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Journal Comment

Mine Planning and Equipment Selection (MPES 2015)

The Southern African Institute of Mining and Metallurgy (SAIMM) hosted the *23rd International Symposium on Mine Planning and Equipment Selection (MPES 2015)* at the Sandton Convention Centre in Johannesburg from 9 to 11 November 2015. This was the first time that South Africa has hosted the MPES in its 25-year history. This conference's theme was '*Smart Innovation in Mining*' in order to recognize technological innovations and new ideas that are required to prepare the industry for the mine of the future.

This special edition of the *Journal* contains a selection of papers from MPES 2015 that discuss issues and present ideas on innovation that span the mine value chain. The world is still experiencing depressed commodity prices since the global financial crisis of mid-2008, and the industry continues to face downsizing challenges. It is imperative that innovation becomes second nature to the mining industry if we are to survive and emerge stronger from the current economic abyss. Papers in this edition of the Journal provide some insight into ideas applicable for innovation.

The paper on productivity is highly relevant in the current context. Mining productivity has declined by nearly 50% since 2001, according to the US Bureau of Labour Statistics and a McKinsey Report published in May 2015. Two of the papers examine issues pertaining to mechanization in the hard rock mining environment – but, is mechanization a panacea for the woes that bedevil the industry?

Uncertainty in mining systems is covered in two papers that discuss the use of simulation as a handy tool to address uncertainty in order to better manage risk. This approach supports the notion that it is better to be approximately right than to be precisely wrong. A major challenge facing simulation projects is how to communicate a wide range of answers where a single answer is required for decision-making. We need to apply our minds to how we can resolve this challenge.

Two other papers discuss economic or financial aspects related to mining. These are also pertinent in the realm of mine planning and equipment selection because we need to mine economically if we are to sustain our operations.

Sometimes we need to go back to basics to be innovative. This is where the two papers on improving our understating of rock masses and rock behaviour are relevant. Arguably, we occasionally need to take that one step back in order to make that leap forward. This takes us forward to the vision of the mine of the future.

What do we need as an industry to prepare ourselves for the mine of the future? Certainly, it will be business unusual in the mine of the future. Innovation and technology as articulated by the papers in this edition of the *Journal* are critical for propelling the mining industry forward, but they also bring an attendant increase in the complexity of mining organizations in terms of data and systems integration. We are not, and will not in the future, be immune to 'Big Data' challenges. As can be gleaned from the paper on integrating load and haul equipment selection and replacement decisions across multiple periods, we will have to be smarter about integrating our systems to facilitate decisions across multiple periods and organizational silos.

It is my hope that everyone will find the papers in this volume insightful as they contribute towards innovative ideas that are required to imagine, and bring to reality, the mine of the future.

C. Musingwini
Organizing Committee Chairman

A few personal memories of David Rankin

David Rankin passed away on 20 December 2015, just two days short of his 84th birthday.

He was a member of the SAIMM for 59 years, having joined as a student member in November 1956. He subsequently became a Member in 1970 and was elected a Fellow in January 1971.

Dave Rogans, a former work colleague and friend of David Rankin, writes:

Dave joined Amcoal, the Anglo Coal Division, in 1973 and when I followed him a year later he was busy commissioning Kriel and Arnot collieries to supply the adjacent Eskom power stations. At this stage our production was less than 11 million tons per annum and much of it came from highly labour-intensive mines (lots of shovel work). At the end of his career, Dave left a company with a capacity of 60 million tons per annum with more modern mining methods in use.

When Jim Braithwaite retired in 1980, David became Managing Director of the Division and I was appointed as his deputy. We sat in two corner offices with an office for our secretary in between, which meant that our contact was always close. There was much to do and he was always very fair in the distribution of the workload and, more importantly, never selfish.

His achievements included the negotiation with SAR to build the line to Richards Bay which enabled us to produce coal for export. This required two new mines and the upgrading of a third. More significantly, it involved difficult negotiations with our overseas customers, especially the Japanese, who had much more experience in this field when buying coal for their steel industry.

His relationship with Eskom was excellent and resulted in two more large collieries and coal supply contracts which both parties liked! A senior Eskom engineer mentioned to me that the then chairman was concerned that Anglo always had the best proposals and he felt that they were becoming too dependent on us! He asked his department to tickle up other suppliers to match Anglo's proposals.

New Vaal Colliery was unique in that it was mining coal, using opencast methods, that had previously been mined by underground methods. When one mines a 4-metre seam using pillar methods and takes only the bottom 2 metres, then a lot of coal is left behind. As I remember, Dave calculated that a field that looked mined out had 94% of the coal left in it!

New Denmark colliery was also remarkable in that it was the first time that longwall mining was used in South Africa, so that all the coal in the seam was recovered.

New Vaal and New Denmark each supplied 3600 megawatt power stations.

I quote the mines that Dave was responsible for developing to illustrate his great imagination, drive, and courage. I found it astonishing that I never knew him to show any sign of irritation or lose his temper … we had a strong Chairman!

