisms. The predictions from the proposed failure criterion agree well with the experimental data.

Keywords

rock mechanics, tension fracture, shear failure, stress triaxiality, failure criterion.

Introduction

Determination of rock mass strength is important in most rock engineering analyses that are based on solid mechanics. Better understanding of rock mass strength will facilitate proper engineering design of structures. The loading condition of rock is complex, and a rock failure criterion represents the strength of rock under different loading conditions. Many failure criteria have been proposed by researchers during the past decades. In rock engineering practice, the linear Mohr-Coulomb (MC) criterion and the nonlinear Hoek-Brown (HB) criterion are widely used in conditions where $\sigma_2 = \sigma_3$, largely due to their simplicity in formulation and the large amount of experimental data available (Melkoumian, Priest, and Hunt, 2009), despite the fact that the physical sense of the former is obscure and the latter is of a purely empirical character. Both the abovementioned criteria suffer limitations arising from ignoring the effect of the intermediate principal stress (σ_2) on rock strength (Chang and Haimson, 2012; Haimson, 2006; Tiwari and Rao, 2004). Much of the experimental evidence accumulated so far strongly suggests that σ_2 has a considerable effect on rock strength (Chang and Haimson, 2012; Tanapol, Chaowarin, and Kittitep, 2013; Tarasov and Potvin, 2013). To incorporate the influence of σ_2, several general failure criteria have been proposed, among which the Drucker-Prager criterion, the modified Lade criterion, the 3D Hoek-Brown criterion, and the unified strength criterion are well-known (Xie and Chen, 2004). However, these failure criteria are not commonly employed in practice. According to several comparative studies, none of the existing 3D failure criteria has a significant advantage over others, from both mathematical and practical points of view.

A perceived shortcoming of the traditional failure criteria is that they are established only on the basis of macroscopic experiments and combined theoretical analysis, and they do not attempt to microscopically analyse the failure mechanism and failure criterion of rock under different loading conditions. In fact, the study of deformation and failure of rock on the micro-meso scale can reveal the innate character and relationship between the macroscopic response, in the form of deformation or failure, and the intrinsic microscopic mechanism (Adelinet *et al.*, 2013; Lin *et al.*, 2015; Zhou and Linn, 2013; Zhou and Linn, 2014).

* *School of Civil Engineering and Architecture, Xi'an University of Technology, Xi'an, China.*

 Paper received Jun. 2015; revised paper received Dec. 2015.

Over the past few decades, many researchers have devoted considerable effort to the study of failure and deformation of rock under different loading conditions from both the microscopic and mesoscopic points of view (Arora and Mishra, 2015; Cai and Liu, 2009; Kittitep and Decho, 2012; Loaiza *et al.*, 2012; Tang and Hudson, 2010; Wong and Baud, 2012; Xie *et al.*, 2012; Yang *et al.*, 2008; Yang, Jing, and Wang., 2012; Zhong, Liu, and Ma., 2015). When the confining pressure is zero or very low, macroscopic failure is typically associated with axial splitting. As the confining pressure is increased, the failure process changes to a macroscopic shear failure. Loaiza (2012) also observed two deformation modes of basaltic rock under triaxial loading. At low confining pressure, shear localization occurs. At high confining pressure, shear-enhanced compaction appears. Amann, Kaiser, and Button (2012) suggested that brittle failure processes tend to be suppressed when the confining pressure is increased beyond a threshold. Xie (2004) outlined the failure modes of rock under several common loading conditions including splitting under uniaxial compression, shear failure under conventional triaxial compression with medium confining pressure, and plastic failure under conventional triaxial compression with high confining pressure. Several laboratory studies on artificial materials with inclined pre-existing flaws under uniaxial and biaxial compressive loading conditions have been utilized to investigate the influence of σ_3 on fracture propagation and the resulting failure mode. The results suggest that confinement suppresses the growth of propagating tensile microcracks (Salvador, Rafael, and Alexandra, 2013; Yang, Jing, and Wang, 2012). Szwedzicki (2007) asserted that the variations in uniaxial compressive strength values of samples from the same lithology depend on the failure mode. Within the same failure mode, the variations may be relatively small. When various failure modes take place in similar samples, variations can often be very large. This means that the failure mode affects the resultant strength of the sample. These experimental studies collectively show that the stress state has a strong influence on the failure processes and the resulting failure mode, and the way in which the sample fails affects the obtained strength of rock samples. This should be considered in the development of a failure criterion. The strength value is a function of the failure mode. An ideal failure criterion should be able to predict not only the stress state at failure but also the failure mode. The dominant factors for different failure modes also vary, so it is difficult to correctly predict different kinds of failure using a single failure criterion. However, the relationship between the failure mode and strength parameters has not been investigated, and a failure criterion that can incorporate the effect of different failure modes is rarely applied. The effect of the stress state on the rock failure and deformability has also been inadequately investigated.

It has been pointed out that a common deficiency in assessment of the rock strength is that it does not take into account the failure mode of rock. This could explain the large range of experimental results obtained on rock samples. In the present study, an attempt will be made to establish a new failure criterion based on three different failure modes proposed in previous research. Here, the stress triaxiality is selected to measure different failure modes for a given type of rock. The factors that influence the occurrence of different failure modes are investigated and the parameters that control failure are determined. Therefore, this failure criterion is widely applicable to different failure modes. Furthermore, it can predict not only the stress state at failure but also the failure mode. In the proposed criterion, seven material parameters are used. The physical meaning of these parameters and the procedures for determining them are presented. The accuracy of this proposed criterion is demonstrated by examining the experimental data on sandstone in this study and other rock types from the literature.

Different failure mechanisms of rock in various stress states

Stress states in terms of stress triaxiality

Let σ_{ij} be the stress tensor and σ_1, σ_2, and σ_3 the principal stresses, and assume that $\sigma_1 \geq \sigma_2 \geq \sigma_3$. The mean stress and the equivalent (Von Mises) stress can be expressed as

$$\sigma_m = (\sigma_1 + \sigma_2 + \sigma_3)/3 \quad [1]$$

$$\sigma_e = \sqrt{(\sigma_1 - \sigma_2)^2 + (\sigma_2 - \sigma_3)^2 + (\sigma_3 - \sigma_1)^2}/\sqrt{2} \quad [2]$$

We here consider an infinitesimal volume element subjected to a three-dimensional stress state and introduce the $(\sigma_1,\sigma_2,\sigma_3)$ coordinate system. The stress state σ_{ij} can be conveniently represented as a point $P(\sigma_1, \sigma_2, \sigma_3)$ in the principal stress space, as shown in Figure 1a. *ON* is a line passing through the origin and is at equal angles with the coordinate axes. This is known as the isoclinal line, and every point along this line corresponds to a hydrostatic stress state. The planes perpendicular to *ON* are termed isoclinal planes, and the hydrostatic stress is constant on these planes. The plane perpendicular to *ON* and passing through the origin is referred to as the π plane and the hydrostatic stress is zero on this plane. Considering an arbitrary stress state at the point *P* with σ_1, σ_2, and σ_3, the stress vector *OP* can be decomposed into the components $\vec{\alpha}$ parallel to *ON* and $\vec{r}$ perpendicular to *ON*. The magnitude of $\vec{\alpha}$ and $\vec{r}$ can be obtained by:

$$a = \sqrt{3}\sigma_m \quad [3]$$

$$r = \sqrt{2}\sigma_e/\sqrt{3} \quad [4]$$

where σ_m and σ_e are the mean stress and the equivalent stress respectively.

To quantify the effect of the stress state on the failure mode of rocks, a dimensionless parameter called stress triaxiality is introduced to indicate the infinitesimal volume element (see Figure 1a) and to reveal stress states under different loading conditions. Stress triaxiality is represented R_σ by and defined as

$$R_\sigma = \sigma_m/\sigma_e \quad [5]$$

R_σ is given by:

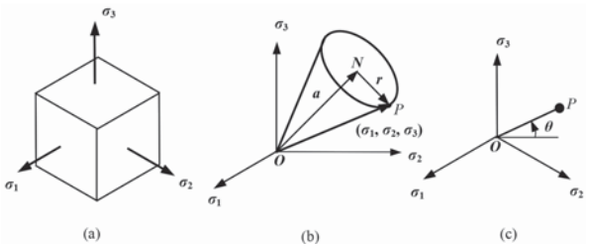


Figure 1—(a) Stress state of an infinitesimal volume element, (b) stress states with the same stress triaxiality ratio, (c) projection of a stress state on the π -plane

$$R_\sigma = \sqrt{2}a/(3r) \qquad [6]$$

It is obvious that, for a given stress triaxiality ratio R_σ, we have an infinite number of stress states, each of which corresponds to a point on the conical surface with the axis *ON* (Figure 1b). For a stress state with specific stress components, σ_1, σ_2, and σ_3, R_σ can be determined uniquely, with its value reflecting the ratio of the volumetric strain to the elastic shear strain.

The calculation results for stress triaxiality in several different stress states are given in Table I. It should be noted that the principal stresses in the stress states of triaxial tension and conventional triaxial compression in Table I are assumed.

As shown in Table I, with the stress states changing progressively from triaxial tension to uniaxial tension, uniaxial compression, and triaxial compression, the value of stress triaxiality decreases constantly. If the value of stress triaxiality is positive, the stress state is tension and the larger the value of stress triaxiality, the stronger the degree of tension. If the value of stress triaxiality is negative, the stress state is compression and the smaller the value, the stronger the degree of compression. It has been widely postulated that the failure mechanism depends upon the stress state for a given type of rock. The one-to-one relationship between each stress state and the value of its stress triaxiality is obvious and perfect, so it can reasonably reveal the influence of different stress factors on deformation and failure of rocks and the resulting failure mechanism. For a given type of rock, within a certain range, if the stress triaxiality is large, the volumetric deformation is relatively large due to the tension-dominated stress state, and thus the rock tends to fail by tensile fracture. If the stress triaxiality is small, the elastic shear strain is relatively large due to the compression-dominated stress state, and thus shear failure tends to take place. Under the stress state of triaxial equivalent compression with the value of stress triaxiality as –∞, the shear stress is too little, and shear failure is difficult to achieve. This is mainly because, as is generally believed, that it is the tension stress that causes tension fracture and the shear stress dominates in the occurrence of shear failure, but compressive stress usually cannot lead to failure, but results in welding. Moreover, once stress triaxiality is introduced to describe the stress state, there is a uniform standard for comparing and combining the experimental results in conventional triaxial compression ($\sigma_1 \geq \sigma_2 \geq \sigma_3$) with other loading conditions where all three principal stresses differ more in value.

Table I

The values of stress triaxiality in different stress states

Stress state	Principal stress			R_σ
	σ_1	σ_2	σ_3	
Triaxial inequivalent tension	σ	σ	0.5 σ	1.67
Biaxial equivalent tension	σ	σ	0	0.67
Uniaxial tension	σ	0	0	0.33
Pure shear	σ	0	$-\sigma$	0
Uniaxial compression	0	0	$-\sigma$	–0.33
Conventional triaxial compression	–0.2 σ	–0.2 σ	$-\sigma$	–0.58
Conventional triaxial compression	–0.4 σ	–0.4 σ	$-\sigma$	–1.00
Conventional triaxial compression	–0.6 σ	–0.6 σ	$-\sigma$	–1.83
Triaxial equivalent compression	$-\sigma$	$-\sigma$	$-\sigma$	–∞

Notes: The values of stress triaxiality in different stress states. In Table I R_σ denotes the value of stress triaxiality corresponding in different stress states, σ_1, σ_2, and σ_3 are the principal stresses, and it is assumed that $\sigma_1 \geq \sigma_2 \geq \sigma_3$ according to the positive-negative prescription of stress in elastic mechanics

In following sections we present experimental and numerical investigations to evaluate the effects of stress state on the deformation and failure of rock. These effects will be evaluated by R_σ.

The relationship between failure modes and stress triaxiality

Various failure mechanisms and failure modes of rocks have been put forward in many published documents (Tien, Kuo, and Juang, 2006; Wong and Baud, 2012; Xie *et al.*, 2011). However, these different failure modes have not been classified, and different stress states have not been considered in interpreting the experimental results, nor has the relationship between failure modes and strength parameters been investigated. Laboratory testing proved that many factors, including loading condition, confining pressure, and extent of microscopic discontinuities, could contribute to different failure modes of a rock sample.

It is known that different failure mechanisms and failure modes will manifest under various loading conditions. Under uniaxial compressive stress, due to localized stress concentrations around microscopic discontinuities, rock samples may fail in tension or in shear. In general, tension failure tends to occur in the most brittle rock, and shear failure will occur in fine-grained rock (Szwedzicki, 2007; You, 2009). Under conventional triaxial compression with higher confining pressures, shear failure tends to occur in most rock types, and this has been verified by many experiments (Tanapol, Chaowarin, and Kittitep, 2013; Tarasov and Potvin, 2013; Tiwari and Rao, 2004). Shear failure has been extensively observed in a certain stress state for a given type of rock. Which failure mode will appear in many other loading conditions – tension or shear failure? However, to our knowledge, this has not been reported yet. Therefore, we need to further clarify the complex relationship between failure mode and stress state.

In order to investigate the influence of stress state and its relationship with failure mode, conventional triaxial compression tests were conducted on sandstone and the

fracture surfaces of samples tested under different confining pressures were studied by scanning electron microscopy (SEM). Sandstone specimens, from Tongchuan in Shaanxi Province, China were chosen for the experimental study in this research. The sandstone has a crystalline and blocky structure, which is macroscopically very homogeneous. The main minerals in the sandstone are calcite, quartz, and feldspar, and the main chemical components are Si and Ca. For this research, a total of 80 sandstone samples were prepared for carrying out conventional triaxial compression tests at confining pressures of 0, 5, 10, 15, 20, 25, 30, 35, 40, 45, 50, and 55 MPa. The samples were drilled from two rectangular blocks. During drilling, the sandstone samples were machined along the same direction in order to avoid the influence of anisotropy on the experimental results. In order to obtain exact results as well as the best comparison, all the experiments were performed on dry specimens at room temperature. In accordance with the method suggested by the ISRM, the length-to-diameter ratio of test samples should be in the range of 2.0–3.0 in order to minimize the influence of the end friction effects on the results. Therefore, all tested sandstone samples were cylindrical with 50 mm diameter and 100 mm length. All the conventional compression tests were conducted at a loading rate of 0.002 mm/s. The failure modes of tested samples under different confining pressures are shown in Figure 2.

From Figure 2, we can see that the failure modes under different confining pressures are noticeably different. Cracks propagated parallel to the specimen's axis, and the samples failed by splitting, which represented typical brittleness with confining pressures ranging from 0 MPa to 5 MPa, as shown in Figure 2a and Figure 2b. The failure mode was transformed into shear failure with increasing confining pressure. With confining pressures in the range of 10–40 MPa, although the failure mode was shear failure on a macroscopic scale, most of these samples displayed traces of tension, illustrating that shear failure was influenced by tension and shear deformation acting together. Moreover, with increasing confining pressure the effect of tension on failure declined gradually and the angle between the fracture surface and the specimen's axis increased, as shown from Figures 2c to 2i. When the confining pressures increased to 45 MPa and higher, the failure surfaces became flat and smooth in macroscopic view and powder-like materials were produced by severe friction near the shear fracture surfaces, indicating that failure was the result of shear stress. The fracture surfaces were about 45 degrees from the specimen's axis, as shown from Figures 2j to 2l. The axial strength increased with increasing confining pressure. The test results and calculation results under different confining pressures are presented in Table II.

It can be seen from Table II that as the confining pressure increased, the value of stress triaxiality decreased and the compression degree increased, and the failure mode changed progressively from tension fracture to local shear failure and to general shear failure. Table II indicates that two samples failed by splitting at axial failure stresses of 61.8 MPa and 77.6 MPa respectively, and three samples failed by general shear at axial failure stresses of 263.7, 270.1, and 275. 5 MPa. The axial failure stresses of the samples that failed by

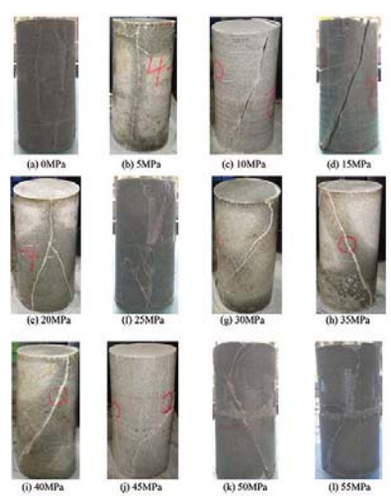


Figure 2—Failure modes of sandstone under different confining pressures. (a) 0 MPa, (b) 5 MPa, (c) 10 MPa, (d) 15 MPa, (e) 20 MPa, (f) 25 MPa, (g) 30 MPa, (h) 35 MPa, (i) 40 MPa, (j) 45 MPa, (k) 50 MPa, (l) 55 MPa

Table II

The failure parameters in conventional triaxial compression tests for sandstone

Number	σ_3 (MPa)	Confining stress $= \sigma_1 = \sigma_2$ (MPa)	τ_{max} (MPa)	R_σ	Failure mode
1	–61.8	–0.0	30.9	–0.33	Splitting
2	–77.6	–5.0	36.3	–0.40	
3	–130.3	–10.0	60.15	–0.42	Local shear
4	–153.4	–15.0	69.2	–0.44	
5	–176.3	–20.0	78.15	–0.46	
6	–200.3	–25.0	87.65	–0.48	
7	–216.1	–30.0	93.05	–0.49	
8	–238.3	–35.0	101.65	–0.51	
9	–250.6	–40.0	105.30	–0.52	
10	–263.7	–45.0	109.35	–0.54	General shear
11	–270.1	–50.0	110.05	–0.56	
12	–275.5	–55.0	110.25	–0.58	

Notes: The failure parameters in conventional triaxial compression tests for sandstone. In Table II, σ_1, σ_2, and σ_3 are the principal stresses at failure, in conventional triaxial compression tests, both σ_1 and σ_2 are the confining stresses, σ_3 is the axial stress at failure, τ_{max} is the maximal shear stress at failure, and R_σ denotes the values of stress triaxiality in different stress states. The unit of σ_1, σ_2, and σ_3 and τ_{max} is MPa, and the stress triaxiality is dimensionless

local shear are also presented in Table II. Therefore, the axial failure stresses of samples probably depended on their failure mode. For the same failure mode, the variations in axial failure stress may be relatively small. However, when various failure modes take place the variations may be very large.

Areas on fracture surfaces were examined by SEM under high magnification. SEM micrographs of fracture surfaces of samples that failed in various modes under various confining pressures are presented in Figure 3. It can be clearly seen that the fracture surface changed from coarse to flat and smooth with increasing confining pressure. At zero confining pressure the fracture surface was ragged and the fracture directions at different points on the fracture surface were strongly random, as shown in Figure 3a, indicating that this kind of fracture was influenced mainly by inherent flaws in the sandstone. When the confining pressure increased to 10 MPa, although the failure mode was shear failure on a macroscopic scale (Figure 2c), the fracture at different points showed a certain level of directionality (Figure 3b), thus indicating that the shear failure was influenced by the inherent cracks and flaws. As the confining pressure increased to 30 MPa, the failure mode was also shear failure on a macroscopic scale (Figure 2g). As shown in Figure 3c, there was little random fracture initiation near the flaws and microcracks, and slip bands appeared among the original cracks, leading to an enhanced directionality of the fracture at different points. The failure mode shown in Figures 3b and 3c was shear failure but affected by tensile stress, so it was defined as local shear failure. When the confining pressure continued to increase to 45 MPa, the failure mode was also shear failure (Figure 2j). Figure 3d indicates that there was almost no random fracture initiation near the flaws and microcracks, numerous slip bands appeared among original cracks, almost all of which had the same slip direction, and the fracture at different points showed stronger directionality. At a confining pressure of 55 MPa, numerous uniformly distributed parabolic dimples appeared on the fracture surface along the same direction. The shear failure shown in Figures 3d and 3e was controlled by shear stress and not affected by tensile stress, so it was defined as general shear failure. In this case, the samples failed in general shear failure due to the slip of shear bands, and the failure was no longer affected by voids and cracks.

Other researchers have reported similar changes of failure mode with stress states. Wang *et al.* (2012) found that the failure behaviour of siltstone in conventional triaxial compression tests was influenced by tension and shear deformation acting together at low confining pressures; and with increasing confining pressures, the effect of tension on failure declined to an extent that failure was completely controlled by shear deformation. That is to say, the failure mode of siltstone was local shear failure under low confining pressures; in this case, the volumetric strain had some effect on the failure, and this effect declined gradually with increasing confining pressure. Once a certain confining pressure had been reached, the failure was general shear failure and it depended entirely on the maximum shear stress. Su and Fu (2014) showed that for sandstone, splitting occurred under uniaxial compression; the maximum shear stress at failure increased with increasing confining pressure until a certain confining pressure was reached, after which the maximum shear stress at failure was almost constant.

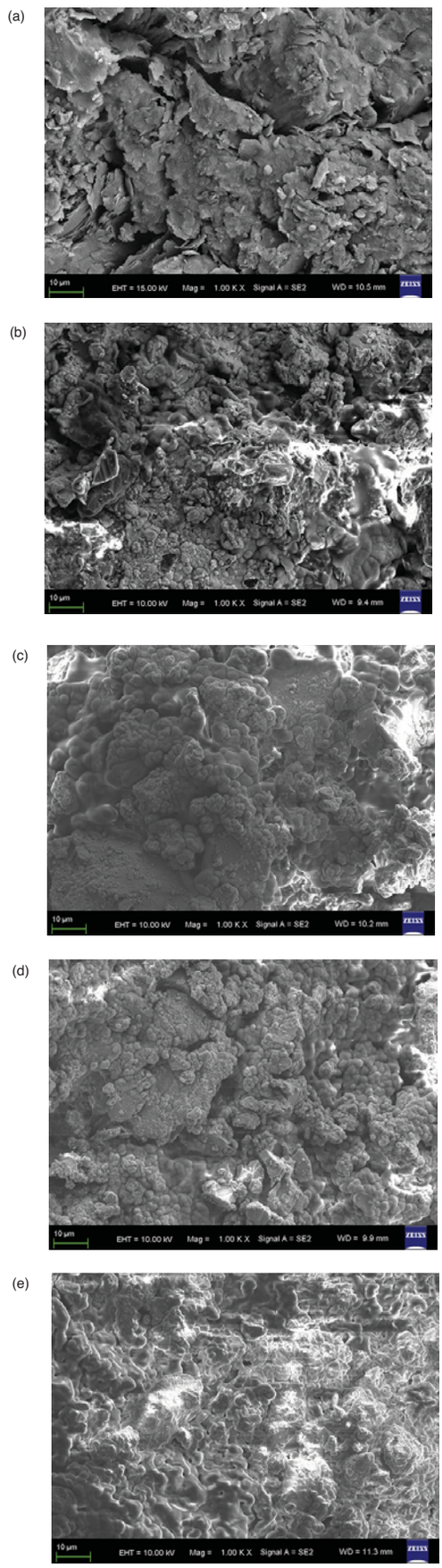


Figure 3—SEM images of fracture surfaces of sandstone under different confining pressures (×1000). (a) 0 MPa, (b) 10 MPa, (c) 30 MPa, (d) 45 MPa, (e) 55 MPa

From the above test results, three distinct failure modes were identified for isotropic rock samples: tension fracture, in which tensile stress predominated; local shear failure, in which shear stress dominated; and general shear failure, in which shear stress also dominated. Both local shear failure and general shear failure fall within the scope of shear failure dominated by the shear stress. What makes them different is that the former is dominated by shear stress and influenced by tensile stress, but the latter is dominated only by shear stress, and is not influenced by tensile stress. The experimental data, including conventional compression test results for sandstone (Table II) and other published test results (Su and Fu, 2014; Wang *et al.*, 2012) provides evidence that for a given rock sample, the failure mode may change progressively from tension fracture to local shear to general shear failure as loading conditions change from unixial compression to biaxial compression and triaxial compression, and also with increasing confining pressures under conventional triaxial compression.

In this article, the stress triaxiality is applied to reveal the degree of tension and compression in different stress states. Combining the calculation results in Table I, the value of stress triaxiality decreases as the loading condition changes from axial compression to triaxial compression, and it also decreases with increasing confining pressure under conventional triaxial compression. Therefore, these observations prove that the corresponding relationship between failure modes and stress states can be as follows: for a given rock type, as the stress state changes from tension to unixial compression, biaxial compression, and triaxial compression, and also as the stress triaxiality decreases, the failure mechanism will change progressively from tension fracture to local shear failure and general shear failure. This suggests that the failure mode for a given rock sample is controlled mainly by the stress triaxiality at the weakest position. Moreover, two demarcation values of the stress triaxiality exist among these three failure mechanisms. We take R_1 and R_2 to be respectively the demarcation values of the stress triaxiality between tension fracture and local shear failure, and between local shear failure and general shear failure. Accordingly, the complete relationship between the failure mechanism and the stress triaxiality can be written as follows: the failure mechanisms at the weakest position will be tension fracture when $R_\sigma, \geq R_1$ local shear failure when $R_2 \leq R_\sigma < R_1$ and general shear failure when $R_\sigma < R_2$.

The relationship between failure mode and material characteristics

Similar to the stress state, the material characteristics also have a significant effect on the failure mechanism of rock. For different types of rock, different failure modes will occur even in the same stress state. Uniaxial compression tests of red sandstone and granite were conducted in our laboratory. Under uniaxial compression, red sandstone underwent shear failure, as shown in Figure 4a; but granite failed by longitudinal splitting, as shown in Figure 4b. Different rock properties result in varying failure modes. In the same stress state, the failure mode depends upon the degree of homogeneity; as the rock texture becomes finer and more isotropic the failure mechanism will change progressively from tension to shear failure. Coarse-grained rocks contain many flaws, and the influence of these flaws is greatest in the state of greater stress triaxiality, decreasing in the state of lower stress triaxiality. Specifically, the larger the stress triaxiality is, the higher the probability that flaws will affect the failure mode and strength.

A new failure criterion corresponding to different failure modes

The condition at failure in a rock sample can be described by the general equation

$$f = f_c \qquad [7]$$

where f_c denotes the allowable value of f. When $f \geq f_c$, failure will occur. The definition of f will be achieved in the form of tension fracture, local shear failure, and general shear failure, referred as to f_1, f_2, and f_3 respectively.

Tension fracture

The tension fracture observed in experiments can be classified as tension fracture under tensile stress, and splitting under uniaxial compression and triaxial compression at low confining pressures. It is easy to understand tension fracture under tensile stress. Splitting under uniaxial compression and triaxial compression at low confining pressures has been documented by various investigators (Paterson, 2005; Szwedzicki, 2007; Fakhimi and Hemami, 2015), but what causes the splitting is still obscure and there is no model for evaluating this phenomenon. In terms of failure mechanism, splitting is essentially tension fracture, but there is no tensile stress in macroscopic stress fields under loading conditions of uniaxial compression and triaxial compression at low confining pressures. Therefore, traditional failure criteria expressed by stress no longer apply.

Different rock types have unique characteristics, which also play a critical role in the whole failure process. Owing to its brittle nature, the capability of rock to withstand tensile strain is very weak. When the tensile strain at certain point reaches a critical value, the micro-crack will propagate quickly. On the other hand, due to the natural flaws and fissures inherent in rock, tension fracture is sensitive to the microstructure, and failure is obviously influenced by the local effect. For these reasons, splitting is difficult to describe in terms of the macroscopic stress field, or more

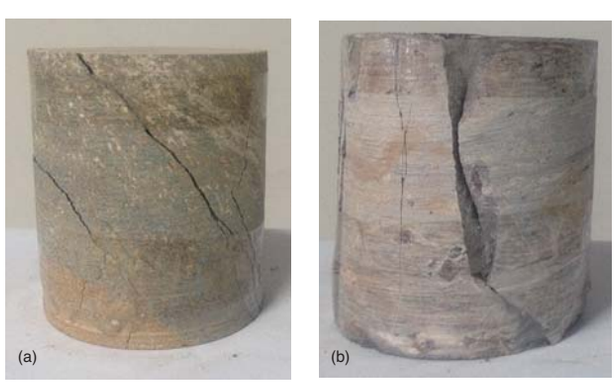


Figure 4—Different failure modes in uniaxial compression. (a) Shear failure of red sandstone, (b) longitudinal splitting of granite

appropriately, the strain field. The splitting observed in many experiments appeared initially along the direction of maximal tension strain, suggesting that the maximal tension strain ε_1 should be the dominant factor leading to the tension fracture. Also, the influence of other principal strains ε_2 and ε_3 on tension fracture should be included in complex stress states. Based on the above analysis, f_1 can be defined in terms of ε_1, ε_2, and ε_3 as

$$f_1 = \varepsilon_1 + b_1\left(\varepsilon_2 + \varepsilon_3\right) = f_{c1} \quad \text{when } R_\sigma \geq R_1 \qquad [8]$$

Tension fracture usually occurs at the position of greatest stress concentration, and failure is dominated by the maximal tensile strain. Therefore, it is assumed that the most likely position for tension fracture is where the stress triaxiality is greatest in the stress field, and the direction of fracture initiation is consistent with the maximum tensile strain at this position. According to the newly proposed criterion for tension fracture, it is inferred that failure will occur when f_1 exceeds its maximum limit value f_{c1}. Let ε_1, ε_2, and ε_3 represent three principal strains at the weakest position, and assume that $\varepsilon_1 \geq \varepsilon_2 \geq$ and ε_3, b_1 represents the influence coefficient of ε_2 and ε_3 on ε_1 and f_{c1} denotes the maximum limit value of f_1. Both b_1 and f_{c1} are constants that depend on the rock characteristic and can be calculated from at least two sets of experimental data in the range of tension fracture. It should be noted that Equation [8] applies only to tension fracture of rock; that is, under the condition of $R_\sigma \geq R_1$ and $\varepsilon_1 > 0$ at the weakest position. A large number of experimental results (Fakhimi and Hemami, 2015; Su and Fu, 2014; Wang *et al.*, 2012) indicate that Equation [8] can well explain and predict the splitting of rock under uniaxial compression and triaxial compression at low confining pressures.

Local shear failure

Local shear failure is dominated by the maximal shear stress and influenced by the tensile stress. We have found that the maximal shear stress at failure among samples of the same lithology varies with the stress triaxiality. Furthermore, within the same mode of local shear failure, the influence of tensile stress on local shear failure decreases progressively with decreasing stress triaxiality. The relationship between the maximal shear stress at failure and the stress triaxiality is, of course, not entirely linear. However, for convenience of application, a linear approximation is reasonable within a limited range of mean stress. Therefore f_2 can be expressed as

$$f_2 = \tau_{\max} + b_2\left(R_\sigma - R_2\right) = f_{c2} \quad \text{when } R_2 \leq R_\sigma < R_1 \qquad [9]$$

where, $\tau_{\max}$ is the maximal shear stress at the weakest position, f_{c2} denotes the limit value of f_2, b_2 represents the influence of stress state on local shear failure, and R_2 is the demarcation value between local shear failure and general shear failure. Both b_2 and f_{c2} are constants that depend on the rock characteristics, and at least two experimental data points are required in order to determine these two parameters. They can be obtained by conducting at least two tests in different stress states within the range of local shear failure, and more testing for confirmation is highly recommended. As regards the local shear failure, because much plastic deformation in the process of failure can relax the stress concentration around the flaws and voids, the voids are difficult to grow and coalesce along the direction of the maximum tensile stress. Therefore, the shape of voids will grow into an ellipsoid or a long strip, and local shear bands will be produced among ligaments of voids. Shear-linking of voids along the direction of the maximal shear stress and the propagation of shear bands are what lead to this kind of failure. Local shear bands usually appear in the position with higher degrees of stress concentration and their propagation is dominated by the maximal shear stress. Failure is therefore modelled in terms of the maximal shear stress and the stress triaxiality, and the weakest position can also be considered as the position where the stress triaxiality is the largest in the stress field. It is important to note that Equation [9] can apply to local shear failure of rock only when $R_2 \leq R_\sigma < R_1$.

General shear failure

General shear failure is caused by the propagation of shear bands. Unlike local shear failure, tension deformation has no influence on this kind of failure. General shear failure is not affected by tensile stress and is dominated by the maximal shear stress. Many experimental results from rock under conventional triaxial compression (Tanapol, Chaowarin, and Kittitep, 2013; Tien, Kuo, and Juang, 2006; Yang, Su, and Xu, 2005) have shown that once a certain confining pressure is reached, the maximal shear stress at failure (referred to as general shear failure in this article) is almost constant. When described by the stress triaxiality, in terms of general shear failure, the maximal shear stress at failure remains almost constant with changing stress triaxiality. Based on the considerations discussed above, the failure function for predicting general shear failure is directly modelled by the maximal shear stress and can be written as

$$f_3 = \tau_{\max} = f_{c3} \qquad \text{when } R_\sigma < R_2 \qquad [10]$$

where $\tau_{\max}$ is the maximal shear stress at the weakest position and f_{c3}, a material constant, f_3 denotes the limit value of f_3 corresponding to the onset of general shear failure. When general shear failure occurs in the specimen, the stress triaxiality is lower than in the case of local shear failure, and more plastic deformation is produced. Therefore there is almost no stress concentration around flaws in the whole failure process, shear bands are caused by the maximal shear stress, and the unstable propagation of shear bands leads to general shear failure. Shear bands tend to appear in the position with higher shape deformation energy, and their propagation is controlled by the maximum shear stress. Therefore, general shear failure is modelled only by the maximal shear stress under the assumption that the weakest position is located where the shape deformation energy is largest in the stress field, and the direction of fracture initiation is consistent with the plane of the maximum shear stress. Equation [10] can be used for general shear failure of rock only when $R_\sigma < R_2$.

In Equations [8], [9], and [10], the demarcation values of stress triaxiality R_1 and R_2 are constant for the same rock type and vary for different rock types. For a given rock type, R_1 and R_2 can be determined by a series of experiments with different failure modes in a wide range of stress triaxiality.

In Equations [9] and [10], f_{c2} and f_{c3} are referred as to allowable values of failure strength for local shear failure and general shear failure respectively, and they should be constant for the same rock type. As the effect of stress state on local shear failure has been included in Equation [9], f_{c2} and f_{c3} should be equal if the demarcation value of stress triaxiality between local shear failure and general shear failure is accurately determined.

Experimental verification

In order to validate the performance of the proposed failure criterion, comparisons were made between the experimental data on sandstone obtained in this investigation, the test data for rock salt and granite taken from the literature, and the predictions obtained from the proposed criterion based on Equations. [8], [9], and [10].

Sandstone

Based on the experimental data for sandstone under conventional triaxial compression tests shown in Table II the failure mode changed progressively from tension fracture to local shear failure and general shear failure with increasing confining pressure. More specifically, the failure mode was tension fracture when the confining pressure was zero and 5 MPa, local shear failure at confining pressures n the range 10–40 MPa, and general shear failure at confining pressures of 45 MPa and higher. The calculation results of the stress triaxiality in line with each confining pressure and the maximal shear stress at failure are shown in Table II. Using the test data, it can be established that the demarcation value of stress triaxiality R_2 between local shear failure and general shear failure is R_2 = –0.53. Equations [9] and [10] are applied to verify the shear failure of sandstone, and the material constants b_2 and f_{c1} in Equation [9] and f_{c3} in Equation [10] are calculated by the least squares method using the third to the twelfth groups of data in Table II. The calculation results are as follows: b_2 = 448.7 MPa, f_{c2} = 109.4 MPa, and f_{c3} = 109.4 MPa. Therefore, the local shear failure criterion for sandstone results in

$$\tau_{\max} + 448.7\left(R_\sigma + 0.53\right) = 109.4 \quad (-0.53 \le R_\sigma \le -0.42) \qquad [11]$$

and the criterion for general shear failure is expressed as

$$\tau_{\max} = 109.4 \quad (R_\sigma < -0.53) \qquad [12]$$

In Equations. [11] and [12], the units for $\tau_{\max}$ are MPa and the stress triaxiality is dimensionless. The comparison between experimental data points for sandstone and criterion predictions using Equations [11] and [12] is presented in Figure 5, with stress triaxiality on the horizontal axis and maximal shear stress on the vertical one. In Figure 5, 'Max shear stress' is the maximal shear stress at the fracture position at failure under different stress states, stress triaxiality denotes the value of stress triaxiality at failure, the solid line corresponds to the predictions obtained from the proposed failure criterion through Equations [11] and [12], and the data points represent experimental results for sandstone in conventional compression tests at different confining pressures. Figure 5 gives the maximal shear stress at failure in different stress states, which is described by the stress triaxiality. As shown, to local shear failure, the maximal shear stress at failure increases approximately linearly with decreasing stress triaxiality, and to general shear failure, the maximal shear stress at failure approaches a constant with decreasing stress triaxiality, illustrating that local shear failure is influenced by the combined actions of maximal shear stress and stress triaxiality, while general shear failure is controlled only by maximal shear stress. From the comparisons shown in Figure 5, the proposed failure criterion is in perfect agreement with experimental results.

Rock salt

According to the experimental data for rock salt under conventional triaxial compression tests (Zhigalkin *et al.*, 2008), the failure mode was always local shear at confining pressures ranging from zero to 20 MPa. The calculated results for stress triaxiality corresponding to each confining pressure and the maximal shear stress at failure are shown in Table III. Considering the lack of data on the failure mode of general shear, R_2 = –0.6 is regarded as the approximate demarcation value of stress triaxiality between local shear failure and general shear failure. Material constants b_2 and

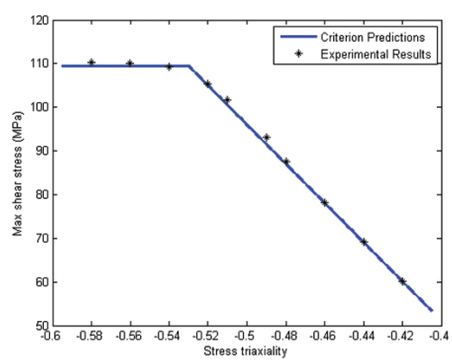


Figure 5—Comparison of experimental data and criterion predictions for sandstone. Max shear stress is the maximal shear stress at the fracture position at failure, stress triaxiality denotes the value of the stress triaxiality at failure, solid line corresponds to the predictions obtained from the proposed failure criteria given by Equations [11] and [12], data points represent experimental results for sandstone samples, the unit of Max shear stress is MPa, and the stress triaxiality is dimensionless

Table III

Failure parameters in conventional triaxial compression tests for rock salt

σ_3 (MPa)	Confining stress $\sigma_1 = \sigma_2$	$\tau_{\max}$ (MPa)	R_σ
–21.4	0	10.7	–0.33
–34.7	–1.0	16.9	–0.36
–45.8	–2.5	21.9	–0.39
–61.2	–5.0	28.1	–0.42
–67.3	–7.5	29.9	–0.46
–75.8	–10.0	32.9	–0.49
–109.0	–20.0	44.5	–0.56

Notes: σ_1, σ_2, and σ_3 are the principal stresses at failure in conventional triaxial compression tests, σ_1 and σ_2 are the confining stresses, σ_3 is the axial stress at failure, $\tau_{\max}$ is the maximal shear stress at failure, and R_σ denotes the values of stress triaxiality in different stress states. The unit of σ_1, σ_2, σ_3, and $\tau_{\max}$ is MPa, and the stress triaxiality is dimensionless

f_{c2} in Equation [9] are calculated by the least squares method using all the experimental data points in Zhigalkin *et al.* (2008), and the calculation results are as follows: b_2 = 150 MPa, and f_{c2} = 50.9 MPa. Therefore, the criterion for local shear failure of rock salt in Zhigalkin *et al.* (2008) can be expressed in terms of τ_{max} and R_σ as

$$\tau_{max} + 150(R_\sigma + 0.6) = 50.9 \ (-0.56 \le R_\sigma \le -0.33) \qquad [13]$$

In Equation [13], the units for τ_{max} are MPa and the stress triaxiality is dimensionless. The criterion predictions using Equation [13] are compared with experimental data on rock salt from the literature in Figure 6. The solid lines correspond to the predictions obtained from the proposed failure criterion, while the data points represent experimental results. As shown in Figure 6, to local shear failure, the maximal shear stress at failure increases approximately linearly with decreasing stress triaxiality. The proposed failure criterion gives very good agreement with the experimental results.