I was close to Dave for some 16 years until my retirement in 1990. I have always believed that you can learn a lot about a person by knowing the children. My wife and I watched Jane and Ann grow into wonderful people and Dave must have been very proud of them; it was much his doing.

D. Rogans

President's Corner

Sustainability

When I travel on aeroplanes, I love to look out of the window, either at the popcorn-shaped cumulus clouds or the striated repeating ripple-patterned undulatus clouds, or the wispy feathery cirrus clouds. While enjoying this ephemeral beauty, I marvel at the atmospheric phenomena (and their governing mathematical equations) that are behind these structures. From the vantage point of 10 km up in the sky, the miniature-looking features on the ground can also be enjoyed. There are hills and valleys, snow-topped peaks, wide open deserts, forests and fields, rivers and lakes. I find that I can easily flip between seeing the world as fragile or as resilient, for both are true.

Apart from continents drifting slowly apart, and the occasional impact of an asteroid, or volcanic explosion, the earth has been relatively stable for a very long time, perhaps as long as 4.5 billion years, and scientific estimates say that we have another 6 billion years to go until the expanding sun eventually burns out our planet.

Yet, we are so reliant on our planet being positioned where it is relative to the sun, so that we receive sufficient radiant energy to keep our world in the very narrow band of temperature and pressure that supports life. The unique properties of water (H_2O) allow life to exist on planet Earth, and we need fresh water on a daily basis in order to survive. When seen from afar, the atmosphere that provides the air we breathe seems like a very thin skin surrounding our world. The planet Venus is often seen as having some similarities to Earth – a solid surface, almost the same diameter, a similar mass, similar rock composition, and a comparable distance from the sun; but instead of our life-supporting mixture of oxygen and nitrogen in just the right amounts, the atmosphere of Venus is dominated by carbon dioxide and is completely covered by clouds of sulphuric acid. The temperature and pressure of the Venusian atmosphere are much too high to support human life.

My university final-year design project involved the production of a chloro-fluorocarbon (CFC) refrigerant that was seen at the time as a safe, harmless substance. Only a few years later, it was seen as responsible for the destruction of part of the ozone layer that protects our world. Bill Bryson, in his book *A Short History of Nearly Everything*, tells the story of a hapless Ohio inventor called Thomas Midgely, Junior, who set out to create a gas that was stable, non-flammable, non-corrosive, and safe to breathe. The CFCs he invented went into rapid production in the 1930s and were used in numerous applications, from air-conditioners to deodorant sprays. It took half a century before it was noticed that these long-lived CFCs were destroying ozone at a prodigious rate, and they were subsequently banned. This same inventor had also earlier in the 1920s invented tetraethyl lead, an inexpensive compound that was effective at stopping engines from knocking. Unfortunately, the widespread use of lead (a known neurotoxin) as a gasoline additive led to the death of numerous workers and to damaging the health of countless other people, despite the denials of harmfulness from the producers. One of Midgely's other inventions led directly to his own death in 1944, when he was strangled in the cord of a device he had created for turning him and lifting him in bed (as he was crippled by polio).

One of the controversies of our present age revolves around the extent to which mankind is causing climate change, and what can be done about it. The rapid increase in human population with the onset of industrialization led to the consumption of vast quantities of fossil fuels, with the result that the atmosphere now contains around 400 parts per million of carbon dioxide, whereas the pre-industrial atmosphere contained only about 280 parts per million. Much evidence has been presented to show that this has led to a progressive warming of the surface of the planet through the greenhouse effect.

Sustainability *(continued)*

The current rate of change in the mean temperature of the Earth is at levels that are unprecedented in the past 100 000 years, and it is this rate that distinguishes today's climate change from past fluctuations. The climate is also becoming more variable, with more extreme conditions becoming more frequent than in the past. Will human beings, and the intricately connected food chain and environment, be able to adapt to these changes fast enough?

Jared Diamond wrote a book in 2005 entitled *Collapse*, and subtitled *How Societies Choose to Fail or Survive*. *Collapse* presents an attempt to understand why so many past societies collapsed, leaving behind only ruined or abandoned temples, pyramids, and monuments. Diamond examines why ancient once-productive societies, including the Maya, the Anasazi of the desert areas of the American Southwest, the Easter Islanders, and the Viking colonies of Greenland, as well as modern ones such as Rwanda, have fallen apart. Diamond identifies five factors that contribute to collapse: climate change, hostile neighbours, collapse of essential trading partners, environmental problems, and failure to adapt to environmental issues because of various political, economic, and social factors. Deforestation was a major factor in the decline of Easter Island's Polynesian society, famous for erecting giant stone statues. Because trees take so long to re-grow, deforestation has more severe consequences than crop failure, and can trigger disastrous erosion, leading to the wind blowing off the island's thin topsoil, resulting in starvation. It's hard to imagine what the person who cut down the island's last tree was thinking. Environmental factors are not the only ones at play: policy also matters, as can be seen in the two countries sharing the Caribbean island of Hispaniola – Haiti is a failure relative to the Dominican Republic. Diamond also argues that humanity collectively faces similar challenges to those outlined for individual societies, and rightly warns of alarming trends in biodiversity, soil loss, freshwater limits (China is depleting its aquifers at a breakneck rate), and over-fishing (much of the developing world relies on the oceans for protein).

I have yet to meet someone who, when genuinely challenged, would seriously suggest that we should stop all industrial activity such as the mining of metals. For example, there is no doubt in my mind that the use of stainless steel in kitchens and hospitals has brought about a better standard of health and life for mankind in general. Perhaps the challenge for us in the mining industry is to use the resources of our planet responsibly, mining the minerals we need, but refraining from the profligate and greedy exploitation that has sometimes taken place. We should be mindful of the impact that mining has on the environment and society, and should take care to clean up after ourselves as best as possible, building in responsible mine closure and rehabilitation to our plans. We should admit that there are places in the world where mining should not take place, even if the pure profit motive would dictate otherwise. The conservation of our natural environment, scenery, and wildlife is something we owe to our children and grandchildren.

This resilient but fragile planet is the only home we have, so let's look after it well.

R.T. Jones
President, SAIMM

South African National Committee on Tunnelling

Final breakthrough at the Hallandsås tunnel marks an outstanding pioneering achievement

HERRENKNECHT Tunnelling Systems

The unshakable will of clients, powerful construction companies, and innovative tunnelling technology make it possible to deal with even the greatest challenges in tunnel construction. This is demonstrated by the successful completion of tunnelling work at the Hallandsås Tunnel in Sweden. At the beginning of September 2013, tunnelling on one of the world's most complex and, due to the geological conditions, the most challenging tunnel projects was completed. In the Hallandsås Tunnel project, the Herrenknecht Multi-mode-TBM cut through the last metres of rock to the target shaft, and this second breakthrough marked a victory of the people and the innovative tunnelling technology involved against a particularly intractable mountain massif. After 8 years of unyielding tunnelling, this is a grand triumph for all those involved in the project. The mountain massif had previously proved to be invincible against other tunnel construction methods.

Båstad, Sweden / Schwanau, Germany, September 23, 2013

The railway route along the Swedish west coast from Malmö to Göteborg is one of the vital arteries for the country's passenger and freight traffic. When expanding this route, the Hallandsås mountain range south of Båstad in Sweden was the decisive bottleneck, since it could be passed only in one-track operation until now. With the breakthrough of the Herrenknecht TBM, the shell of the twin-bore Hallandsås Tunnel – which will increase capacity – was completed on 4 September 2013. Transport Minister Catharina Elmsäter-Svärd, representatives from the construction companies, the Swedish Transport Administration Trafikverket, and from the local and national administrations, as well as numerous representatives from the media celebrated this outstanding project success. 'We have shown that it is possible to build a tunnel of high quality trough the complicated Hallandsås, while at the same time meeting the high environmental requirements. Our competent and dedicated co-workers are today worthy of every recognition for our common achievement', said Per Rydberg, project manager of the Swedish Transport Administration at the breakthrough ceremony.

Due to its geology, the project occupies a top position on the list of tunnel projects with extremely complex ground conditions. Large sections of the very abrasive rock formations (mainly gneiss and amphibolite) with high rock strengths of up to 250 MPa are extremely fissured. In addition, the tunnel is exposed to extreme groundwater pressures of more than 10 bar on large parts of the route. Earlier attempts to construct a tunnel failed to pass this hurdle and led to strict environmental requirements that limited the quantity of groundwater (to the litre) allowed to drain during tunnel construction between Förslöv and Båstad, for example.

For the mechanized tunnelling – which was the last resort to implement the project – Herrenknecht developed and delivered a specially adapted tunnel boring machine (TBM) for the two remaining 5.5 km long sections of the overall 8.7 km long Hallandsås Tunnels. The high-tech colossal machine (Multi-mode TBM, Ø 10 530 mm) was designed to work in both the closed slurry mode with hydraulic removal of excavated material and the open hard-rock mode with belt conveyor removal. Permanently installed drilling and injection tools ensured water inflow could be controlled by grout injection when needed. As part of comprehensive test series, the sealing system of the machine was designed to withstand a groundwater pressure of up to 13 bar. Werner Burger, Head of Design Department Traffic Tunnelling at Herrenknecht said: 'The machine design for Hallandsås was both a response to the extreme project requirements and a large technological advance: the concept aimed to provide a hard-rock machine with the potential to work safely and efficiently in loose rock and even under high groundwater pressure if needed. Hallandsås has set the right course for later projects.'

The Swedish-French joint venture between the companies Skanska and Vinci started tunnelling on the first (eastern) tunnel in September 2005. Tunnelling teams and engineers approached the optimum working

Final breakthrough at the Hallandsås tunnel marks an outstanding pioneering achievement *(continued)*

method in the tunnel in close and trusting cooperation of machine supplier and construction companies. The best results were obtained with the machine named 'Åsa' in the open mode with cement injections that kept the groundwater at bay. High abrasivity and blocky rock on parts of the route caused an extremely high wear of material at the cutter head, and the maintenance and tool change intervals were correspondinglyshort. However, soon after the start of tunnelling, site reports showed regular progress, although time-consuming cement grouting and the required service intervals meant limited speed. Nevertheless, man and machine were able to force some hundred metres of tunnel per month.