Marble

Based on the experimental data for medium-grained marble in conventional triaxial compression tests (Yang, Su, and Xu, 2005), the failure mode underwent a transition from local shear to general shear with increasing confining pressure. More specifically, the failure mode was local shear at confining pressures in the range of 0–30 MPa, while general shear failure occurred when the confining pressure increased to 35 MPa and higher. Because test data for different stress states obtained from the previous experiments is sparse, it is difficult to accurately determine the demarcation value of stress triaxiality R_2 between local shear failure and general shear failure. We can only conclude that the range of the demarcation value of stress triaxiality R_2 between local shear failure and general shear failure is $-0.5 \le R_2 \le -0.48$. However, $R_2 = -0.5$ is obtained by fitting all the test data points and the result agrees well with the data. Due to the limited data, $R_2 = -0.5$ is estimated to be the approximate demarcation value of stress triaxiality between local shear failure and general shear failure. Material constants b_2 and f_{c2} in Equation [9] and f_{c3} in Equation [10], calculated by the least squares method using all the experimental data points in Yang, Su, and Xu (2005), are as follows: b_2 = 326 MPa, f_{c2} = 106.8 MPa, and f_{c3} = 106.8 MPa. Therefore, the local shear failure criterion for medium-grained marble has the form

$$\tau_{max} + 326(R_\sigma + 0.5) = 106.8 \ (-0.50 \le R_\sigma \le -0.33) \qquad [14]$$

and the criterion for general shear failure is expressed as

$$\tau_{max} = 106.8 \ (R_\sigma \le -0.50) \qquad [15]$$

In Equations [14] and [15], the units for τ_{max} are MPa. The comparison between experimental data points for medium-grained marble and criterion predictions using Equations [14] and [15] is presented in Figure 7. As shown, within the range of shear failure, the maximal shear stress at failure also approaches a constant value with decreasing stress triaxiality.

For coarse-grained marble (Yang, Su, and Xu, 2005), the failure mode was always local shear at the confining pressure between zero and 30 MPa. Due to a lack of experimental results on general shear failure, $R_2 = -0.6$ is considered as the approximate demarcation value of stress triaxiality between local shear failure and general shear failure. Material constants b_2 and f_{c2} in Equation [9] were calculated by the least squares method using all the experimental data points in Yang, Su, and Xu (2005), and the calculation results are as follows: b_2 = 201.5 MPa, and f_{c2} = 80 MPa. Therefore, the local shear failure criterion for coarse-grained marble (2005) can be expressed by

$$\tau_{max} + 201.5(R_\sigma + 0.6) = 80 \ (-0.54 \le R_\sigma \le -0.37) \qquad [16]$$

In Equation [16], the units for τ_{max} are also MPa. The comparison between experimental data points for marble with coarse grains and criterion predictions using Equation [16] is given in Figure 8. As shown, to local shear failure, the maximal shear stress at failure also increases approximately linearly with decreasing stress triaxiality, which also illustrates that high stress triaxiality will speed up the occurrence of local shear failure within a certain range.

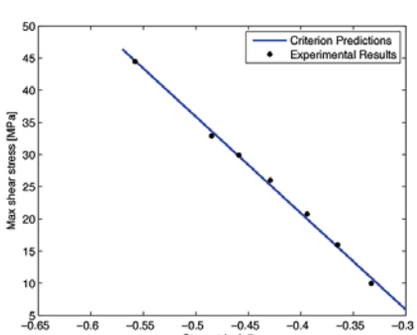


Figure 6—Comparison of experimental data (Zhigalkin *et al.*, 2008) and criterion predictions for rock salt. Max shear stress is the maximal shear stress at the fracture position at failure, stress triaxiality denotes the value of the stress triaxiality at failure, solid line corresponds to the predictions obtained from the proposed failure criterion given by Equation [13], data points represent experimental results for salt rock, the unit of Max shear stress is MPa, and the stress triaxiality is dimensionless

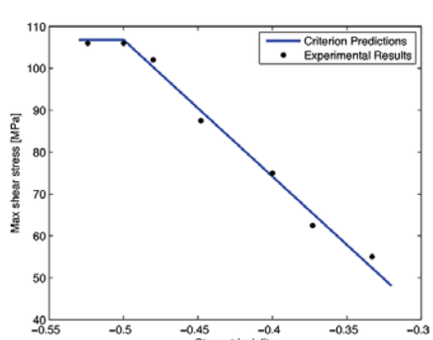


Figure 7—Comparison of experimental data (Yang, Su, and Xu, 2005) and criterion predictions for medium-grained marble. Max shear stress is the maximal shear stress at the fracture position at failure, stress triaxiality denotes the value of the stress triaxiality at failure, solid line corresponds to the predictions obtained from the proposed failure criteria given by Equations [14] and [15], data points represent experimental results for medium-grained marble, the unit of Max shear stress is MPa, and the stress triaxiality is dimensionless

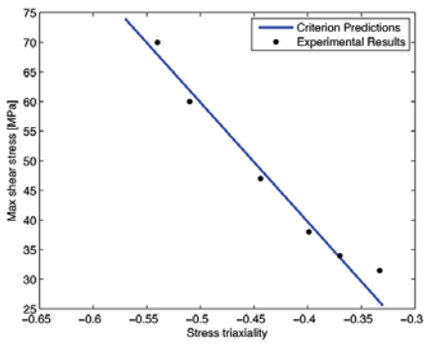


Figure 8—Comparison of experimental data (Yang, Su, and Xu, 2005) and criterion predictions for coarse-grained marble. Max shear stress is the maximal shear stress at the fracture position at failure, stress triaxiality denotes the value of the stress triaxiality at failure, solid line corresponds to the predictions obtained from the proposed failure criterion given by Equation [16], data points represent experimental results for coarse-grained marble, the unit of Max shear stress is MPa, and the stress triaxiality is dimensionless

The comparisons shown in Figures 5-8 prove that the proposed failure criterion is able to accurately predict the failure strength of isotropic rock of various types in various stress states.

As can be seen from the comparisons in Figure 7 and Figure 8, the test results for coarse-grained marble are relatively dispersive, and the experimental data for medium-grained marble match the criterion well. Therefore the deviation between experimental data points and criterion prediction for medium-grained marble is smaller than that for coarse-grained marble. When the value of stress trialiaxity is reduced from -0.33 to -0.55, coarse-grained marble always fails by local shear failure, but the failure mode of medium-grained marble changes from local shear to general shear. As discussed above, material characteristics also have a significant influence on the failure mechanism and failure criteria of rock.

Discussion

From the above analyses and comparisons, it can be clearly seen that the relationship between failure mode and stress state is well described using stress triaxiality and the proposed failure criterion can accurately predict the failure of rock in different stress states. We previously investigated the fracture mechanisms of metals in various stress states (Li, Shi, and Tang, 2014), and found that the variation law for macroscopic fracture modes with stress triaxiality for metals is similar to that for rock, but there appears a great difference between the two materials in terms of the mesoscopic mechanism. The primary factor that affects the fracture mechanism of metals is the amount of plastic deformation during the fracture process, while the major factor that influences the failure mode of rock tends to be the evolution of flaws in different stress states.

However, there are still some limitations in this method. For example, the values of stress triaxiality in the experiments were limited within a certain range. All the available data is based on conventional triaxial compression tests on rock, hence the values of stress triaxiality can only cover the range from -1 to -0.33 owing to the limited range of confining pressures generated by triaxial compression equipment. Because the values of stress triaxiality are within this range, failure modes are mainly local shear and general shear, and tension fracture appears rarely. Therefore, the experimental verification in this article is confined to the analysis of local shear failure and general shear failure. Nevertheless, the failure criterion is based on three different failure mechanisms and can be applied all failure behaviours in complex stress states.

For a given rock type, demarcation values of stress triaxiality R_1 and R_1 are constant and should be obtained by a series of experiments with different failure modes covering a wide range of stress states. Due to a lack of continuous experimental data covering a wide range of stress triaxiality, the value of R_2 in this article is estimated using the available data. For this reason, more continuous experiments over a wider range of stress triaxiality are required in order to accurately determine the demarcation values of stress triaxiality.

Moreover, the existence of microscopic discontinuities is responsible for 'size effects', that is, strength reduction with increased sample size. The larger the sample, the higher the probability that the discontinuities will affect the failure mode and the strength. Therefore, the results presented in this paper are reliable for some relatively homogeneous rock samples that can be regarded as isotropic. On the other hand, for some anisotropic and inhomogeneous rock samples, there are too many randomly distributed unknown flaws and micro-cracks in the rock, and the failure mode will be greatly influenced by these flaws and cracks, especially in states of high stress triaxiality.

Conclusion

In this article, the concept of stress triaxiality was introduced to reveal different stress states, and failure mechanisms and strengths of rock under various complex loading conditions were investigated. The major results obtained are as follows.

1. The introduction of stress triaxiality to reveal different stress states makes it possible to determine whether the rock samples fail in tension, in local shear, or in general shear. For a given rock type, with decreasing stress triaxiality, the failure mechanism will change progressively from tension fracture to local shear and general shear. Two demarcation points of stress triaxiality, expressed as R_1 and R_2, exist between tension fracture and local shear failure, and between local shear and general shear. Accordingly, the failure mode at the weakest position will be tension fracture when $R_{\sigma}, \geq R_1$ local shear failure when $R_2 \leq R_{\sigma} < R_1$ and general shear failure when $R_{\sigma <} R_2$.
2. Failure of rock by splitting under axial compression is essentially tension fracture. However, from a macroscopic viewpoint, there is no tensile stress in a specimen under axial compression and conventional triaxial compression at low confining pressures. The splitting fracture of rock is essentially caused by tensile strain because of rock's high brittleness and weak capability to withstand tensile strain. For these reasons, the splitting is difficult to describe using the macroscopic stress field, or more appropriately, the strain field.

3. It has been proved that the way the sample fails (the failure mode) affects the obtained strength of the sample. The controlling parameter is a function of the failure mode. Considering tension fracture, local shear failure, and general shear failure, different parameters that dominate these three failure modes were taken into account, and failure criteria corresponding to these three failure modes were developed and presented to evaluate the failure behaviour of rock in various complex stress states. Furthermore, predictions of the failure criterion were compared with experimental data for sandstone, rock salt, and marble under conventional triaxial compression tests. The comparison indicates that the newly proposed failure criterion offers consistent results for these three rock types in a range of stress triaxiality from uniaxial to triaxial compression.

Understanding the effect of failure mode on the failure criterion allows an in-depth interpretation of experimental results obtained in various stress states. The relationship between failure mode and failure criteria of rock samples in various stress states has been elucidated in this paper, with the aim of inducing researchers in rock mechanics to conduct further investigations and provide a theoretical framework to geotechnical applications.

Acknowledgement

This work was supported by the National Natural Science Foundation of China (Grant no. 11302167) and the research programme of Shaanxi Provincial Education Department (no. 2013JK0609).

References

Adelinet, M., Fortin, J., Schubnel, A., and Guéguen, Y. 2013. Deformation modes in an Icelandic basalt: From brittle failure to localized deformation bands. *Journal of Volcanology and Geothermal Research*, vol. 255. pp. 15–25.

Amann, F., Kaiser, P., and Button, E.A. 2012. Experimental study of brittle behavior of clay shale in rapid triaxial compression. *Rock Mechanics and Rock Engineering*, vol. 45. pp. 21–33.

Arora, S. and Mishra, B. 2015. Investigation of the failure mode of shale rocks in biaxial and triaxial compression tests. *International Journal of Rock Mechanics and Mining Sciences*, vol. 79. pp. 109–123.

Cai, M.F. and Liu, D.M. 2009. Study of failure mechanisms of rock under compressive–shear loading using real-time laserholography. *International Journal of Rock Mechanics and Mining Sciences*, vol. 46. pp. 59–68.

Chang, C.D. and Haimson, B. 2012. A failure criterion for rocks based on true triaxial testing. *Rock Mechanics and Rock Engineering*, vol. 45. pp. 1007–1010.

Fakhimi, A. and Hemami, B. 2015. Axial splitting of rocks under uniaxial compression. *International Journal of Rock Mechanics and Mining Sciences*, vol. 79. pp. 124–134.

Haimson, B. 2006. True triaxial stresses and the brittle fracture of rock. *Pure and Applied Geophysics*, vol. 163. pp. 1101–1130.

Kittitep, F. and Decho, P. 2012. Effects of cyclic loading on mechanical properties of Maha Sarakham salt. *Engineering Geology*, vol. 112. pp. 43–52.

Li, Z.H., Shi, J.P., and Tang, A.M. 2014. Investigation on fracture mechanisms of metals under various stress states. *Acta Mechanica*, vol. 225, pp. 1867–1881.

Lin, P., Wong, R. H.C., and Tang, C.A. 2015. Experimental study of coalescence mechanisms and failure under uniaxial compression of granite containing multiple holes. *International Journal of Rock Mechanics and Mining Sciences*, vol. 77. pp. 313–327.

Loaiza, S., Fortin, J., Schubnel, A., Gueguen, Y., Vinciguerra, S., and Moreira, M. 2012. Mechanical behavior and localized failure modes in a porous basalt from the Azores. *Geophysical Research Letters*, vol. 39. pp. L19304.

Melkoumian, N.S., Priest, D., and Hunt, S.P. 2009. Further development of the three-dimensional Hoek–Brown yield criterion. *Rock Mechanics and Rock Engineering*, vol. 42. pp. 835–847.

Paterson, M.S. 2005. Experimental Rock Deformation in the Brittle Field. Springer-Verlag, Berlin, Heidelberg. pp. 132–135.

Salvador, S., Rafael, J., and Alexandra, R. 2013. Numerical simulation of the influence of small-scale defects on the true-triaxial strength of rock samples. *Computers and Geotechnics*, vol. 53. pp. 142–156.

Su, C.D. and Fu, Y.S. 2014. Experimental study of triaxial compression deformation and strength characteristics of sandstone. C*hinese Journal of Rock Mechanics and Engineering*, vol. 33. pp. 3164–3169.

Szwedzicki, T. 2007. A hypothesis on modes of failure of rock samples tested in uniaxial compression. *Rock Mechanics and Rock Engineering*, vol. 40. pp. 97–104.

Tanapol, S., Chaowarin, W., and Kittitep, F. 2013. True-triaxial compressive strength of Maha Sarakham salt. *International Journal of Rock Mechanics and Mining Sciences*, vol. 61. pp. 256–265.

Tang, C.A. and Hudson, J.A. 2010. Rock failure mechanisms: explained and illustrated. Taylor and Francis, London. pp. 231–239.

Tarasov, B. and Potvin, Y. 2013. Universal criteria for rock brittleness estimation under triaxial compression. *International Journal of Rock Mechanics and Mining Sciences*, vol. 59. pp. 57–69.

Tien, Y.M., Kuo, M.C., and Juang C.H. 2006. An experimental investigation of the failure mechanism of simulated transversely isotropic rocks. *International Journal of Rock Mechanics and Mining Sciences*, vol.43. pp. 1163–1181.

Tiwari, R.P. and Rao, K.S. 2004. Physical modeling of a rock mass under a true triaxial stress state. *International Journal of Rock Mechanics and Mining Sciences*, vol. 41. pp. 1–6.

Tiwari, R.P. and Rao, K.S. 2007. Response of an anisotropic rock mass under polyaxial stress state. *Journal of Materials in Civil Engineering*, vol. 19. pp. 393–403.

Wang, D., Liu, C.W., Wang, D., Li, X.D., and Xu, Y.H. 2012. Research on the tension-shear deformation and failure criterion of rock under complex stress. *Journal of Sichuan University* (*Engineering Science Edition*), vol. 44. pp. 31–35.

Wong, T.F. and Baud, P. 2012. The brittle-ductile transition in porous rock: A review. *Journal of Structural Geology*, vol. 44. pp. 25–53.

Xie, H.P. and Chen, Z.H. 2004. Rock Mechanics. Science Press, Beijing. pp. 129–135.

Xie, H.P., Liu, J.F., Ju, Y., Li, J., and Xie, L.Z. 2011. Fractal property of spatial distribution of acoustic emissions during the failure process of bedded rock salt. I*nternational Journal of Rock Mechanics and Mining Sciences*, vol. 48. pp. 1344–1351.

Xie, N., Zhu, Q.Z., Shao, J.F., and Xu, L.H. 2012. Micromechanical analysis of damage in saturated quasi brittle materials. *International Journal of Solids and Structures*, vol. 49. pp. 919–928.

Yang, S.Q., Jiang, Y.Z., Xu, W.Y., and Chen, X.Q. 2008. Experimental investigation on strength and failure behavior of pre-cracked marble under conventional triaxial compression. *International Journal of Solids and Structures*, vol. 45. pp. 4796–4815.

Yang, S.Q., Jing, H.W., and Wang, S.Y. 2012. Experimental investigation on the strength, deformability, failure behavior and acoustic emission locations of red sandstone under triaxial compression. *Rock Mechanics and Rock Engineering*, vol. 45. pp. 583–606.

Yang, S.Q., Su, C.D., and Xu, W.Y. 2005. Experimental investigation on strength and deformation properties of marble under conventional triaxial compression. *Chinese Journal of Rock and Soil Mechanics,* vol. 26. pp. 475–478.

You, M.Q. 2009. True-triaxial strength criteria for rock. *International Journal of Rock Mechanics and Mining Sciences*, vol. 46. pp. 115–127.

Zhigalkin, V.M., Semenov, V.N., Usol'tseva, O.M., Tsoi, P.A., Asanov, V.A., Baryakh, A.A., Pankov, I.L., Toksarov, V.N., and Evseev, A.V. 2008. Deformation of quasi-plastic salt rocks under different conditions of loading. Report II: regularities of salt rock deformation under triaxial compression. *Journal of Mining Science*, vol. 44. pp. 15-23.

Zhong, J.H., Liu, S.X., and Ma, Y.S. 2015. Macro-fracture mode and micro-fracture mechanism of shale. *Petroleum Exploration and Development*, vol. 42. pp. 269–276.

Zhou, Y. and Lin, J.S. 2013. On the critical failure mode transition depth for rock cutting. *International Journal of Rock Mechanics and Mining Sciences*, vol. 62. pp. 131–137.

Zhou, Y. and Lin, J.S. 2014. Modeling the ductile–brittle failure mode transition in rock cutting. *Engineering Fracture Mechanics*, vol. 127. pp. 135–147.◆

Conference Announcement

SAIMM
THE SOUTHERN AFRICAN INSTITUTE OF MINING AND METALLURGY

MMMA

The Southern African Institute of Mining and Metallurgy
&
Mine Metallurgical Managers Association is proud to host the

WATER 2017 CONFERENCE

lifeblood of the mining industry

10–11 July 2017

Emperors Palace, Hotel Casino Convention Resort Johannesburg

BACKGROUND

The mining industry is faced with a number of challenges regarding the use, recycling and management of their water resources. Some affected parties are unaware that the legislation around the use of water has become more onerous and strict controls have been put in place. This includes the requirements for the application of water use licences.

The scarcity of water in the Southern African region is a fact and the availability of water is a major consideration in the development of mining ventures across the sub-continent. The water authorities throughout the region have developed strategies to address the needs of the mines and their surrounding communities.

Acid Mine Drainage has been a reality for quite some time and with the 'closure' of mines on the Witwatersrand it has become a major issue for communities in Gauteng. A number of initiatives have been put in place to address the challenge and the enormity of the task has taken many by surprise.

The use of fresh water alone is no longer an option and users have to consider alternatives in the treatment and recycling of water. Major advances have been made in the processing of water yet these options have not been shared with the engineers on the mines.

OBJECTIVE

- To sensitise the mining and metallurgical industry to the requirements of the new legislation
- Share the overall water distribution strategy across the sub-continent
- Introduce new technology for the processing and recycling of water
- Report on various initiatives in the reclaiming of water
- Update interested parties on the status of the Acid Mine Drainage threat.

WHO SHOULD ATTEND

- Senior and operational management of mines
- Engineers responsible for mine water management
- Regional and national officials from DoE, DMR, DWS, and DEA
- Companies and individuals offering water related solutions
- Researchers
- Environmentalists and NGOs
- Agricultural sector.

TOPICS

The Conference will include but not be limited to the following topics:

- Legal requirements, amendments, and compliance
- What is required to obtain a water licence
- Acid mine drainage
- Status of water supply
- New technology in proccessing and recovering of water
- Treatment plants
- Water analysis
- Wetlands
- Agriculture *vs.* Mining
- Case studies
- Research.

Sponsor LANXESS Energizing Chemistry LANXESS Chrome Mining (Pty) Ltd.

SAIMM
THE SOUTHERN AFRICAN INSTITUTE OF MINING AND METALLURGY

For further information contact:
Camielah Jardine Conference Co-ordinator • Saimm
P O Box 61127, Marshalltown 2107
Tel: +27 (0) 11 834-1273/7 E-mail: camielah@saimm.co.za
Website: http://www.saimm.co.za

http://dx.doi.org/10.17159/2411-9717/2017/v117n3a7

CFD study of the effect of face ventilation on CH_4 in returns and explosive gas zones in progressively sealed longwall gobs

by S.A. Saki*, J.F. Brune*, G.E. Bogin Jr.*, J.W. Grubb*, M.Z. Emad†, and R.C. Gilmore*

Synopsis

The main purpose of coal mine ventilation design is to provide a sufficient quantity and quality of air to the workers and to dilute methane and other contaminants. It is generally perceived that additional air along the longwall face will improve methane dilution at the face and in the tailgate. However, computational fluid dynamics (CFD) modelling efforts at the Colorado School of Mines (CSM) under a National Institute for Occupational Safety and Health (NIOSH) funded research project have found that higher flow velocities along the longwall face will increase the pressure differential between the gob and longwall face and allow more methane to be swept from the gob into the active face and tailgate area, thereby diminishing the dilution effect. Increased pressure along the headgate side also allows more oxygen to ingress into the gob area, thereby increasing the amount of oxygen available to form explosive methane-air mixtures in the gob and to support spontaneous combustion of the coal. In this paper, a parametric study is presented to discuss the effect of face air quantity on methane concentrations in the tailgate and formation of explosive gas zones (EGZs) in the gob. Counter to conventional wisdom, increased air quantities at the longwall face may increase the explosion hazard as they result in increased EGZ volumes in the gob, along with increased methane quantities in the tailgate return.

Keywords

computational fluid dynamics, explosive gas zones, spontaneous combustion, gob ventilation boreholes.

Introduction

Longwall mining is an underground mining method that maximizes safe coal production from coal beds that do not have major geological discontinuities. In the western United States, the majority of underground-mined coal is produced through longwall mining. The initial step in developing a longwall mine is to drive development entries around the panel using the continuous miners, leaving behind a solid block of coal with dimensions ranging from 900–6 000 m (3 000 –20 000 ft) in length and 300–500 m (1 000–1 500 ft) in width. Once panel development is complete, longwall machinery is placed along the face of block and longwall mining begins. A continuous flow of fresh air is provided to the face and working sites by the ventilation system. The intake side of the longwall panel is referred to as the headgate, and the ventilation return is through the tailgate. During longwall mining operation, the coal is continuously cut by a shearer and extracted via an armoured face conveyor. Following extraction, the shield roof supports are advanced. As the shields move forward along the panel, the immediate roof is allowed to collapse into the void behind the shields. The collapse of roof rock forms a pile of rock fragments known as the gob. The passing longwall face also disturbs the strata overlying the gob, leading to subsidence.

Methane ignition, explosion, and spontaneous combustion ('spon-com') are major safety hazards in underground longwall coal mines. Safety systems to monitor and control these hazards include; ventilation design, air quality monitoring systems, methane degasification, gob ventilation boreholes, and inertization of the gob. Ventilation systems must be designed to provide sufficient air to the face for workers and to dilute hazardous gases like methane to acceptable levels. The face air can enter into the gob by leaking past the face support shields and may form explosive mixtures as it combines with methane present in the gob. The methane-air mixtures in the explosive range are called explosive gas zones and present an explosion and fire hazard. As the Upper Big Branch (UBB) disaster in 2010 has shown, methane explosions can also lead to coal dust explosions, which can spread to a larger area underground. The fatalities caused by some major methane fires and explosions in underground coal mines of the United States between 2000 and 2010 are shown in Table I. Goodman *et al.* (2008) summarized the

* *Colorado School of Mines, Golden, USA.*

† *University of Engineering and Technology, Lahore Pakistan.*

 Paper received Sep. 2015; revised paper received Apr. 2016.

Table I

Methane ignition accidents in US underground coal mines from 2000 to 2010 (Saki *et al.*, 2015a)

Methane ignition accidents		
Year	**Underground coal mine**	**Fatalities**
2000	Willow Creek Mine	2
2001	Jim Walter Resources, No. 5 Mine	13
2003	McElroy Mine	3
2006	Sago and Darby Mine	17
2010	Upper Big Branch Mine	29

following recent methane ignition accidents. In July 2000, a methane gas explosion occurred in the gob at Willow Creek mine, which caused two fatalities and eight serious injuries. In September 2001, an explosion accident occurred at Jim Walter Resources No. 5 mine in Alabama, though not in the gob, which resulted in 13 fatalities and 3 injured workers. In 2006, an explosion in a sealed, mined-out area at the Sago Mine in West Virginia caused 12 fatalities and 1 injury. In May 2006, a similar accident at the Kentucky Darby Mine resulted in 5 deaths. The most recent explosion at Upper Big Branch in 2010 killed 29 miners. The investigation report by Page *et al.* (2011) revealed that methane from the gob was ignited by the shearer cutting near the tailgate and set off a coal dust explosion which spread through almost 50 miles of mine entries.

Current methane monitoring technologies do not provide information on the gas distribution and flow patterns inside the gob areas. In order to obtain information about gas compositions in the gob, the authors modelled the gas flows inside the gob using computational fluid dynamics (CFD). CFD modelling can predict the gas flow patterns inside the gob and the formation of EGZs. It can also determine the volume and locations of EGZs and oxygen ingress into the gob.

Previous CFD modelling research

Various researchers have used CFD to model gas flows in longwall gobs under different ventilation schemes. Ren, Balusu, and Humphries (2005) used CFD modelling of longwall gobs and concluded that reduction in the face air velocity on the intake side of the goaf will help to reduce the risk of spon-com. Krishna, Balusu, and Manoj (2012) used CFD to study the effect of buoyancy on methane gas distribution in the tailgate region and concluded that a back return system was helpful in reducing the tailgate methane concentration level to below 1%. Brune and Sapko (2012) used CFD to analyse the ventilation and potential methane accumulation and mixing patterns in the tailgate corner area, and recommended keeping the immediate longwall tailgate entry open at least to the nearest inby crosscut so that positive ventilation is maintained inby the face to dilute and carry away any methane released behind the shields.

Researchers at the Colorado School of Mines, under a project funded by the National Institute for Occupational Safety and Health (NIOSH), have modelled various longwall panels and ventilation configurations using the CFD software FLUENT® by ANSYS. Marts *et al.* (2013) determined that a lower face ventilation quantity will reduce the size of EGZs in longwall gobs. They also found that nitrogen injection is more effective from the headgate side than it is from the tailgate side, and that there is a point of diminishing returns where additional nitrogen injection will no longer reduce EGZ volume. Marts *et al.* (2013) recommended a back return arrangement on the tailgate side, which is effective in pushing the tailgate EGZ away from face and reducing the likelihood of ignitions near the face due to the shearer cutting hard rock. Marts *et al.* (2014) also reported that the nitrogen injection and reduction in the face air velocity can effectively control spon-com risk in the sealed gobs. Gilmore *et al.* (2014a) concluded that a bleeder ventilated gob will carry a contiguous explosive fringe on all sides surrounding the gob. Marts *et al.* (2014) found that headgate nitrogen injection may form a 'dynamic seal' separating the oxygen-rich face zone from the methane-rich interior gob without forming an explosive fringe in the case of progressively sealed gob. In this paper, the results of modelling studies with varying quantities of air at the face to dilute the tailgate methane are presented. Also, the communication of gob gases with face air and tailgate return, and the performance of gob ventilation boreholes (GVBs) under varying face air quantities, are analysed.

CFD model creation

The CFD model grid was created using ventilation and air quality data collected from two cooperating mines in the western United States. The collected data consisted of mine layouts, geometric dimensions, lithology, overburden caving characteristics, ventilation operating conditions, and gas concentration measurements. The CFD model panel is 314 m (1030 ft) in width and 39 m (128 ft) in height to account for the caved and fractured zone above the coal seam. Gob ventilation boreholes (GVBs) extend into the fractured zone and terminate at 18 m (60 ft) above the top of coal seam. GVBs are 18 m (60 ft) away from the gateroads, with the first GVB being located about 66 m (220 ft) inby the face. The spacing between the GVBs is 60 m (200 ft). A vertical cross-section of the model geometry is shown in Figure 1. The communication of flow between the gob and the face is modelled through shield leakage.

Ventilation layout

The CFD model was constructed with a 'U' ventilation design for a progressively sealed panel, as shown in Figure 2. In the United States, regulations require the use of a bleeder system (Mine Safety & Health Administration, n.d). Exceptions may be granted if the coal has a tendency to spontaneously combust. Many western US coals are prone to spon-com and several mines in this area use progressive sealing (also referred to as bleederless panels) to reduce the oxygen ingress into the gob. This is accomplished by progressively sealing the headgate side crosscuts as the panel advances. Gilmore *et al.* (2014b) concluded that U-ventilation is an effective method to limit oxygen ingress into the gob to prevent spon-com. The fresh intake air enters the longwall face through the headgate, travels across the face, and leaves through the tailgate.

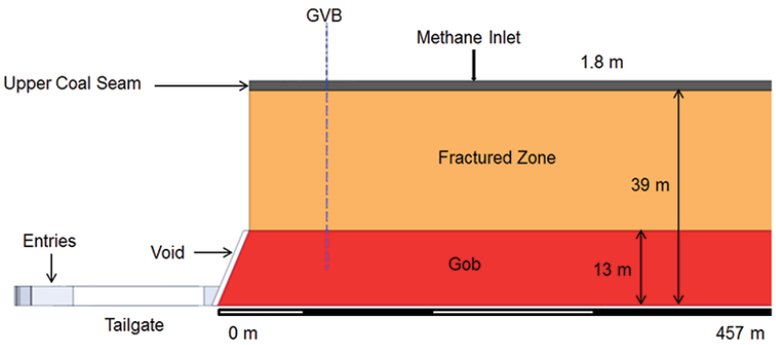


Figure 1—Vertical cross-section of model geometry (Saki, 2016a)

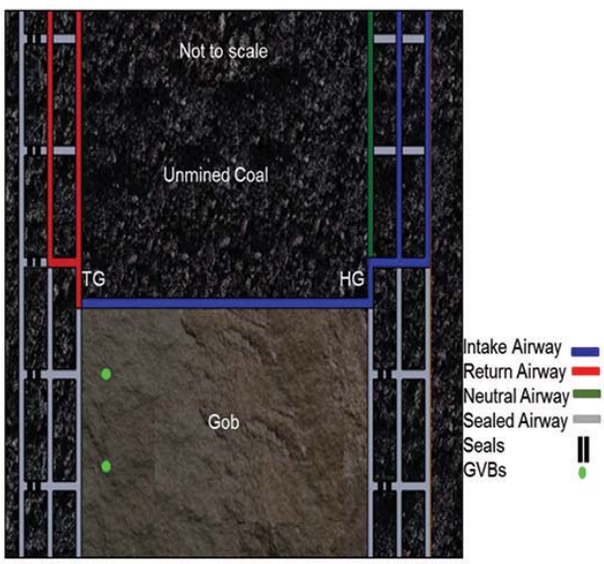


Figure 2—Progressively sealed, U-type ventilation design (Saki *et al.*, (2015b)

Boundary conditions

In the CFD modelling efforts, the longwall face ventilation air quantity was varied from 23 m^3/s (50 000 cfm) to 61 m^3/s (130 000 cfm), with nitrogen injection on both the headgate and tailgate of 0.19 m^3/s (400 cfm). The GVBs were operated at a flow rate of 0.17 m^3/s (350 cfm). A rider coal seam 40 m (130 ft) above the mined horizon was modelled as methane inlet, based on the actual stratigraphy of the cooperating mine. The methane inlet amount for the model was determined from mine measurements of methane released by the ventilation systems and GVBs. The methane liberation rate used for the 460 m (1500 ft.) panel length is 0.5 m^3/s (1040 cfm). The gob permeability is in the range of $2.0*10^{-7}$ to $5.1*10^{-6}$ m^2 and the porosity is 14 to 40%, based on geomechanical modelling using actual mine lithology, shield loading information, as well as final and dynamic subsidence data (Marts *et al.*, 2014b). More information about meshing, model set-up, grid independence, turbulence, convergence, and solution is presented in a previous publication of CSM coal mine ventilation and safety research group (Gilmore *et al.*, 2015).

Explosion hazard characterization

The explosive potential for the mixtures of methane and air is presented using a colouring scheme based on Coward's triangle (Coward and Jones, 1952), as depicted in Figure 3. There are four distinct regions: the explosive region (red), a fuel-rich inert region that can become explosive when fresh air or oxygen is added (yellow), an inert region where no explosive composition is possible (green and dark green), and a fuel-lean inert region (blue). The orange colour denotes an arbitrary, near-explosive zone. Worrall *et al.* (2012) developed an algorithm as a user-defined function (UDF) in FLUENT to correlate the methane-air mixtures with these colours. The contour plots for the plan view of the model are shown so as to visualize the EGZs in a plane at a height of 1.5 m (5 ft) above the mine floor, which is approximately in the middle of the coal seam height.

Model validation

The authors validated the model predictions with the gas measurements conducted by the mine. The cooperating mine used a tube bundle gas analyser system that allowed for measurements along the fringe of the gob. The mine also provided the data for measurements of gases through seal sampling tubes and from manual readings at the face.

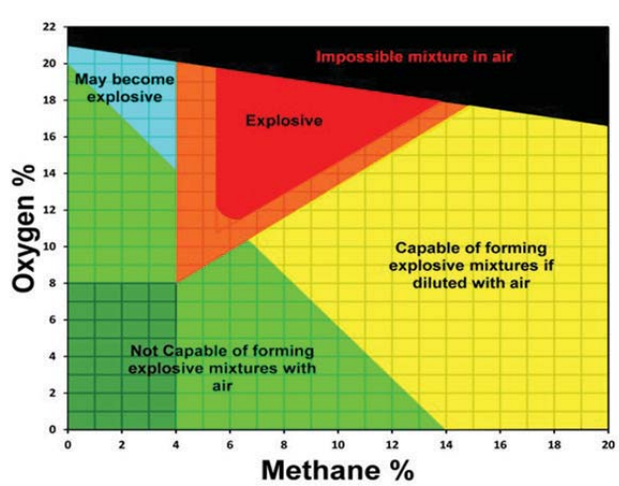


Figure 3—Modified Coward's triangle (Saki *et al.*, 2016b, after Coward and Jones, 1952)

Additionally, discussions with mine personnel were used to validate the results. Oxygen ingress in the model was compared with an actual mine as shown in Figure 4 at actual operating conditions. Figure 4a shows the 12% oxygen contour plot from actual mine data, while Figure 4b shows the oxygen contour plots from the model. The model predictions agree with oxygen measurements taken in the mine. The general results of the CFD models agreed with operator experience. The methane concentration at the tailgate return (0.5%) matched with the mine measurements. The methane concentration of gob ventilation boreholes exhaust predicted by the models also matched with the GVBs methane quality at mine. According to the operator, methane enters the face primarily at the tailgate corner as shown by the models in Figure 8. The operator also reported low oxygen concentration behind the shields inby the tailgate corner, which matched the models results shown in Figure 4b.

Discussion

Figure 5 shows the explosive mixture plots on the left and oxygen ingress plots on the right. All four cases use nitrogen injection of 0.19 m³/s (400 cfm) on both the headgate and tailgate sides. Two GVBs are operating on the tailgate side. Figure 5a shows gas mixtures at a face air quantity of 33 m³/s (70 000 cfm) and Figure 5b shows the results for a face air quantity of 61 m³/s (130 000 cfm). In Figure 5a, a yellow (fuel-rich, oxygen-deficient) zone is visible behind the shields near the tailgate corner. If the oxygen content of this yellow zone increases, it will be capable of forming an explosive mixture. Figure 5a also shows that the nitrogen injection from the headgate (dark green) forms a separation between the oxygen-rich zone near the face and the methane-rich zone in the centre of the gob. Figure 5b shows that with a higher quantity of air on the face (61 m³/s; 130 000 cfm), more face air ingresses into the gob and forms an explosive mixture, visible as a narrow red fringe between the green and yellow areas. The explosive zone extends behind the shields and presents an immediate fire and explosion hazard. The oxygen plots in Figure 5c and Figure 5d show that with 33 m³/s (70 000 cfm) of air on the face, oxygen ingress into the gob is low, whereas with 61 m³/s (130 000 cfm) of air flow on the face, the oxygen ingresses much deeper into the gob. If the coal is present and has a spon-com propensity, such deep oxygen ingress could cause a fire, which would also present an ignition source for the EGZ. In all cases of face air variation from 23 m³/s (50 000 cfm) to 61 m³/s (130 000 cfm), the total EGZs

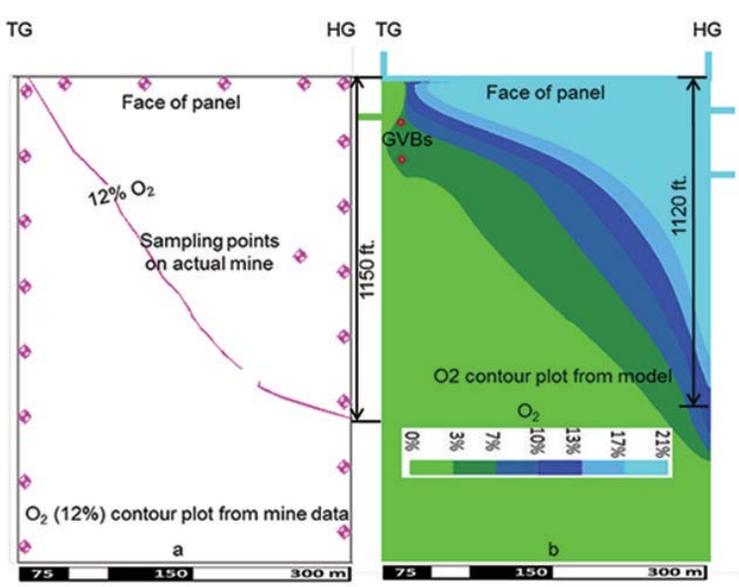


Figure 4—Validation of CFD model predictions of oxygen concentration profiles with oxygen measurements from a mine

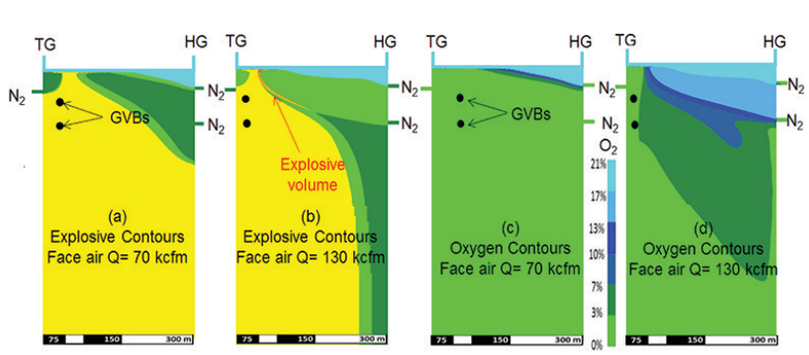


Figure 5—Contour plots of explosive gas and oxygen ingress

volume increased linearly, as shown in Figure 6. It should be noted that the yellow zone, while not explosive, still presents a fire hazard, as fuel-rich mixtures of methane and air can burn with a diffusion flame. Most of the deeper gobs contain little or no oxygen, as shown in oxygen contours plots in Figure 5b and Figure 5c.