Then, spring 2008 saw the first victory of many with the breakthrough into the cavern of a mid-adit, which had been excavated with conventional methods. On that occasion the highly worn cutter head was replaced by a new one with larger disc cutters (19 inch instead of 17 inch). Finally, in August 2010 the entire first tunnel was successfully completed. The machine was completely refurbished and again equipped with a new cutter head for excavating the second (western) tunnel. The Herrenknecht Field Service rendered active assistance on the site with competent staff and the supply of spare and wear parts: they assisted in the assembly of the machine on the site, during tunnelling in both tunnels, and in the restructuring and refurbishment work for the second drive.

From February 2011 onwards, specialists from the Skanska-Vinci joint venture drove the machine from Förslöv towards Båstad for the second tunnel, where they were welcomed and celebrated by a large and festive party at final breakthrough on 4 September 2013. Dr.-Ing. E.h. Martin Herrenknecht was present at this outstanding event: 'This breakthrough is an absolutely great moment in my life. A big success for all those involved. It is like the moon landing of tunnel construction. Nobody else has been here before us.' After almost 8 years of tunnelling this finish marked an outstandingly pioneering achievement in the construction of underground infrastructure. Thanks to state-of-the-art tunnelling technology and trusting cooperation of all project partners, even a tunnel project which seemed to be impossible could be managed safely for people and the environment.

Hallandsås Tunnel

Location	Förslöv, Sweden	Herrenknecht Multi-mode TBM S-246	
Application	Railway	Diameter	10 530 mm
Tunnel length TBM	1st tunnel 5480 m 2nd tunnel 5445 m	Installed power	4000 kW
Geology	Gneiss, amphibolite, diabase	Torque	20 370 kNm
Contractor	Trafikverket (formerly Banverket)	Customer	Skanska/Vinci HB (Skanska Sweden AB; Vinci Construction Grands Projets)

SANCOT

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

Papers – Mine Planning and Equipment Selection Conference

Estimating cost of equity in project discount rates: comparison of the Capital Asset Pricing Model and Gordon's Wealth Growth Model
by A.S. Nhleko and C. Musingwini. 215

This study considers the commonly applied Capital Asset Pricing Model (CAPM) and Gordon's Wealth Growth Model to estimate the cost of equity, and explores how differences in the cost of equity obtained by these two methods can be explained for a mining environment.

Estimating mine planning software utilization for decision-making strategies in the South African coal mining sector
by B. Genc, C. Musingwini, and T. Celik. 221

A new methodology has been developed to define and measure mine planning software utilization in the South African coal mining sector within an evolving data-set framework. The methodology can be used to review existing software combinations and inform decision-making strategies for software utilization.

Determination of value at risk for long-term production planning in open pit mines in the presence of price uncertainty
by M. Rahmanpour and M. Osanloo. 229

A procedure is presented in this paper to determine the value at risk (VaR) in any possible mine planning alternative. The VaR is considered, together with downside risk and upside potential, to select the most profitable and least risky plan. The model is tested on a small iron ore deposit.

Impact of the South African mineral resource royalty on cut-off grades for narrow, tabular Witwatersrand gold deposits
by C. Birch 237

This study investigates the impact of the mineral resource royalty on mineable reserves and the resultant life-of-mine and total income for various gold mines, as well as on a single gold mine, at various profitability ratios. Each individual mine is affected differently, due primarily to differences in grade-tonnage curves and profitability ratios.

Rock Strength and Geometallurgical Modelling, Mogalakwena Mine
by J. P. Germiquet and R.C.A. Minnitt 247

This work investigates the domaining of grain-size-adjusted uniaxial compressive strength (UCS) to allow more accurate scheduling of drilling rigs through increased knowledge of rock strength in various areas. Successful rock strength domaining has the potential to be incorporated into blast indexing and in predicting crushing/milling performance.

Microscopic analyses of Bushveld Complex rocks under the influence of high temperatures
by G.O. Oniyide and H. Yilmaz. 251

The effect of temperature on the physical and chemical properties of selected rocks from the Bushveld Complex was studied by means of optical microscopy and scanning electron microscopy using energy-dispersive X-ray spectrometry (SEM-EDX). Minor physical changes were observed, but the temperature range considered (50–140°C) is not high enough to induce changes in the chemical composition.

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

Papers – Mine Planning and Equipment Selection Conference *(continued)*

The effect of geological uncertainty on achieving short-term targets: a quantitative approach using stochastic process simulation
by M. Soleymani Shishvan, and J. Benndorf 259

This paper presents a stochastic-based mine process simulator capturing different sources of uncertainties. It aims to quantify the effect of geological uncertainty and its impacts on the ability to deliver contractually defined coal quantities and qualities, as well as the efficiency of a large continuous mining system in terms of equipment utilization. A case study of a large continuous coal mine is used to evaluate the results.

The importance of people in the process of converting a narrow tabular hard-rock mine to mechanization
by D. Vogt and T. Hattingh 265

A published model for technological progress from 'art' to 'science' is presented and adapted to the mining industry. The paper argues that a mechanized mining approach requires a different mix of skills from the workforce, and in particular a management system that allows for higher levels of personal initiative.