Explosive gas zones (EGZs) and methane in tailgate returns

The authors documented the relative volume of explosive mixtures and concentrations of methane in the tailgate return as a function of varying face air quantity. Normally, one would expect linear dilution of the tailgate and face methane with increased face air quantities. However, Figure 6 shows that there is no significant reduction of the tailgate methane concentration (red line) above a certain face air quantity (about 33 m^3/s; 70 000 cfm in this case). Figure 6 shows that the EGZ volume (blue line) in the gob increases with increasing face air quantity. This phenomenon is explained as follows. As the face ventilation quantity is increased, a higher pressure differential is created between the face and the gob and more air (oxygen) will migrate into the gob through the gaps between the shields. Oxygen ingress into the gob will enhance the formation of explosive mixtures. Where the coal has a high propensity for spontaneous combustion, the oxygen ingress may also increase the spon-com potential. As shown in Figure 5b, if that EGZ is located just behind the face, small changes to the gob atmospheric pressure due to a drop in barometric pressure or a roof fall could push the EGZ into the face area.

Gob communication with face and tailgate return

The authors further investigated the phenomenon that increased air flow along the face will not lead to a proportional dilution in tailgate methane concentrations, as shown in Figure 6. There are two related phenomena occurring at two different locations along the longwall face. An increase in pressure as the flow enters the active area will force more oxygen into the gob, but as the flow is developed along the longwall face, the higher velocity causes a greater pressure drop, allowing methane to enter the active face further downstream from the headgate towards the tailgate side. Figure 7 shows that increasing the face air quantity increases the volumetric flow of fresh air into the gob on the headgate side (blue line). This air ingress or leakage flow sweeps increasing amounts of methane from the gob towards the face and into the tailgate. The red line shows the concentration of methane in the flow from the gob towards the face. The trend lines show a near-linear relationship between the face air quantity and the air and gas exchange phenomena in the gob; this observed trend provides insights as to why the concentration of methane does not drop further after a certain face air quantity. The path lines of air flow in the gob are shown in Figure 8, which confirms the flow into and out of the gob towards the tailgate.

Gob ventilation borehole performance

The model contains two GVBs on the tailgate side that are operated at 0.17 m^3/s (350 cfm). The concentration of methane in the GVB exhaust from the modelling predictions is plotted in Figure 9. Methane concentration in both GVBs is near 100% at face air quantities of 33 m^3/s (70 000 cfm) or below. Since GVB 1 (red line) is located near the nitrogen injection point on the tailgate side crosscut, some nitrogen ends up in the GVB, reducing the methane concentration slightly. As the face air quantity is increased above 33 m^3/s (70 000 cfm), the methane concentration in the GVBs decreases due to the face air ingressing into the gob and moving into the GVBs. As the face air quantity is increased, the GVBs start exhausting more air and the methane concentration in the exhaust subsequently decreases. GVB performance provides additional evidence that higher face air quantities lead to deeper oxygen penetration into the gob.

Summary and conclusions

The following conclusions are based on observed trends in the above CFD parametric study on face ventilation.

- By increasing the quantity of air on the longwall face, additional air from the face is pushed into the gob near

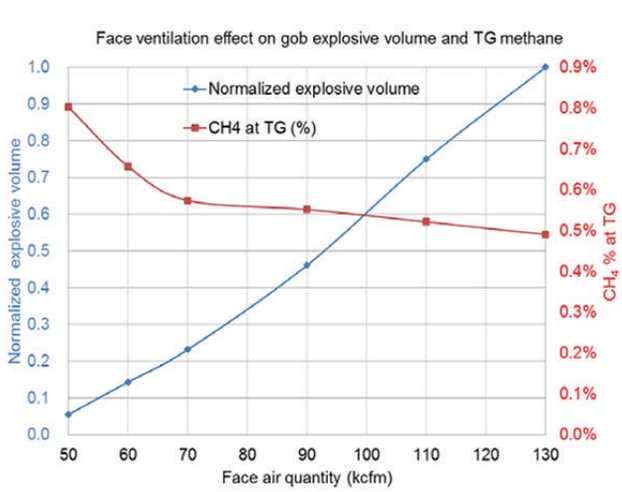


Figure 6—Explosive gas zones volume and CH_4 concentrations for varying face air quantities

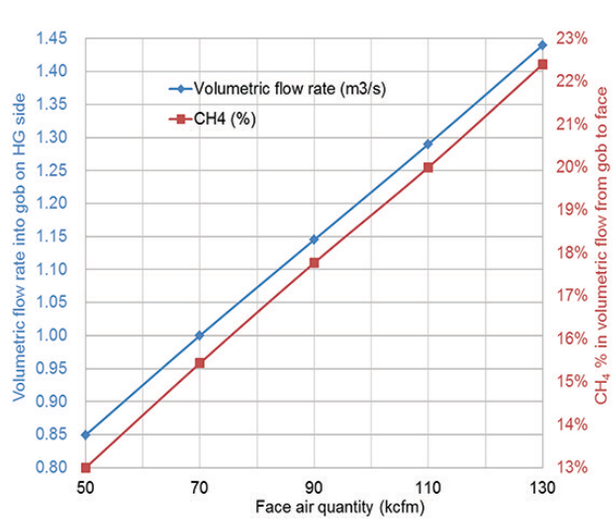


Figure 7—Flow from face to gob and gob to face

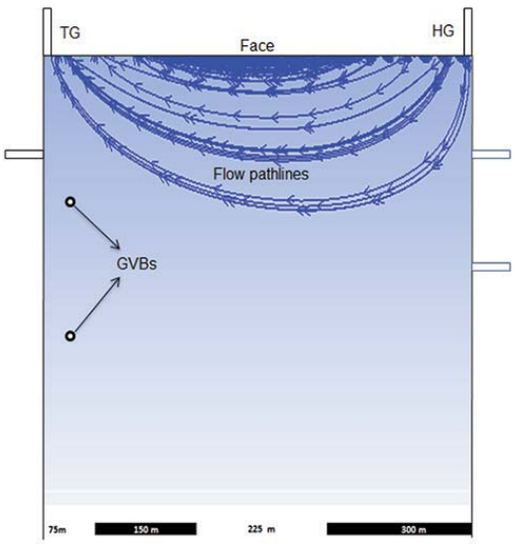


Figure 8—Flow pathlines in the gob

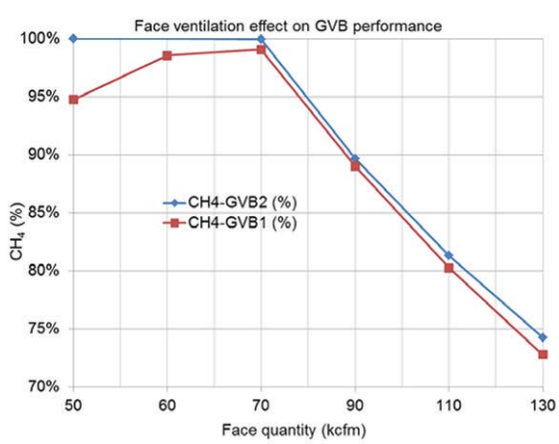


Figure 9—Gob ventilation boreholes performance

the headgate side. In progressively sealed panels, this fresh air ingress tends to sweep methane from the gob into the tailgate. Hence an increased face air quantity does not result in linear dilution of methane on the face. As the face air quantity is increased, a point is reached where the reduction in the tailgate methane concentration becomes insignificant

- As fresh air from the face ingresses into the gob, the volume of EGZs and the explosion hazard increase
- Ingress of more air and oxygen from face into the gob can lead to spontaneous combustion hazards
- Air ingress into the gob may decrease the effectiveness of gob ventilation boreholes.

Acknowledgments

The researchers at Colorado School of Mines gratefully acknowledge the financial support from NIOSH under contract number 200-2009-31409 and the cooperation of mines in the western United States for providing the data for model construction and validation. The modelling skills learned and equipment necessary to complete this project would not be possible without this support.

References

Brune, J.F. and Sapko, M. 2012. A modeling study on longwall tailgate ventilation. *Proceedings of the 14th Annual North American Ventilation Conference*, Salt Lake City, UT.

Coward, H.F. and Jones, G.W. 1952. Limits of flammability of gases and vapours. *Bulletin no*. 503. US Bureau of Mines,

Gilmore, R.C., Marts, J.A., Brune, J.F., Bogin, G.E., Grubb, J.W., and Saki, S.A. 2014a. CFD modeling explosion hazards - bleeder vs. progressively sealed gobs. *Proceedings of the 10th International Mine Ventilation Congress*. Mine Ventilation Society of South Africa, Johannesburg, South Africa.

Gilmore, R.C., Marts, J.A., Brune, J.F., Saki, S., Bogin, G.E., and Grubb, J.W. 2014b. An innovative meshing approach to modeling longwall gob gas distributions and evaluation of back return using computational fluid dynamics. *Proceedings of the SME Annual Meeting and Exhibit*, Salt Lake City, UT.

Gilmore, R.C., Marts, J.A., Brune, J.F., Saki, S., Bogin, G.E., and Grubb, J.W. 2015. Simplifying CFD modeling of longwall gob with modular meshing approach. *Mining Engineering*, vol. 67, no. 3. pp. 68–2.

Goodman, G.V.R., Karacan, C.Ö., Schatzel, S.J., Krog, R.B., Taylor C.D., and Thimons, E.D. 2008. NIOSH research for monitoring and controlling methane at U.S. underground coal mining operations. *Proceedings of the 21st World Mining Congress*, Krakow, Poland.

Krishna, T., Balusu, R., and Manoj, K. 2012. Effect of buoyancy on methane gas distribution and gas control strategies at tailgate region in a gassy coal mine. *Proceedings of the 9th International Conference on CFD in the Minerals and Process Industries*, Melbourne, Australia. CSIRO.

Marts, J., Brune, J., Gilmore, R., Worrall, D., and Grubb, J. 2013. Impact of face ventilation and nitrogen inertization on hazardous gas distribution in bleederless longwall gobs. *Mining Engineering*, vol. 65, no. 9. pp. 71–77.

Marts, J., Gilmore, R., Brune, J., Bogin, G., Grubb, J., and Saki, S. 2014a. Accumulations of explosive gases in longwall gobs and mitigation through nitrogen injection and face ventilation method. *Proceedings of the Aachen International Mining Symposium (AIMS)*, Aachen, Germany. RWTH Aachen.

Marts, J.A., Gilmore, R.C., Brune, J.F., Bogin, G.E., Grubb, J.W., and Saki, S. 2014b. Dynamic gob response and reservoir properties for active longwall coal mines. *Mining Engineering*, vol. 66, no. 12. pp. 41–48.

Mine Safety & Health Administration. Code of Federal Regulations - Title 30: Part 75.334-f. Mine Safety and Health Administration, Arlington, VA. 965 pp.

Page, N.G., Watkins, T.R., Caudill, S.D., Cripps, D.R., Godsey, J.F., Maggard, C.J., Moore, A.D., Morley, T.A., Phillipson, S.E., Sherer, H.E., Steffey, D.A., Stephan, C.R., Stoltz, R.T., Vance J.W., and Brown, A.L. 2011. Report of Investigation, fatal underground mine explosion, April 5, 2010, Upper Big Branch Mine-South, Performance Coal Company, Montcoal, Raleigh County, West Virginia, ID No. 46-08436.

Ren, T.X., Balusu, R., and Humphries, P. 2005. Development of innovative goaf inertization practices to improve coal mine safety. *Proceedings of the Underground Coal Operators' Conference*, University of Wollongong, Australasian Institute of Mining and Metallurgy, Melbourne.

Saki, S.A. 2016a. Gob ventilation borehole design and performance optimization for longwall coal mining using computational fluid dynamics. PhD dissertation, Colorado School of Mines.

Saki, S.A., Marts, J.A., Gilmore, R.C., Brune, J.F., Bogin, G.E., and Grubb, J.W. 2016b. Optimization of gob ventilation boreholes completion parameters. *Proceedings of the SME Annual Meeting*, Phoenix, AZ, 21–24 February.

Saki, S.A., Marts, J.A., Gilmore, R.C.. Brune, J.F., Bogin, G.E., and Grubb, J.W. 2015a. CFD study of face ventilation effect on tailgate methane concentration and explosive mixture of gob in underground longwall coal mining, *Proceedings of the SME Annual Meeting*, Denver, CO, 15–18 February.

Saki, S.A., Brune, J.F., Bogin, G.E., Gilmore, R.C., Grubb, J.W., Zipf, R.K., and Marts, J.A. 2015b. Gob ventilation boreholes design optimization for longwall coal mining. *Proceedings of the 15th North American Mine Ventilation Symposium*, Blacksburg, VA.

Worrall, D.M., Wachel, E.W., Ozbay, U., Muñoz, D.R., and Grubb, J.W. 2012. Computational fluid dynamic modeling of sealed longwall gob in underground coal mine – a progress report. *Proceedings of the 14th Annual North American Ventilation Conference*, Salt Lake City, UT. ◆

http://dx.doi.org/10.17159/2411-9717/2017/v117n3a8

Narrow-reef mechanized mining layout at Anglo American Platinum

by F. Fourie*, P. Valicek*, G. Krafft†, and J. Sevenoaks†

Synopsis

Anglo American Platinum (AAP) is constantly striving towards safe, sustainable, productive, and cost-effective operations. There are many different initiatives, one of which is mechanization, and in particular, mining with extra-low profile (XLP) or ultra-low profile (ULP) equipment in pre-developed stoping areas.

This paper considers the best practices that have already been implemented within AAP's mechanized mining operations and details new-generation XLP and ULP equipment that is anticipated to be able to achieve monthly production rates in excess of 4000 m^2, inclusive of the panel and advanced strike drive areas. The new-generation XLP equipment has completed its production trial and the ULP equipment is entering its proof-of-concept phase.

This paper will discuss the results obtained during the XLP production trial as well as the progress that has been made on the ULP equipment. Emphasis is placed on the higher production levels and greater efficiencies that can be achieved using this equipment. The paper will highlight the importance of the mining cycle as well as the availability of the equipment. It will also examine the new skills sets that will be required in the industry.

Keywords

mechanization, narrow-reef layout, extra-low profile, ultra-low profile, mining cycle.

Introduction

Anglo American Platinum (AAP) is constantly striving towards safe, sustainable, productive and cost-effective operations. There are many different initiatives, among which mechanization, and in particular, mining with ultra-low profile (ULP) and extra-low profile (XLP) equipment in pre-developed stoping areas has been identified.

This paper considers the best practices that have already been implemented in AAP's mechanized mining operations and details the new-generation XLP and ULP equipment that is currently in the production trial and proof-of-concept phases respectively. This equipment is able to achieve a monthly production rate in excess of 4000 m^2, inclusive of the panel and advanced strike drive (ASD) areas, when operating in the narrow-reef mechanized mining layout that was developed by AAP.

Emphasis is placed on the higher production levels and greater efficiencies that can be achieved using this equipment. The paper also highlights the importance of the mining cycle as well as the availability of the equipment. It examines the new skills sets that are required within the industry, and will discuss some of the interim results that have been achieved during the production trials and proof-of-concept trials.

The paper builds on initial analysis and modelling work that was conducted to understand and optimize AAP's existing XLP sections. The learning outcomes were then applied in the development of the ULP equipment.

The extensive modelling exercises demonstrated that improved safety and value can be achieved. The layout and technology discussed in this paper are currently being implemented in test sections at AAP. This paper is an expanded and updated version of the paper presented at the Sixth International Platinum Conference in 2014, and includes the results of the XLP production trial.

Purpose of the mine design

The mine design provides the parameters to be used in the layout of a new logistical mining method. The design criteria will be as practical as possible, and will endeavour to conform with and improve upon existing standards and best mining practices. The stoping method that will be used is an on-reef scattered breast mining system, utilizing ULP mining equipment that has been designed to operate in stoping widths of 0.90–1.20 m. It is important to note that the same mining method can be conducted using XLP mining equipment, which is operating in stoping widths ranging from 1.30–1.70 m.

This paper serves as a guideline to the ULP mining method; however, a 'fit-for-purpose' study must be conducted for each mine and circumstance prior to implementing the method.

* *Anglo American Platinum.*

† *Cyest Analytics.*

Objectives of the new mine design

The introduction of the latest generation ULP and XLP equipment is expected to present the following opportunities.

- Safe operations through the reduction of personnel in the high-risk zone of the stopes
- Separation of machines and personnel through the use of remotely operated machines
- Creation of focused mining on primary development from the production stoping, thus having dedicated teams on development and stoping
- Primary development ahead of the stoping activity.

  This enables:
- Better understanding of the geology in advance
- Ability to install tipping points ahead of stoping, creating 'immediate stoping reserves', thereby enhancing flexibility and reducing production risk
- Improved productivity by having adequate faces available and reduction in re-development to establish faces through the adoption of a scattered breast mining system
- Low capital requirement due to recovering revenue from on-reef development
- Layout allows flexibility between stope sections, thereby enabling better resource sharing and scheduling
- Optimizing of the long-term ratio of machine makeup of the fleet in relation to the production outputs
- Creation of a flexible mining layout, allowing rapid response to market pressures
- Low stoping widths, resulting in higher head feed grades
- High-productivity stopes, resulting in high square metres mined per employee.

Mining method

The mining method is based on the concept that on-reef development takes place on the strike, prior to stoping, thereby ensuring that all the necessary services and infrastructure are in place prior to stoping. This results in an improvement in the overall efficiency of the section and will assist in providing a better understanding of the geology, which in turn will help to ensure better planning for the section before stoping commences. The stoping method that will be used is an on-reef scattered breast mining system, which will utilize ULP equipment that has been designed to operate in stoping widths of 0.90–1.20 m. The method that is described in this paper is based on a reef dipping up to 10°, although initial testing suggests that angles up to 16° can be explored.

The mining method described is suitable for both the UG2 and Merensky orebodies within the abovementioned range. As this is an on-reef mining method, the more uniform the ground conditions and the more limited the reef rolling conditions the better. On steeper dipping reefs, apparent-dip tramming roadways are required.

AAP's existing operations currently operate in reef dip angles ranging between zero and 30° with various ranges in depth. Where applicable, the on-reef pre-development layout outlined in Figure 1 provides an opportunity to choose the stoping technology that is most suited to the specific operations requirements. For the purpose of this paper, we will focus on a scattered breast mining method utilizing ULP equipment. However, it is important to emphasize that the mining method can change based on the operation's stoping width, mine depth, and angle of reef dip, resulting in condition-based mine design criteria (Figure 2).

Table I gives a high-level summary of the various mechanized mining technologies that can be used for stoping.

Narrow-reef mechanized mining layout

The mine layout is based on a pre-developed seven-panel strike section that is mined from either side of each raiseline (scattered breast mining) and is capable of producing >4000 m² per strike.

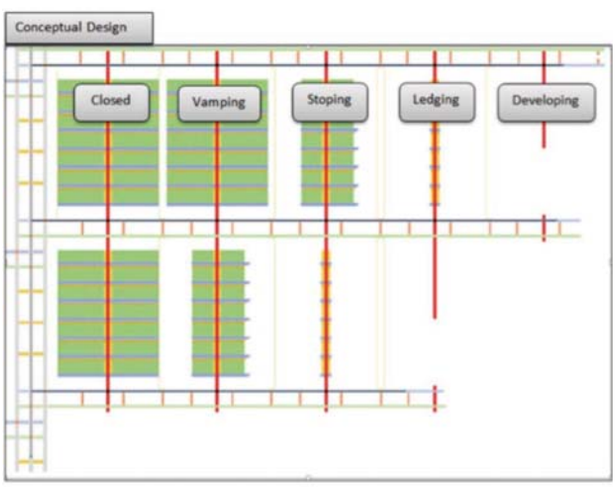


Figure 1—Conceptual design of the mining method

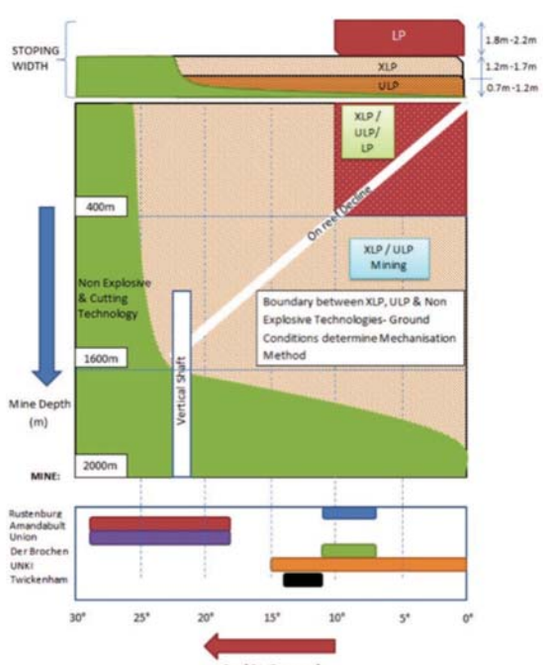


Figure 2—Application of the on-reef pre-development layout at different depths utilizing different stoping methods

Table I

Summary of stoping methods

	LP	XLP	ULP
Depth	0–400 m	350–1800 m	350–1800 m
Stoping width	1.8–2 m	1.3–1.7 m	0.9–1.2 m
Mining method	Bord and pillar	Breast mining	Breast mining
Dip	Approx. 10°	0–22°	0–22°
Production	2 100–3 000 m^2	2 100–3 000 m^2	2 000–3 000 m^2
Orebody	Consistent ore deposits without major faults	Consistent orebody, high extraction ratio	Fairly consistent orebody, high extraction ratio
Advantages	Low-level operating complexity	Less dilution when compared to LP Less waste introduced into plant	High-grade ore. Able to deal with orebody complexities
Disadvantages	High-volume tonnages. Low grade High operating cost. Complex infrastructure	Robust equipment. Labour-intensive. Highly skilled workforce required. Complex infrastructure	Advanced technology. Highly skilled workforce required. Technology in POC phase

Twenty years ago, underground hard-rock mining traditionally made use of conventional mining methods. As the technology used in open pit mining evolved, the knowledge gained was applied to develop underground hard-rock mechanized mining equipment. This resulted in the introduction of low-profile (LP) equipment into AAP's shallow operations (shallow operations are defined as operations up to 400 m in depth with a dip less than 10°), and subsequently in the development of the first generation of XLP equipment. The LP equipment was used for bord and pillar mining, and over the course of the past 15 years there have been significant improvements in both production and safety, with the average monthly production per crew currently being approximately 2400 m^2.

Due to the historically high capital costs of the XLP equipment, as well as the inconsistent results (one month of high production followed by low production the next month) due to inefficiencies (from a logistical and geological perspective) in the traditional breast mining layout, the profit margins of an XLP operation were minimal compared to LP operations.

However, through the introduction of the latest generation XLP and ULP technologies as well as a clearer understanding of mechanized mining principles that was garnered through extensive analysis and modelling, XLP and ULP mining below 400 m utilizing the narrow-reef mechanized mining layout has been shown to create significantly more value from both a financial and an operational perspective, due to the extraction ratios that can be achieved using these methods coupled with the overall efficiencies (>4000 m^2 production per month) of the XLP/ULP equipment.

The focus has therefore been on:

- The change management required to introduce the new XLP and ULP technologies
- Training in mechanization best practices through dedicated training centres
- Introduction of the narrow-reef mechanized mining layout.

Figure 2 provides a high-level overview of the various applications of the narrow-reef mechanized mining layout using various stoping methods. At present, significant work is being conducted within AAP in partnership with Anglo American in developing non-explosive technologies.

The layout consists of three main elements:

- On-reef pre-development
- Ledging
- Stoping (scattered breast mining).

The on-reef pre-development is separate from the stoping activity, which forms an integral part of the stoping layout. The mining layout is designed around the principle of having four active raiselines (development, ledging, stoping, sweeping and vamping) as illustrated in Figure 3. To assist with the rapid development a mobile crushing unit has been developed.

Introduction to the XLP and ULP mechanized mining equipment

The mine design makes extensive use of mechanized mining equipment for on-reef pre-development as well as for the scattered breast mining stoping method. This section provides a brief introduction to the XLP/ULP mining cycle and equipment. The XLP equipment has been available within the industry for a number of years, and therefore greater emphasis is placed on the ULP equipment.

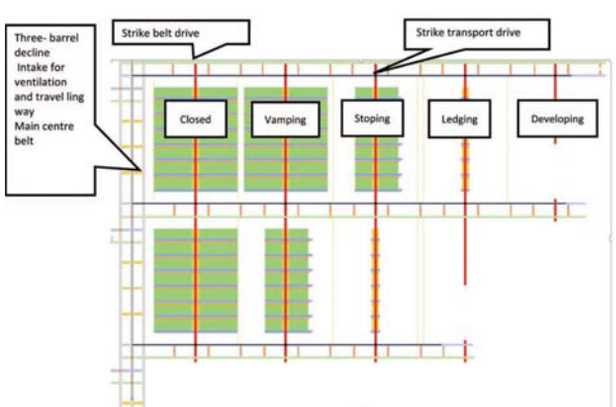


Figure 3—On-reef mechanized layout

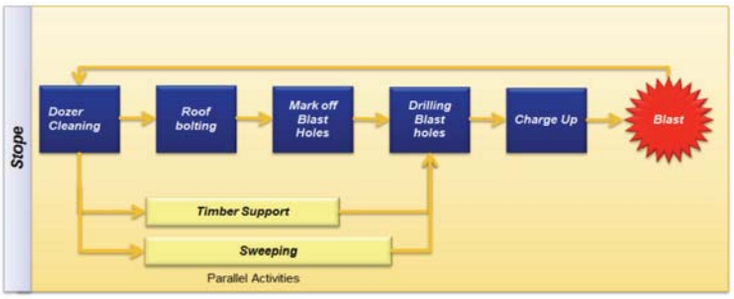


Figure 4—XLP/ ULP mining cycle

XLP/ULP mining cycle

An overview of the mining cycle is given in Figure 4. It is important to note that the XLP/ULP equipment will be used to mine the panels and create the ventilation holings, whereas the low-profile (LP) mechanized equipment will be required to develop the advance strike drives (ASDs) and sidings.

XLP equipment

In 2003 AAP partnered with Atlas Copco, Dok-ing, and Sandvik to develop equipment that was capable of operating in stoping widths ranging between 1.2–1.4 m (Figures 5 and 6).

In 2007 the first XLP test trial was conducted at Amandebult mine. The equipment was later moved to Bathopele mine, where >3000 m^2 has been achieved in a production section using the traditional (single-sided) XLP breast mining method.

Figure 5—XLP drill rig

Figure 6—XLP roofbolter

ULP equipment

In 2003 the first XLP dozer was trialled at AAP. Following this trial, the need to develop a mechanized suite of equipment that would be able to operate in low stoping widths (<1.2 m) was identified. In 2007 a suitable original equipment manufacturer (OEM) was appointed and in 2011 an agreement was signed by AAP to develop the first ULP fleet.

The ULP technology is currently capable of operating in stoping widths ranging between 0.9–1.2 m at depths ranging from 350–1800 m at dip angles of up to 22°. The equipment has been designed to operate in fairly consistent orebodies. Figure 7 shows the ULP fleet

Key milestones in the progression of the ULP technology are:

- Full understanding of mechanization principles
- Modelling and optimization (mining cycle) of XLP and LP mining methods
- Development of a new on-reef mechanized layout (incorporating optimal face length and number of panels).

ULP dozer

The ULP dozer is designed to clean mined material from underground places of work and remove it into the ASD. Figure 8 illustrates the ULP dozer cleaning the panel.

ULP roofbolter

The ULP roofbolter (Figure 9) is a double-boomed roofbolter that has been designed to operate in stoping widths between 0.9–1.2 m at a nominal bolting speed of less than eight minutes between collaring of successive 1.6 m holes.

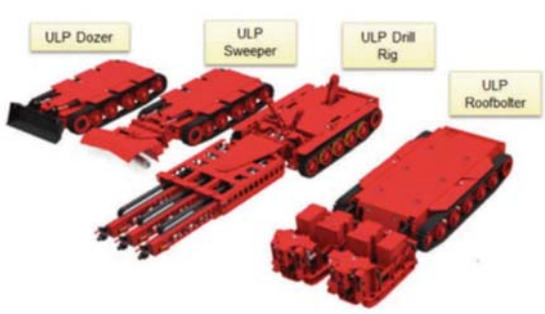


Figure 7—The ULP fleet

Figure – 8 ULP dozer cleaning a panel

Figure 9—ULP roofbolter

Figure 10—ULP drill rig

ULP drill rig

The ULP drill rig (Figure 10) is a three-boomed drill rig that has been designed to operate in stoping widths between 0.9–1.2 m.

KPIs:

- The drill rig has been designed to drill a minimum of two panels in a five-hour working shift
- Power for the drill rig movement and positioning is provided by an onboard electric battery. Depending on the deployed drilling technology and if technically possible, the same battery should be used as a source of energy for drilling. If this is not possible, an electro-hydraulic drilling system shall be deployed. In that case, this system shall be powered by the mine power grid

ULP sweeper

The ULP sweeper (Figure 11) is a cleaning tool intended to facilitate sweeping tasks in locations that the ULP dozer cannot reach. These sweeping tasks are currently performed manually.

KPIs:

The mobile sweeper shall clean the following locations:

- The footwall of the stope between the face and the first line of support
- The accumulated remnant broken rock between the support structures of the first line of support
- The corner where the rock face and the footwall meet.

The unit is thus intended to sweep 10 m behind the face position. The sweeper has been designed to remove remnant rock in the work area.

Mining layout

Main access design

The orebody will be accessed via a barrel decline cluster as illustrated in Figure 12 (for the purpose of this paper, a three-barrel decline cluster has been used). The decline consists of a centre main dip conveyor decline and two transport declines, all of which are used for intake ventilation.

The dip of the three-barrel decline will be a maximum of 10º to accommodate trackless mobile equipment (TME). The spacing of the three-barrel decline cluster and regional and protection pillars will be in accordance with rock engineering requirements.

Figure 11—ULP sweeper cleaning a panel

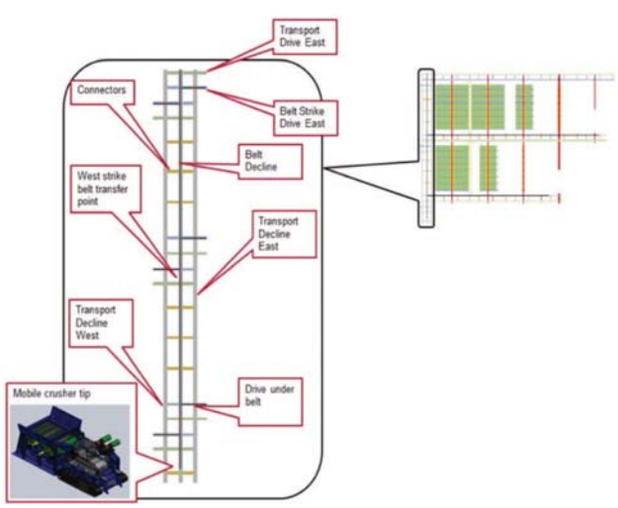


Figure 12—Typical three-barrel decline

One of the transport declines will be a dedicated mine entry transport route and the other a dedicated mine exit transport drive. The drive under the belts will connect the decline transport system to the strike transport system.

Pre-development

The purpose of pre-development is to ensure that all the necessary services and infrastructure are in place prior to ledging or stoping taking place. The pre-development incorporates two strike drives, a raiseline, as well as pre-installed footwall (FW) tips. The layout has been designed to improve the overall efficiency as well as the logistics within the section. By pre-developing the strike before stoping takes place one is also able to gain a greater understanding of the geology, thereby improving the overall planning for the area.

The pre-development layout comprises the following design elements:

- Strike belt drives
- Strike transport drives
- Winze and raiselines
- Connectors
- Double-sided footwall tip.

The strike belt drive and transport drive are developed concurrently at intervals above and below the seven-panel mining area with a footwall tip situated in the strike belt drive on the top and bottom of each raiseline.

The strike belt drive is a designated drive for the belt and mine personnel, while the strike transport drive is a designated drive for machinery. The design makes use of connectors (interconnecting roadways) to facilitate movement between the strike belt drive and strike transport drive during pre-development. The interconnecting roadways ensure the tramming distances during the development of the strike belt, transport drive, and raiseline are kept to a minimum. This is achieved through the introduction of a mobile crushing unit fitted with a sacrificial belt, which is positioned in the strike belt drive.

A more detailed description of the footwall tip and mobile crushing unit is given in the section that follows; with the key design elements summarized in Table II.

Footwall tip

The double-sided footwall tip has been designed to establish flexibility between stope sections, thereby enabling better resource sharing. This is achieved by placing the tip in the footwall, thus making it accessible from either side of the raise. The tip has been designed to allow mechanized equipment to drive over the deck plates, allowing for quick and easy access into other stope sections. The tip has been designed to accommodate tipping by LHDs from both the updip and downdip sides of the raiseline. In order to reduce the tipping time as well as the need to go into the hangingwall, each LHD is fitted with a pushplate. Figures 14 to 16 illustrate the footwall tip and spoon feeder assemblies.

The spoon feeder has been designed to be applied in all AAP's underground operations. The design of the spoon feeder allows material from underground workings to be tipped onto the strike belt at a low trajectory angle by a vibrating or oscillating feed. The oscillating action of the spoon feeder controls the amount of ore that is deposited on the belt, ensuring that there will be no lumping and thus improving the lifespan of the belt.

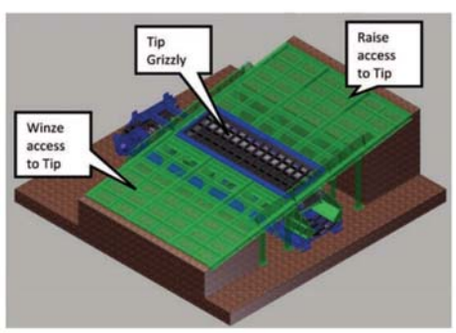


Figure 14—Footwall tip design

Figure 15—Footwall tip spoon feeder assembly

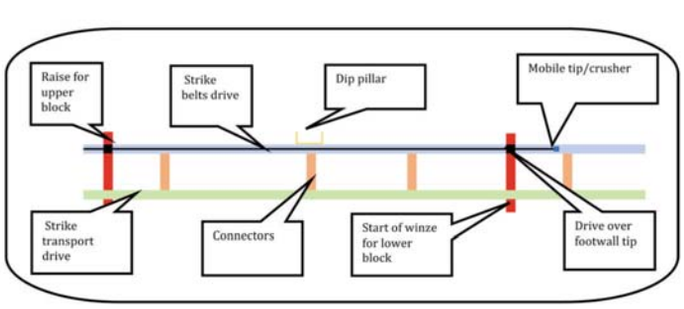


Figure 13—Plan view of pre-development layout

Table II

Key design elements of the new layout

Design element	Safety	Productivity
Pre-developed and proposed stoping layout	• Minimized interaction between machinery and personnel • Designated areas for labour, machinery, and handling lines • Creates safe operations through the reduction of personnel in the high-risk zone of stopes • In-stope remotely operated equipment Minimizes travelling, people, and machinery in an excavation due to development of designated drives for travelling and for the belt	Pre-developed layout ensures logistics infrastructure is ahead • Tips are always in place • Separate pre-development and production tipping points • Max. distance of face to tip not exceeded • Designated crew and fleet for primary development (focused mining) • Better appreciation of geology due to pre- development being well ahead of stoping • Separate primary and secondary development (in conjunction with stoping). Checkerboarding, increasing productivity and utilization • Specialist development teams • Assists planning (better geological information) • Improved m^2 • Better face availability • Construction crew is part of the development crew **Raiseline** Optimal tramming distance of LHDs, resulting in a better utilization due to multiple tipping points (updip and downdip tips resulting in less congestion in the raiseline) **Tips** Tips are able to be leapfrogged once the tip is no longer required to serve the section Ore handling line transporting ore from primary development and stoping continuously
Mobile crusher	**Safety Standards** • Mine Health and Safety Act – Act 29 of 1996 • SANS 10104 • ISO 7731: 2003 • EN 954-1998 • AFRS GUIDELINES • SRMP GUIDELINES Dials and gauges are visible	Design reduces or eliminates re-handling of big rocks • Mobile crusher unit • Modular design ensures that major components can be easily replaced
Double-sided tips	• Designed to create a safer environment around tip **Safety standards:** • Mine Health and Safety Act – Act 29 of 1996 • SANS 10104 • ISO 7731: 2003 • EN 954-1:1998 • AFRS GUIDELINES • SRMP GUIDELINES	• Double-sided tips, therefore accessible either side of the raise • Handles large LHDs without any delay • Quick, easy, and cost-effective tip moves • Major components are easily accessible • Tips are always in place • Pre-development and production tipping points are separated, therefore reducing congestion
Ventilation	Equipment to use low-sulphur diesel Sufficient time to ensure the necessary ventilation infrastructure is in place prior to stoping.	• Ventilation-on-demand system • Machines equipped with catalyst, particulate filters, and Tier-2 type diesel engines
Rock mechanics	Pre-development layout provides the ability to predict ground conditions and potholes timeously	• Development and stoping are separated, ensures that the support always meets the support requirements
Strike transport drive	Designated drive for machinery (strike pillar	Due to pre-development, roadway quality can be separates the two drives) assured, resulting in better tramming speeds and fewer breakdowns
Strike belt drive	Designated drive for the belt and for mine personnel to travel through	• Dedicated area for the belt • Services run • Strike pillar protects the belt

The tip grizzly is designed to allow blasted rock smaller than 300 mm by 300 mm to pass through the tip into the spoon feeder (unless otherwise stipulated). The tip is equipped with a mobile pecker, which is able to break oversized rock into manageable pieces that will fit through the grizzly.

Figure 16 provides a cross-section of an installed footwall tip in the strike belt drive.

Mobile crusher

The mobile crusher (Figure 17) has been designed for underground mines, and will be installed in the centre of the strike belt drive. The crusher is intended to feed crushed material from development workings in a low-trajectory angle

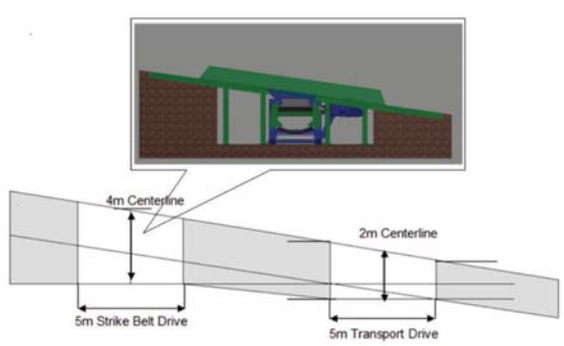


Figure 16—Footwall tip, strike belt drive, and transport drive (section view)

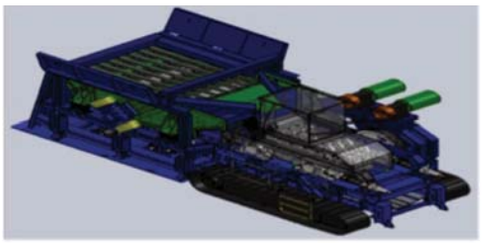


Figure 17—Mobile crusher

onto the strike belt. The mobile crusher has been designed to reduce the tramming distances of the LHD, thereby improving the overall development cleaning cycle. The mobile crusher is accessed from the transport drive via the interconnecting roadway as shown in Figure 18.

Figure 19 illustrates the LHD tipping onto the mobile crusher. The LHD should be fitted with a pushplate to reduce the tipping time.

Pre-development requirements

Pre-development will follow the mining cycle that is summarized in Figure 18. It is important to note that pre-development is done parallel with and independent of stoping.

The pre-development crew's role is to ensure that everything is in place for the stoping team to commence its work. Thus the crew will be responsible for the following:

- Development ends
- Cubbies
- Winzing and raising of the raiseline
- Satellite workshops
- Crosscuts (interconnecting roadways)
- Movement and installation of the tip and belts
- Reticulation extension
- Ventilation extension
- Vent holings (if required)
- Initial drainage.