Improvement in the overall efficiency of mining equipment: a case study
by H. Fourie 275

This case study describes an initiative to improve equipment performance which developed into a comprehensive turnaround plan for the Mogalakwena platinum mine and put it at the forefront of performance achievement.

Trends in productivity in the South African gold mining industry
by P.N. Neingo and T. Tholana 283

The performance of the South African gold mining industry, in terms of labour productivity and the industry cost curve position, is analysed for the period 2006–2013 to assess the impacts of both the global financial crisis and labour unrest.

Paper of General Interest

Critique of the South African squat coal pillar strength formula
by M. Mathey and J.N. van der Merwe 291

The arguments which have been proposed in favour of the South African squat coal pillar strength formula are discussed critically in this paper. As a result, an alternative design criterion for squat pillars in South Africa is suggested.

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Estimating cost of equity in project discount rates: comparison of the Capital Asset Pricing Model and Gordon's Wealth Growth Model

by A.S. Nhleko* and C. Musingwini*

Synopsis

Since the Global Financial Crisis (GFC) in mid-2008, capital has been more difficult to access. Mining projects must contend with projects from other industries for scarce capital. A decision to invest available capital in mineral projects requires that valuation be conducted to assess the expected return on the projects. The discounted cash flow (DCF) analysis is generally applied for the valuation of mining projects, whereby future cash flows are discounted to present value using an appropriate discount rate. The discount rate significantly affects the outcome of a valuation. Economic and finance theory provides tools to calculate discount rates. The discount rate must account for factors such as the risk and stage of development of the mineral project; hence the appropriate discount rate to utilize in a project is often a subject of debate.

The discount rate is the weighted sum of the cost of debt and equity. There are several methods for determining the cost of equity. This study considers the commonly applied Capital Asset Pricing Model (CAPM) and Gordon's Wealth Growth Model because of their simplicity and availability of parameters required to estimate the cost of equity. This study explores how differences in the cost of equity obtained by these two methods can be explained for a mining environment. Data for empirical analysis were collected from the I-Net Bridge, McGregor BFA, and Bloomberg databases. It was found that Gordon's Wealth Growth Model provides better estimates of the cost of equity compared to the CAPM under depressed market conditions. Therefore, this research recommends that Gordon's Wealth Growth Model be used to estimate the discount rates for mining projects during periods of depressed market conditions.

Keywords
Capital Asset Pricing Model, Gordon's Wealth Growth Model, discount rate, cost of equity, mining projects.

Introduction

Ever since the Global Financial Crisis (GFC) in mid-2008, investment capital for mining projects has been a scarce resource as mining companies have to compete with projects from other sectors (Njowa *et al.*, 2013) while markets remain depressed. It is imperative that valuation of mining projects is treated with caution as decisions to invest in mineral projects are based on the expected return derived from valuations. The discounted cash flow (DCF) analysis is commonly accepted as the principal method for valuing mining projects (Park and Matunhire, 2011; Smith *et al.*, 2007; Janisch, 1976). In DCF analysis, an appropriate discount rate is applied to discount future cash flows to present value. The discount rate is important because it significantly impacts the outcome of a valuation. Economic and finance theory provides valuable tools to calculate discount rates. However, there is often uncertainty on an appropriate discount rate to apply to a project, as the discount rate must account for such factors as the risk and stage of development of the mineral project. The Weighted Average Cost of Capital (WACC) is the discount rate that is used for cash flows with risk profiles similar to that of the overall company. The WACC is the average after-tax cost of capital, which is computed as the weighted sum of the cost of debt and equity. The cost of debt is derived from the interest rate adjusted for the tax rate, normally fixed for the length of the loan. The cost of equity can be calculated using the commonly applied Capital Asset Pricing Model (CAPM) or Gordon's Wealth Growth Model, even though there are other less commonly used methods such as the Arbitrage Pricing Theory (APT).

The cost of equity is defined as the expected return on an asset's common stock in capital markets (Witmer and Zorn, 2007). There is a risk that the investor may not receive the expected return; therefore, investors are expected to take the risk of the investment into account when determining the returns they want to receive. There is a relationship between risk and expected return on a stock; the greater the risk, the greater the expected return on investment the investor expects (Fehr, 2010).

It is vital, then, that a proper analysis of a stock is done in order to determine its true value and forecast future returns. According to Witmer and Zorn (2007), estimating the cost of equity is not a straightforward exercise; different assumptions and methods result in

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

different answers. Hence, this study undertook an analysis of the cost of equity estimation by comparing the commonly applied CAPM and the Gordon's Wealth Growth Model.

The CAPM relies on historical data to estimate the beta (β) value, which is used to calculate forward-looking returns. This process is based on the premise that past performance of an entity is a good estimator of expected future returns. However, this proposition is not entirely correct because there are periods in the past where unexpected returns were realized due to events not captured by the β-value. Therefore, unreliability in estimated β-values results in the need to explore the use of forward-looking models such as Gordon's Wealth Growth Model in estimating a more appropriate figure for the cost of equity.