Ledging

- On a true dip layout the stoping team will perform ledging. For ledging on apparent dip a separate ledging team will be required

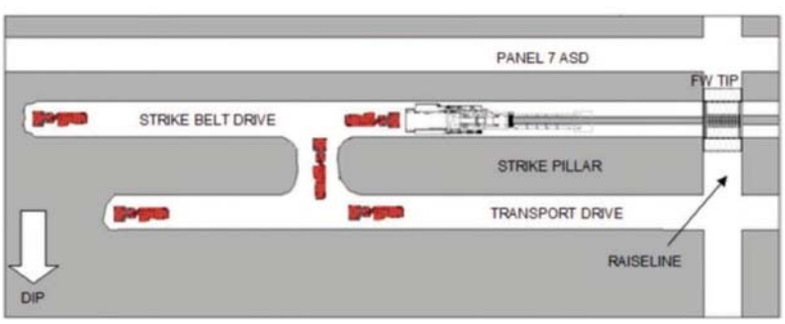


Figure 18—Development in conjunction with a mobile crusher

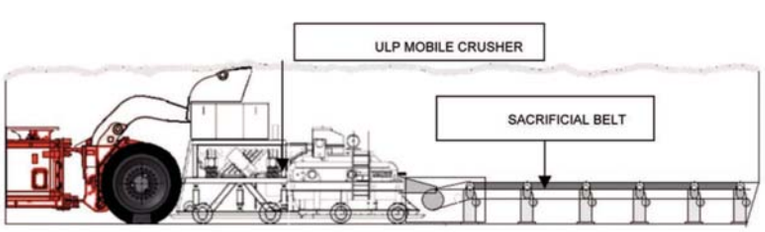


Figure 19—Typical crusher installation indicating the position of the mobile crusher

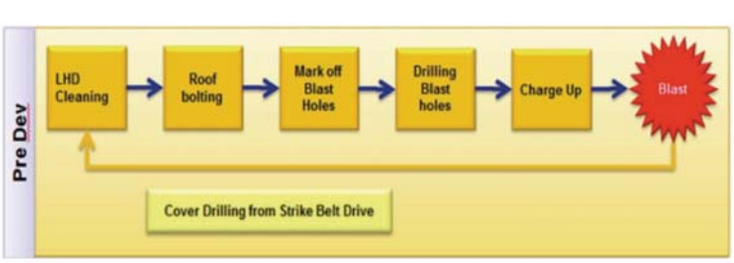


Figure 20—Pre-development mining cycle

- Ledging will proceed from the top to the bottom of the raiseline
- Checkerboarding will be used for ledging to reduce long unsupported spans. The concept of checkerboarding is illustrated in Figure 21
- Off the raiseline, the ASDs and sidings will be blasted 6 m in (two blasts) using LP equipment
- At each alternative ledge face, a 1 m round followed by a 3 m round will be drilled by a LP drill rig positioned in the raiseline. This will be drilled to the required stope height between 0.9–1.2 m
- The blast will incorporate a ramp from the bottom ASD into the ledge so that the ULP equipment can access the ledge. This will predominately be throw-blasted into the ASD, the remainder will be cleaned with the ULP dozer
- The first 1 m of ledging will be supported using the LP bolter

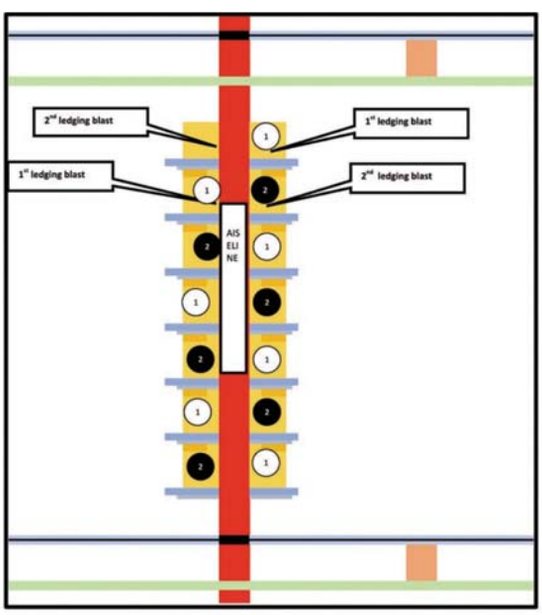


Figure 21—Ledging off the raiseline using checkerboarding

- Once cleaned, the ledge will be supported using the ULP roofbolter (according to rock engineering specifications)
- From the third round onwards, the ULP equipment will be used to advance the ledge until the final primary support has been installed
- While the ledging is taking place the ASDs are kept at least 2 m, but not more than 10 m, ahead of the ledge using the LP equipment
- Permanent strike support is then installed along the raise according to rock mechanics standards
- Once the ledge is completed, then the ledge on the opposite side of the raise can be completed
- Once all ledging has been completed on either side of the raiseline, equipping can commence.

Stoping

The mining method that has been adopted for this layout is scattered breast mining utilizing ULP equipment with stope panels lagging the ASDs. Emulsion explosives will be used for charge-up and blasting.

The stoping layout described in this paper is based on a mining depth of >500 m and therefore a siding has been incorporated into the layout. The stope face is to be drilled using at least a 2 m drill steel and advanced at the very least 1.8 m per blast while operating within stoping widths between 0.9–1.2 m (in this paper a stoping width of 1.1 m has been used).

The ASDs and sidings will advance at least 2.8 m per blast and should be drilled using a 3 m drill steel. Two 12-hour shifts will be worked per day. Blasting is planned to take place on both shifts, although the layout offers an opportunity to create multi-blast conditions. Re-entry period will be as determined by the ventilation department. This results in an effective face time of eight hours and thirty minutes per shift.

The stope panels and ventilation holings will be drilled using a ULP three-boom drill rig. A LP single-boom drill rig will be used to drill the ASD and siding. The ASD will be carried 2–10 m ahead of the stope. An overview of the stoping layout is given in Figure 22 and Figure 23.

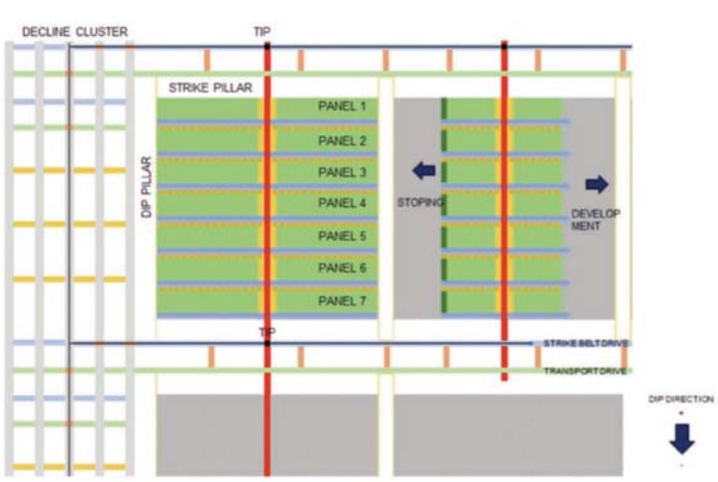


Figure 22—Stoping layout

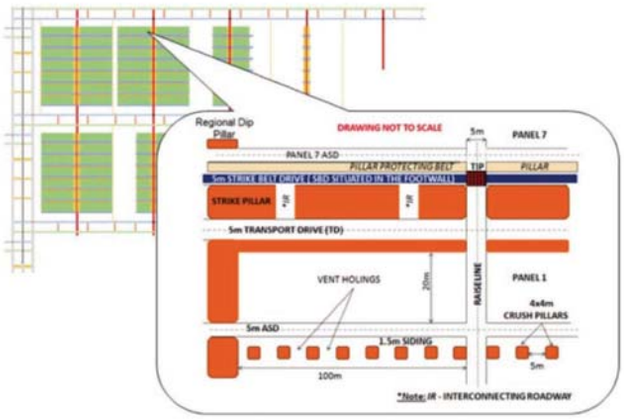


Figure 23—ULP stoping mining

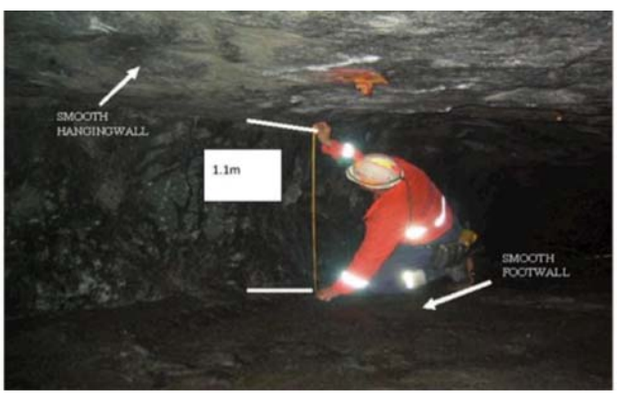


Figure 24—ULP breast mining, stoping width 1.1 m

After the blast, the remaining ore that has not been throw-blasted into the ASD is dozed into the ASD from the stope panels using the ULP dozer. This ore is then trammed using the LHDs to either the updip or downdip footwall tips (refer to Figure 26).

During the first 30 minutes of stope face dozing the LHD must continue with other work so as not to interrupt the dozer cleaning the panel.

Stoping production outlines the anticipated production figures.

XLP/ ULP mining cycle

The mining method will follow the ULP mining cycle that is summarized in Figure 27. It is important to note that the pre-development is done in parallel to and independent of the stope and ASD development.

To blast 4000 m^2, inclusive of the panel and ASD areas, 3.5 to 4 blasts per day (depending on the face configuration) are required for a two-shift cycle operating over 23 days per month. To achieve these blasts the resources available need

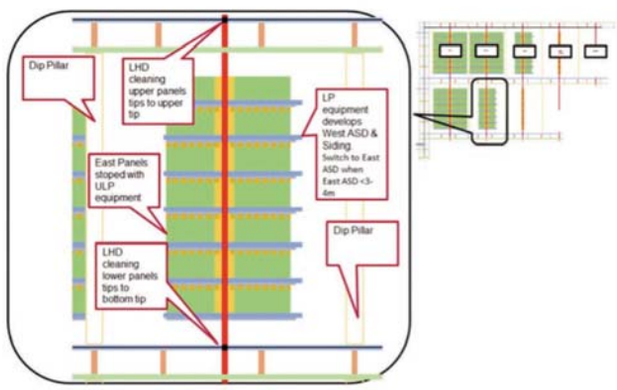


Figure 25—Schematic of ULP stoping layout

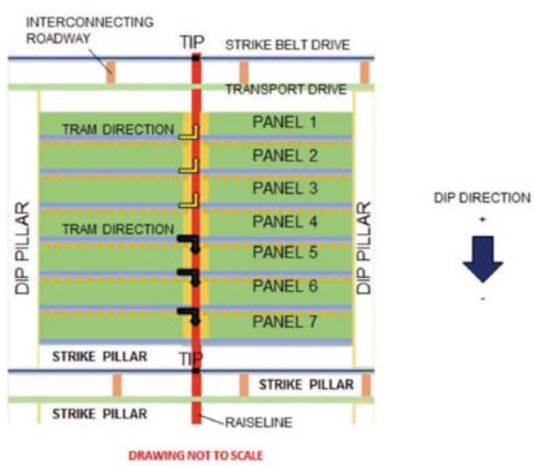


Figure 26—LHD tramming

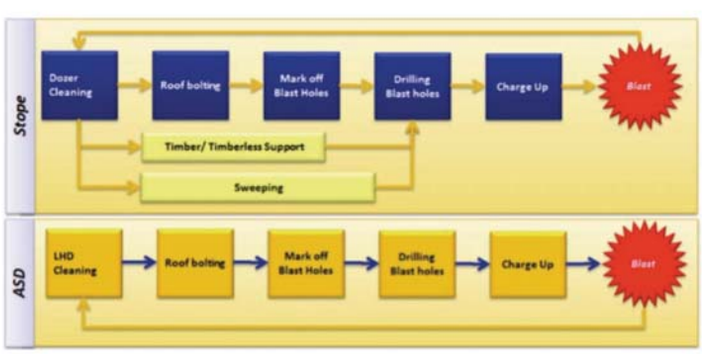


Figure 27—Mining cycle

to be scheduled to best fit within the seven-panel layout and mining logic. Consideration also needs to be given to the respective machines cycles and availabilities.

The general stope mining cycle is summarized in the ULP mining cycle depicted in Figure 27.

Some key sequencing concepts that were used to arrive at the monthly production figure of 4000 m² are as follows:

- Panels 1 to 3 to be trammed to the upper tip, 5 to 7 trammed to the lower tip, and panel 4 can tram to either tip. This is done to reduce and optimize the average tramming distance
- Throw-blasting of 11 m is assumed in the stope panels due to:
  - Breast stopes
  - Electronic detonators
  - Improved explosive technologies (emulsion explosives as well as tamping)
- The ASD needs to be of sufficient size so as to not choke the throw-blasted ore. The LHD needs to load the panel ore from the ASD for at least 2 hours to create sufficient space within the ASD for the dozer to doze the remaining ore into the ASD
- Note that in mechanized mining optimization, when a panel activity is complete, *e.g.* stope roofbolting needs to start at the next planned panel even if the activity will not be completed in the remainder of the shift. The object is to get as much work as possible done by the machines within the available face time. To optimize and control this scheduling, an intelligent management operating system (preferably a real-time system) is required to dispatch the machines and resources optimally. This should be managed from a control room.

Table III

LHD cleaning cycle

LP cleaning cycle Description	Units	LP ASD	LP Siding
Tram tons	t	5 516.1	12 752.57
Effective loader payload	t	6	6
Load time	min	1.5	1.5
Tip time	min	1.5	1.5
Av. speed	km/h	4	4
One-way tram distance	m	138.4	138.4
Cycle time	min/cycle	7.2	7.2
Production trial	min/cycle	√	√
	5.4	5.4	
Cycles per hour	h	8.4	8.4
Tons per hour	t/h	50.3	50.3
LHD hours per month	h	109.6	253.4
Plus 15%	h	126.0	291.4
Scalping and re-handle	h	0.02	0.02
LHD hours per month	h	128.6	297.2
Face advance	m	2.8	1.8
m² per month	m²	891.9	3 108.1
No. of blasts per month		49.0	77.7
Hours per blast	h	2.6	3.8
Hours per loader per blast	h	1.3	1.9

Table III detail the cycle times for the cleaning, support, and drilling cycles. Existing LP and XLP equipment was modelled and a lost time factor was added to the total time based on underground time studies that had been conducted on both the LP and XLP equipment. This time includes time delays caused by changing drill bits, connecting water and power cabling to the machines, and other operational delays.

The design called for at least two 8 t LHDs to be used as optimization studies as well, as Arena modelling showed that the tons per hour reduces as the tram distance increases, which impacts the total tons that can be moved (Figure 28 and Figure 29).

The total face cycle time for both the LP and ULP equipment is a representative time that would apply to the total time it would take to support and drill panels 2 to 7, as these panels all carry a siding as well as vent holing. For the first panel, only the ASD and panel cycle times would be applicable as the first panel does not carry a siding or vent holing (Figure 22 and Figure 23).

Training and development of new skills

Training, as well as the development of new skill sets, has been identified as a vital component for the implementation of mechanization in the mining industry.

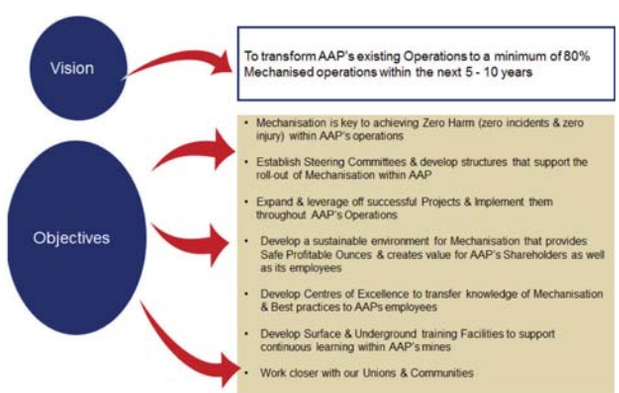


Figure 28—Strategy for the creation of new skills sets within our industry and communities

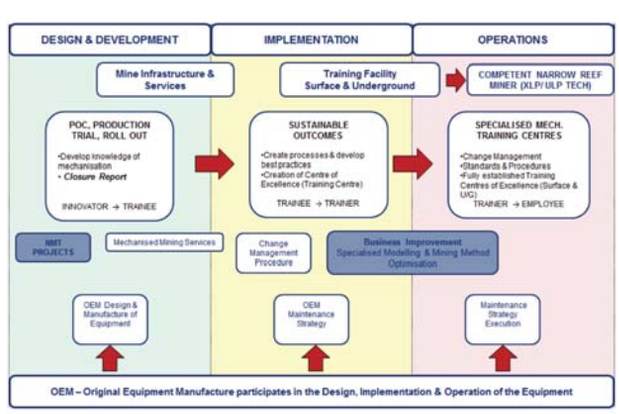


Figure 29—Strategy for the creation of new skills sets within our industry and communities

The results thus far achieved from the XLP production trial clearly demonstrate that higher production levels (>4000 m^2 per month) and greater efficiencies can be achieved with the new-generation equipment that has been developed. However, in order to maximize this potential, new skill sets (Figures 28 and 29) are required to be developed within the industry.

Operators of the future will need to be:

- Strongly focused on safety
- Goal-orientated
- Adaptable and flexible
- Able to think and act strategically
- Effective communicators
- Have a strong sense of teamwork and collaboration
- Visual spatial thinkers
- Capable of performing maintenance on their machines.

In order to create these skills for the future AAP has embarked on the creation of Centres of Excellence as well as experiential learning. Coupled to this, AAP is constantly looking at other innovative and creative ways to identify and garner these skills within the surrounding communities as well as applying best-practice measures taken from our peers in industry.

Conclusions and way forward

The following key milestones in the progression in narrow-reef mechanized mining have been met:

- Full understanding of mechanization principles
- Modelling and optimization (mining cycle) of XLP and LP mining methods
- Development of a new on-reef mechanized layout (incorporating optimal face length and number of panels).

The narrow-reef mechanized mining layout has been designed around (and focuses on) the following four key areas:

- Safety
- Sustainability
- Productivity
- Cost-effectiveness.

Safety

The mining layout is designed to remove personnel from the high-risk zone of the stopes.

- The ULP technology has been designed to separate machines and personnel through the use of remote-controlled technology
- The layout introduces a dedicated drive for machinery (strike transport drive) as well as a dedicated drive for personnel and belt system (strike belt drive)
- The layout allows for future automation of the mining process, which will be managed using a management operating system based in a control room on surface
- Through pre-development the orebody is better understood, resulting in better planning
- The layout requires a skilled workforce that is highly trained and highly safety-conscious
- The mechanized mining method improves underground conditions through the introduction of well-controlled hangingwall and sidewall conditions.

Sustainability, productivity, and cost-effectiveness

- The layout supports concentrated and dedicated mining within a section by divorcing the pre-development from the secondary development and stoping activities. The primary development team is responsible for ensuring that all the necessary services and infrastructure are in place before stoping commences within the section. The layout also offers the potential to switch between explosive (LP/XLP/ULP) and non-explosive stoping methods depending on the dip, depth, and resource width of the orebody
- Higher production levels and greater efficiencies can be achieved using mechanized equipment
- On-reef mechanization delivers early ounces (higher head grade). A majority of the capital and development costs are offset by revenue generated by on-reef development
- Flexible layout – the 14-panel layout has been designed to create flexibility within operations (the layout improves productivity by having adequate faces available and reduces the amount of re-development that is required to re-establish faces). The mining layout is designed around the principle of having four active raiselines (development, ledging, stoping, and sweeping and vamping)
- The layout incorporates logistical elements (optimal tramming distances, improved footwall conditions, tips, services, and IT infrastructure are in place prior to stoping)
- Pre-development results in upfront geological information
- The layout provides effective ventilation and support
- The ULP technology is able to operate in stoping widths of 0.9–1.2 m, thereby reducing the dilution and increasing the grade of the feed to the plant.

Through proper planning and the understanding of mining principles as well as the introduction of new technology, safer, more productive, and sustainable operations are possible in narrow-reef tabular mining systems.

References

Anglo American Platinum Limited. 2011. Building Foundations for the Future through Mining. Anglo American Integrated Report 2011.

Cambitsis, A. and Lane, G. 2012. A framework to simplify the management of throughput and constraints. Cyest Corp. Internal white paper

Valicek P., Fourie, F., Krafft, G., and Sevenoaks, J. 2012. Optimization of mechanized mining layout within Anglo American Platinum. *Proceedings of the Fifth International Platinum Conference: A Catalyst for Change*, Sun City, South Africa, 17–21 September 2012. Southern African Institute of Mining and Metallurgy, Johannesburg. pp. 1–34.

Fourie, F., Valicek, P., Krafft, G., and Sevenoaks, J. 2014. Narrow reef mechanized mining layout at Anglo American Platinum. *Proceedngs of the 6th International Platinum Conference, 'Platinum–Metal for the Future'*, Sun City, South Africa, 20–22 October 2014. Southern African Institute of Mining and Metallurgy, Johannesburg. pp. 1–20. ◆

http://dx.doi.org/10.17159/2411-9717/2017/v117n3a9

Geotechnical characterization of ore related to mudrushes in block caving mining

by R.L. Castro*, K. Basaure*, S. Palma*, and J. Vallejos*

Synopsis

Mudrushes are sudden inflows of mud ore into underground mining facilities. They may cause harm to people and equipment, production delays, dilution of ore, and mine closure. The aim of this work is to characterize, by geotechnical laboratory tests, mud from a block cave and to derive mechanical reasons for the failure of this material while it is being drawn. We used ore samples from extraction points closed due to mudrush potential from the Diablo Regimiento Sector at El Teniente Mine, which represent the three types of mud ore. Characterization showed that the samples comprise gravel and sand with silt and clay grain sizes, whereas the fine particles are classified as low-plasticity silt and clay. Only small differences in fines content, density, and packing were found. The effects of density and degree of saturation on ore strength in unconfined conditions were evaluated. It was found that relative density is the most relevant variable that governs the unconfined strength. Triaxial consolidated tests showed a linear relationship between deviatoric strength and effective confining stress. Deviatoric strength is related to the capacity to flow under triaxial conditions, and consequently it could be related to mudrushes. Unconsolidated, saturated material exhibited a very low deviatoric maximum strength followed by static liquefaction, resulting in residual strength values close to zero. In terms of mining, the geotechnical tests were related to the draw rate and the saturation conditions: a high rate of draw could cause unconsolidated conditions as the strength of the mud decreases close to zero, while a low extraction rate could be related to consolidated conditions where the ore is allowed to release the pore pressure. If the ore is under unsaturated conditions it presents a lower risk as it will develop uniaxial strength.

Keywords

mudrush, block caving, geotechnical characterization.

Introduction

Mining is the process of extracting minerals from the earth, and different mining methods have been developed over time (Lacy and Lacy, 1992). Block/panel caving is an underground mining method based on the action of gravity for ore breakage and transport. It has the lowest cost and the highest production rates of underground methods (Douglas, 1992; Heslop, 2000). In addition, block/panel caving is the safest underground mining method, although it has associated operational risks such as rockbursts, air blasts, and mudrushes (Heslop, 2000). A mudrush is a sudden inflow of mud from drawpoints or other underground openings inside the mine (Butcher *et al.*, 2005). Mudrushes can cause damage to equipment, ore dilution, production delays, injuries, and fatalities.

In block/panel caving mines, an undercut level is developed in the base of the orebody and a production level is constructed some distance below. An arrangement of openings called drawbells is excavated between the undercut level and the production level, and drawpoints are located on the bases of the drawbells. A horizontal ore layer is extracted from the undercut level in order to collapse the orebody above, and broken ore then falls into the drawbells and is managed on the production level. As broken ore is removed from drawpoints, the ore above continues to break and cave (Douglas, 1992). This method is inherently susceptible to mudrushes, owing to the fact that it has all the elements that are necessary: mud-forming material, disturbances, discharge points, and water (Butcher *et al.*, 2000). Mud is formed by fine granular particles and water (Becerra, 2011; Hubert *et al.*, 2000; Jakubec *et al.*, 2012; Lacy and Lacy, 1992; Samosir *et al.*, 2008). Water comes from the surface or from underground sources, and fine granular particles come from surface or from abrasion of ore during caving. The discharge points are the drawpoints on the production level.

* *Univeristy of Chile, Santiago, Chile.*
 Paper received Nov. 2015; revised paper received Jul. 2016.

Mudrushes are triggered by disturbances, which may be dynamic, as in blasting, equipment movements, or earthquakes (Call and Nicholas, 1998; Jakubec *et al.*, 2012), or static as in stable arch collapse above drawbells (Butcher *et al.*, 2000, 2005; Jakubec *et al.*, 2012) and ore drawing (Butcher *et al.*, 2000, 2005; Call and Nicholas, 1998; Jakubec *et al.*, 2012). Previous investigators have postulated two mechanisms for the occurrence of mudrushes. In one mechanism, the ore loses shear strength: an increase in the water content of mud can change its properties (Butcher *et al.*, 2005; Jakubec *et al.*, 2012), decreasing its shear strength. In the other mechanism, stress induces a high water pressure in the mud pores (Call and Nicholas, 1998; Hubert *et al.*, 2000), resulting in a sudden decrease in the shear strength – a phenomenon called static liquefaction (Yamamuro and Lade, 1998) or flow failure (Yoshimine and Ishihara, 1998). The proposed mechanisms have not been proven quantitatively to date.

Call and Nicholas (1998) geotechnically characterized the ore from six samples involved in mudrushes in IOZ mine, Freeport, Indonesia. They found that ore involved in mudrushes had more than 20% of particles less than 2 mm, and could be classified as 'well graded fine gravel with coarse to fine sand', 'poorly graded to silty fine gravel', and 'poorly graded fine gravel to silty fine gravel' depending on each sample. They also compared the samples with other sandy gravel materials and concluded that, at greater than 80% saturation, the material is prone to liquefy due to the excess pore pressure. They performed five unsaturated, consolidated, undrained triaxial tests on a composite from two of the samples, using initial conditions of confining stresses from 50 kPa to 200 kPa, loose compaction, and water contents from 1% to 9%. Results showed that all samples reached 100% saturation after consolidation, and three of them developed liquefaction: instability with an increase in pore pressure (Yamamuro and Lade, 1998). Nevertheless, they did not establish the specific conditions in which liquefaction occurs, because of the small number of triaxial tests and the unspecific initial set of conditions.

In the present paper we study geotechnical properties of mud-type ore from the Diablo Regimiento Sector (DR) of Codelco's El Teniente Mine, Chile. The aim is to obtain geotechnical parameters to describe the ore, characterize the strength and behaviour under different conditions, and use the results to evaluate and quantify the mechanisms for occurrence of mudrushes. We first describe concepts for understanding the results and interpreting the experimentation. This is followed by a description of the material samples and experimental methods. Finally, we discuss the results and the proposed mechanisms for mudrushes based on experimental results.

Framework

Mud ore is known to be a moist, fine-sized granular material; its void ratio and porosity are uncertain because of the density changes caused by the ore flowing through the column and drawbells until it reaches the drawpoints, where the material is extracted, (Kvapil, 1992). Water content is also a variable of mud ore in mine conditions, depending on the permeability of the ore and water sources (Call and Nicholas, 1998). In this section we discuss the physical relationships between water, void, and solid phases in a granular assemblage in order to describe compaction and water content in mud ores. Also, we present concepts to understand the behaviour of granular materials in relation to stress and pore pressure. Minimum and maximum packing densities are properties of granular materials that can be measured in a laboratory, but their values depend on the procedure used (Lambe and Whitman, 1969). Equation [1] represents the relative density, an index used to compare packing of different materials depending on their individual packing properties (Lambe and Whitman, 1969). e_{max} and e_{min} are maximum and minimum possible void ratios for a granular material, respectively, and e is the current void ratio, where void ratio is the ratio of void volume to solid volume (Lambe and Whitman, 1969). At zero relative density the material is in its loosest possible packing state, and a relative density of 100% indicates that the material is in the densest possible state.

$$RD[\%] = \frac{e_{max} - e}{e_{max} - e_{min}} \cdot 100 \qquad [1]$$

Water in a granular material is retained in the voids between particles, and thus the quantity of water that a granular material can hold depends on the available voids. Saturation (Equation [2]) is a concept used to quantify water in a granular material in relation to voids available (Lambe and Whitman, 1969):

$$S = \frac{V_\omega}{V_v} \qquad [2]$$

where $V\omega$ is the volume of water in the granular material and V_V is the volume of voids between solid particles.

Equation [3] shows the relationship between void ratio and saturation, and water content ω and specific gravity G_s, where specific gravity is the unit weight of solid divided by the unit weight of an equal volume of water, and the water content is the ratio of water weight to solid weight (Lambe and Whitman, 1969). This equation is used to describe the relationship between volume and mass indexes.

$$G_s \cdot \omega = S \cdot e \qquad [3]$$

The pore pressure concept mentioned in the previous section in relation to mudrush mechanisms refers to the pressure of water inside pores or voids of a saturated granular material such as soil or other aggregate. Pore pressure depends on the water level above the point where the pore pressure is evaluated, and on the stresses acting on

the saturated granular material. Pore pressure is related to the concept of effective stress, as presented in Equation [4], where effective stress σ' is calculated by subtracting the pore pressure (u) from the total stress (σ). Additionally, it represents stresses supported only by the solid structure of the granular material.

$$\sigma' = \sigma - u \quad [4]$$

For granular materials, there are three types of triaxial tests depending on the conditions of drainage and initial consolidation. These are unconsolidated-undrained (UU), consolidated-undrained (CU), and consolidated-drained (CD) tests (Das, 2002). The drainage condition depends on the rate of shear stress and the permeability conditions in the field. The drained test is used for low-rate loads or high-permeability aggregates in which pore pressure can dissipate when the sample is under stress. The undrained test is for high-rate loads or low permeability, in which pore pressure cannot be dissipated or changed during the application of stress. Consolidation is the process in which water content decreases, with consequent reduction in volume due to the confining stress acting on a saturated granular material (Lambe and Whitman, 1969). Consolidation is complete when all pore pressure has been dissipated from the material, and consequently the water content and volume stay constant. In order to understand the mechanical behaviour, stresses developed in triaxial tests are interpreted using deviatoric (q) and mean stress (p), both depending on the principal stresses. Mean stress is the isotropic (spherical) component of stress, and deviatoric stress is the component that causes the shearing (Schofield and Wroth, 1968). Triaxial tests have well-defined principal stresses, with vertical compression (σ_v) as the major principal stress and confining stress (σ_c) as the minor and intermediate principal stresses (Lambe and Whitman, 1969). Mean stress (p) can also be expressed as mean effective stress (p'). Equations [5] and [6] show the mean effective stress and the deviatoric stress for triaxial test conditions (Schofield and Wroth, 1968), respectively.

$$p' = \frac{\sigma_v + 2 \cdot \sigma_c}{3} - u \quad [5]$$

$$q = \sigma_v - \sigma_c \quad [6]$$

Granular materials such as silty sands can exhibit different deformation behaviours, depending on the initial density and the confining stress. Dilative and contractive behaviour indicate increasing or decreasing rates of volume change, respectively. In undrained conditions, dilative and contractive behaviour can be identified by the decrement or increment of pore pressure respectively. Consequently, according to Equation [4], dilative and contractive behaviour are indicated by the increment or decrement in mean effective stress respectively. Loose sand tends to contract until the mean effective stress reaches a minimum and the behaviour changes from contractive to dilative at a point termed the phase transformation (Yoshimine and Ishihara, 1998). Flow failure or static liquefaction occurs when a confined loose granular material with contractive behaviour exhibits instability or loss of shear strength before the phase transformation (Yamamuro and Lade, 1998).

Methods and material

Samples

Samples for this study were obtained from Diablo Regimiento (DR) Sector at El Teniente mine, which is the largest underground copper mine in the world. It is located 50 km east of Rancagua city in Chile, between 2100 and 2800 m above sea level. The mining method is mechanized block/panel caving using load haul dump (LHD) machines. Mud has appeared in several drawpoints in El Teniente and there have been seven mudrush events since 1989 (Becerra, 2011). In DR, three kinds of mud ore have been identified, according to visual characterization by mine personnel: grey mud associated with sulphide ores, yellow mud associated with oxide ores, and a mixture of yellow and grey mud. Three samples of mud ore were collected from different drawpoints classified as constituting a critical risk (Becerra, 2011). Each sample is representative of one type of mud ore according to mine personnel, as shown in Table I.

Sampling was carried out by mine personnel in charge of water content. Samples were extracted using a hand shovel to take increments of mud on an imaginary horizontal line across the drawpoint 1.5 m from the floor. This method was developed by El Teniente in order to take samples that are representative of ore in drawpoints. Samples were transported in sealed plastic bags to the laboratory, where they were dried and prepared for testing.

Test work

Samples were tested in the Solids Laboratory facilities of the Civil Engineering Department of Universidad de Chile. Tests were carried out to obtain a geotechnical characterization and determine the geomechanical behaviour of mud-type ore. Given the known characteristics of mud ore, ASTM standard tests for soils were selected for geotechnical characterization, one for each sample. Table II lists a summary of the tests.

The tests listed in Table II have different objectives: grain size distribution, specific gravity, liquid and plastic limits; the

Table I

Samples and visual description

Sample	Description	Composition
1	Grey color	Sulphides
2	Yellow color	Oxides
3	Mixed color	Mixture

Table II

ASTM standards for geotechnical characterization

Parameter	Test	ASTM standard	Number of tests for each sample	Total number of tests
Grain size distribution	Sieve analysis	D6913	1	3
Specific gravity	Water pycnometer Water immersion	D854 C127	1 1	3 3
Liquid limit	Casagrandes's spoon	D4318	1	3
Plastic limit	Rolling	D4318	1	3
Minimum density	Pouring in cylinder of known volume	D4254	1	3
Maximum density	Modified Proctor	D1557	1	3
Permeability coefficient	Constant-head permeability	D 2434	3	9
Unconfined strength	Unconfined compression	D2166	9	27
UU confined behaviour	UU triaxial	D2850	4	12
CU confined behaviour	CU triaxial	D4767	9	27

maximum and minimum densities are determined to characterize and classify mud-type ore, and to obtain index geotechnical parameters for use in the geomechanical tests. Unconfined compression tests were carried out to determine unconfined and partially saturated strengths of mud ore, and how the strength changes with different degrees of saturation and compaction. Finally, triaxial tests were performed in order to obtain geomechanical behaviour of mud-type ore in saturated confined conditions, and determine how it changes with different initial compaction grade, initial consolidation, and confining stress.

Grain size distributions were obtained using sieve analysis on 32 kg subsamples separated by quartering. Material was washed with water to separate fines using a 200 mesh (75 μm) sieve as filter. Finally, material was placed on the uppermost sieve and the sieve assembly was vibrated. The fines from cleaning and grains retained on sieves were weighed in order to calculate the grain size distribution of the samples.

Specific gravity testing was carried out using two different methods, depending on the grain size. For particle sizes greater than 4.75 mm we used the water pycnometer method, in which a volume of water is weighed, and then the same volume is filled with a known mass of solid material submerged in water. The mass difference was used to obtain the soil density, which divided by water density gives the specific gravity of the solid. For particle sizes over 4.75 mm, a dry portion of material was weighed and then submerged and its weight under water measured; the dry weight is divided by the difference between dry and submerged weights to obtain the specific gravity.

Liquid limit testing was performed on the fine fraction of samples (less than 0.425 mm). A wet layer of material is added over a specially designed standard spoon, and a groove is cut in the middle. The spoon is then repeatedly dropped from a standard height until the groove is closed. The aim of this procedure is to obtain the water content of two subsamples that require from 15 to 25 drops, two from 25 to 35 drops, and one close to 25 drops, then plot the number of drops *versus* water content and construct a straight line between the points. The liquid limit is the intercept of 25 drops and the straight line. The minimum density test method consists of filling a container of known volume and weight with material using a pouring device to place it as loosely as possible, and then weighing the container with the loose material inside to obtain the density. This procedure is repeated at least three times and the lowest density is selected as the minimum density.

Maximum density is carried out using the modified Proctor test, in which a wet portion of material is compacted in layers in a container of known volume. This procedure is repeated with different water contents to obtain the maximum wet density. Finally, the dry density is calculated from maximum wet density and water content to obtain the maximum density. Maximum and minimum void ratios are calculated using minimum and maximum density respectively with specific density for each sample. The plastic limit was determined only for the fine fractions of samples (less than 0.425 mm). This method involves forming an ellipsoidal mass with 1.5 to 2 g of wet material and rolling it with the hand against a ground-glass plate until it has a uniform diameter of 3.2 mm, then breaking it into several pieces and squeezing it together to repeat the procedure. The mass of material slowly dries with each cycle, and when the mass crumbles after reaching a diameter of 3.2 mm it is placed in a container and its water content, which is the plastic limit of the sample, determined.

Three constant-head permeability tests were performed on each sample in order to determine the permeability coefficient at different relative densities. Samples were compacted to RD values of 30%, 60%, and 70%. The test consists of measuring the time (in seconds) in which a volume (in mm) of water passes through a cylindrical permeameter filled with a saturated sample. Water

temperature is also measured in order to normalize the water viscosity to 20°C. This procedure is repeated at least three times for each test and the results averaged.

Equations [1], [2], and [3] in combination with index parameters are used to set different conditions of saturation degree and compaction for unconfined and triaxial compression tests. An unconfined compression test involves the axial compression of a cylindrical specimen in a strain-controlled compression device until failure. The test ends when there is a decrease in stress or decrease of more than 20% in axial strain. The maximum stress reached during testing is the unconfined compressive strength. Force and axial strain are measured during testing with a precision of 0.001 kN and 0.001 mm respectively. The initial cross-sectional area of the specimen is calculated as the average of nine diameter measurements with a precision of 0.1 mm in order to calculate the stress.

Nine combinations of relative density and saturation degree, as in Table III, were used to perform unconfined compression tests. Ranges for the relative densities and saturation degrees were selected in order to allow the correct selection of cylindrical specimens for testing in unconfined conditions. Specimens were moulded using the known characteristics of each sample to calculate, using Equations [1] and [3], the pertinent water contents. Finally, the moist sub-sample was compacted in five layers using a metal mould in order to attain the density and saturation set previously.

The triaxial test involves the axial compression of a saturated cylindrical specimen with constant isotropic confining stress imparted by the water pressure until 20% axial strain is reached. The specimen is separated from the confining water by a rubber membrane. At the base and top of the specimen, porous discs allow water drainage from the inside of the specimen. Drainage is connected to a valve in order to allow or close water drainage. For a consolidated test there is an initial consolidation stage in which the drainage valve is opened and water outflow is measured with a precision of 1 ml. In the compression test stage, the drainage valve is closed for undrained triaxial conditions, and the pore pressure is measured with precision of 0.1 kPa. Triaxial tests were carried out using three controlled variables: confining stress, initial relative density, and consolidation. Samples were fully saturated for all cases. For consolidated tests, three confining stresses and three initial relative densities were selected. For unconsolidated tests, pore pressure equals confining stress and the initial effective stress is zero for any confining stress. Consequently, the behaviour should not change for different confining stresses, and only two confining stresses are tested to verify this. Samples compacted to 40% relative density showed a very low strength (near zero), therefore tests at zero relative density were not performed and there are only four unconsolidated tests for each sample. Table IV summarizes the conditions for the 13 triaxial tests performed on each sample.

Table III

Relative density and saturation combinations for unconfined compression test on each sample

Relative density [%]	Saturation [%]
20	45
20	60
20	75
60	45
60	60
60	75
90	45
90	60
90	75

Results and discussion

Geotechnical characterization

Geotechnical characterization is summarized using index values from the test results. Table V shows the indexes of ore from the three samples for each test. Grain size distributions are presented in terms of percentage weight passing each size in Figure 1. It can be seen that all samples have similar distributions: well-graded, with coarse sizes mostly under 50 mm and fines with sizes less than 75 μm comprising over 10% by weight of the samples. Sample 2 contains more fines (less than 75 μm) than others – over 20% by weight – and sample 1 has the lowest percentage of fines. Sample 3 has the largest maximum particle size, between 150 mm and 75 mm, compared with the other samples whose maximum particle size is between 75 mm and 50 mm. Specific gravity results from Table V show small differences for the three samples, with sample 2 having the lowest specific gravity. We calculated the specific gravity as the average of the results obtained using the coarse and fine methods, taking note of

Table IV

Conditions of consolidation, relative density, and confining stress for triaxial tests

Consolidation	Initial RD [%]	Confining stress [kPa]
Yes	0	196
Yes	0	392
Yes	0	588
Yes	40	196
Yes	40	392
Yes	40	588
No	40	196
No	40	588
Yes	80	196
Yes	80	392
Yes	80	588
No	80	196
No	80	588

Table V

Summary of geotechnical characterization

Parameter	Test	ASTM standard	Results		
			Sample 1	Sample 2	Sample 3
Grain size [mm]			Percentages in weight passing		
150	Sieve analysis	D6913	100%	100%	100%
75			100%	100%	98.0%
50			98.0%	99.5%	93.0%
37.5			94.0%	97.7%	85.7%
25			82.8%	94.0%	75.5%
19			74.0%	90.0%	69.8%
9.5			55.0%	74.6%	54.2%
4.75			39.8%	62.8%	44.8%
2.36			29.8%	52.7%	36.7%
0.6			19.1%	37.6%	25.7%
0.3			15.9%	31.5%	21.6%
0.15			13.5%	26.6%	18.0%
0.075			11.5%	22.5%	14,8%
Specific gravity	Water pycnometer, water immersion	D854, C127	2.76	2.68	2.72
Liquid limit	Casagrandes's spoon	D4318	21.7%	25.7%	26.1%
Plastic limit	Rolling	D4318	16.9%	21.%	19.1%
Minimum void ratio	Pouring in cylinder of known volume	D4254	0.27	0.28	0.22
Maximum void ratio	Modified Proctor	D1557	0.90	1.00	0.93

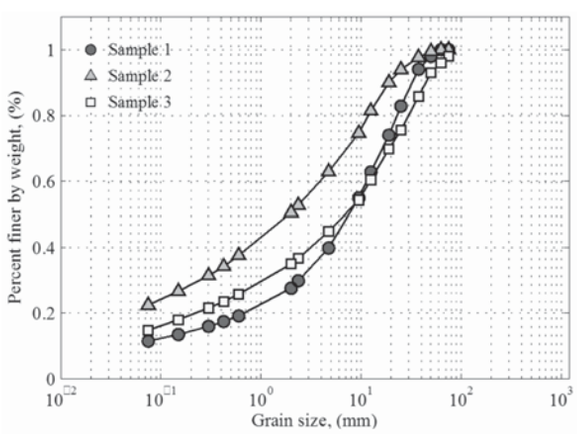


Figure 1—Grain size distribution curves

the minimal differences between measured values of coarse and fine specific gravity (less than 2%) for all samples. Therefore specific gravity does not depend on grain size for the analysed material. Plastic and liquid limits are similar for all samples, which are all classified in the same group according to ASTM (D2487-00) standard: low-plasticity silt and clay. However, the liquid limits of samples 2 and 3 are, respectively 4% and 4.4% higher than the liquid limits of sample 1, and the plastic limits of samples 2 and 3 are 4.1% and 2.2% higher than the plastic limit of sample 1. These differences mean that, with the same water content, fines from sample 1 are more prone to flow than the other samples. Minimum and maximum void ratios are very similar for the three samples, with values between silty sand (from e_{max} = 0.9 to e_{min} = 0.3) and fine to coarse sand (from e_{max} 0.95–0.2) according to typical values (Douglas, 1992). Sample 2 has the highest minimum and maximum void ratios: the maximum void ratio of sample 2 is 12%, 8% higher than samples 1 and 3. Moreover, the minimum void ratio of sample 2 is 6%, 25% higher than samples 1 and 3. This indicates that sample 2 is prone to looser density states, because its both its maximum and minimum density are lower than the values for the other samples. It must be noted that the above indexes could not be related to the strength of the granular mass.

Permeability results are presented in Figure 2. The permeability coefficient is inversely related to relative density (RD). Sample 1 has the lowest values for all RD conditions, which means water can flow faster through sample 1 at the same compaction condition.

Geomechanical behaviour

Unconfined compression tests were carried out at various relative densities and saturation levels. The results show small changes in strength with saturation varying in the range 45% to 75%, and larger strength changes with relative density variations. Averaged unconfined strength values for each sample and RD, at different degrees of saturation, are plotted in Figure 3. For all samples, strength increases with compaction, and the exponential model of Equation [7] is fitted to each sample:

$$S_u = \alpha e^{\beta RD} \qquad [7]$$

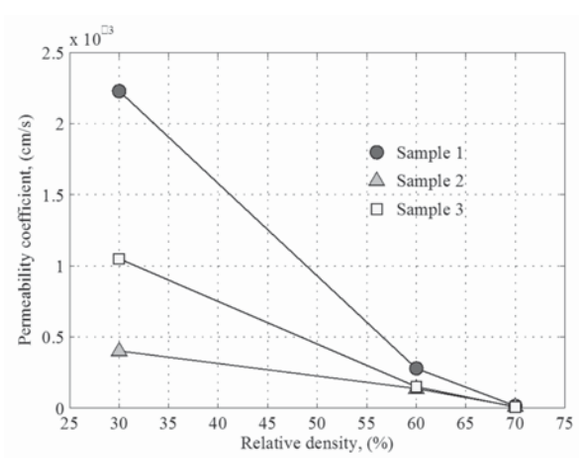


Figure 2—Permeability coefficients as a function of relative density. The solid lines are for visual aid purposes

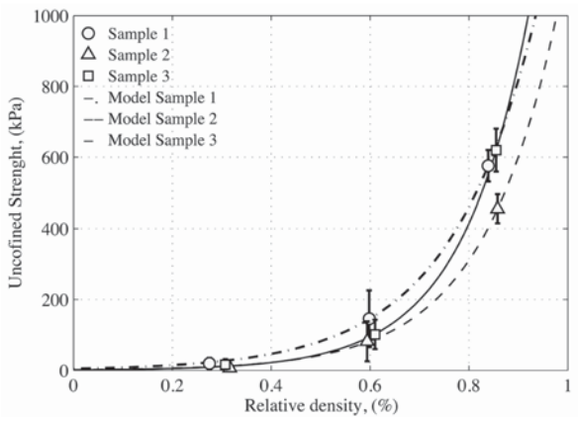


Figure 3—Unconfined compression results. Each point represents the averaged value of strength for three saturations at the same compaction state. Error bars represent the standard deviation for the averaged values at each point

where Su is unconfined strength, α and β are model constants, and RD is the relative density. Sample 2 has less strength than the others at high relative densities. Using the exponential models we can see that samples 1 and 3 have strengths 30% and 49% higher than sample 2 at 95% relative density. Table VI contains the detail of the parameters fitted to unconfined strength values for each sample using Equation [7].

For triaxial testing the compaction conditions in Table IV were set. The real initial relative densities for tests conducted after saturation compaction changes are listed in Table VII, which also shows the final relative densities after consolidation The triaxial test results show the same behaviour for all samples. In order to explain material behaviour, only results from sample 2 are presented. In Figure 4a we observe a rapid initial increase in deviatoric stress for all consolidated tests. The deviatoric stress then stabilizes before passing 5% axial strain, followed by a low rate of increase. In some cases deviatoric stress reaches a maximum value and then slowly decreases to less than 8% of the maximum. Figure 4b shows deviatoric stress *versus* mean effective stress, and a contractive behaviour can be seen with an initial fast increase in deviatoric stress, which stabilizes when mean effective stress reaches the minimum at phase transformation for each test. After phase transformation, a contractive behaviour is exhibited with small increases in deviatoric stress. According to Figure 4a and 4b, the consolidated saturated triaxial strength at phase transformation and maximum strength depends on mean effective stress. It is also possible to see a relationship between relative density and strength. Final relative density after consolidation depends more on confining stress than initial relative density, as seen in Table IX, where we observe that the standard deviation values of initial relative density are 5 to 12 times greater than those of final relative density for the same confining stress. Consequently, the strength at phase transformation and maximum strength depend only on effective confining stress for consolidated undrained conditions. We can use Equation [8] to fit the strength at phase transformation and maximum strength separately for each sample in a linear relationhip that depends only on mean effective stress.

$$q = M \cdot p' \qquad [8]$$

Table VI

Parameters of exponential model for unconfined strength

Sample	A	β	R^2
1	4.660	5.742	0.999
2	1.587	6.596	0.999
3	1.163	7.344	0.999

Table VII

Nominal and real values for relative density in triaxial tests

Initial consolidation	Nominal initial RD	Confining stress [kPa]	Sample 1		Sample 2		Sample 3	
			Real initial RD	Final RD	Real initial RD	Final RD	Real Initial RD	Final RD
Yes	0%	196	31.0%	69.0%	24.8%	71.7%	35.5%	70.2%
Yes	0%	392	27.8%	74.6%	32.4%	76.5%	29.6%	72.9%
Yes	0%	588	28.0%	80.3%	30.4%	78.7%	31.1%	73.9%
Yes	40%	196	40.5%	71.5%	47.2%	73.6%	41.9%	71.8%
Yes	40%	392	42.0%	77.8%	40.6%	78.2%	38.8%	74.9%
Yes	40%	588	45.3%	80.9%	43.3%	80.4%	40.0%	77.5%
Yes	80%	196	58.7%	72.7%	62.8%	74.7%	66.1%	76.3%
Yes	80%	392	68.3%	77.9%	60.5%	79.1%	64.5%	77.6%
Yes	80%	588	63.0%	83.0%	59.9%	81.2%	67.9%	78.9%
No	40%	196	41.4%	41.4%	42.5%	42.5%	40.1%	40.1%
No	40%	588	41.0%	41.0%	42.6%	42.6%	42.3%	42.3%
No	80%	196	65.5%	65.5%	62.5%	62.5%	61.9%	61.9%
No	80%	588	67.0%	67.0%	60.3%	60.3%	60.4%	60.4%

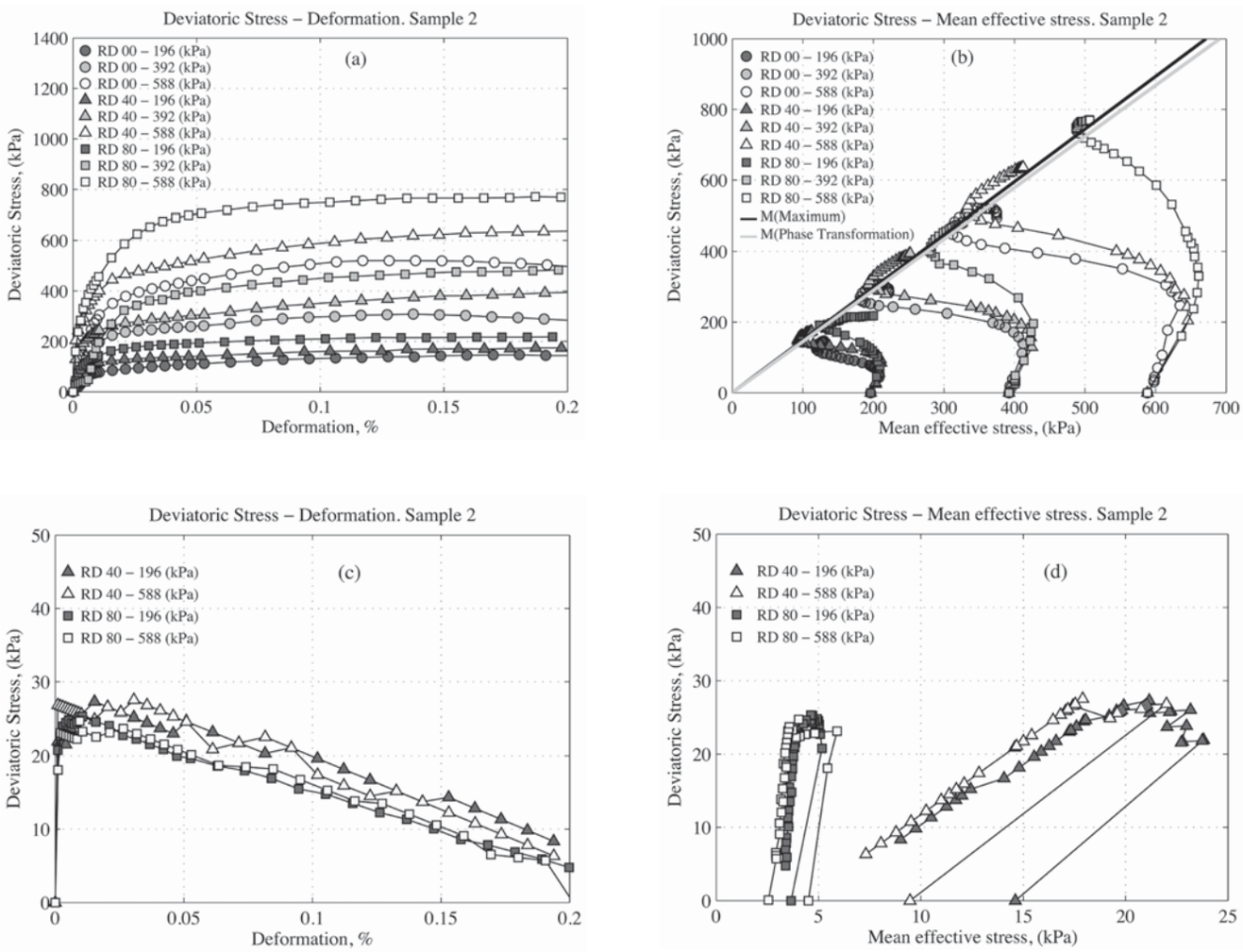


Figure 4—Triaxial results for sample 2. Consolidated and unconsolidated tests are plotted separately in order to improve the clarity of results. Panels (a) and (b) depict the consolidated test results, and panels (c) and (d) represent the unconsolidated test results. Panels (a) and (c) show the deviatoric stress as a function of the deformation curves. Panels (b) and (d) show the deviatoric stress as a function of the mean effective stress

Table VIII

Analysis of average and standard deviation of initial and final relative density for triaxial tests

Confining stress [kPa]	Statistical measure	Sample 1		Sample 2		Sample 3	
		Real initial RD	Final RD	Real initial RD	Final RD	Real initial RD	Final RD
196	Average	43.4%	71.0%	44.9%	73.3%	47.8%	72.8%
	Standard deviation	14.1%	1.9%	19.1%	1.5%	16.1%	3.2%
392	Average	46.0%	76.8%	76.8%	44.5%	44.3%	75.1%
	Standard deviation	20.5%	1.9%	14.4%	1.3%	18.1%	2.4%
588	Average	45.4%	81.4%	44.5%	80.1%	46.3%	76.8%
	Standard deviation	17.5%	1.4%	14.8%	1.3%	19.2%	2.6%

Table IX

Phase transformation and maximum strength models for saturated triaxial tests

Constant	Sample 1	Sample 2	Sample 3
M (maximum)	1.547	1.489	1.544
M (phase transformation)	1.493	1.447	1.448

Fitted parameters of the model for all samples are presented in Table IX. They indicate that all samples have a similar failure curve for strength at phase transformation and maximum strength. The M parameter is obtained by the best fit of Equation [8] to the results. M is 4% higher in samples 1 and 3 than in sample 2 for maximum strength. On the other hand, for residual strength, the M factor is 3% higher in sample 1 than in samples 2 and 3. Unconsolidated tests for

sample 2 are plotted in Figure 4c and Figure 4d. The deviatoric stress of unconsolidated mud increases rapidly to a maximum value and then decreases at a constant rate until it reaches almost zero strength, with maximum values around 30 kPa and residual values between 0.1 kPa and 10 kPa for all samples. These results show a loss of strength after phase transformation, consequently monotonic liquefaction or flow failure occurs due to the high pore pressures developed. Nevertheless, maximum and residual deviatoric strengths obtained in unconsolidated tests are minimal compared with those of consolidated tests, and do not depend on initial compaction or confining stress. Samples with dense compaction show a greater variation in pore pressure during testing, but mean effective stress is very low for all tests, due to the similar pore pressures and confining stress values that were applied in the experiments. Given these points, we conclude that mud-type ore in a saturated condition is more prone to flow under low mean effective stress than under high effective stress. A low mean effective stress condition is possible in the case of low confining stress and also when pore pressure is high according to Equation [4]. In unconsolidated triaxial and unconfined compression conditions, the effective confining stress is zero (or near zero for triaxial conditions). Nevertheless, the strength for unconsolidated material under triaxial conditions is comparable with only the lowest values for unconfined tests, even when packing for unconsolidated tests is medium to high. Values of unconsolidated strength are closer to those for loose unconfined tests, which can be explained by the partial saturation of unconfined tests. When saturation is not full, capillary forces are developed inside the granular structure, giving the ore an apparent cohesion. On the other hand, unconsolidated tests are fully saturated, and consequently there are no capillary forces acting. Also, high pore pressures are developed and flow failure or static liquefaction is present in this condition.

Mudrush mechanisms

In the first sections of this paper we identified different triggers and mechanisms for mudrushes from several authors. Here, considering the experiments conducted so far, we analyse the geomechanical behaviour in order to determine the feasibility of these different mechanisms and triggers. Deviatoric stress has been described as the stress that generates shear in a material, and a material 'flows' or deforms when the shear stresses acting on it exceed the shear strength of the material. Consequently, we assume that the deviatoric strength of the ore is directly related to its capacity to flow. We presented two mechanisms by which mud can lose sufficient shear strength to flow: liquefaction, and increase in water content. Additionally, we established that Diablo Regimiento mud is affected by static liquefaction or flow failure phenomena only in an unconsolidated condition, which means low mean effective stress. Also, we have shown that strength variations with saturation are minimal compared with compaction (RD) changes, and saturation depends not only on water content, but also on void ratio, as can be seen in Equation [3]. At constant saturation, the greater the volume of voids, the greater the water content, hence the results in Figure 3 also show indirectly an exponential increase in strength with a decrease in water content.

The shear strengths of the mud in a saturated condition and in partial saturation depend on effective confining stress and compaction (packing), respectively. Consequently, ore can lose shear strength only if it loses effective confining stress under saturated conditions or becomes looser under unsaturated conditions. Extraction from drawpoints involves loosening of the flowing ore, as demonstrated by the scale models of Kvapil (1992). Therefore the distribution of vertical stresses in zones under draw would reach low vertical stresses or low confined conditions. The greater the mass drawn in a cycle, the less the vertical load to be expected.

This also means that the zones that are not extracted will experience higher vertical loads, which implies compaction (for partial saturation) or consolidation (for full saturation). As mentioned previously, UU and CU conditions for triaxial tests represent extreme cases of consolidation before shearing. We hypothesize that, in the field, a very high rate of draw means no full consolidation of ore, and a low rate of draw gives enough time for full consolidation. As we have seen previously, no consolidation means zero (or very low) effective confining stress. It is important to note that the process of consolidation is always occurring, and a true non-consolidated mud in the field is improbable. We can, however, have cases that are closer to non-consolidated conditions as a result of extraction and the low permeability of the mud. Given these points, high rates of draw can be an important trigger mechanism for mudrush occurrences. To summarize, if ore is saturated and draw is too rapid to allow consolidation, it is then more susceptible to flow than a consolidated ore. On the other hand, if ore is not saturated, it also loses strength with drawing due to the packing becoming looser. Nevertheless, saturated strengths are lower than partial saturation strength, and ore in a saturated condition is always more prone to flow than in an unsaturated condition. In saturated conditions, mean effective stress can also decrease due to increasing pore pressure; this is possible if the water level is increased inside or over the drawbell owing to inhibited drainage of the water from the granular mass. A very compacted mud in the base of drawbells is a very low-permeability medium, which favours the accumulation of water. It is important to realize that mud ore can always flow if it is exposed to a disturbance that exceeds its shear strength.

Conclusions

Geotechnical parameters have been obtained for three samples of mud-type ore, comprising sand and gravel with

silt and clay, from Diablo Regimiento Sector, El Teniente. There are differences between samples in the finest size fraction (less than 1 mm). The oxide sample is finer than the other two samples, and its liquid and plastic Atterberg limits are higher, which means that it is less susceptible to changing its state from solid to plastic and from plastic to liquid with increasing water content. This means that, given the differences in the samples, water content should not be considered as a flow parameter directly, because its influence can change depending on the characteristics of each kind of mud. Unconfined compression tests show no important variation in strength with an intermediate variation of saturation degree (from 45% to 75%), but a very important increase in strength related to an increase in packing density is developed for all samples, which was modelled using an exponential function for each sample. Triaxial tests showed a clear correlation between deviatoric strength and effective confining stress. Tests on unconsolidated material showed almost no deviatoric strength compared with the consolidated condition, and material in this condition is the only one prone to suffer static liquefaction or flow failure. For a mudrush it is necessary for the ore strength to be exceeded under a given stress state. At a high draw rate from the drawpoint, consolidation is not full and stresses are also low. Therefore ore strength is lower than in the case of fully consolidated ore. It is thus easier for a mudrush to occur with a high drawing rate. Water accumulation inside pores of ore within and over drawbells can increase pore pressure in the ore, lowering the effective confining stress and consequently making it easier for its strength to be exceeded and a mudrush to develop with a high level of water over the drawbell. Research on the mudrush phenomenon is in an initial stage and will require, from a geomechanical point of view, a thorough understanding of stresses acting in granular media during flow.

Acknowledgements

This work was supported by Innova Corfo with collaboration of Codelco Chile, El Teniente. The authors also acknowledge the support of the Department of Mining Engineering and the Advanced Mining Technology Center of the University of Chile.

References

Becerra, C. 2011. Controlling drawpoints prone to pumping. *Proceedings of Geomin 2011*, Session 7: Mine Production Geology / Geo-metallurgy.

Butcher, R., Joughin, W., and Stacey, T.R. 2000. Methods of combating mudrushes in diamond and base metal mines. SIMRAC OTH601 Project. Safety in Mines Research Advisory Committee, Johannesburg

Butcher, R., Stacey, T., and Joughin, W. 2005. Mud rushes and methods of combating them. *Journal of the South African Institute of Mining and Metallurgy*, vol. 105. pp. 807–824.

Call & Nicholas, Inc., Freeport McMoRan Copper and Gold, Co., and Hydrologic Consultants, Inc. 1998. IOZ wet muck study.

Das, B. 2002, Soil Mechanics Laboratory Manual (6th edn). Oxford University Press. Chapter 18.

Douglas, J. 1992. Block caving. *SME Mining Engineering Handbook* (2nd edn). Society for Mining, Metallurgy and Exploration, Littleton, CO. Chapter 20.3.

Heslop, T. 2000. Block caving — controllable risks and fatal flaws. *Proceedings of Massmin 2000*, Brisbane, Australia, 29 October–2 November. Australasian Institute of Mining and Metallurgy, Melbourne.

Hubert, G., Dirdjosuwondo, S., Plaisance, R., and Thomas, L. 2000. Tele-operation at Freeport to reduce wet muck hazards. *Proceedings of Massmin 2000*, Brisbane, Australia, 29 October–2 November. Australasian Institute of Mining and Metallurgy, Melbourne.

Jakubec, J., Clayton, R., and Guest, A. 2012. Mud rush risk evaluation. *Proceedings of Massmin 2012*, Sudbury, Ontario, 10–14 June 2012. Canadian Institute of Mining, Metallurgy and Petroleum, Montreal.

Kvapil, R. 1992. Sublevel caving. *SME Mining Engineering Handbook* (2nd edn). Society for Mining, Metallurgy and Exploration, Littleton, CO. Chapter 20.2.

Lacy, W. and Lacy, J. 1992. History of mining. *SME Mining Engineering Handbook* (2nd edn), Society for Mining, Metallurgy and Exploration, Littleton, CO. Chapter 1.1.

Lambe, W. and Whitman, R. 1969. Soil Mechanics. Wiley, New York. Chapters 3, 9, and 27.

Samosir, E., Basuni, J., Widijanto, E., and Syaifullah, T. 2008. the management of wet muck at PT Freeport Indonesia's Deep Ore Zone Mine. *Proceedings of the 5th International Conference and Exhibition on Mass Mining*, Lulea Sweden, 9–11 June 2008. Schunnesson, H. and Nordlund, E. (eds). Lulea University of Technology

Schofield, A. and Wroth, P. 1968. Critical State Soil Mechanics. McGraw-Hill.

Yamamuro, J. and Lade, P. 1998. Steady-state concepts and static liquefaction of silty sands. *Journal of Geotechnical and Geo-environmental Engineering*, vol. 124, no. 9. DOI: 10.1061/(ASCE)1090-0241(1998)124:9(868)

Yoshimine, M. and Ishihara, K. 1998. Flow potential of sand during liquefaction. *Soils and Foundations*, vol. 38, no. 3. pp. 189–198. DOI: 10.3208/sandf.38.3_189 ◆

http://dx.doi.org/10.17159/2411-9717/2017/v117n3a10

A mining perspective on the potential of renewable electricity sources for operations in South Africa

Part I—The research approach and internal evaluation process

by R.G. Votteler* and A.C. Brent†

Synopsis

The business performance of mining corporations in South Africa is adversely affected by the constantly increasing electricity costs. The most commonly used power sources are the national utility supplier via a grid connection, and on-site diesel generators. Previous research has identified the renewable sources of solar photovoltaic (PV), onshore wind, and geothermal energy in hybrid versions with current sources as possible opportunities to counteract rising electricity costs. To provide a clear understanding of the new, developing market of renewable energy sources, this research is divided into two papers. This first paper investigates the internal business structure of mining corporations in order to evaluate electricity generating sources. The multi-criteria decision analysis (MCDA) method is selected as the most suitable approach for this. The paper identifies the criteria used by mining corporations to evaluate possible electricity generation sources for self-generation by means of the corporations' own investment. Four mining corporations with several mining locations were interviewed. The interviews revealed both new and identical evaluation criteria when the findings were compared to earlier MCDA adaptations analysed in the literature review. The second paper (Part 2) combines current knowledge about the external macroeconomic environment with the findings about the internal environment described here. MCDA is adapted in Part 2 and implemented to analyse and compare current to hybrid renewable sources from the perspective of mines.

Keywords

Multi-criteria decision analysis, South Africa, renewable electrical energy, mining.

Introduction

Mining corporations in South Africa are currently adversely affected by increasing electricity prices. Electricity is supplied mainly by Eskom[1] and, especially in remote locations, by on-site diesel generators (Boyse *et al.*, 2014). Electricity price increases, emanating from diesel price hikes and Eskom, have escalated the total operational expenditure on electricity by the largest mining corporations from 8% to 20% of total operating costs in the past seven years (EIUG, 2015). The reliability of electricity supplied by Eskom has decreased drastically (Govender, 2008), and the prices will increase annually by at least 13% until 2018 (Numbi *et al.*, 2014; Eskom, 2015b). In addition, the South African government plans to introduce a carbon tax on greenhouse gas emissions (Alton *et al.*, 2014).

The use of renewable sources of energy has the potential to be an opportunity for mining corporations to reduce long-term electricity costs, diversify energy supply, be less affected by fuel price volatility, decrease greenhouse gas emissions, and show 'green leadership' (Nicolas, 2014). The combination of technological progress regarding renewable sources and factors in the external environment, like increasing fossil fuel prices and social pressure to become greener, increases the attractiveness of renewable sources for mining corporations (Roehrl and Riahi, 2000).

Mining corporations are relatively new customers for renewable energy companies, whose current target customers are mostly governmental organizations and smaller private bodies. Moreover, mining corporations have to be more informed about renewable energy technologies and the possible fit to their specific needs. To optimize the learning process for mining corporations to understand renewable energy technologies, and for energy companies to learn how to approach these potential new customers, more research has to be conducted (Steinhaeuser *et al.*, 2012). One of the outcomes of several renewables and mining summits[2] worldwide is the realization that mining corporations have to become better educated about the concept of renewable energy sources in the context of their unique operational demands (Judd, 2014).

1. South Africa's state-owned electricty provider (Eskom, 2015a
2. Summits of mining corporations and renewable energy companies started in 2013 to develop and discuss the market of renewables in mining operations (Energy and Mines, 2015).

* *School of Public Leadership, Faculty of Economic and Management Sciences, Stellenbosch University, South Africa.*

† *Centre for Renewable and Sustainable Energy Studies, Department of Industrial Engineering, Stellenbosch University, South Africa and Sustainable Energy Systems, Engineering and Computer Science, Victoria University of Wellington, New Zealand.*

 ISSN 2225-6253. Paper received Sep. 2015; revised paper received Jun. 2016.

The purpose of this paper is to contribute more knowledge to this learning process. Previous research has been directed only at analysing the external macroeconomic framework for renewable sources relating to mining operations in South Africa (Votteler and Brent, 2016). The contribution of this paper is to investigate the internal business approach of mining corporations in evaluating electrical energy sources. A strategic method is used to structure the research, as the existing theory and previous applications create a greater research foundation through experience, and which ensures that all aspects are considered in order to to achieve the research objectives. The findings of this paper about the internal evaluation process represent the foundation for future research; with the aim of ultimately combining it with current knowledge about the external framework.

This is the first of two papers on these topics. The reason for dividing the work is initially to create a solid foundation, which is reflected in the structure of this paper. The next section illustrates the research methodology, and this is followed by the selection of an appropriate research approach and an examination of how past applications can contribute to this research. The internal evaluation process of mining corporations is then investigated. Finally, the research results are presented. The objectives are illustrated in Figure 1. The second paper implements the selected research approach from a mining perspective, with the final result of a clear evaluation and comparison of how renewable and current sources perform for mining corporations in South Africa.

Research methodology

Literature review

The first objective was to identify the most suitable method. Based on the work of Petticrew and Roberts (2006), a systematic review was used to comprehensively investigate possible options. Firstly, the paper states the requirements that the method has to fulfil to obtain the best possible solution to the main objective. Research was conducted to select three methods that are most likely to contribute to the requirements. The three methods are introduced and analysed according to the requirements. The last section introduces the selected method and provides the reasoning.

The second objective of this paper was to identify previous approaches of method selection in energy planning in similar cases. Based on Petticrew and Roberts (2006), a state-of-the-art review was conducted. Firstly, previous publications in the energy planning field were sourced and key characteristics contained therein were tabulated. The aim was to investigate the areas in which research had already been conducted, and those in which new adaptations had to be made for the purpose of this research. Secondly, an overview is provided of the results generated in the publications identified. The overview was of assistance in the interviews with mining corporations, as basic background knowledge of applications in similar cases.

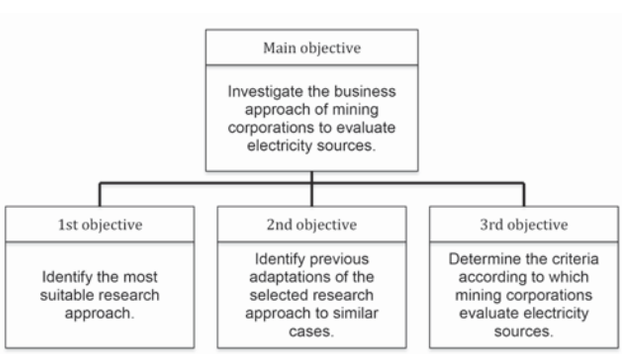


Figure 1—Research objectives

The aim was to provide a substantial overview of the selected research methods in energy planning that were implemented between 2001 and 2015. Subsequently, 26 different papers were identified; 13 purely from a renewable energy perspective, and 13 from a mixed conventional/renewable energy perspective.

Semi-structured interviews

A qualitative research focus was applied in the form of semi-structured interviews. The reason for this approach was the exploratory form of the research. The research included a sample of four different mining corporations. The respondents were decision-makers or people who knew all about the criteria that the corporation used to evaluate possible electricity sources. The respondents were firstly interviewed face-to-face to elicit the information about the evaluation process. The main part of the interview used a post-it session to generate all information in a structured manner. The semi-structured interview questionnaire ensured that all aspects were considered in the session (Newton, 2010).

The identified criteria were used to construct a decision criteria table, which illustrated all the relevant criteria that the mining corporation used to evaluate possible electricity sources. The transformation from the mind-map to the table was conducted according to the requirements for criteria, which are enumerated later. A Delphi technique was used, which meant sending the constructed table back to the respondents via e-mail to obtain confirmation that it reflected their practices accurately. The Delphi technique is in essence a series of sequential rounds—interspersed with controlled feedback – that seek to gain the most reliable consensus of opinion of a group of experts (Powell, 2003).

The questionnaire

A semi-structured interview questionnaire was set up to collect the necessary information for the third research objective. As no approach using the selected method for energy planning from a corporate perspective was found, a qualitative exploratory research technique was used to identify the evaluation criteria. Firstly, the questionnaire would draw accurate information from respondents. Secondly, it would provide structure to the interview. Thirdly, it would provide standards from which facts, comments, and attitudes could be recorded. Lastly, it would facilitate data processing, as answers would be recorded in a common place on each questionnaire (Hague and Jackson, 1995).

Method selection

Requirements of the method

The reason for using a strategic method as the foundation of this research is firstly the existing theory. Extensive academic

research has been done to ensure that the method considers all aspects regarding its overall purpose. Therefore, it provides more assurance in respect of this research that all aspects are investigated. Secondly, previous applications in similar cases contribute to a greater understanding of how the objectives can be met.

This section describes the requirements of the strategic method to best address the main research objective of this paper, namely: to investigate the business approach of mining corporations in evaluating and selecting electricity sources. The requirements were to be used to find the most appropriate basis for research. To be able to successfully investigate the main objective, the strategic method needed to fulfil the following requirements:

- To provide information to mining corporations about the concept of renewable energy in the context of their own unique usage patterns. As described in the Introduction, recent developments have increased the attractiveness of renewable energy. Consequently, research has to be conducted to illustrate to mining corporations how this could fit their specific needs
- To add to the knowledge of renewable energy companies about the needs and business structure of this new type of customer, namely mining corporations. The approach has to create a greater understanding of renewable energy companies and how best to customize the information about electricity sources to the specifications of mining operations
- To investigate the strengths and weaknesses of selected electricity sources. The approach has to be able to contribute to the reasoning why a selected electricity source has or has not been considered appropriate for mining operations. The weaknesses should illustrate what potentially has to change to make it more attractive
- To compare different selected electricity sources according to the specific needs of mining operations. It should be possible to illustrate why certain sources have provided a better fit for mining operations than others. The comparison between currently used and new technologies adds to a better understanding.

Possible methods

This section lists different popular methods that could possibly assist to structure the research. All methods listed in Table I were considered and investigated. The MCDA, the balanced scorecard (BSC), and strategic planning methods were found to be most likely to contribute to the characteristics listed in the previous section. Table I illustrates which requirements the three selected methods fulfilled. The numbers in the top row are linked to the requirements described in the previous section.

Each of the three selected methods is briefly introduced, a possible utilization is described, and a contribution according to these characteristics is discussed.

Strategic planning approach

A strategic planning (SP) model is a tool for businesses to organize their current operations in order to realize the desired future. The model can be seen as a roadmap for the business to get from where it is now to where it wants to be. It is of importance for businesses to create a plan, as it provides clarity on how to achieve the planned goal (King and Cleland, 1987). Every strategic planning model should incorporate seven elements: plan-to-plan (rarely used); mission; needs assessment; strategic objectives; outcome measures; strategies; and performance feed-forward (De Beer, 2000).

The SP approach can be adapted and implemented according to the main research objective. A mining corporation should express in its strategic mission statement the aim to diversify its electricity mix and should specifically set out its approach to adopting renewable sources (Cetindamar *et al.*, 2013).

A shortcoming of the SP approach is that it does not identify strengths and weaknesses, nor does it compare the selected technologies according to the specific demand requirements of mining corporations. Furthermore, it will be influenced strongly by the mining corporations' strategic aims. Another shortcoming – relevant to this paper – is the more strategic nature of the SP approach to improve a company's business coordination (Jakhotiya, 2013). The objective of this research is oriented towards a once-off decision.

Balanced scorecard approach

The balanced scorecard (BSC) is a tool that converts the strategy and mission of an organization into qualitative and quantitative performance indicatorss. The indicators provide the structure for an effective, dynamic, and timely strategic management and measurement system to achieve the overall strategy (Westermann and Sehl, 2006). The scorecard approach identifies elements and requirements that have to be considered in order to follow the strategy with the best possible outcome. The original Balanced Scorecard of Robert Kaplan and David Norton (1992) entails four scoring

Table I

Possible strategic methods

Method	Requirement 1	Requirement 2	Requirement 3	Requirement 4
Multi-criteria decision analysis[1]	✓	✓	✓	✓
Balanced scorecard[2]	✓	×	✓	×
Strategic planning[3]	✓	✓	×	×

(1) Ishizaka and Nemery, 2013; (2) Westermann and Sehl, 2006; (3) King and Cleland, 1987

elements: financial, customer, internal business processes, and learning and growth (Linard and Yoon, 2000).

The BSC principle could have been used to fulfil the main research objective of this paper, with some adaptation for mining corporations that have the goal of becoming involved in renewable energy sources. The adaptation and application would contribute to providing information to mining corporations, as it would show all the criteria that a mining corporation has to fulfil to realize such a project. The BCS approach would contribute to informing mining corporations about renewable energy by illustrating the main requirements that have to be fulfilled when realizing on-site renewable energy projects. The mining corporation would be able to understand in which areas adjustments have to be made or, perhaps, that no realization of objectives would be possible as the requirements are simply not achievable.

The shortcoming here is that the requirements are not based on the specific needs of a mining corporation. Furthermore, the fact that the requirements are based on what the technology can supply, rather than on the demands of the mining corporation, make comparisons difficult as the requirements might differ.

Another problem is that the BSC approach requires the mining corporations to have the initial strategic goal of getting involved in renewable energy (Person, 2013). As the market is still relatively new, mining corporations first have to be informed about the possibilities of renewable energy in catering for their specific needs (Chislett, 2014). Consequently, with adapting and applying the BSC, the criteria used are based on the technology's specification and not the mining corporations' perspective, which limits the informative data. The approach should not illustrate the requirements to realize the technology, but rather how it would work based on the mining corporations' needs.

Multi-criteria decision analysis

Multi-criteria decision analysis (MCDA) is a method that is utilized in making complex decisions. When making complicated decisions it is necessary for the decision-makers to handle a large number of criteria that influence the decision. The MCDA method assists the decision-makers to select the best possible alternative (Ishizaka and Nemery, 2013). The MCDA process is generally divided into three main steps: problem structuring, model building, and approval of the model (Stewart and Belton, 2002).

It was found that the MCDA method could be adapted to the context of the main research objective as it could illustrate to mining corporations which selection among electricity sources would be most suitable. The method would achieve this by utilizing the mining corporations' own evaluation criteria. Firstly, the research would identify the criteria that mining corporations use to evaluate possible electricity sources. Secondly, the type of electricity source and possible uncertainties in the internal and external environment would be identified. Thirdly, based on the identified criteria, an MCDA method would have to be developed to analyse different electricity-generating technologies. The most likely and attractive technologies could then be analysed and evaluated.

The MCDA method developed for the purpose of this research would contribute to providing information to mining corporations by indicating the possible fit of renewable energy. The corporations would be able to understand, according to their own evaluation criteria, what the use of renewable energy entails. However, the method would be developed according to a specific type of electricity source to ensure that the same criteria can be used—which would limit the applicability to technologies. Possible different types of technologies are self-generating sources, like diesel generators, and tri-generation systems, like combined heat and power (CHP).

The renewable energy company would gain more understanding of how mining corporations evaluate such projects. Consequently, it would ease communication about a possible project realization, as information packages about the technology can be customized from the beginning.

The MCDA method enables the mining corporation to identify the strengths and weaknesses of the selected technologies from its own point of view (Stewart and Belton, 2002). A clearer understanding of how the new technologies would perform compared to the present ones would be obtained. It would be possible to compare, according to each criterion, how the different alternatives perform. Close attention has to be paid, however, to ensuring that no external influences are neglected.

Selection of strategic method

The MCDA technique was selected as the most appropriate method to investigate the main research objective. The technique evaluates and analyses electricity options from the perspective of mining corporations. The other two approaches would require the initial aim of the mining corporations to be to implement renewable energy, and would not analyse the technologies according to the mines' own criteria. As the market for renewable sources of electricity is still in its infancy (Chislett, 2014), the MCDA method was considered more suitable.

The renewable energy company would be able to gain the most valuable information from the MCDA method, as it identifies the criteria and structure that are important to mining corporations. In addition, as the technology is new for mining corporations and not for the renewable energy company, it would be more beneficial to analyse the strengths and weaknesses and to compare the technologies from the perspective of the mining corporations.

Another reason is that the SP and BSC approaches are more oriented towards the successful realization of a business strategy than of a physical project. The BSC aims to continually measure, according to selected criteria, how close the performance is to the overall strategic goal. The research content of this paper, on the other hand, aims for a once-off decision.

Lastly, the SP approach involves the investigation of the different technologies, the internal issues, the external influences, the market potential, and a possible scenario analysis. As the market is still in its infancy, the key focus should be on the education of the mining corporations.