Gordon's Wealth Growth Model is based on the principle that dividends grow at a constant rate to perpetuity. It is difficult to realize this proposition because of volatility in earnings and uncertainty in estimates of expected inflation and real growth in the economy (Stowe *et al.*, 2007; Damodaran, 2002).

The assumptions upon which the CAPM and Gordon's Wealth Growth Model are based may result in difficulty when applied to real investment problems. Consequently, caution must be exerted to appreciate the constraints of the underlying assumptions. This raises the question: '*How can differences in the cost of equity obtained by these two methods be explained in a mining environment*?' Selection of an appropriate discount rate is central to an accurate and valid assessment of the value of any mineral project. The discount rate is applied to cash flows in estimating the present value of an asset and is used as the rate of return for an investment. A discount rate that is lower than the true rate will overvalue the project, resulting in the commissioning of an uneconomic project. A discount rate that is higher than the true rate will undervalue the project, resulting in the rejection of a financially viable project. For this reason, it is essential to estimate the discount rate as close to the true discount rate as possible. Therefore, valuation should incorporate a thorough and objective analysis to obtain an appropriate discount rate reflecting the acceptable returns matching with the project's risk profile and market conditions (Ballard, 1994). It is vital to identify a model that can be employed to reliably estimate the appropriate discount rate.

Capital Asset Pricing Model

The prices of securities result from different analyses of different information accompanied by different conditions and preferences relevant to a particular investor. Therefore, it is necessary to employ some standard principles when estimating prices of securities. The CAPM describes the relationship between risk and return in an efficient market. An efficient market is one where the market price is an unbiased estimate of the intrinsic value of the investment (Damodaran, 2002). The CAPM is regarded as a single factor model because it is based on the premise that the expected rate of return can be predicted by using a single factor, the systematic risk.

The systematic risk predominant in any investment is represented by beta (β), which is calculated as the historical volatility of a company's share prices compared to the market and is therefore a proxy for risk. A minimum level of return required by the investor is realized when the expected return $E(R_i)$ is equal to the actual return on an asset; this is known as risk-free return (R_f). The CAPM assumes that an investor will only hold a market portfolio. A market portfolio (m) is defined as a portfolio in which an investment into any asset is equal to the market value of that asset divided by the market value of all risky assets in the portfolio. Equation [1] is the CAPM equation for estimating the rate of return.

$$E(R_i) = R_f + \beta_i\left[E(R_m) - R_f\right] \qquad [1]$$

where $E(R_i)$ is the expected return (cost of equity) on an asset, i, and R_f is the risk-free rate and can be obtained from a totally safe investment (Rudenno and Seshold, 1983). When estimating the risk-free rate it is important to use a rate on long-term Treasury bonds (T-bonds) because mining stocks are long-term securities; Treasury bills are more volatile than T-bonds. When using the CAPM to estimate the cost of equity, the theoretical holding time horizon is the life of the project. Therefore, it is a logical choice to use the rate on long-term T-bonds as a proxy for the risk-free rate. The term $E(R_m)- R_f$, represents the market risk premium (Brigham and Ehrhardt, 2007).

Gordon's Wealth Growth Model

Gordon's Wealth Growth Model was initially developed by Gordon and Shapiro in 1956, and later refined by Gordon in 1962, based on the premise that dividends grow at a constant rate in perpetuity. Nonetheless, this assumption does not hold in reality because projections of dividends cannot be made for an indefinite period; hence, various versions of the dividend discount model have been developed. These models were developed based on different assumptions concerning future growth. The simplest form of the dividend discount models is Gordon's Wealth Growth Model and it is used to value a stock of a company that has stable growth and pay dividends regularly (Stowe *et al.*, 2007; Damodaran, 2002).

This model assumes that the stock is equal to the present value of all its future dividend payments. The predicted dividends are discounted back to their present values. Gordon's Wealth Growth Model is useful when evaluating entities having well-established policies on dividend payouts and a growth rate equivalent to or lower than the small growth in the economy (Damodaran, 2002). Companies may have different expected growth rates, but there is evidence that dividends growth rates for mature companies are similar to the nominal gross domestic product (GDP) rate. Nominal GDP is given by real GDP plus inflation (Brigham and Ehrhardt, 2007). The model's expected rate of return is calculated by using Equation [2].

$$r = \frac{D_{t-1}(1+g)}{P_t} + g = \frac{D_t}{P_t} + g \qquad [2]$$

where

R represents the investors' required rate of return (discount rate)

D_{t-1} represents dividends at the present time (paid in the previous period)

D_t represents dividends at the next consecutive time (paid in the next period)

P_t represents current stock price
g represents the constant growth rate of the dividend stream.

In order to obtain a reliable estimate of the expected rate of return, it is imperative that the expectations of investors are reflected on the stable growth rate and future dividends. Estimating reliable and impartial forecasts of future dividends, their timing, and growth patterns for deriving cost of equity is regarded as the main challenge in using dividend discount models.