Existing applications to similar cases

Previous MCDA approaches to energy planning

To be able to gain an overview and an understanding of how MCDA approaches have been used in energy planning in

recent years, a literature review was conducted. Twenty-six other approaches were identified and summarized in the same way as the MCDA approaches were used, namely according to publication year, energy alternatives selected, final ranking of alternatives, type of criteria used, types of alternatives, main stakeholder (perspective), project size, source of criteria used, and the area of implementation. The results are represented in Table II.

The points below introduce each column in Table II and provide a brief summary of the findings:

- Column 1 firstly enumerates each article of the 26 articles, with the purpose of using them to review the criteria used in energy-planning MCDA approaches (Table III). Secondly, the selection of past MCDA approaches was divided in half: 13 mixed electricity alternatives and 13 purely renewable alternatives. The main reason was to illustrate if there were differences in the selection of the evaluation criteria, which are shown in Table III. Mixed alternatives, besides renewable energy, include other non-renewable energy sources like nuclear, coal, and/or other fossil fuels
- Column 2 gives the author and the publication year of the study. Nineteen (73%) of the 26 articles were published between 2009 and 2014, with the oldest in 2001
- Columns 3 and 4 name all electricity alternatives that were analysed by the MCDA. In the mixed alternatives, *i.e.* articles 1 to 13, the most frequently nominated alternative was wind with 12, followed by solar (11), hydro (11), gas (11), nuclear (10), and coal (9). In nine of the 13 articles, rankings and preferred choices were presented. A maximum of the top four choices are indicated. It is noteworthy that no non-renewable sources featured among the top four approaches. Wind was listed nine times, followed by hydro (8) and solar (6).
- The renewable energy sources (RES), *i.e.* articles 14 to 26, most recurrently selected as an alternative were wind (10 times), followed by solar (9), hydro (9), and biomass (8). In seven of the 13 articles, rankings and preferred choices were presented. Again, a maximum of the top four choices are indicated in Table II. Wind was listed in all seven top ratings, solar in six, and hydro in two. Again, the same types of electricity sources as in the mixed articles were the preferred choices
- Column 5 represents the genre of criteria that the study used to evaluate the alternatives. However, only 16 of the 26 articles clearly subdivided the criteria into genres. With 14 listings, technological and environmental genres are used most frequently, followed by economic (11), socioeconomic-political (6) and social (6)
- Column 6 firstly shows the size of possible projects the MCDA was dealing with. It is divided into small-scale (up to 5 MW) and utility-scale (>5 MW). Secondly, the main stakeholder for the MCDA selection is given. In 19 publications the purpose was to find the best utility-scale alternative for a country's electricity demand, and in one publication for small-scale residential buildings. The project size was utility-scale in 20 articles and small-scale in one. The other articles did not specify the size or main stakeholder
- Column 7 indicates the types of decision-makers who were involved in identifying the evaluation criteria. In 19 MCDA approaches, decision-makers were not specified and a literature review was used. The other seven approaches used different types of decision-makers depending on the objectives
- Columns 8 and 9 state the topic of the MCDA approach and the physical area of implementation. A regional implementation was found in 15 publications. Turkey had the highest implementation rate (5), followed by Spain (2) and Greece (2). Almost half (7) were within the European Union.

Evaluation criteria in previous MCDA approaches

This section provides an overview of the evaluation criteria that were used in the MCDA approaches from articles no. 1 to 26 in Table II. The listing of publications in column 1 of Table II is repeated in Table III to indicate how often criteria appeared in the literature. The most frequently recurring criteria are listed, and the number specifies the publication in which they appeared. In cases where criteria were used less than three times, they were categorized under 'Other'.

The criteria in Table III are divided into categories: technical, economic, and environmental and socio-political. This was based on the results of 'types of criteria' in column 6 of Table II. The technical category addresses the physical characteristics of the electricity alternative. The economic category investigates the financial feasibility. The environmental category evaluates the impact of the alternative on nature. Lastly, the socio-political category measures the influence on the quality of life of people affected by the project (Mateo, 2012).

The criteria used in the MCDA approaches are further subdivided into those used in approaches evaluating, on the one hand, mixed sources and on the other, only renewable energy sources. Criteria are noted only when they occur in more than three papers. The following provides a brief summary of the findings.

- Technical – the criteria indicate that overall 'efficiency' (n=10), 'capacity factor' (n=8), 'reliability' (n=8) and 'maturity' (n=7) are most frequently evaluated. A strong difference in application between mixed and renewable sources can be seen with 'maturity'. It shows a higher regularity in the case of renewable energy sources
- Economic – this category shows that 'investment cost', with n=18, is predominantly used, followed by 'fixed and variable operation, and maintenance costs' (n=11) and 'electricity costs' (LCOE). A significant difference can be seen with the criteria 'service life' and 'implementation period', which are used mostly with the renewable energy sources
- Environmental – the most frequently used criterion is 'external costs', with n=16. The external costs include different types of emissions. Some MCDA approaches specified the various emissions, while others summarized this aspect into one criterion. Further criteria are 'land use' (n=10) and 'noise' (n=4)
- Socio-political – the criteria illustrate that 'social acceptability' and 'job creation' were most regularly used, with n=10. In addition, 'loss of life expectancy' was used in n=6. The criterion 'social benefit' was used only in mixed MCDA approaches.

Table II

Review of MCDA approaches in energy planning

Article/ source	Reference	Energy alternatives	Outcome	Type of criteria	Scale/main stakeholder	Decision-makers identify criteria	Application topic	Area
1 Mix	Stein (2013)	Solar (PV); wind; hydro; biomass; geothermal; nuclear; coal; oil; natural gas	1. Solar PV 2. Wind 3. Hydro 4. Geo-thermal	Financial; technological; environmental; socioeconomic-political	Utility-scale / country	Financial; operational; government; community	A comprehensive multi-criteria model to rank electrical energy production technologies	
2 Mix	Boran *et al.* (2013)	Fossil fuels; geothermal; wind; hydro; natural gas	1. Hydro 2. Wind 3. Gas		Utility-scale / country	Not specified / literature review	A multidimensional analysis of electricity-generation options with different scenarios	Turkey
3 Mix	Ribeiro *et al.* (2013)	Coal; natural gas; hydro gas; wind; hydro			Utility-scale / country	Electrical engineers; mechanical engineers; environmental engineers	Evaluating future scenarios for the power generation sector using a MCDA tool	Portugal
4 Mix	Brand and Missaoui (2014)	Nuclear; coal; solar; wind		Financial; Supply security; socioeconomic; environmental	Utility-scale / country	National electricity utility STEG; Ministry of Industry; Ministry of Environment; National Agency of Energy Conservation; Ministry of Planning and Regional Development; Ministry of Finance	Multi-criteria analysis of electricity-generation mix scenarios	Tunisia
5 Mix	Diakoulaki and Karangelis (2007)	Coal; natural gas; wind; solar			Utility-scale / country	Regulatory Authority for Energy; public power corporation; Climate Change Abatement (National Observatory of Athens)	MCDA and cost-benefit analysis of alternative scenarios for the power generation sector	Greece
6 Mix	Maxim (2014)	Coal; natural gas; CHP; position engine; fuel cell; hydro; wind; geothermal; solar PV; solar thermal; biomass; nuclear	1. Hydro 2. Onshore wind 3. Solar PV	Economic; technological; environmental; sociopolitical	Utility-scale / country	Energy industry professionals	Sustainability assessment of electricity-generation technologies using weighted MCDA	
7 Mix	Afgan and Carvalho (2002)	Coal; solar thermal; geothermal; biomass; nuclear; solar PV; wind; ocean; hydro; gas			No size specified	Not specified / literature review	Multi-criteria assessment of new and renewable energy power plants	
8 Mix	Mamlook *et al.* (2001)	Fossil fuel; hydro; wind; solar; nuclear	1. Solar 2. Wind 3. Hydro		Utility-scale / country	Not specified / literature review	A neuro-fuzzy programme approach for evaluating electrical power generation systems	Jordan
9 Mix	Chatzimouratidisa and Pilavachi (2008)	Coal; fossil fuels; natural gas; nuclear; hydro; wind; solar PV; biomass; geothermal	1. Geo-thermal 2. Wind 3. Biomass 4. Solar PV	Social; socioeconomic-political	No size specified	Utility company; affected communities	Multi-criteria evaluation of power plants' impact on the living standard using the analytic hierarchy process	
10 Mix	Chatzimouratidisa and Pilavachi (2009a)	Coal; fossil fuels; natural gas; nuclear; hydro; wind; solar PV; biomass; geothermal	1. Hydro 2. Geo-thermal 3.Wind 4. Biomass	Technological; economic	No size specified	Not specified / literature review	Sensitivity analysis of technological, economic, and sustainability evaluation of power plants using the analytic hierarchy process	
11 Mix	Chatzimouratidisa and Pilavachi (2009b)	Coal; fossil fuels; natural gas; nuclear; hydro; wind; solar PV; biomass; geothermal	1. Hydro 2. Geo-thermal 3.Wind	Technological; economic	No size specified	Not specified / literature review	Technological, economic and sustainability evaluation of power plants using the analytic hierarchy process	
12 Mix	Topcu and Ulengin (2004)	Hydro; wind; solar PV; biomass; fossil fuels; natural gas; nuclear	1. Wind 2. Hydro 3. Solar PV 4. Biomass	Technological; environmental; economic; political	Utility-scale / country	Not specified / literature review	Energy for the future: an integrated decision aid for the case of Turkey	Turkey
13 Mix	Streimikiene *et al.* (2012)	Nuclear; fuel cell; hard coal; lignite; oil; natural gas; hydro; biomass; solar PV; wind	1. Hydro 2. Solar 3. Wind 4. CHP	Economic; technological; social; political	Utility-scale / country	Not specified / literature review	Prioritizing sustainable electricity production technologies: MCDM approach	European Union
14 RES	Cristobal (2011a)	Wind; hydro; solar PV, thermo; biomass; biofuels	Wind 10–50 MW		Utility-scale / country	Not specified / literature review	A multi-criteria data envelopment analysis model to evaluate the efficiency of the renewable energy technologies	Spain

Article/ source	Reference	Energy alternatives	Outcome	Type of criteria	Scale/main stakeholder	Decision-makers identify criteria	Application topic	Area
15 RES	Heo *et al.* (2010)	Not specified		Technological; market; economic; environmental; policy	Utility-scale / country	Not specified / literature review	Analysis of the assessment factors for renewable energy dissemination programme evaluation using fuzzy AHP	Korea
16 RES	Troldborg *et al.* (2014)	Wind; hydro; geothermal; solar PV; biomass; heat-pump; energy from waste; wave; tidal	1. Solar PV 2. Heat pumps 3. Off-shore wind	Technical; environmental; socioeconomic	Utility-scale / country	Not specified / literature review	Assessing the sustainability of renewable energy technologies using multi-criteria analysis: Suitability of approach for national-scale assessments and associated uncertainties	Scotland
17 RES	Ertay *et al.* (2013)	Hydro; wind; solar PV; biomass; geothermal	1. Solar PV 2. Wind	Technological; environmental; economic; sociopolitical	Utility-scale / country	Not specified / literature review	Evaluation of renewable energy alternatives using MACBETH and fuzzy AHP multi-criteria methods	Turkey
18 RES	Datta *et al.* (2011)	Solar PV; wind; biomass; hydro			No size specified	Not specified / literature review	Green energy sources (GES) selection based on MCDA	
19 RES	Cristobal (2011b)	Wind; hydro; solar-thermal; biomass; bio fuels	1. Biomass 2. Wind 3. Solar-thermal		Utility-scale / country	Not specified / literature review	Multi-criteria decision-making in the selection of a renewable energy project in Spain: the VIKOR method	Spain
20 RES	Kaya and Kahraman (2010)	geothermal; solar PV; wind; hydro; biomass	1. Wind 2. Solar PV 3. Biomass 4. Geo-thermal	Technical; economic; environmental; social	Utility-scale / country	Not specified / literature review	Multi-criteria renewable energy planning using an integrated fuzzy VIKOR and AHP methodology for the case of Istanbul	Istanbul/ Turkey
21 RES	Ahmad and Tahar (2014)	Hydro; solar PV; biomass; biogas; wind; multi solid waste	1. Solar PV 2. Biomass 3. Hydro 4. Wind	Technical; economic; social; environmental	Utility-scale / country	Not specified / literature review	Selection of renewable energy sources for sustainable development of electricity generation system using AHP for the case of Malaysia	Malaysia
22 RES	Tsoutsos *et al.* (2009)	Wind; biomass; hydro; solar PV		Technical; economic; environmental; social	Utility-scale / country	Local authorities; potential investors; local communities; academic institutions; environmental groups; government and European Union	Sustainable energy planning by using multi-criteria analysis for application on the island of Crete	Crete/ Greece
23 RES	Boran *et al.* (2012)	Solar PV; wind; hydro; geothermal	1. Hydro 2. Wind 3. Geo-thermal 4. Solar PV		Utility-scale / country	Not specified / literature review	Evaluation of renewable energy technologies for electricity generation in Turkey using intuitionistic fuzzy TOPSIS	Turkey
24 RES	Lee *et al.* (2012)	Wind		Machine characteristic; economic; environmental; technical	Utility-scale / country	Not specified / literature review	A wind turbine evaluation model under a multi-criteria decision-making environment	Taiwan
25 RES	Wang *et al.* (2008)	Stirling engine; gas turbine; gas engine; solid oxide fuel cell		Technical; economic; environmental; social	Small-scale (residential building)	Not specified / literature review	A fuzzy multi-criteria decision-making model for tri-generation system	
26 RES	Cavallaro (2009)	Parabolic trough; parabolic DSG; SCR molten salt; SCR saturated St.; SCR phoebus; solar hybrid gas; dish-Stirling	1. SCR molten sal 2. Parabolic DSG		Utility-scale	Not specified / literature review	Multi-criteria decision aid to assess concentrated solar thermal technologies	

Impact on the MCDA approach for mining corporations

The purpose of this paper, as stated in the Introduction, is to investigate the internal evaluation process regarding possible electricity sources for mining corporations in South Africa. This will contribute to the foundation of an MCDA approach for mining corporations in South Africa to optimize the evaluation of electricity generation sources. The literature review provides an overview of how MCDA approaches have been used in energy planning in past years. The following points indicate how they contributed to, or affected, the identification of the criteria of this paper.

- Not one MCDA approach in energy planning could be found from the perspective of a corporate entity. Consequently, the previously used criteria can be used only as an indication
- The only MCDA approach to energy planning on the African continent was in Tunisia. No such approach to energy planning was found in South Africa
- The overview of previously used evaluation criteria assists in selecting the criteria for mining corporations. As the marketing of renewable energy to mining corporations is relatively new, respondents might not include all important criteria for renewable sources. The listed criteria of energy planning serve as a checklist and might indicate that further investigations have to be conducted
- The types of criteria used in previous publications provide a solid indication about which are important for the evaluation of electricity generation sources
- The overall preferred selection of solar, wind, and hybrid technologies indicates the renewable alternatives that should be included in the MCDA approach for mining corporations.

Foundations to the internal evaluation process

Type of alternatives to be evaluated

As the main purpose of this paper is to provide more

Table III

Review of criteria used in energy planning MCDA approaches

Criterion	General electricity	Renewable energy	Frequency
Technical			
Efficiency	2, 6, 7, 8, 10, 11	18, 20, 21, 23	10
Reliability	4, 5, 8	15, 16, 17, 18, 22	8
Capacity factor	1, 6, 10, 11, 13	14, 16, 26	8
Maturity		15, 16, 18, 21, 22, 25, 26	7
Available power during peak load	4, 5 ,6	20	4
Safety	8, 9	24	3
Primary energy ratio	10, 11	25	3
Energy efficiency	1	20, 25	3
Others	5	15, 17, 19, 24, 25	
Economic			
Investment cost	1, 2, 3, 5, 7, 10, 11, 13	14, 17, 19, 20, 21, 22, 23, 24, 25, 26	18
Fixed and variable operation, and maintenance cost	1, 3, 5, 10, 11	14, 19, 20, 22, 24, 26	11
Electric cost (LCOE)	2, 4, 6, 7, 12, 13	16, 18, 26	9
Fuel cost	1, 3, 10, 11	20	5
Service life		14, 15, 18, 19, 21	5
Implementation period		14, 17, 19, 21	4
Fuel reserve years (sustainability)	1, 8, 12	23	4
Net present value (NPV)	4	17, 24, 25	3
Net import % of energy (stability)	1, 3, 12		3
Contribution to energy independence	3, 4, 6		3
Others	3, 4, 7, 13	15, 16, 17, 21, 25	
Environmental			
CO_2 emission	*1, 2, 3, 4, 5, 7, 9*	*14, 19, 20, 21, 22, 25*	*13*
NO_x emission	*1, 4, 5, 9*	*20, 25*	*6*
SO_2 emission	*1, 4, 5, 9*		*4*
Particles emission	*1, 4, 9*		*3*
External cost	1, 2, 3, 4, 5, 6, 7, 9, 10, 11, 12, 13	14, 16, 17, 18, 19, 20, 21, 22, 23, 25, 26	23
Land use	3,6,7,9	16,17,20,21,24,25	10
Noise	3	16,24,25	4
Visual impact	3	16,24	3
Others	4	15,18,24	
Socio-political			
Social acceptability	3, 4, 6, 9	15, 16, 17, 20, 21, 22	10
Job creation	1, 3, 4, 6, 9, 13	17, 20, 21, 22	10
Loss of life expectancy (LLE)	1, 6, 9, 13	23, 25	6
Social benefits	3, 4, 8		3
Others	9, 13	17, 25	

information about the criteria used by mining corporations to evaluate possible electricity sources, with the final outcome of being able to compare renewable with current sources, it is important to define the exact characteristics of the types of alternatives available. The purpose of specifying the selection standards for alternatives is to create more transparency in the process of analysing and evaluating them by comparing them with each other (Stewart and Belton, 2002). In cases where alternatives are too different in nature, it becomes more difficult and less informative to compare them according to the same criteria (Keeney, 1996).

Earlier research by Votteler and Brent (2016) investigated the external macroeconomic environment to reveal the potential of renewable sources for mining corporations in South Africa. Based on this research and the previous MCDA applications in energy planning, solar PV,

onshore wind power, and 'hot dry rock' geothermal power were selected as the renewable sources with the greatest potential. Owing to the intermittency, especially of solar PV and wind power, and the constant electricity demand of mining operations, hybrid versions with current electricity sources were identified as the best option.

In addition, based on the current legislative and regulatory framework in South Africa, the business model of self-generation[3], in the form of own investment or a power purchase agreement, has the greatest potential. As the purpose of the development of the MCDA for mining corporations was to create more transparency, own investment was selected. The reason for this choice was to focus the attention on the performance of the technology and not on third parties (Boyse *et al.*, 2014).

Therefore, decision-makers at mining corporations were asked to list the evaluation criteria that they would use to evaluate the following electricity sources:

- Diesel generator
- Hybrid diesel generator / solar PV
- Hybrid diesel generator / onshore wind power
- Hybrid diesel generator / geothermal power
- Eskom grid connected
- Hybrid Eskom grid connected / solar PV
- Hybrid Eskom grid connected / onshore wind power
- Hybrid Eskom grid connected / geothermal power.

Stakeholders

The business model has great influence on the type and number of stakeholders involved in establishing an electricity source for mining operations. All stakeholders and their influence on the decision-making process have to be identified to ensure that the best possible solution can be found (Boyse *et al.*, 2014). The selected model of self-generation and own investment involved three main stakeholders: the mining corporation, the project developer, and the legislative and regulatory body.

The mining corporation was the main stakeholder for the purpose of this research, as it is the decision-maker regarding a possible project realization. The decision-makers have to cover two main areas: operations and finance. Decision-makers selecting an alternative from an operational perspective will make use of criteria that ensure that the generating source will satisfy the electricity demands of the mine (Cookie *et al.*, 2007). Decision-makers selecting an alternative from a financial perspective will make use of criteria to find the most feasible alternative for electricity generation at the mining location (Goh *et al.*, 2014).

The project development company is responsible for the realization of the alternative. The companies represent the different electricity sources and will provide the data necessary to feed the evaluation criteria to execute the MCDA approach. The developer has no direct influence on the decision-making, and can only affect the attractiveness of the project (Lerro, 2011; Aslani, 2014).

The regulatory and legislative body in South Africa dictates the framework regarding the business model of how electricity projects will be realized. As previously stated, according to this framework the model of self-generation was selected as the most lucrative. It further influences the actual development of the project with factors like compulsory environmental assessments, which also consider the surrounding communities (Lerro, 2011; Frost, 1995).

Research results

Profile of mining corporations

To reveal the criteria that mining corporations use to evaluate possible electricity generation sources, four different mining corporations were included in this study, as illustrated in Table IV. In three the respondents were electrical engineers, and in one case the manager of a utility supply chain with responsibility for six mines. Emphasis was placed on gaining information from a variety of mines with different resources, sizes, and targets in order to gain optimal insight. The resources mined were gold, coal, chrome, and zircon. The average annual electricity consumption per mine varied from 4.2 GWh to 2 752 GWh. As the Eskom tariff varies according to factors like season or time of day, the yearly average price was stated for the sake of simplicity. All mines were connected to the grid, with diesel generators as backup systems. One of the mines had installed a 1 MW solar PV plant, and two other corporations were conducting solar PV and onshore wind power feasibility studies.

Criteria characteristics and requirements

To structure the internal evaluation process, a decision table was developed. The table of mining corporations evaluating different alternatives for electricity sources had to be constructed according to certain characteristics. The decision table was based on the results of the post-it mind-maps from the interviews with decision-makers. To be able to identify a clear structure and to use the criteria for further analysis in a MCDA approach, specific requirements had to be fulfilled by each criterion (Stewart and Belton, 2002).

- Value relevance – the decision-maker has to be able to relate the concept to the aim of the mining corporation, which enables him or her to define a clear preference for the criterion
- Understandability – the criterion has to be clearly identified and explained. Each person involved in the decision-making process has to know the exact meaning to prevent any confusion and misleading results (Edwards *et al.*, 2007)
- Measurability—it has to be possible to measure each criterion in a consistent manner according to the alternatives being analysed. As the decision table is the foundation of an MCDA analysis, this requirement is important to create meaningful results
- Non-redundancy—there should not be more than one criterion measuring the same factor. A negative result would be to have faulty results, *e.g.* one factor has too much weight because it was counted more than once (Edwards *et al.*, 2007)
- Judgmental independence—one criterion should not have significant influence on the performance of another criterion (Loken, 2007)
- Balancing completeness and conciseness—all aspects of

3. The mining corporation develops its own on-site renewable generation source (Boyse *et al.*, 2014).

Table IV

Characteristics of mining corporations interviewed

Characteristic	Corporation 1	Corporation 2	Corporation 3	Corporation 4
Job description	Energy engineer	Manager: Utilities: Supply chain	Energy engineer	Energy engineer
Type of resources	Gold	Coal	Chromium ores	Zircon
Number of mines represented	3	6	1	1
Average annual electricity consumption	2 752 GWh	701 GWh	4,2 GWh	16,8 GWh
Electricity costs as a percentage of total operational costs	20%	–	–	-
Price per kWh to Eskom	0.72 cent per kWh (yearly average)	0.71 cent per kWh (yearly average)	Not set yet, as connection had been established only recently	No connection
Percentage of electricity supplied by Eskom	100%	100%	70% (30% solar PV)	0%
Current on-site sources	Diesel	Diesel	Diesel / solar PV	Diesel
Backup system or full-time use	Backup	Backup	Diesel backup Solar PV full-time	Base load
Reason to invest	Save costs and more independence	Not feasible	Diesel savings	Diesel savings
Year of realization	Feasibility stage (10 MW Solar PV)	-	2012 (PV)	Feasibility stage (Wind power)

alternatives in a decision process have to be addressed by the selected criteria. However, when selecting too many criteria the researcher has to beware of omitting to consider the previously mentioned requirements (Edwards *et al.*, 2007; Loken, 2007)

- Operationally—criteria should not be only theory-based, but should also be practically proven.

The decision criteria

The interviews with the mining corporations showed that possible electricity sources were evaluated based on their being independent and able to supply electricity (including the necessary fuel) on their own, and within the investment capability of the corporation. The criteria are listed and described in Table V. The first column provides the category and the second the criteria.

The economic category included two criteria that measured the economic value of the electricity source, namely: levelized electricity costs and net present value. These criteria used several values as part of the calculation. The prediction of fuel costs was new in regard to the literature review. Further new criteria were supply 24/7, service level, corporate image, and effect on community.

It is important to mention that the criteria listed in Table V represent a summary of all criteria used during the evaluation process, and that most had been applied in other energy planning evaluations at various points in time. The sequence started with technological criteria, aimed to ensure that the potential electricity source could satisfy the electricity requirements of the mining operation. Thereafter, technologies that passed the technical criteria were analysed according to economic criteria. Lastly, environmental and social criteria were evaluated.

Mining corporations have peculiarities not found in previous MCDA evaluations. Firstly, mining operations are more profit-oriented in comparison to national electricity providers. Therefore, criteria like initial investment cost, which can have a considerable influence on the balance sheet in the first years, have higher priority for mining corporations than for electricity providers. Secondly, owing to the limited mix of electricity sources and the requirement of constant electricity supply, the criteria of reliability and supply 24/7 are more important to mines. Both were therefore used as prerequisites in the choice of the type of electricity source to be evaluated in Table V. Another criterion that had to be considered, which did not feature in previous evaluations, was the predicted lifespan of the mine, as this factor has a considerable influence on the economic criteria, and consequently on the feasibility of the project.

Conclusion

The preparation of this paper is justified by two facts. Firstly, the difficulties with current electricity sources in South Africa have increased the attractiveness of the steadily advancing renewable technologies for mining operations in the country.

Table V

Evaluation criteria of mining corporations

First hierarchy criteria	Second hierarchy criteria	Description
Economy	Investment cost	Includes all costs regarding the planning, purchase, and installation of the electricity source.
	Operating and maintenance costs	Operating costs entail employees' salaries, the money spent on the energy (fuel), and the products and services for the system's operation. Maintenance costs ensure that the system is in operating condition, in order to prolong the system's life and avoid failures that result in downtime.
	Prediction of fuel costs	Provides a prediction of the price of fuel consumed to produce electricity.
	Prediction of initial investment costs	Provides an estimation of how the initial investment cost will develop. If the technology is relatively new, possible price drops can be expected.
	Annual cost of electricity	Measures the annual cost of electricity per kWh. It compares annual costs to annual output, without initial investment costs.
	Levelized electricity cost	Measures the rand cost per kWh, including all costs incurred by the initial investment till the end of the predicted lifetime – which is placed in relation with the projected output of kWh in the same time span.
	Net present value	This is a financial method to define the total present value of a series of annual cash inflows and outflows during the lifespan of the asset. The cash flows are discounted back to their present and summed. The final present amount is compared to the initial investment cost.
Technology	Efficiency	Indicates the ratio of output energy to input energy. It explains how much useful energy can be generated.
	Safety	Relates to the degree of safety for employees working on site.
	Implementation period	The amount of time needed to realize the project.
	Reliability	The capacity of a system to perform as designed and planned.
	Supply 24/7	Most mining operations need a 24-hour electricity supply.
	Maturity	Refers to the development stage of the technology. The stages range from 'only tested in laboratories' to 'close to reaching the theoretical limits of efficiency'.
	Service level	Measures the availability of experts and spare parts to repair damaged equipment.
Environment	CO_2 emission	Represents the measurement of the emission of a colourless, odourless and tasteless gas, which is mainly emitted through the combustion of coal, oil and gas.
	Noise	The machine-created sound that disrupts human and animal daily life.
	Land requirement	Represents the amount of land that the electricity source requires to produce at a certain capacity.

Table V

Evaluation criteria of mining corporations

Social	Job creation	The number of people employed during the life cycle of an energy system.
	Corporate image	The possible impact of the electricity source on the corporate identity in the minds of diverse publics, such as customers, investors, and employees.
	Effect on community	Refers to the possible impact on the surrounding residents, after the decision to close the mine. The community could further utilize the electricity source.

Secondly, previous research has been directed only at investigating the external influences on this market.

The contribution of this paper is an investigation of the internal business approach of mining corporations towards evaluating electricity generation sources. This research has shown that the MCDA method is the most suitable approach. No application of the MCDA method in energy planning from the perspective of corporate or mining entities could be found – most MCDA adaptations were from the perspective of governmental bodies or general electricity source evaluations without a specific perspective. In addition, no adaptation of MCDA methods in energy planning has been conducted in a South African context. Consequently, in the absence of data, qualitative interviews were conducted with four mining corporations operating in South Africa to reveal the criteria that these organizations applied in order to evaluate possible electricity sources. The differences found between the selection criteria used by the four corporations and those of previous approaches in the literature are the result of different perspectives. Mines are profit-oriented business entities and electricity generation is not their core business, while previous approaches were more focused on the technological and environmental factors.

This paper has established the basis for evaluating and comparing current and renewable electricity-generating options from the perspective of mining corporations in South Africa. The reason for subdividing the work into two papers was to create a basis of two components: the first being the current knowledge about the external framework; while the second reveals the internal framework. Part 2 will fuse the external and internal components in order to apply the adapted MCDA framework and to feed the model with real-time data.

References

Afgan, N.H. and Carvalho, M.G. 2002. Multi-criteria assessment of new and renewable energy power plants. *Energy*, vol. 27, no. 8. pp. 739–755.

Ahmad, S. and Tahar, R.M. 2014. Selection of renewable energy sources for sustainable development of electricity generation system using analytic hierarchy process: A case of Malaysia. *Renewable Energy*, vol. 63. pp. 458–466.

Alton, T., Arndt, C., Davies, R., Hartley, F., Markrelov, K., Thurlow, J., and Ubogu, D. 2014. Introducing carbon taxes in South Africa. *Applied Energy*, vol. 116, March. pp. 344–354.

Aslani, A. 2014. Private sector investment in renewable energy utilization: Strategic analysis of stakeholder perspectives in developing countries. *International Journal of Sustainable Energy*, vol. 33, no.1. pp. 112–124.

Boran, F.E., Boran, K., and Menlik, T. 2012. The evaluation of renewable energy technologies for electricity generation in Turkey using intuitionistic Fuzzy TOPSIS. *Energy Sources, Part B: Economics, Planning, and Policy,* vol. 7, no. 1. pp. 81–90.

Boran, F.E., Dizdar, E., Toktas, I., Boran, K., Eldem, C., and Asal, Ö. 2013. A multidimensional analysis of electricity generation options with different scenarios in Turkey. *Energy Sources, Part B: Economics, Planning, and Policy,* vol. 8, no. 1. pp. 44–55.

Boyse, F., Causevic, A., Duwe, E., and Orthofer, M. 2014. Sunshine for mines: Implementing renewable energy for off-grid operations. Carbon War Room, Washington, DC.

Brand, B. and Missaoui, R. 2014. Multi-criteria analysis of electricity generation mix scenarios in Tunisia. *Renewable and Sustainable Energy Reviews*, vol. 39. pp. 251–261.

Cavallaro, F. 2009. Multi-criteria decision aid to assess concentrated solar thermal technologies. *Renewable Energy*, vol. 34, no. 7. pp. 1678–1685.

Cetindamar, D., Daim, T.D., Beyhan, B., and Basoglu, N. 2013. Strategic Planning Decisions in the High Tech Industry. Springer Verlag, London.

Chatzimouratidisa, A.I. and Pilavachi, P.A. 2008. Multicriteria evaluation of power plants impact on the living standard using the analytic hierarchy process. *Energy Policy*, vol. 36, no. 3. pp. 1074–1089.

Chatzimouratidisa, A.I. and Pilavachi, P.A. 2009a. Sensitivity analysis of technological, economic and sustainability evaluation of power plants using the analytic hierarchy process. *Energy Policy*, vol. 37, no. 3. pp. 788–798.

Chatzimouratidisa, A.I. and Pilavachi, P.A. 2009b. Technological, economic and sustainability evaluation of power plants using the analytic hierarchy process. *Energy Policy*, vol. 37, no. 3. pp. 778–787.

Chislett, M. 2014. Cleaner energy at stable prices: The value proposition of solar for mines. *Interview*. Toronto, Energy and Mines.

Cookie, R., Cripps, A., Irwin, A., and Kolokotroni, M. 2007. Alternative energy technologies in buildings: Stakeholder perception. *Renewable Energy*, vol. 32, no. 14. pp. 2320–2333.

Cristobal, J.R.S. 2011a. A multi criteria data envelopment analysis model to evaluate the efficiency of the renewable energy technologies. *Renewable Energy*, vol. 36, no. 10. pp. 2742–2746.

Cristobal, J.R.S. 2011b. Multi-criteria decision-making in the selection of a renewable energy project in Spain: The Vikor method. *Renewable Energy*, vol. 36, no. 2. pp. 498–502.

Datta, A., Ray, A., Bhattacharya, G., and Saha, H. 2011. Green energy sources (GES) selection based on multi-criteria decision analysis (MCDA). *International Journal of Energy Sector Management*, vol. 5, no. 2. pp. 271–286.

De Beer, A.J. 2000. The development of a generic model for strategic planning for small and medium manufacturing enterprises in a turbulent environment. MBA dissertation, Stellenbosch University, South Africa.

Diakoulaki, D. and Karangelis, F. 2007. Multi-criteria decision analysis and cost-benefit analysis of alternative scenarios for the power generation sector in Greece. *Renewable and Sustainable Energy Reviews*, vol. 11, no. 4. pp. 716–727.

EDWARDS, W., MILES, R.F., and VON WINTERFELDT, D. 2007. Advances in Decision Analysis: From Foundations to Applications. Cambridge University Press, Cambridge.

EIUG (Energy Intensive User Group of Southern Africa). 2015. Electricity cost as a percentage of annual expenditure. http://www.eiug.org.za/about/membership/ [accessed 11 Jun. 2015].

ENERGY and MINES. 2015. Introduction to renewable energy with mining operations. http://energyandmines.com [accessed 26 May 2015].

ERTAY, T., KAHRAMAN, C., and KAYA, I. 2013. Evaluation of renewable energy alternatives using MACBETH and fuzzy AHP multi criteria methods: The case of Turkey. *Technological and Economic Development of Economy*, vol. 19, no. 1. pp. 38–62.

ESKOM. 2015a. Company information. http://www.eskom.co.za/OurCompany/CompanyInformation/Pages/Company_Information_1.aspx [accessed 10 Jun. 2015].

ESKOM. 2015b. Tariff and charges. http://www.eskom.co.za/CustomerCare/TariffsAndCharges/Pages/Tariffs_And_Charges.aspx [accessed 13 Jun. 2015].

FROST, F.A. 1995. The use of stakeholder analysis to understand ethical and moral issues in the primary resource sector. *Journal of Business Ethics*, vol. 14, no. 8. pp. 653–661.

GOH, H.H., LEE, S.W., CHUA, Q.S., GOH, K.C., KOK, B.C., and TEO, K.T.K. 2014. Renewable energy project management, challenges and risk. *Renewable and Sustainable Energy Reviews,* vol. 38. pp. 917–932.

GOVENDER, S. 2008. Energy saving mechanisms in the mining industry: A case study of switching off non-essential power. MBA thesis, Stellenbosch University, South Africa.

HAGUE, P.N., and JACKSON, P. 1995. Do Your Own Market Research. Kogan Page, London.

HEO, E., KIM, J., and BOO, K. 2010. Analysis of the assessment factors for renewable energy dissemination program evaluation using fuzzy AHP. *Renewable and Sustainable Energy Reviews*, vol. 14, no. 8. pp. 2214–2220.

ISHIZAKA, A., and NEMERY, P. 2013. Multi-criteria Decision Analysis, Wiley, Chichester, UK.

JAKHOTIYA, G.P. 2013. Strategic Planning, Execution, and Measurement (SPEM). Taylor & Francis, Boca Raton, FL.

JUDD, E. 2014. Sincerity and patience required for renewables in mining sector. Energy and Mines, Toronto.

KAPLAN, S., and NORTON, D.P. 1992. The balanced scorecard – measures that drive performance. *Harvard Business Review*, vol. 70, no. 1. pp 71-79.

KAYA, T., and KAHRAMAN, C. 2010. Multi criteria renewable energy planning using an integrated fuzzy VIKOR & AHP methodology: The case of Istanbul. (Report). *Energy*, vol. 35, no. 6. pp. 2517–2527.

KEENEY, R.L. 1996. Value-focused thinking: Identifying decision opportunities and creating alternatives. *European Journal of Operational Research*, vol. 92, no. 3. pp. 537–549.

KING, W. and CLELAND, D. 1987. Strategic Planning and Management Handbook. Van Nostrand Reinhold, New York, NY.

LEE, A.H.I., HUNG, M., KANG, H., and PEARN, W.L. 2012. A wind turbine evaluation model under a multi-criteria decision making environment. (Report). *Energy Conversion and Management*, vol. 64. pp. 289–300.

LERRO, A. 2011. A stakeholder-based perspective in the value impact assessment of the project ''Valuing intangible assets in Scottish renewable SMEs''. *Measuring Business Excellence*, vol. 15, no. 3. pp. 3–15.

LINARD, K.L. and YOON, J. 2000. The dynamics of organizational performance development of a dynamic balanced scorecard. *Proceedings of the 1st International Conference on Systems Thinking in Management*, Geelong, Australia, 8-10 November.

LOKEN, E. 2007. Multi-criteria planning of local energy systems with multiple energy carriers. Doctoral thesis, Norwegian University of Science and Technology, Norway.

MAMLOOK, R., AKASH, B.A., and MOHSEN, M.S. 2001. A neuro-fuzzy program approach for evaluating electric power generation systems. *Energy*, vol. 26, no. 6. pp. 619–632.

MATEO, J.R.S.C. 2012. Multi Criteria Analysis in the Renewable Energy Industry. Springer Verlag, London.

MAXIM, A. 2014. Sustainability assessment of electricity generation technologies using weighted multi-criteria decision analysis. *Energy Policy*, vol. 65. pp. 284–297.

NEWTON, N. 2010. The use of semi-structured interviews in qualitative research: Strengths and weaknesses. http://www.academia.edu/1561689/The_use_of_semi-structured_interviews_in_qualitative_research_strengths_and_weaknesses [accessed 24 May 2014].

NICOLAS, F. 2014. Africa-focused miners positioned to embrace 100% renewable energy. *Mining Weekly*, 21 May. http://www.miningweekly.com/article/africa-focused-miners-positioned-to-embrace-100-renewable-energy-2014-05-21 [accessed 2 Jun. 2015].

NUMBI, B.P., ZHANG, J., and XIA, X. 2014. Optimal energy management for a jaw crushing process in deep mines. *Energy*, vol. 68, April. pp. 337–348.

PERSON, R. 2013. Balanced Scorecards & Operational Dashboards with Microsoft Excel. 2nd edn. Wiley, Indianapolis.

PETTICREW, M., and ROBERTS, H. 2006. Systematic Reviews in the Social Sciences: A Practical Guide. Blackwell, Oxford.

POWELL, C. 2003. The Delphi technique: Myths and realities. *Journal of Advanced Nursing*, vol. 41, no. 4. pp. 376–382.

RIBEIRO, F., FERREIRA, P., and ARAUJO, M. 2013. Evaluating future scenarios for the power generation sector using a multi-criteria decision analysis (MCDA) tool: The Portuguese case. *Energy*, vol. 52. pp. 126–136.

ROEHRL, R. and RIAHI, K. 2000. Technology dynamics and greenhouse gas emissions mitigation: A cost assessment. *Technological Forecasting & Social Change*, vol. 63, no. 2. pp. 231–261.

STEIN, E.W. 2013. A comprehensive multi-criteria model to rank electric energy production technologies. (Report). *Renewable and Sustainable Energy Reviews*, vol. 22. pp. 640–654.

STEINHAEUSER, I., SETHIA, A., JAISWAL, S., ROCHA OLIVEIRA, G., SEKINE, Y., and ALVES, L.C. 2012. Global Corporate Renewable Energy Index (CREX). Bloomberg New Energy Finance & Vestas, Copenhagen.

STEWART, T.J. and BELTON, V. 2002. Multi Criteria Decision Analysis: An Integrated Approach. Kluwer, Dordrecht.

STREIMIKIENE, D., BALEZENTIS, T., KRISCIUKAITIEN, A.I., and BALEZENTIS, A. 2012. Prioritizing sustainable electricity production technologies: MCDM approach. (Report). *Renewable and Sustainable Energy Reviews*, vol. 16, no. 5. pp. 3302–3311.

TOPCU, Y.I. and ULENGIN, F. 2004. Energy for the future: An integrated decision aid for the case of Turkey. *Energy*, vol. 29, no. 1. pp. 137–154.

TROLDBORG, M., HESLOP, S., and HOUGH, R.L. 2014. Assessing the sustainability of renewable energy technologies using multi-criteria analysis: Suitability of approach for national-scale assessments and associated uncertainties. *Renewable and Sustainable Energy Reviews*, vol. 39. pp. 1173–1184.

TSOUTSOS, T., DRANDAKI, M., FRANTZESKAKI, N., LOSIFIDIS, E., and KIOSSES, I. 2009. Sustainable energy planning by using multi-criteria analysis application in the island of Crete. *Energy Policy*, vol. 37, no. 5. pp. 1587–1600.

VOTTELER, R.G. and BRENT, A.C. 2017. A mining perspective on the potential of renewable electricity sources for operations in South Africa: Part 2. A multi-criteria decision assessment. *Journal of the Southern African Institute of Mining and Metallurgy*, vol. 117, no. 3. pp. 299–312.

WANG, J., JING, Y., ZHANG, C., SHI, G., and ZHANG, X. 2008. A fuzzy multi-criteria decision-making model for trigeneration system. *Energy Policy*, vol. 36, no. 10. pp. 3823–3832.

WESTERMANN, G., and SEHL, I. 2006. Developing a balanced scorecard based benchmarking approach for tourist destinations. *Proceedings of the European Innovation Pressure Conference,* Tampere, Finland, 15–17, March. ◆

AfriRock 2017

ISRM International Symposium

'Rock Mechanics for Africa'

30 September–6 October 2017

Cape Town Convention Centre, Cape Town

Keynote Speakers

Nick Barton
Sergio Fontoura
Luís Lamas
Dick Stacey
Nielen van der Merwe
Paul Buddery

BACKGROUND

The 2017 ISRM International Rock Mechanics Symposium is to be held in Cape Town. The conference theme is 'Rock Mechanics for Africa'.
Mining has traditionally been a mainstay of African economies, while Oil and Gas industries are rapidly growing throughout Africa. Infrastructure is being developed to support these industries. Rock engineering design is and therefore will continue to be essential for the growth of the continent. Prior to the conference, the ISRM Board, Council and Commission meetings will take place. Technical visits are being arranged for after the conference.

Sponsors

GEOBRUGG
IDS INGEGNERIA DEI SISTEMI
MINOVA
New Concept Mining
ROCBOLT TECHNOLOGIES
REUTECH MINING

WHO SHOULD ATTEND

- Rock engineering practitioners
- Researchers
- Academics
- Mining engineers
- Civil engineers
- Petroleum engineers
- Engineering geologists.

TECHNICAL VISITS

The following technical visits are confirmed for the conference:

- Palabora Mine
- Tau Tona Mine
- Chapman's Peak

TENTATIVE PROGRAMME

Saturday 30/09/2017		Sunday 1/10/2017		Monday 2/10/2017	Tuesday 3/10/2017	Wednesday 4/10/2017	Thursday 5/10/2017	Friday 6/10/2017
ISRM Board Meeting	Workshop	ISRM Board Meeting	Workshop	ISRM Commission Meetings	Technical Session Morning Refreshments Technical Session Lunch Technical Session Afternoon Refreshments Technical Session			Technical Visits
	Workshop		Workshop	ISRM Council Meetings				
		Board Dinner		Network Function		Conference Dinner		• Palabora Mine • Tau Tona Mine • Chapman's Peak

EXHIBITION/SPONSORSHIP

Sponsorship opportunities are available. Companies wishing to sponsor or exhibit should contact the Conference Co-ordinator.

For further information contact:

Raymond van der Berg, Head of Conferencing
SAIMM, P O Box 61127, Marshalltown 2107
Tel: +27 (0) 11 834-1273/7 · E-mail: raymond@saimm.co.za
Website: http://www.saimm.co.za

Supported by

ISRM
SANIRE SOUTH AFRICAN NATIONAL INSTITUTE OF ROCK ENGINEERING
SAIMM THE SOUTHERN AFRICAN INSTITUTE OF MINING AND METALLURGY
South African National Committee on Tunnelling

Conference Announcement

http://dx.doi.org/10.17159/2411-9717/2017/v117n3a11

A mining perspective on the potential of renewable electricity sources for operations in South Africa:

Part 2 – A multi-criteria decision assessment

by R.G. Votteler* and A.C. Brent†

Synopsis

The first in this series of two papers, on the potential of renewable electricity sources for mining operations in South Africa, investigated the internal structure of mining corporations to evaluate feasible alternative electricity sources that were identified as through earlier research. The purpose of this paper is to combine current knowledge about the external macroeconomic and the internal environments to produce a clear picture of how renewable sources of electricity could perform from the perspective of mining corporations in South Africa. The multi-attribute value theory (MAVT) approach was adapted to structure the research and results. The model was fed with real-time data provided from five different energy companies and four mining corporations operating in South Africa. The results show that the performance of hybrid versions of the currently used electricity sources (diesel generators and Eskom grid connection) with solar PV and onshore wind is favourable compared with the current sources alone. The advantage of diesel generators is significantly greater than that of the Eskom grid connection. By combining the macroeconomic influences with the MAVT results, hybrid solar PV versions are identified as having the greatest potential. In second place are hybrid wind solutions, which have the shortcoming that good wind conditions occur only in coastal regions where there are fewer mining activities. Geothermal hybrid versions are the least favourable owing to the lack of service infrastructure and high initial investment costs.

Keywords: Multi-criteria decision analysis, multi-attribute value theory, renewable electrical energy.

Introduction

Recent developments in the macroeconomic environment necessitate an investigation of the internal structure of mining corporations to evaluate possible alternative electricity-generating sources (Roehrl and Riahi, 2000). To be able to combine current knowledge of macroeconomic factors and the internal structure of mining corporations, the research process is presented in two papers. The first paper (Votteler and Brent, 2016) investigated the internal structure and argued that a strategic tool should be used to evaluate electricity sources from the perspective of mining corporations. The multi-criteria decision analysis (MCDA) approach was selected as the most appropriate strategic tool. Based on a literature review of previous similar MCDA approaches in energy planning, it was concluded that no adaptation of MCDA methods could be found from a corporate, mining, or South African perspective. Finally, the paper investigated the internal evaluation structure in collaboration with mining corporations, by identifying the criteria they use to evaluate electricity sources.

In this second paper we set out to identify the best MCDA method to investigate a potential fit for renewable energy sources at mining operations. We then implement the selected MCDA method. Previous macroeconomic research identified non-grid-connected solar PV, onshore wind, and geothermal power in hybrid versions along with the current on-site diesel generators and grid connection to Eskom as the most lucrative. The choice of these hybrid versions is based on the constant electricity demand of mining operations and the intermittency of the renewables. The business model identified was self-generation via own investment (Votteler and Brent, 2016). The first paper describes the evaluation criteria used by mining corporations to determine this choice. All further information for implementing the MAVT method was gathered in cooperation with mining corporations and renewable and conventional energy companies in order to use real-time data.

The purpose of this paper is to analyse and compare the strengths and weaknesses of these potential electricity sources, according to a possible fit to the specific needs of mining corporations and from their perspective. To optimize the learning process for mining corporations in order to equip them with an understanding of renewable energy technologies and for energy companies to learn

* *School of Public Leadership, Faculty of Economic and Management Sciences, Stellenbosch University, South Africa.*

† *Centre for Renewable and Sustainable Energy Studies, Department of Industrial Engineering, Stellenbosch University, South Africa and Sustainable Energy Systems, Engineering and Computer Science, Victoria University of Wellington, New Zealand.*

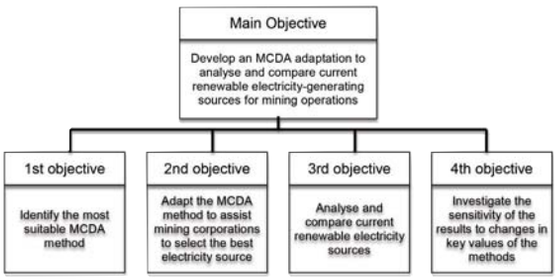


Figure 1—Research objectives

how to approach these potential new customers, more research has to be conducted (Steinhaeuser *et al*., 2012).

The research objectives are illustrated in Figure 1. The first objective, to investigate the best-suited MCDA, method was based on a literature review. The ensuing objectives were based on primary research investigations. The second objective was to adapt the selected MCDA method. The data was gathered in cooperation with energy companies supplying the electricity sources and mining corporations using them. Based on the third objective, the results of the adapted MCDA method were analysed to identify strengths and weaknesses of the selected sources. The last section investigated the sensitivity of the method to possible changes to the results when alternating key input data.

MCDA method selection

Research approach

To conduct secondary research, data was used that was gathered and recorded by others prior to the current project. The advantages were the assurance of readily available data and the relatively quick and inexpensive acquisition of such data. The disadvantages were that the information may be outdated, that there may have been a variation in the definition of terms, and that different units of measurement may have been used. Cross-checking the data could reduce these disadvantages; this would entail comparing similar data (Zikmund and Babin, 2010). The media used for the literature review were books, the internet, conference proceedings, journal articles, Master's and doctoral dissertations, and case studies.

Petticrew and Roberts (2006) identified six different types of literature review. This paper made use of a 'conceptual review', which aims to synthesize areas of conceptual knowledge that can contribute to a better understanding of the issues studied. The objective of such a review is to provide an overview of the literature in a given field, including the main ideas, models, and debates.

The first objective was to identify the most suitable MCDA method for the research context of this paper, which was based entirely on secondary literature. The literature is discussed in subsequent sections. Firstly, requirements of this research to identify the most suitable MCDA method are given and explained. By fulfilling the requirements it was possible to ensure that all the areas of the investigation were addressed. Secondly, the different MCDA 'schools of thought' are introduced and their possible use according to the requirements discussed. The value measurement school of thought was selected. The next step was to use the same requirements to analyse the possible contribution of each method. Three well-established and comprehensive methods were investigated.

The second objective was to adapt the selected MCDA method to the decision structure of mining corporations, evaluating different electricity-generating sources. The secondary literature provided the background knowledge for the selected MCDA method and the foundation for the adaptation.

Requirements of the MCDA method

This section describes the requirements of the MCDA method to best address the main research objective, namely to analyse and compare potential electricity sources for mining corporations. The required elements were used to scan different MCDA categories and methods to identify the one best-suited for adaptation in the context of this investigation. It is important to mention that it was often difficult to justify the selection of a method. None of them are perfect, nor is it possible to apply them to all decision problems. Each method entails its own limitations, characteristics, principles, and perspectives (Ishizaka and Nemery, 2013). Consequently, the following requirements were used to make the best possible selection.

1. *Once-off decision* – Specialist knowledge of the characteristics of the MCDA method is required in order to contribute successfully to this type of decision. The decision to select the best-suited electricity source has to be made once and is not recurring. Only after years of usage a re-evaluation may be necessary as price structures or performance levels may have changed (Khatib, 2008)
2. *Investigate the evaluation structure* – The implementation of the MCDA method has to bring about a clear understanding of the internal evaluation structure of mining corporations. It has to show how each criterion contributes to the overall decision. This can contribute to renewable energy companies having a better understanding of a potential new type of customer, namely mining corporations
3. *Analyse alternatives separately* – The method has to deliver the basis for a separate analysis of each alternative. It should be possible to illustrate and explain the strengths and weaknesses of an alternative independently of the other alternatives. The decision-maker has to understand the implications of selecting a certain option
4. *Compare alternatives* – It has to be possible to compare alternatives according to their strengths and weaknesses. The results should demonstrate the advantages and disadvantages of selecting one electricity source in preference to the others. The results ought to provide the decision-maker with the necessary information to optimize his or her selection
5. *Incorporate unknown alternatives* – As the concept of renewable electricity sources is relatively new for mining corporations in South Africa (Boyse *et al*., 2014), the method has to be able to incorporate the analysis of alternatives not known to the decision-maker. The basis of the adapted method, before implementing the data of alternatives, should not include any predispositions towards a specific outcome.

Possible MCDA methods

The purpose of this section was to select the MCDA method that is best suited to contribute to the research objectives. This was achieved by investigating the categories and methods according to the requirements listed above (see Table I). For structural purposes the reasoning was subdivided into the selection of the most appropriate MCDA category, and then the method.

Selection of MCDA category

It is generally accepted that MCDA methods can be divided into three broad categories, or schools of thought, namely the value measurement methods, the goal, aspiration, or reference level methods, and outranking methods (Stewart and Belton, 2002). The list below introduces the theory of each category, followed by an examination of which requirements are or are not satisfied. Lastly, the category is selected and reasons are stated.

- *Goal, aspiration, or reference methods* – These methods establish, in coordination with the decision-makers, desirable or satisfactory levels of achievement for each evaluation criterion. The results of the implementation identify the alternative that is closest to realizing these goals and aspirations (Roy and McCord, 1996). The method is preferably used for decision problems of a repetitive nature or familiar to the decision-maker (Stewart and Belton, 2002). This contradicts requirement 1 and 5. In addition, the methods necessitate that performance measures are available in quantitative form (Chang, 2011). This eliminates requirements 2, 3, and 4, as selected criteria of this study are of qualitative nature
- *Outranking methods* – These methods compare alternative courses of action in a pairwise approach. This is initially done on a criterion-to-criterion basis in order to state the preference for one over the other. Thereafter the methods aggregate such preferences of all selected criteria in order to identify the level of evidence favouring one alternative over the others. Partial and complete rankings are constructed (Geldermann and Schöbel, 2011). Consequently, the methods investigate the internal evaluation structure and compare alternatives that are new or known to the decision-maker, which fulfils requirements 2, 3 and 4. Requirement 3 is not satisfied, as pairwise comparisons are used, which makes it impossible to analyse alternatives separately. The methods are used for discrete choice problems (Bouyssou *et al.*, 2002), which satisfies requirement 1
- *Value measurement methods* – The methods create numerical scores for each alternative analysed to illustrate the preferences associated with each alternative. Initially, scores are established for each selected criterion separately. Thereafter, scores are synthesized on the basis of relative importance. This, in turn, effects aggregation into higher-level preference methods—which enables the drawing up of a complete ranking with scores. The basis of the methods differs as some are built on pairwise comparisons and others on preference functions (Keeney, 1992). The foundation of constructing preference functions for each criterion fulfils requirements 2 and 3. As the decision-maker does not have to formulate any pre-set ambitions, requirement 5 is satisfied. The methods are suited for once-off decision problems, to fulfil requirement 1 (Triantaphyllou, 2000).

For the purpose of this study, the value measurement school of thought was selected, while the outranking methods were a close second. The value measurement methods satisfied all requirements. Firstly, the value measurement methods were better suited than the outranking methods in regard to requirement 2. By creating value functions for each selected criterion, incorporating relative weights, the internal structure of mining corporations in order to to evaluate electricity sources is illustrated in detail. The outranking methods did not satisfy this requirement to the same extent, as criteria were not investigated separately. In addition, the outranking methods were not able to fulfil requirement 3, as the results of the analysis were not separate for each alternative, but appeared in relation to each other. Lastly, the goal, aspiration, or reference school of thought could not fulfil any of the requirements.

Selection of value measurement method

The selected value measurement school of thought, also known as the 'full aggregation approach', was the most detailed and comprehensive MCDA option (Eliasson and Lee, 2003). Within this school of thought different methods exist. This section investigates three established method designs that produce the most detailed results of the value measurement methods (Ishizaka and Nemery, 2013; Linkov and Moberg, 2012). In the following list, the choice of the best-suited method for the adaptation to the context of this paper is identified and discussed:

Table I

Selection process for MCDA method

MCDA categories	Req. 1.	Req. 2.	Req. 3.	Req. 4.	Req. 5.
Goal, aspiration or reference level methods	✖	✖	✖	✖	✖
Outranking methods	✔	✔	✖	✔	✔
Value measurement methods	✔	✔	✔	✔	✔
Selection					
Value measurement methods					
Measuring attractiveness by a categorical based evaluation technique method (MACBETH)	✔	✔	✖	✔	✔
Analytical hierarchy process (AHP)	✔	✔	✖	✔	✔
Multi-attribute value theory (MAVT)	✔	✔	✔	✔	✔
Final choice					
Multi-attribute value theory					

- *Measuring attractiveness by the categorical-based evaluation technique method (MACBETH)* – This method consists of three steps. The first step is to structure the problem, which is followed by constructing a judgement matrix on the basis of interval pairwise comparisons. If consistency of the matrix is proven, the attractiveness can be calculated (Ertay *et al.*, 2013). As the method uses pairwise comparisons, it is difficult to analyse alternatives separately, which contradicts requirement 3. The other requirements can be fulfilled
- *Analytical hierarchy process (AHP)* – This method comprises three steps, similar to MACBETH. Firstly, the problem is structured, followed by the creation of a judgement matrix based on ratio pairwise comparisons. Again, if results are consistent, the attractiveness can be calculated. A sensitivity analysis can be conducted to reduce uncertainty (Stein, 2013). As this method also uses pairwise comparison, even though on a ratio scale, requirement 3 is difficult to accomplish. However, all other requirements can be satisfied
- *Multi-attribute value theory (MAVT)* – This method entails five steps. The first step is to structure the problem. The second is to determine the criteria that the decision-maker uses to evaluate the decision problem. Thirdly, a scale is developed to measure each criterion. Fourthly, a value function is created for each criterion. Lastly, the data for each alternative is implemented and results can be analysed and compared. A sensitivity analysis can then be conducted (Stefanopoulos *et al.*, 2014). As a preference function is constructed for each criterion, alternatives can be analysed separately and in comparison to each other.

The MAVT was selected as it satisfies all the requirements. The method can be used for once-off decisions (Ferretti *et al.*, 2014). It is the most comprehensive MCDA method and the most detailed way to investigate the internal structure of mining corporations, as a preference function is created for each criterion. The MACBETH and AHP methods are less satisfying on this requirement as each criterion is not treated separately (Ishizaka and Nemery, 2013:6). Consequently, only the MAVT is able to analyse alternatives separately, as results are not based on a pairwise comparison. The three methods enable the decision-maker to choose between unfamiliar alternatives, as no aspirations or goals are required.

The multi-attribute value theory approach

The first step is to structure the problem. As discussed in the preceding paper and again briefly in the introduction of this paper, the type of decision for mining corporations is of a once-off nature, as the electricity source will most likely be used for the life of the mine. The implemented method intends to assist and provide more structure in the decision-making of mining corporations in South Africa to enable them to understand how renewable energy could perform as a solution for their unique needs. Based on past macroeconomic research, the choice entails the following.

- The electricity sources available were diesel generator; hybrid diesel generator/solar PV; hybrid diesel generator/onshore wind power; hybrid diesel generator/geothermal power; Eskom grid connected; hybrid Eskom grid connected/solar PV; hybrid Eskom grid connected/onshore wind power; hybrid Eskom grid connected/geothermal power (Votteler and Brent, 2016)
- The business model used to realize the potential project was self-generation in the form of own investment (Votteler and Brent, 2016).

Determine criteria

The second step was to identify the criteria that mining corporations are currently using to evaluate the decision problem stated above. The criteria, in Table II, were identified in the preceding paper in cooperation with four mining corporations.

Identify data

The third step was to develop a scale for each criterion. These scales are summarized in Table II. A local numerical scale was established for the 11 measurable criteria. A local scale entails all values of alternatives analysed, from the worst to the best. An example is provided in Figure 2 in the following section. The reason for developing scales is the fast-changing environment where values, which are based on factors like technological progress (Stewart and Belton, 2002), alter. This makes it impossible to use global scales.

A global qualitative scale was developed for the remaining five criteria to make comparability possible (Stewart and Belton, 2002). A pilot study was conducted with one expert from the renewable field and one expert from the mining industry to ensure that the qualitative scales were interpreted consistently. The data for the electricity sources to feed all criteria and to develop the scale were revealed in cooperation with four mining corporations and five different energy companies that have specialist knowledge of one or more of the sources. The data was backed up with professional literature to ensure accuracy.

Define value function and importance weight

The fourth step was to develop the value functions and assess the relative importance weights of each criterion. The value function reflects the preference of the mining corporation as the decision-maker (Stewart and Belton, 2002). Figure 2 provides an example of the operating and maintenance (O&M) cost value function. The coloured sections are explained in the section 'Criteria and value functions' below. The vertical axis represents the value to the respondent, from worst (0) to best (100). The horizontal axis gives the scale for the specific criteria. The worst value is situated on the left with zero value points while the best value is situated on the right with 100 value points.

The procedure for all four mining corporations was the same. All respondents had to go through all criteria and were asked three questions. Firstly, they had to identify the point on the scale that represents halfway in value (50) for them. If there was no preference the point would stay in the middle and a linear function would result. In the case of the examples, an initial cost reduction was more important – which results in a convex function. A more convex curve would represent a stronger preference. A possible reason for the more convex curve could be a tight budget, and the tighter the budget the more convex the function would be, as the higher prices cannot be afforded. Question two and three followed the same procedure between the value points of 0

Table II

Evaluation criteria of mining corporations

First hierarchy criteria	Second hierarchy criteria	Description	Scale
Economy	Investment cost	Investment cost includes all costs regarding the planning, purchase, and installation of the electricity source.	Numerical
	Operating and maintenance costs	Operation costs entail employees' salaries, the money spent on the energy (fuel), and the products and services for the system's operation. Maintenance costs ensure that the system is in operating condition, in order to prolong the system's life and avoid failures that result in downtime.	Numerical
	Prediction of fuel costs	This criterion provides a prediction of the price of fuel consumed to produce electricity.	Numerical
	Prediction of initial investment costs	This criterion provides an estimation of how the initial investment cost will develop. If the technology is relatively new, possible price drops can be expected.	Numerical
	Levelized electricity cost	This criterion measures the rand cost per kWh, including all costs incurred by the initial investment till the end of the predicted lifetime, which is placed in relation with the projected output of kWh in the same timespan.	Numerical
	Net present value	This is a financial method to define the total present value of a series of annual cash inflows and outflows during the lifespan. The cash flows are discounted back to the present value and summed. The final present amount is compared to the initial investment cost.	Numerical
Technology	Safety	Safety relates to the degree of safety for employees working on site	Qualitative
	Implementation period	The implementation period is the amount of time needed to realize the project.	Numerical
	Maturity	Maturity refers to the development stage of the technology. The stages range from 'only tested in laboratories' to 'close to reaching the theoretical limits of efficiency'.	Qualitative
	Service level	Service level measures the availability of experts and spare parts to repair damaged equipment.	Qualitative
Environment	CO_2 emission	CO_2 emission represents the measurement of the emission of a colourless, odourless and tasteless gas, which is mainly emitted through the combustion of coal, oil and gas.	Numerical
	Noise	Noise is the machine-created sound that disrupts human and animal daily life.	Numerical
	Land requirement	Land requirement represents the amount of land the electricity source requires for a certain capacity.	Numerical
Social	Job creation	Job creation means the number of people employed during the life cycle of an energy system.	Numerical
	Corporate image	Corporate image represents the possible impact of the electricity source on the corporate identity in the minds of diverse publics, such as customers, investors, and employees.	Qualitative
	Effect on community	The effect on the community refers to the possible impact on the surrounding residents after the mine has closed. The community could further utilize the electricity source.	Qualitative

and 50, and 50 and 100. For each value function the average of the four responses was taken. Outliers were defined as a variation of more than 15% of the numerical scale that they are measured on. No outliers occurred. The reason for translating all criteria into scales from 0–100 was to reveal the respondents' value and to equalize numbers for result calculations (Stewart and Belton, 2002). The value score (vertical axis) for each criterion was determined at the rectangular crossing point on the value function, based on its numerical value (horizontal axis). In Figure 2, an electricity source with O&M costs of 15 euros would obtain a value score of 73.

The importance weight shows the magnitude of influence that a single criterion contributes to the final decision. The procedure entailed two stages. In the first stage, the importance weight within each category, namely economy, technology, environment, and social, was identified. The questioning was always conducted in the same way. The

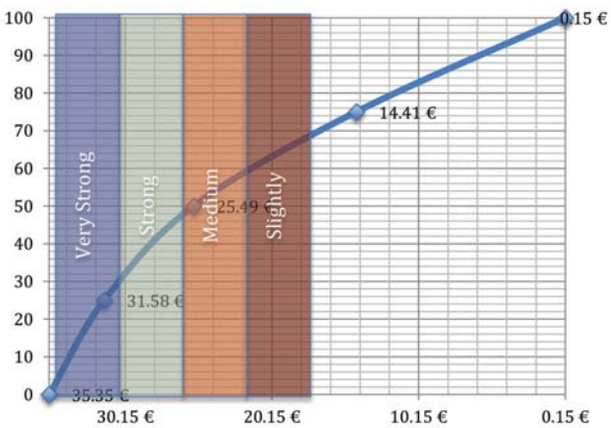


Figure 2—Example of O&M cost value function

respondent first had to identify the most important criterion; this received 100 points, followed by the second most important with less than 100 points, depending on how much less it was influencing the decision. The second stage entailed rating the categories themselves, again starting with the most important one with 100 points. To obtain the overall relative importance weight, the weight of each criterion was multiplied with the corresponding category weight (Ishizaka and Nemery, 2013).

Determine results

This investigation produced two forms of results, which are discussed in detail in a later section. First, an overall score for each electricity source is presented. The score was calculated by multiplying the relative importance weight of each criterion with the corresponding value score. The multiplied scores of each criterion were added to obtain the overall value of the electricity source. The scores were normalized and measured against the best-performing source with 100 points. The second result is represented in two matrix diagrams for each electricity source. The matrix illustrates the performance of all criteria for each source. One matrix includes the importance weights, and the other excludes them.

The MAVT for mining corporations to evaluate electricity sources

Selected electricity sources

This section introduces the electricity sources that were considered for possible solutions at mining operations. As mentioned earlier, previous research by the authors argued that the selected renewable sources in hybrid versions with current sources presently have the highest potential for mining corporations in South Africa. Table III briefly describes each source and provides specifications of the exact type of technology used. The energy companies participating in this study recommended the types of technologies.

Key values used for calculations

This section presents two different types of data tables. Table

Table III

Electricity sources

Type	Description	Specification
Diesel generator	Baseload diesel generators were considered for the purpose of this research. The reason for this was to consider the source as its own and not as a backup.	The type of diesel generators considered had 16 cylinders, 50 Hz and 1 500 rev/min. This type has a baseload output of 1 500 kVA, with a range of 100 kVA depending on the exact model.
Eskom	Eskom is South Africa's state-owned electricity producer and supplier. The megaflex tariff was used, as it applies for large-scale customers of greater than 1 MVA.	A 50 km distance to the next Eskom-Hub was assumed. A 66 kV line was considered to establish connection.
Solar PV	Horizontal single-axis solar tracker devices were used as the data for this research. The reason for this was the increased production in comparison to fixed-tilt devices, but still a simpler and robust handling in comparison to multi-tilt devices.	The data calculated on an average annual radiation level of 2 000 kWh/m^2 were selected. A capacity factor of 26% was used.
Wind	On-shore 2 MW wind turbines with an electrical frequency of 50 Hz were used.	The data were calculated on an average annual wind speed of 600 W/m^2. A capacity factor of 30% was used.
Geothermal	The 'hot dry rock' was selected. The method uses the generation heat due to radioactive decay of granites and gneisses.	The drilling depth of 4 000 m was estimated to calculate the data for South Africa. A capacity factor of 85% was used.

IV represents the general values, which were used for all electricity sources analysed, especially for the economic criteria. By applying the values it was ensured that the same foundation was used to compare sources. Owing to the relatively young market and the experimental stage in which mining corporations find themselves regarding renewables, a smaller project size of 10 MW was selected. Each value was discussed with all mining and energy companies contributing to this study and all were found to be most suitable. Where required, the values were double-checked with official sources to ensure that they were in line with international standards.

Table V provides the specific values for single and hybrid electricity sources. The values were used to calculate the data to feed the criteria of the MAVT method of this paper. The values were again provided by the energy companies and double-checked with professional, official sources. It is important to mention that the current sources, namely diesel generators and the grid connection to Eskom, are considered as the main sources. Technological standards, especially the intermittency of renewables, require the current sources to contribute continuously to the supply to ensure a stable system. Diesel generators have higher ratios than the grid connection to Eskom, because of their minimum load ratios. For illustration purposes, a 10 MW hybrid diesel-solar PV project would entail a 10 MW diesel generator and a 7.5 MW solar PV plant.

Criteria and value functions

This section represents an overview, illustrated in Table VI, of all relevant data regarding the adaptation of the MAVT method for South African mining corporations to analyse and compare current with hybrid renewable electricity sources. The left column of Table VI shows the criteria investigated, followed by the specific data for the electricity source options and lastly the results for the importance weight and value function. Most of the numerical measured criteria were again double-checked with professional, recognized official sources, including the results in Tables IV and V.

The numerical criteria data for the hybrid electricity options were calculated for fixed costs according to the project ratios in Table V. For example, the investment cost for a hybrid diesel-solar PV of €1 425.00 was calculated by: 1*€450.00+0,75*€1 300.00. The two variable cost criteria, namely the actual and predicted fuel costs, were calculated according to the electricity contribution, where the renewable electricity has priority and the conventional source is used to satisfy the remaining demand. The levelized electricity cost and net present value entail scenarios of both fixed and variable costs, and consequently both were considered in the calculations.

The relative importance weight was normalized according to the most important criterion, namely investment cost with a score of 100. The number indicates by how much more, or less, weight a criterion influences the final decision. The number in brackets indicates the ranking for illustration purposes. The last column specifies the level of preference in the value function regarding each criterion. All functions were linear or convex, consequently the column indicates the extent to which the function is convex. The extent was categorized into four levels, as can be seen in Figure 2. The

Table IV

Basic values of electricity sources

Description	Value
Project size in MW	10
Straight-line depreciation rate	15%
Tax rate[4]	28%
Discount rate[1,2]	10%
Debt percentage	50%
Equity percentage	50%
Years of loan payment[1,2]	15
Cost of debt[1,3]	13,4%
Number of years[1,2]	20

(1) Waissbein et al., 2013; (2) ETH zürich SusTec, 2014; (3) SAPVIA, 2013; (4) SARS, 2015.

Table V

Specific values for electricity sources

Description	Diesel	Eskom	Solar	Wind	Geothermal
Annual change of fuel costs , %[1,2,3,4,6,11,13]	7%	9%	0%	0%	0%
O&m cost escalation , %[6,12]	2%	2%	2%	2%	2%
Annual system degradation[4,6,7]	1%	0%	1%	1%	1%
Project ratio diesel	1		0,75	0,8	0,8
Project ratio Eskom		1	0,95	0,95	0,95
CO_2 emission kg/kWh[5,6]	0,85	1,005	0	0	0
Land requirement m^2/MW[8,9,10]	200	15 000	24 000	30 000	16 000

(1) IRENA, 2014a; (2) OPEC, 2014; (3) South African Reserve Bank, 2015; (4) Jung and Tyner, 2014; (5) Eskom, 2015a; (6) Lazard, 2014; (7) EIA, 2013; (8) NREL, 2013; (9) NREL, 2009; (10) Kagel et al., 2007; (11) Wouter, 2014; (12) Masters, 2013; (13) Eskom, 2015b.

Table VI

Summary of final criteria data

Criteria	Diesel	Diesel & Solar	Diesel & Wind	Diesel & Geothermal	Eskom	Eskom & Solar	Eskom & Wind	Eskom & Geothermal	Relative importance weight	Value function
Economic										
Investment cost per kW[1,3,5,10]	€450,00	€1 425,00	€1 330,00	€4 210,00	€400,00	€1 635,00	€1 445,00	€4 865,00	100 (1)	Medium
O&m costs per kW[5,8,3,10]	€0,15	€15,15	€10,55	€26,55	€4,00	€23,00	€16,35	€35,35	47 (8)	Medium
Fuel costs per kWh[4,8,3]	€0,290	€0,230	€0,217	€0,082	€0,058	€0,044	€0,041	€0,011	86 (3)	Medium
Prediction of fuel costs / electricity cost in 5 yrs[4,12]	€0,407	€0,323	€0,304	€0,116	€0,089	€0,067	€0,064	€0,017	85 (4)	Medium
Prediction of initial investment costs[3]	0%	12,3%	7,28%	0,0%	0%	13,6%	7,96%	0,0%	21 (15)	Medium
Levelised electricity cost (kWh)[1,3,5,]	€0,373	€0,311	€0,292	€0,158	€0,094	€0,088	€0,082	€0,076	37 (12)	Medium
Net present value	-€254 725 448	-€212 271 668	-€199 272 705	-€107 782 002	-€70 067 085	-€65 116 461	-€60 552 526	-€55 189 202	92 (2)	Slight
Technology										
Safety	9	9,5	7,5	9	8	9	7	8,5	62 (5)	Strong
Implementation period in months	3	12	15	36	36	36	36	36	47 (8)	No
Maturity	9	8	8,5	6,5	7	7	7,5	5,5	43 (10)	Medium
Service level	9	8,5	8	7	10	9	8,5	7,5	50 (7)	Medium
Environment										
Total lifetime CO_2 emission kg	1 342 766 122	1 067 145 708	890 465 955	325 090 745	1 758 999 240	1 324 091 520	1 089 910 440	253 549 440	52 (6)	Medium
Noise (1 MW)[6,9,10,11]	110	100	109	20	20	42	53	20	10 (16)	No
Total land requirement for project m^2/10 MW	2 000	182 000	242 000	130 000	150 000	378 000	435 000	302 000	8 (17)	Medium
Social										
Job creation (10 MW)[2,7,9]	12	34	24	14	1	23	13	3	42 (11)	No
Corporate image	5	10	10	10,5	8	11,5	11,5	12	34 (13)	Strong
Effect on community	5	6,5	6	5	5	6,5	6	5	31 (14)	Strong

(1) IRENA, 2014a; (2) IRENA, 2013; (3) IRENA, 2014b; (4) OPEC, 2014; (5) Fraunhofer ISE, 2013; (6) Jongens, 2007; (7) Jacobson et al., 2013; (8) Lazard, 2014; (9) Matek and Gawell, 2014; (10) Kagel et al., 2007; (11) Vestas, 2015; (12) Wouter, 2014.

point moving on 50 points of the value scale was considered. For a linear function the point is in the middle of the horizontal axis. As the point moves further to the left, the convexity of the curve increases and therefore there is a preference not to pay the high O&M costs.

To enable understanding of all data measurements, the following list introduces the criteria measured according to a qualitative scale. A global scale was used, as the scale is set and unchanging. The respondents of mining corporations were asked to rate the corporate image and effect on community criteria, as they are specific to the mining industry. All other data for the qualitative criteria was gathered from the energy companies, because of their expertise in the field. The scales that were developed and measured are presented in Table VII.

Results

Overall ranking of electricity sources

This section contains the analysis and evaluation of the results of the MCDA implemented for the purposes of this paper. The results are based on the input data introduced in the previous sections.

Table VII

Qualitative scales

Safety		
(1) Incidents that hospitalise workers occur frequently	(5) Incidents occur that force workers to rest at home	(9) Incidents causing light injuries occur infrequently
(2) Between 1 and 3	(6) Between 5 and 7	(10) Between 9 and 11
(3) Severe injuries that hospitalise workers occur sparingly	(7) Hard work causes frequent light injuries, but work can be continued	(11) No incidents of any form are expected during the lifetime of the system
(4) Between 4 and 5	(8) Between 7 and 9	
Maturity		
(1) Only performed in pilot plants, where the demonstrative goal is linked to the experimental one, in respect of operating and technical conditions	(5) Well established in the market but further efficiency improvements are expected	(9) Close to reaching the theoretical limits of efficiency
(2) Between 1 and 3	(6) Between 5 and 7	(10) Between 9 and 10
(3) Available on the market, but the improvement rate in coming years is expected to be very high	(7) Technologies could still be improved but are close to maximum efficiency	(11) Technological plateau is reached
(4) Between 3 and 5	(8) Between 7 and 9	
Service level		
(1) More than 4 weeks	(5) 1 week	(9) 1 day
(2) 4 weeks	(6) 4 days	(10) less than 1 day
(3) 3 weeks	(7) 3 days	
(4) 2 weeks	(8) 2 days	
Corporate image		
(1) Negative influence through public media	(7) Negative influence within own company	(13) Positive influence on business partners
(2) Between 1 and 3	(8) Between 7 and 9	(14) Between 13 and 15
(3) Negative influence within the industry	(9) No impact	(15) Positive influence within the industry
(4) Between 3 and 5	(10) Between 9 and 11	(16) Between 15 and 17
(5) Negative influence on business partners	(11) Positive influence within own company	(17) Positive influence through public media
(6) Between 5 and 7	(12) Between 11 and 13	
Effect on community		
(1) The pollution is negatively affecting the health standards	(4) Between 3 and 5	(7) Uplift community by leaving a source of electricity after mine closure
(2) Between 1 and 3	(5) No impact	(8) Between 7 and 9
(3) No effect on health, but factors like noise is lowering quality of life	(6) Between 5 and 7	(9) Uplift community by leaving electricity source and increase quality of life

As explained previously, two types of results were generated: firstly, the overall ranking of electricity sources analysed, and secondly, the individual performance matrix of each source based on the evaluation criteria. The overall ranking is illustrated in Figure 3. The hybrid version consisting of Eskom with solar PV was ranked as the best-performing source, followed closely by another hybrid version consisting of Eskom with onshore wind, and Eskom alone. The three hybrid versions with diesel generators, and the hybrid consisting of Eskom with geothermal power had very similar overall scores. The lowest-performing source was a diesel generator alone.

It is important to note from the results in Figure 3 that the hybrid versions of wind and solar PV along with current sources were always ranked more highly than current sources only. The difference between hybrid applications and only diesel generators was marked. However, the advantage of using only Eskom was relatively small.

Individual performance of electricity sources

The individual performances of electricity sources are presented in this section. For illustration purposes two sources are always represented in each figure, giving the score for the currently used source only and for the corresponding hybrid version. Consequently, it is possible to identify and evaluate how exactly hybrid versions performed differently from the sources used currently. The solid shaded area represents the result including the importance weights. The dotted lines represent the criteria scores only, excluding the importance weights.

The performance matrix for diesel generators and a hybrid version with solar PV is illustrated in Figure 4. The advantages of diesel generators were the low initial investment cost, the small space requirement, and the short implementation period. However, the running costs, including the expected diesel price increases, were a serious weakness. A hybrid version with solar PV increased initial investment and lowered fuel consumption. The net present value indicated that about €44 million can be saved over a period of 20 years using a hybrid version. Further advantages of a hybrid version were lower levelized costs, fewer CO_2 emissions, a better corporate image, and a more positive effect on the community.

The performance matrix for diesel generators and a hybrid version with onshore wind is illustrated in Figure 5. This hybrid was very similar to the hybrid version with solar PV above. The implementation period of 15 months for the hybrid onshore wind version was slightly longer than the 12 months required for the solar PV hybrid version. Moreover, wind had a slightly lower job creation potential. The long-term net present value savings of a hybrid onshore wind version in comparison to only diesel generators amounted to about €55 million, with a possible levelized cost reduction of €0.081.