According to Foerster and Sapp (2005) and Whitcutt (1992), the growth rate (g) can also be estimated using the nominal GDP since it is argued that GDP is the maximum sustainable growth rate for a company's dividend. However, using GDP growth rates to approximate long-term growth rate in dividends seems to work well at estimating the dividends for the stock of a mature and dividend-paying company. There is evidence that the dividend and GDP growth rates have a positive correlation (Foerster and Sapp, 2005).

Approach to comparing CAPM and Gordon's Wealth Growth Model

This study is limited to mining companies listed on the Johannesburg Securities Exchange (JSE). The JSE was selected because it:

- Is a member of the World Federation of Exchanges
- Complies with the global standards and legislative requirements
- Acts as a regulator to its members, ensuring that markets operate in a transparent manner, safeguarding investors
- Ensures accurate and adequate disclosure of all information relevant to investors (City of Johannesburg, 2014)
- JSE data can be accessed for an adequate length of time.

Prior to June 1995, the sectors of the JSE were defined differently; therefore to use data prior to this period, one would have to reconstruct each sectoral index in order to obtain comparable data. Reconstruction of the data is beyond the scope of this study, thus only data post-1995 was considered. The time horizon period for the study is January 1998 to December 2012 because the JSE was illiquid prior to 1998. The study considered top mining companies, by market capitalization, as the smaller companies are thinly traded and may experience long periods of mispricing, which would adversely affect the findings if it occurs.

In order to apply Gordon's Wealth Growth Model effectively, a company has to have stable growth and pay dividends regularly. Small mining enterprises do not pay dividends frequently and have variations in growth rates, thus it will be futile to determine their discount rates using this model. According to Tholana *et al.*, (2013) gold, platinum, and coal are the most economically vital minerals in South Africa; hence, the focus of this study is restricted to these commodities. Coal-mining companies (excluding the multi-commodity companies) were not considered as they cannot be classified as 'stable growth' companies due to failure to pay dividends regularly. Based on the market capitalization, the top three mining companies in the gold and platinum sectors quoted on the JSE over the period of the study were selected.

The six mining companies used were:

- Platinum: Anglo American Platinum Limited, Lonmin Plc, and Impala Platinum Holdings Limited
- Gold: AngloGold Ashanti Limited, Harmony Gold Mining Company Limited, and Gold Fields Limited.

The databases used to source data for this study are I-Net Bridge, McGregor BFA, and Bloomberg. The WACC values obtained from the Bloomberg database were split into debt and equity components. The equity component of WACC was used to benchmark against the estimates of CAPM and Gordon's Wealth Growth Model.

The average market premium and risk-free rate (T-bonds) used in South Africa for the year 2013 are 6.8% and 6.4%, respectively (Fernandez *et al.*, 2013). These estimates were adopted for this study in order to reduce the inaccuracy in the estimated cost of equity. The values were calculated for the preceding 60 months using ordinary least squares (OLS) linear regression, covariance of stock returns against the market returns, and adjusted using the Blume's technique. The adjusted coefficient values were used to estimate the cost of equity rates.

Gordon's Wealth Growth Model is based on the premise that dividends grow at a constant rate in perpetuity. Subsequently, the GDP rate was applied as an alternative to company-specific growth rates because the latter are not constant over time. The GDP growth rate in South Africa averaged 3.16% in real terms from 1993 until 2014, as shown in Figure 1.

For the purposes of illustrating the approach followed in comparing CAPM and Gordon's Wealth Growth Model for all the six mining companies, data and analysis is presented on Anglo American Platinum Limited. The discount rate estimates for Anglo American Platinum Limited for the period from financial year 2002 (FY2002) to FY2012 are shown in Figure 2. The estimates are divided, according to market conditions, into three phases: 'A' (market boom), 'B' (recession), and 'C' (steady economic growth). In Phase A, commodity prices soared drastically due to increased global demand for platinum group metals (PGMs). The headline earnings of Anglo American Platinum increased due to high US dollar prices realized for metals sold and weaker rand/US dollar exchange rates.

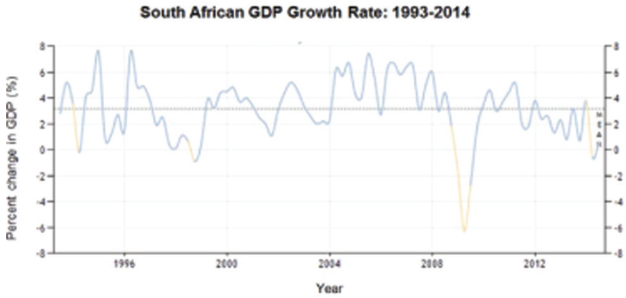

Figure 1—South African real GDP growth rate: 1993–2014 (Taborda, 2014)

Although the demand for PGMs kept on growing, South African producers did not gain as anticipated from the increased metal prices, citing operational challenges as the main cause. These challenges resulted in reduced metal supply into the market. These challenges were, *inter alia*, industrial action, safety-related production stoppages, shortage of skilled labour, and processing bottlenecks. Inability to achieve set production targets and supply demands may impact a company negatively, which is evident in an increasing risk profile. The effect of these challenges can be seen in Figure 2, where values for the equity component of WACC increased steadily during Phase A.