The performance matrix of diesel and a hybrid version with 'hard rock' geothermal power in Figure 6 showed distinctive, different characteristics from the solar PV and onshore wind hybrids. The strong advantages were superior fuel-related costs, levelized costs, net present value, and CO_2 emission benefits. One reason for the advantages was the baseload characteristic of 85% of geothermal electricity

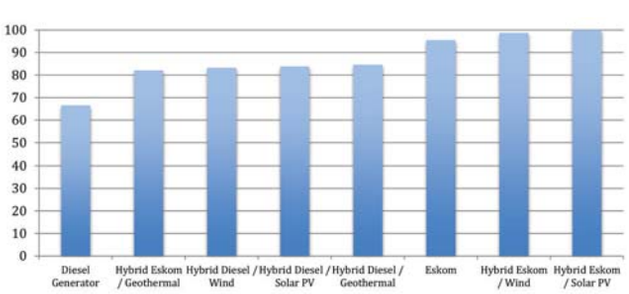


Figure 3—Overall ranking of electricity sources

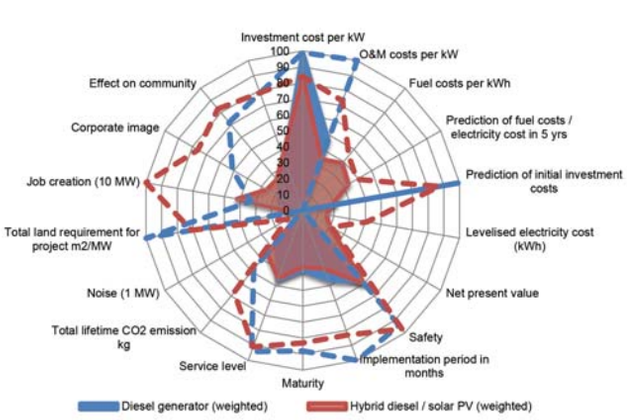


Figure 4— Performance matrix for diesel and hybrid diesel/solar PV

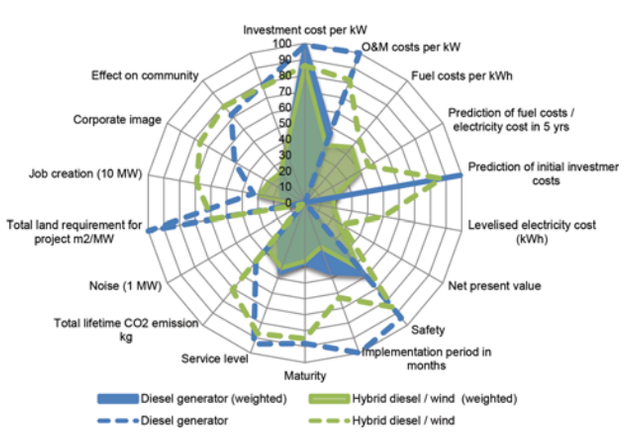


Figure 5 – Performance matrix for diesel/onshore wind

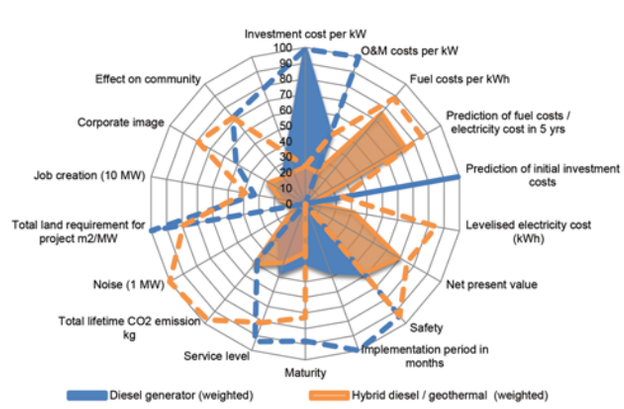


Figure 6—Performance matrix for diesel and hybrid diesel/geothermal

generation. The clear disadvantages were the extremely high initial investment costs and an implementation period of 36 months. The net present value savings amounted to approximately €147 million. All three hybrid versions with diesel showed very similar overall scores in Figure 3 and a considerably improved rating compared to only diesel generators.

Figures 7, 8, and 9 represent Eskom and the renewable hybrid versions. Eskom in hybrid versions with solar PV, onshore wind, and Eskom alone were the best-performing sources. As can be seen in Figure 3, the improvement when using Eskom alone compared with the hybrid versions was not particularly significant. The hybrid version with geothermal was ranked in the second-last position and therefore performed worse than Eskom alone.

The reason for the minimal performance improvement can be seen in the matrix for Eskom and a hybrid version with solar PV. The disadvantage of the hybrid version was again the initial investment cost and the O&M costs. Slight improvements can be seen in fuel costs, levelized costs, and net present value. The hybrid version accumulated a net present value saving of €5 million over the timespan of 20 years. Further improvements can be seen in respect of job creation, corporate image, effect on community, and CO_2 emissions.

The performance matrix for Eskom and a hybrid version with onshore wind is illustrated in Figure 8. The performance on all criteria was very similar to the solar PV hybrid version, which displayed the same pattern as the diesel versions. The wind hybrid version performed slightly better on all economic criteria and in respect of CO_2 emissions. The net present value savings in comparison to Eskom were only €10 million. However, worse performance, especially relating to safety, maturity, job creation, and space requirements, moved this option to the second ranking.

The performance matrix for Eskom and a hybrid version with geothermal power is shown in Figure 9. The overall poor ranking can be explained as being due particularly to the very high initial investment and O&M costs, which represented the worst values in comparison to the other sources, and their high importance weight. However, significant long-term benefits lay in having the lowest fuel costs, and were to be had in respect of levelized costs, net present value, and CO_2 emissions. Net present expenses at €55 million were the lowest, and were €15 million less than only Eskom.

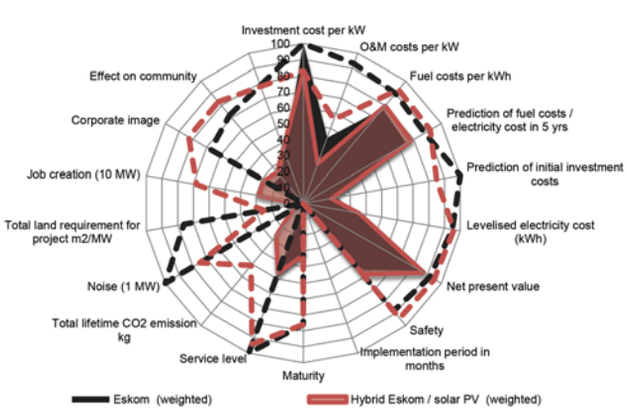


Figure 7—Performance matrix for Eskom and hybrid Eskom/solar PV

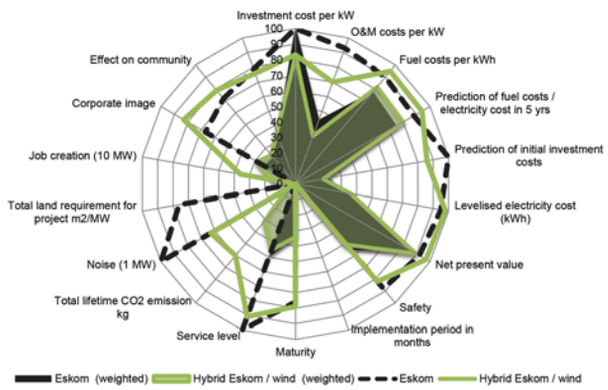


Figure 8—Performance matrix for Eskom and hybrid Eskom/wind

Discussion of results

The results show similar patterns for the use of renewables in hybrid versions with currently used electricity sources. Economically, the move to renewables has a long-term advantage as it requires a shift from constant high operational cost to an initially high capital investment with low operational expenses, especially for mines running on diesel generators. Considering the currently increasing Eskom tariffs, the benefits of using renewables will increase further.

In this study, solar PV had the greatest potential, as a vast number of mining areas are in prime solar radiated regions in central South Africa, the service structure is well developed (Votteler and Brent, 2016), and it is the best performing source from the perspective of mines connected to the Eskom grid and almost identical to all renewable hybrid versions with diesel generators.

Wind power was ranked second owing to the fact that it can be applied only on a limited scale in the coastal regions of South Africa, where there are few mines. The service infrastructure is also well developed (Votteler and Brent, 2016) and performance from the perspective of mines is only slightly behind that of solar PV. However, the economic performance of onshore wind is slightly better than that of solar PV.

Geothermal power had the weakest overall potential for mining operations in South Africa. Although it has long-term benefits from the perspective of mines, a considerable initial investment has to be made. The source did not outperform

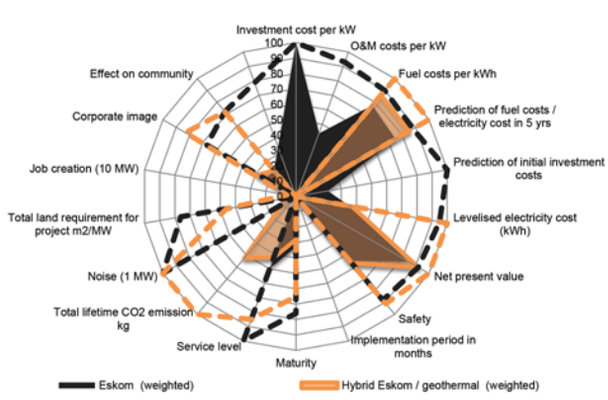


Figure 9—Performance matrix for Eskom and hybrid Eskom/geothermal

Eskom alone and is only minimally ahead of the other diesel hybrid versions. The disadvantage of a relatively young technology, in which there is little experience in South Africa, is that it increases the investment risk. Moreover, service infrastructure has not been developed yet and areas with good geothermal potential in the northeastern region of South Africa are well connected to the Eskom grid (Votteler and Brent, 2016). Nevertheless, the study has shown a great potential for future development, especially for long-term success, which could be realized with a growing geothermal industry in South Africa.

Sensitivity analysis

The following sections address how changes in the most influential input data would influence the overall results. The selected variables are the system lifespan, the predicted fuel cost, and the forecast implementation of a CO_2 tax. The lifespan is of interest for mining corporations, as lifespans of mines differ—which has an impact especially on the economic framework of the project. The prediction of tariff increases by Eskom and fuel and diesel costs have a great influence on economic criteria and were therefore double-checked with official sources. However, exact forecasts for the next 20 years can be imperfect, which necessitates an investigation into consequences of the results for worst- and best-case scenarios. Lastly, the possible impact of a CO_2 tax was considered, as all contributing mining corporations mentioned it as a point of concern.

Lifespan of the project

The lifespans considered in this section are 10 and 5 years. The criteria selected to illustrate the effect on performance are the levelized costs and the net present value, as shown in Table VIII. The options with a diesel generator remain in the same order for both criteria; nevertheless, the shorter the lifespan the smaller the advantage becomes. Options with Eskom, on the other hand, undergo a change in ranking on those two criteria. On a 20-year lifespan all hybrid sources perform better than Eskom alone. On a 10-year span only the hybrid version with onshore wind can compete. Based on 5 years, Eskom alone is the best-performing option.

Nevertheless, the overall ranking on considering all criteria does not change significantly, as the change of lifespan affects only economic criteria and CO_2 emission. The hybrid versions of Eskom with solar PV and wind are in top position, followed by Eskom alone. The only change is the shift of the diesel hybrid geothermal version from fourth to seventh place. The reason for this is the gain in weight of initial investment on a lower calculated lifespan, as there is a smaller saving on fuel costs.

Eskom tariff and diesel price forecast analysis

Changes in fuel and tariff costs of present sources over the next 20 years cannot be forecast precisely. As discussed previously, predictions were made with the aid of experts' opinions and a literature review. However, as the influence of the forecast on the running costs is considerable, a scenario analysis was conducted. The results are presented in Table IX. For illustration purposes, the levelized cost and net present value criteria were considered.

The costs of the diesel generator and hybrid version are presented in the top section of Table IX. A variation of about 5% of the predicted annual 7% diesel price increase was investigated. The different scenarios did not affect the overall ranking of the sources. The levelized costs showed a great sensitivity to a change in the diesel price, with the costs of the worst-case scenario being more than double those of the best case. The net present values showed that the usage of renewables lowers the sensitivity to fuel price changes. The differences in value between worst and best case were significantly less for the hybrid versions.

Again, the overall rankings for all Eskom versions were not affected by the changes. It is noteworthy that only the hybrid version with wind could compete with Eskom alone in the best-case scenario based on the two criteria. In the other scenarios Eskom is the most expensive version. The greater independence from tariff changes was still noticeable, but not as great as with diesel versions.

Table VIII

Project lifespan and costs

	Levelised electricity cost (kWh)			Net present value		
	5 years	10 years	20 years	5 years	10 years	20 years
Diesel generator	€ 0,262	€ 0,299	€ 0,373	-€ 81 553 544,41	-€ 149 266 230,37	-€ 254 725 448,75
Hybrid diesel / Solar PV	€ 0,230	€ 0,255	€ 0,311	-€ 71 635 936,68	-€ 127 492 730,87	-€ 212 271 668,37
Hybrid diesel / Wind	€ 0,216	€ 0,240	€ 0,292	-€ 67 199 275,49	-€ 119 637 232,19	-€ 199 272 705,30
Hybrid diesel / Geothermal	€ 0,156	€ 0,150	€ 0,158	-€ 48 375 876,12	-€ 74 820 983,01	-€ 107 782 002,70
Eskom	€ 0,061	€ 0,071	€ 0,094	-€ 20 289 802,31	-€ 38 009 734,70	-€ 70 067 085,60
Hybrid Eskom / Solar PV	€ 0,072	€ 0,074	€ 0,088	-€ 23 833 906,90	-€ 39 749 667,50	-€ 65 116 461,09
Hybrid Eskom / Wind	€ 0,066	€ 0,069	€ 0,082	-€ 21 828 424,56	-€ 36 655 721,09	-€ 60 552 526,57
Hybrid Eskom / Geothermal	€ 0,102	€ 0,086	€ 0,076	-€ 33 546 884,90	-€ 45 458 865,02	-€ 55 189 202,54

Implications of a CO_2 tax

The possible impact of the implementation of a CO_2 tax in 2016 is again illustrated with the criteria of levelized costs and net present value, as the effect is the greatest. The overall ranking of all electricity sources was not affected. A tax of R120 per ton of CO_2 was considered, with an annual increase of 10%. As it is not yet certain how Eskom is going to handle the tax, the full amount was calculated for illustration purposes (Mbadlanyana, 2013; The Carbon Report, 2015). As can be seen in Table X, the effect on levelized costs for diesel and Eskom alone amounted to just over €0.01 and €10 million net present worth. The usage of hybrid versions reduces the increase to less than €0.01 and €6 million.

Conclusion

The research presented in this paper is based on the results from the previous paper, in which the MCDA method was selected as the most appropriate research framework and the criteria that South African mining corporations used to evaluate the selected electricity sources were revealed. The purpose of this paper was firstly, to identify the most suitable MCDA method for adaptation, therefore the MAVT method was selected, and secondly to analyse and evaluate the selected sources – which was conducted in cooperation with mining and energy companies operating in South Africa.

The main results of the adapted MAVT method showed that the hybrid versions with solar PV and onshore wind were more favourable than diesel generators or the Eskom

Table IX

Eskom tariff and diesel price forecast analysis

	Levelised electricity cost (kWh)			Net present value		
	Worst case (12%)	Normal case (7%)	Best case (2%)	Worst case (12%)	Normal case (7%)	Best case (2%)
Diesel generator	€ 0,591	€ 0,373	€ 0,248	-€ 403 203 031,68	-€ 254 725 448,75	-€ 169 526 278,34
Hybrid diesel / Solar PV	€ 0,484	€ 0,311	€ 0,212	-€ 330 272 273,76	-€ 212 271 668,37	-€ 144 560 748,73
Hybrid diesel / Wind	€ 0,455	€ 0,292	€ 0,199	-€ 310 240 162,02	-€ 199 272 705,30	-€ 135 597 535,84
Hybrid diesel / Geothermal	€ 0,220	€ 0,158	€ 0,122	-€ 149 980 894,69	-€ 107 782 002,70	-€ 83 567 501,64
	Worst case (14%)	Normal case (9%)	Best case (4%)	Worst case (14%)	Normal case (9%)	Best case (4%)
Eskom	€ 0,151	€ 0,094	€ 0,062	-€ 112 187 877,94	-€ 70 067 085,60	-€ 46 281 517,24
Hybrid Eskom / Solar PV	€ 0,131	€ 0,088	€ 0,064	-€ 96 823 003,47	-€ 65 116 461,09	-€ 47 211 809,04
Hybrid Eskom / Wind	€ 0,123	€ 0,082	€ 0,059	-€ 90 656 876,65	-€ 60 552 526,57	-€ 43 552 630,87
Hybrid Eskom / Geothermal	€ 0,088	€ 0,076	€ 0,070	-€ 63 263 408,48	-€ 55 189 202,54	-€ 50 629 706,71

Table X

Impact of carbon emission tax

	Levelised electricity cost (kWh)		Net present value	
	Tax	No tax	Tax	No tax
Diesel generator	€ 0,385	€ 0,373	-€ 263 039 856,58	-€ 254 725 448,75
Hybrid diesel / Solar PV	€ 0,321	€ 0,311	-€ 218 879 434,60	-€ 212 271 668,37
Hybrid diesel / Wind	€ 0,301	€ 0,292	-€ 205 486 631,16	-€ 199 272 705,30
Hybrid diesel / Geothermal	€ 0,161	€ 0,158	-€ 110 145 044,93	-€ 107 782 002,70
Eskom	€ 0,109	€ 0,094	-€ 80 958 808,89	-€ 70 067 085,60
Hybrid Eskom / Solar PV	€ 0,099	€ 0,088	-€ 73 315 235,78	-€ 65 116 461,09
Hybrid Eskom / Wind	€ 0,093	€ 0,082	-€ 68 337 001,48	-€ 60 552 526,57
Hybrid Eskom / Geothermal	€ 0,079	€ 0,076	-€ 57 277 055,41	-€ 55 189 202,54

grid connection alone. The advantages of diesel generators were significantly greater than those of Eskom grid connection. As renewable sources are steadily advancing and the currently used sources are becoming ever more expensive, the trend will shift further towards renewables. In combining the macroeconomic influences with the MAVT results of this paper, hybrid solar versions were identified as having the greatest potential. Hybrid wind solutions were in second place, as good wind conditions occur only in coastal regions where there are fewer mining activities. Geothermal hybrid versions were selected as least favourable owing to the poor service infrastructure and high initial investment costs.

The performance matrixes indicate that the usage of renewable hybrid versions contributes to long-term success, but requires an initial shift from operational to capital expenses. Considering the overall rankings and specifically the levelized costs, renewables are already profitable with a 5-year lifespan and diesel generators. However, for Eskom hybrid versions, only wind is profitable over a 10-year span and the rest over 20 years.

This paper provides objective information for use by the management of mining corporations in South Africa. The aim is to illustrate how the relatively new opportunity of renewables could perform from the perspective of mining corporations, while at the same time considering the macroeconomic influences.

References

BOUYSSOU, D., JACQUET-LAGREZE, E., PERNY, P., SLOWINSKI, R., VANDERPOOTEN, D., and VINCKE, P. 2002. Aiding Decisions with Multiple Criteria: Essays in Honor of Bernard Roy. Kluwer, Dordrecht.

BOYSE, F., CAUSEVIC, A., DUWE, E., and ORTHOFER, M. 2014. Sunshine for mines: Implementing renewable energy for off-grid operations. Carbon War Room, Washington, DC.

CHANG, C.T. 2011. Multi-choice goal programming with utility functions. *European Journal of Operational Research,* vol. 215, no. 2. pp. 439–445.

EIA. 2013. Updated capital cost estimates for utility scale electricity generating plants. US Energy Administration Information, Department of Energy, Washington, DC.

ELIASSON, B., and LEE, Y.Y. 2003. Integrated Assessment of Sustainable Energy Systems in China. Kluwer, Dordrecht.

ERTAY, T., KAHRAMAN, C., and KAYA, I. 2013. Evaluation of renewable energy alternatives using MACBETH and fuzzy AHP multi criteria methods: The case of Turkey. *Technological and Economic Development of Economy,* vol. 19, no. 1. pp. 38–62.

ESKOM. 2015a. Climate change. http://financialresults.co.za/2012/eskom_ar2012/fact-sheets/006.php#carbon [Accessed 14 July. 2015].

ESKOM. 2015b. Tariff & charges booklet 2014/2015. Eskom, Johannesburg.

ETH ZÜRICH SUSTEC. 2014. Derisking renewable energy investment NAMA finance case study exercise. United Nations Development Program, New York.

FERRETTI, V., BOTTERO, M., and MONDINI, G. 2014. Decision making and cultural heritage: An application of the Multi-Attribute Value Theory for the reuse of historical buildings. *Journal of Cultural Heritage*, vol. 15, no. 6. pp. 644–655.

FRAUNHOFER ISE. 2013. Levelized cost of electricity: Renewable energy technologies. Fraunhofer Institute for Solar Energy Systems, Freiburg.

GELDERMANN, J. and SCHOBEL, A. 2011. On the similarities of some multi-criteria decision analysis methods. *Journal of Multi-Criteria Decision Analysis,* vol. 18, no. 3-4. pp. 219–230.

IRENA. 2014a. Analysis of infrastructure for renewable power in Southern Africa. Abu Dhabi: International Renewable Energy Agency.

IRENA 2014b. Renewable power generation costs in 2014. International Renewable Energy Agency, Abu Dhabi.

IRENA. 2013. Renewable energy and jobs. International Renewable Energy Agency, Abu Dhabi.

ISHIZAKA, A. and NEMERY, P. 2013. Multi-criteria Decision Analysis. Wiley, Chichester, UK.

JACOBSON, N., WYDER, J., FRANKLIN, S., and MCCRACKEN, P. 2013. Data collection of diesel generators in South IT Power Australia, Canberra.

JONGENS, A.W.D. 2007. Environmental noise impact assessment for scoping purposes into the establishment of a wind energy facility along the West Coast, north of the Olifants River mouth. Savannah Environmental (Pty) Ltd, Cape Town.

JUNG, J. and TYNER, W.E. 2014. Economic and policy analysis for solar PV systems in India. *Energy Policy*, vol. 74. pp. 123–133.

KAGEL, A., BATES, D., and GAWELL, K. 2007. A guide to geothermal energy and environment. Geothermal Energy Association, Washington, DC.

KEENEY, R.L. 1992. Value-Focused Thinking: A Path to Creative Decisionmaking. Harvard University Press, Cambridge, MA.

KHATIB, H. 2008. Economic evaluation of projects in the electricity supply industry. Institute of Electrical Engineering, London.

LAZARD. 2014. Levelized cost energy analysis – version 8.0. Lazard Asset Management, New York.

LINKOV, I. and MOBERG, E. 2012. Multi-criteria decision analysis: Environmental applications and case studies. Taylor & Francis, London.

MASTERS, G.M. 2013. Renewable and Efficient Electric Power Systems. 2nd edn. Wiley, Hoboken, NJ.

MATEK, B. and GAWELL, K. 2014. The economic costs and benefits of geothermal power. Geothermal Energy Association, Bochum.

MBADLANYANA, T. 2013. The political economy of carbon tax in South Africa. *Africa Insight,* vol. 43, no. 1. pp. 77–90.

NREL. 2009. Land-use requirement of modern wind power plants in the United States. National Renewable Energy Laboratory, Denver, CO.

NREL. 2013. Land use requirements for solar power plants in the United States. National Renewable Energy Laboratory, Denver, CO.

OPEC. 2014. World Oil Outlook. Organization of the Petroleum Exporting Countries, Vienna:.

PETTICREW, M. and ROBERTS, H. 2006. Systematic Reviews in the Social Sciences: A Practical Guide. Blackwell, Oxford.

ROEHRL, R. and RIAHI, K. 2000. Technology dynamics and greenhouse gas emissions mitigation: A cost assessment. *Technological Forecasting & Social Change*, vol. 63, no. 2. pp. 231–261.

ROY, B., AND MCCORD, M.R. 1996. Multicriteria Mythology for Decision Aiding. Kluwer, London.

SAPVIA. 2013. NERSA consultation on refit rates. South African Photovoltaic Industry Association, Johannesburg.

SARS. 2015. Corporate income tax. http://www.sars.gov.za/TaxTypes/CIT/Pages/default.aspx [Accessed 14 July 2015].

SOUTH AFRICAN RESERVE BANK. 2015. Inflation targets and results. https://www.resbank.co.za/MonetaryPolicy/DecisionMaking/Pages/TargetsResult.aspx [Accessed 10 July 2015].

STEFANOPOULOS, K., YANG, H., GEMITZI, A., and TSAGARAKIS, K.P. 2014. Application of the multi-attribute value theory for engaging stakeholders in groundwater protection in the Vosvozis catchment in Greece. *Science of the Total Environment*, vol. 470–471. pp. 26–33.

STEIN, E.W. 2013. A comprehensive multi-criteria model to rank electric energy production technologies. *Renewable and Sustainable Energy Reviews,* vol. 22. pp. 640–54.

STEINHAEUSER, I., SETHIA, A., JAISWAL, S., ROCHA OLIVEIRA, G., SEKINE, Y., and ALVES, L.C. 2012. Global corporate renewable energy index (CREX). Bloomberg New Energy Finance and Vestas, Copenhagen.

STEWART, T.J. and BELTON, V. 2002. Multi Criteria Decision Analysis: An Integrated Approach. Kluwer, Dordrecht.

THE CARBON REPORT. 2015. The proposed South African carbon tax. http://www.thecarbonreport.co.za/the-proposed-south-african-carbon-tax/ [Accessed 20 July 2015].

TRIANTAPHYLLOU, E. 2000. Multi-criteria Decision Making Methods: A Comparative Study. Kluwer, Dordrecht.

VESTAS. 2015. Technical specifications V100-2.0MW. Vestas Wind, Denmark.

VOTTELER, R.G. and BRENT, A.C. 2017. A mining perspective on the potential of renewable electricity sources for operations in South Africa: Part 1. The research approach and internal evaluation process. *Journal of the Southern African Institute of Mining and Metallurgy*, vol. 117, no. 3. pp. 285–297.

WAISSBEIN, O., GLEMAREC, Y., BAYRAKTAR, H., and SCHMIDT, T.S. 2013. Derisking Renewable Energy Investment. United Nations Development Program, New York.

WOUTER, F. 2014. Energy management AGA. Presentation. AngloGold Ashanti, Johannesburg.

ZIKMUND, W.G. and BABIN, B.J. 2010. Exploring Marketing Research. 10th edn. Thomson,.Mason, OH. ◆

INTERNATIONAL ACTIVITIES

2017

20 April 2017 — Proximity Detection and Collision Avoidance Systems in Mining Colloquium 2017
Striving for zero harm from mining mobile machinery
Emperors Palace, Hotel Casino Convention Resort, Johannesburg
Contact: Camielah Jardine
Tel: +27 11 834-1273/7, Fax: +27 11 838-5923/833-8156
E-mail: camielah@saimm.co.za,Website: http://www.saimm.co.za

9–12 May 2017 — 6th Sulphur and Sulphuric Acid 2017 Conference
Southern Sun Cape Sun, Cape Town
Contact: Camielah Jardine
Tel: +27 11 834-1273/7, Fax: +27 11 838-5923/833-8156
E-mail: camielah@saimm.co.za, Website: http://www.saimm.co.za

20–27 May 2017 — ALTA 2017 Nickel-Cobalt-Copper, Uranium-REE and Gold-PM Conference and Exhibition
Pan Pacific Perth, Australia
Contact: Allison Taylor
Tel: +61 (0) 411 692 442
E-mail: allisontaylor@altamet.com.au
Website: http://www.altamet.com.au/conferences/alta-2017/

6–7 June 2017 —Mine Planning Colloquium 2017
Mintek, Randburg, South Africa
Contact: Camielah Jardine
Tel: +27 11 834-1273/7, Fax: +27 11 838-5923/833-8156
E-mail: camielah@saimm.co.za, Website: http://www.saimm.co.za

7–8 June 2017 —AIMS - Aachen International Mining Symposia Second International Conference: Mining in Europe
Aachen, Germany
Contact: Iris Schümmer
Tel: +49-(0) 241-80 95673, Fax: +49-(0) 241-80 92272
E-mail: aims@mre.rwth-aachen.de
Website: http://www.aims.rwth-aachen.de

19–20 June 2017 —Chrome Colloquium 2017
What's next for Chrome? A debate on the tough questions
Mintek, Randburg, South Africa
Contact: Camielah Jardine
Tel: +27 11 834-1273/7, Fax: +27 11 838-5923/833-8156
E-mail: camielah@saimm.co.za, Website: http://www.saimm.co.za

25–28 June 2017 — Emc 2017: European Metallurgical Conference
Leipzig, Germany
Contact: Paul-Ernst-Straße
Tel: +49 5323 9379-0, Fax: +49 5323 9379-37
E-mail: EMC@gdmg.de, Website: http://emc.gdmb.de

27–29 June 2017 —4th Mineral Project Valuation Colloquium
Mine Design Lab, Chamber of Mines Building,
The University of the Witwatersrand, Johannesburg
Contact: Camielah Jardine
Tel: +27 11 834-1273/7, Fax: +27 11 838-5923/833-8156
E-mail: camielah@saimm.co.za,Website: http://www.saimm.co.za

10–11 July 2017 — Water 2017 Conference
Lifeblood of the Mining Industry
Emperors Palace, Hotel Casino Convention Resort, Johannesburg
Contact: Camielah Jardine
Tel: +27 11 834-1273/7, Fax: +27 11 838-5923/833-8156
E-mail: camielah@saimm.co.za,Website: http://www.saimm.co.za

25–26 July 2017 — Entrepreneurship in Mining Forum
A Focus on new Business in the Value Chain
Johannesburg
Contact: Camielah Jardine
Tel: +27 11 834-1273/7, Fax: +27 11 838-5923/833-8156
E-mail: camielah@saimm.co.za, Website: http://www.saimm.co.za

3–4 August 2017 — Building a Robust Mineral Industry
Thriving under prolonged low commodity price environment
Mandel Training Centre, Marlborough, Harare
Contact: Camielah Jardine
Tel: +27 11 834-1273/7, Fax: +27 11 838-5923/833-8156
E-mail: camielah@saimm.co.za,Website: http://www.saimm.co.za

7–9 August 2017 —Rapid Underground Mine & Civil Access Conference 2017
Emperors Palace, Hotel Casino Convention Resort, Johannesburg
Contact: Camielah Jardine
Tel: +27 11 834-1273/7, Fax: +27 11 838-5923/833-8156
E-mail: camielah@saimm.co.za, Website: http://www.saimm.co.za

15–16 August 2017 —The SAMREC and SAMVAL Codes
Advanced Workshop: Can you face your peers?
Emperors Palace, Hotel Casino Convention Resort, Johannesburg
Contact: Camielah Jardine
Tel: +27 11 834-1273/7, Fax: +27 11 838-5923/833-8156
E-mail: camielah@saimm.co.za,Website: http://www.saimm.co.za

22–24 August 2017 — The biennial Southern African Coal Processing Society Conference and Exhibition
'Coal Processing – the key to profitability'
Graceland Hotel, Casino and Country Club, Secunda
Contact: Gerda Craddock, Tel: +27 11 432-8918, E-mail: gerdac@mineralconcepts.co.za, Website: www.sacoalprep.co.za

30 August–1 September 2017 — MINESafe Conference 2017
Striving for Zero Harm—Driving Excellence through Compliance
Emperors Palace, Hotel Casino Convention Resort, Johannesburg
Contact: Camielah Jardine
Tel: +27 11 834-1273/7, Fax: +27 11 838-5923/833-8156
E-mail: camielah@saimm.co.za,Website: http://www.saimm.co.za

11–15 September 2017 — Uranium 2017 International Conference
Extraction and Applications of Uranium — Present and Future
Swakopmund Hotel, Swakopmund, Namibia
Contact: Camielah Jardine
Tel: +27 11 834-1273/7, Fax: +27 11 838-5923/833-8156
E-mail: camielah@saimm.co.za,Website: http://www.saimm.co.za

30 September–6 October 2017 — AfriRock 2017: ISRM International Symposium—*Rock Mechanics for Africa*
Cape Town Convention Centre, Cape Town
Contact: Camielah Jardine
Tel: +27 11 834-1273/7, Fax: +27 11 838-5923/833-8156
E-mail: camielah@saimm.co.za,Website: http://www.saimm.co.za

17–20 October 2017 — AMI Precious Metals 2017
The Precious Metals Development Network (PMDN)
Protea Hotel Ranch Resort, Polokwane
Contact: Camielah Jardine
Tel: +27 11 834-1273/7, Fax: +27 11 838-5923/833-8156
E-mail: camielah@saimm.co.za,Website: http://www.saimm.co.za

18–20 October 2017 — 7th International Platinum Conference
Platinum—A Changing Industry
Protea Hotel Ranch Resort, Polokwane
Contact: Camielah Jardine
Tel: +27 11 834-1273/7, Fax: +27 11 838-5923/833-8156
E-mail: camielah@saimm.co.za, Website: http://www.saimm.co.za

25 October 2017 — 14th Annual Student Colloquium
Johannesburg
Contact: Camielah Jardine
Tel: +27 11 834-1273/7, Fax: +27 11 838-5923/833-8156
E-mail: camielah@saimm.co.za, Website: http://www.saimm.co.za

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
Air Liquide (PTY) Ltd
AMEC Foster Wheeler
AMIRA International Africa (Pty) Ltd
ANDRITZ Delkor (Pty) Ltd
Anglo Operations (Pty) Ltd
Arcus Gibb (Pty) Ltd
Aurecon South Africa (Pty) Ltd
Aveng Engineering
Aveng Mining Shafts and Underground
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
Bell Equipment Limited
Blue Cube Systems (Pty) Ltd
CDM Group
CGG Services SA
Concor Mining
Concor Technicrete
Cornerstone Minerals Pty Ltd
Council for Geoscience Library
Cronimet Mining Processing SA (Pty) Ltd
CSIR Natural Resources and the Environment (NRE)
Data Mine SA
Department of Water Affairs and Forestry
Digby Wells and Associates
DRA Mineral Projects (Pty) Ltd
DTP Mining - Bouygues Construction
Duraset
Elbroc Mining Products (Pty) Ltd
eThekwini Municipality
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 Group
Geobrugg Southern Africa (Pty) Ltd
Glencore
Goba (Pty) Ltd
Hall Core Drilling (Pty) Ltd
Hatch (Pty) Ltd
Herrenknecht AG
HPE Hydro Power Equipment (Pty) Ltd
IMS Engineering (Pty) Ltd
Ivanhoe Mines SA
Joy Global Inc.(Africa)
Kudumane Manganese Resources
Leco Africa (Pty) Limited
Longyear South Africa (Pty) Ltd
Lonmin Plc
Magotteaux (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
MineRP Holding (Pty) Ltd
Mintek
MIP Process Technologies (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
PANalytical (Pty) Ltd
Paterson & Cooke Consulting Engineers (Pty) Ltd
Perkinelmer
Polysius A Division Of Thyssenkrupp Industrial Sol
Precious Metals Refiners
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
Sandvik Mining and Construction RSA(Pty) Ltd
SANIRE
Schauenburg(Pty) Ltd
SENET (Pty) Ltd
Senmin International (Pty) Ltd
Smec South Africa
SMS group Technical Services South Africa (Pty) Ltd
Sound Mining Solution (Pty) Ltd
South 32
SRK Consulting SA (Pty) Ltd
Technology Innovation Agency
Time Mining and Processing (Pty) Ltd
Tomra (Pty) Ltd
Ukwazi Mining Solutions (Pty) Ltd
Umgeni Water
Webber Wentzel
Weir Minerals Africa
WorleyParsons RSA (Pty) Ltd

Forthcoming SAIMM events...

EXHIBITS/SPONSORSHIP

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

For the past 123 years, the Southern African Institute of Mining and Metallurgy, has promoted technical excellence in the minerals industry. We strive to continuously stay at the cutting edge of new developments in the mining and metallurgy industry. The SAIMM acts as the corporate voice for the mining and metallurgy industry in the South African economy. We actively encourage contact and networking between members and the strengthening of ties. The SAIMM offers a variety of conferences that are designed to bring you technical knowledge and information of interest for the good of the industry. Here is a glimpse of the events we have lined up for 2017. Visit our website for more information.

SAIMM
THE SOUTHERN AFRICAN INSTITUTE OF MINING AND METALLURGY

For further information contact:
Conferencing, SAIMM
P O Box 61127, Marshalltown 2107
Tel: (011) 834-1273/7
Fax: (011) 833-8156 or (011) 838-5923
E-mail: raymond@saimm.co.za

SAIMM DIARY

2017

- COLLOQUIUM
  Proximity Detection and Collision Avoidance Systems Colloquium 2017
  20 April 2017, Emperors Palace, Hotel Casino Convention Resort, Johannesburg
- CONFERENCE
  6th Sulphur and Sulphuric Acid 2017 Conference
  9–12 May 2017, Southern Sun Cape Sun, Cape Town
- COLLOQUIUM
  Mine Planning Colloquium 2017
  6–7 June 2017, Mintek, Randburg
- COLLOQUIUM
  Chrome Colloquium 2017
  19–20 June 2017, Mintek, Randburg
- COLLOQUIUM
  4th Mineral Project Valuation Colloquium
  27–29 June 2017, Mine Design Lab, Chamber of Mines Building, The University of the Witwatersrand, Johannesburg
- CONFERENCE
  Water 2017: Lifeblood of the Mining Industry Conference 2017
  10–11 July 2017, Emperors Palace, Hotel Casino Convention Resort, Johannesburg
- FORUM
  Entrepreneurship in Mining Forum
  25–26 July 2017, Johannesburg
- CONFERENCE
  Building a Robust Mineral Industry
  3–4 August 2017, Mandel Training Centre, Marlborough, Harare
- CONFERENCE
  Rapid Underground Mine & Civil Access 2017 Conference
  7–9 August 2017, Emperors Palace, Hotel Casino Convention Resort, Johannesburg
- WORKSHOP
  The SAMREC and SAMVAL Codes—Advanced Workshop: Can you face your peers?
  15–16 August 2017, Emperors Palace, Hotel Casino Convention Resort, Johannesburg
- CONFERENCE
  MINESafe Conference 2017
  30 August–1 September 2017, Emperors Palace, Hotel Casino Convention Resort, Johannesburg
- CONFERENCE
  Uranium 2017 International Conference
  11–15 September 2017, Swakopmund Hotel, Swakopmund, Namibia
- SYMPOSIUM
  AfriRock 2017: ISRM International Symposium *'Rock Mechanics for Africa'*
  30 September–6 October 2017, Cape Town Convention Centre, Cape Town
- CONFERENCE
  AMI Precious Metals 2017 'The Precious Metals Development Network (PMDN)'
  17–20 October 2017, Protea Hotel Ranch Resort, Polokwane
- CONFERENCE
  7th International Platinum Conference
  18–20 October 2017, Protea Hotel Ranch Resort, Polokwane
- COLLOQUIUM
  14th Annual Student Colloquium
  25 October 2017, Johannesburg

THE top line matters...

The current economic climate means companies need to work more efficiently to reap the benefit of an upswing in the market.

"This has been one of the key pillars of the success of MIP Process during the last few years. We are able to maximise our resources and this allows us to provide faster and more cost efficient solutions"

says Philip Hoff, Managing Director

MIP Process is one of few Level 1 BEE Contributors in the industry they serve. We are proud of this and have strategies in place to maintain this status, Hoff affirms.

In order to strengthen the staff complement, a General Manager was appointed together with experienced engineers in various fields. This has enhanced the top structure of the organisation. This strategy together with client partnerships has proved to be successful if one looks at the amount of repeat customers we have, says Hoff.

MIP Process has executed numerous projects outside the African continent. They have supplied a flocculant plant to the Philippines, numerous thickeners and flocculant plants to Asia. MIP recently supplied an order for counter-current decantation (CCD) thickeners for a Copper Project in the Democratic Republic of Congo, which has expanded MIP's thickener solutions offering.

The acquisition of Alliance Dust Control (ADCS) secured in 2013, enhanced MIP's product range and they offer the complete dust extraction including bagfilters, scrubbers and dry cyclones.

The success of this acquisition can be seen in that two scrubber plants, for the removal of sulphur dioxide, was recently supplied for two different Copper/Cobalt projects in the DRC.

A contract for the supply of a taphole extraction system, for a larger chrome producer, with a substantial size bagfilter, is presently being executed.

MIP Process will be exhibiting at Electra Mining again in 2016. This is taking place from 12th to 16th September 2016 at Nasrec and is one of the biggest of its kind in the world. "MIP Process will be in Hall 6, Stand No. H05 as usual", concludes Hoff.

Our Current product range includes:

- Attrition Scrubbers
- Agitators
- Bagfilters
- Clarifiers
- Cyclones for dust extraction
- Depressant Pumps
- Horizontal Linear Screens
- Mixers
- Reagent Make-up Plants
- Screen-covers for vibrating screens
- Slurry Samplers
- Thickeners
- Wet Scrubbers

mip PROCESS TECHNOLOGIES

MIP Process Technologies (Pty) Ltd. | P.O. BOX 677, Kelvin 2054 | Tel + (0) 11 234 1007
Fax: +27 (11) 86 682 2394 | E mail: sales@mipprocess.com | Website: www.mipprocess.com
MIP House, 4A Coombe Place, Rivonia, 2128