During Phase B, the Gordon's Wealth Growth Model trend for the estimate cost of equity rates shows a sharp decrease, while that of CAPM is flat when compared to the increasing trend for the equity component of WACC. The GFC curbed the demand for PGMs, triggering a price decline. Anglo American Platinum experienced a decrease in headline earnings per ordinary share of 95% in FY2009 due to depressed US dollar prices realized on metals sold.

In Phase C, in the second half of 2009, there were signs of market recovery with a consequent increase in metal demand and recovering prices. In FY2009, the company did not pay dividends, citing the need to retain cash for maintenance of operations as the major reason. Anglo American Platinum adopted cost management strategies (curtailing operations that are not adding value such as putting high-cost shaft Siphumelele 3 on care and maintenance), which had an effective contribution to the company's performance (Anglo American Platinum Limited, 2010). It can be seen from Figure 2 that both the CAPM and Gordon's Wealth Growth Model failed to estimate the actual discount rates during periods of economic instability observed throughout the period under study.

Descriptive statistics for Anglo American Platinum

The descriptive statistics tool was adopted in order to analyse the data from the equity component of WACC, CAPM, and Gordon's Wealth Growth Model. Descriptive statistics allow data to be presented in a more meaningful manner, from which simpler interpretation can be performed. The relationship between actual and estimated discount rates was analysed to check how similar the values are by looking at the mean squared error (MSE), mean, standard deviation, range, sum, correlation coefficient, and box and whisker plot.

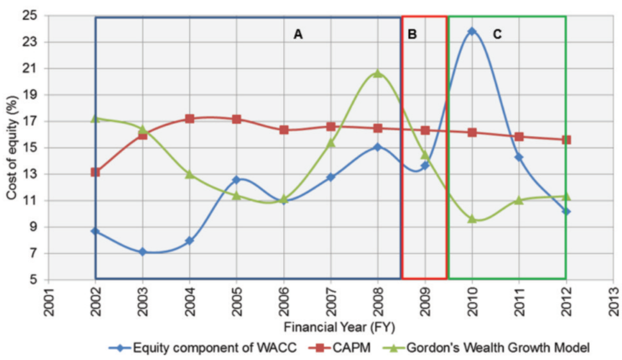

Figure 2—Cost of equity for Anglo American Platinum from FY2002 to FY2012

Estimated discount rates using CAPM and Gordon's Wealth Growth Model are contrasted with the equity component of WACC values in Table I using MSE. The actual cost of equity for Anglo American Platinum in FY2010 was 23.78%, which is higher than in other years. This may be attributed to the reaction to the GFC that curbed the demand for the PGMs, and lower US dollar prices on metals sold in 2009 (Anglo American Limited, 2009).

The mean square error between the equity component of WACC and either of the models must be zero to prove that there is similarity between actual and estimated cost of equity values. From Table I, it can be seen that CAPM and Gordon's Wealth Growth Model produce similar estimates for the MSE that are very close to zero. However, when considering the years individually, Gordon's Wealth Growth Model produces more values of MSE close to zero. The main problem with using the unweighted mean as a measure of analysis is that all values are assumed to have the same weighting. Therefore, when there is a wider range in values, as observed with Gordon's Wealth Growth Model, the calculated mean is biased towards narrowly spread values (Massart *et al.*, 2005). Hence, it is important that other measures are employed in an analysis.

Statistical analysis using some of the measures is presented in Table II for the CAPM and Gordon's Wealth Growth Model against the equity component of WACC. Gordon's Wealth Growth Model has a mean of 0.14, which is close to that of the equity component of WACC (0.12), while CAPM has a mean of 0.16.

The standard deviation for Gordon's Wealth Growth Model is similar to (approximately 74% of) that of the equity component of WACC, while that of CAPM is about 24%. Therefore, the spread of data around the average for the equity component of WACC and Gordon's Wealth Growth Model is almost identical. As with the analysis using MSE, the range of Gordon's Wealth Growth Model is close to that of the equity component of WACC, whereas CAPM differs by approximately 76%. The main cause of this disparity is the wide spreads of Gordon's Wealth Growth Model and the equity component of WACC values, whereas the estimates of CAPM have a narrow spread.

The calculation yields the same results as with the other measures, Gordon's Wealth Growth Model showing superiority over CAPM. The cost of equity estimates from Gordon's Wealth Growth Model vary from 9.62% to 17.25%, while the CAPM estimates vary from 13.13% to 17.18%. Hence, the average discount rates are 13.78% and 16.07% for Gordon's Wealth Growth Model and CAPM, respectively, proving that CAPM yields higher estimates.

Correlation of the equity component of WACC rates with the cost of equity estimates from CAPM and Gordon's Wealth Growth Model was tested. Perfect correlation exists if all values lie on a straight line and the correlation coefficient is unity. The correlation coefficient values are very weak for both models (see Table II); however, Gordon's Wealth Growth Model has a better correlation with the equity component of WACC than CAPM. The spread for CAPM data distribution is narrower, as illustrated in Figure 3. The interquartile ranges (IQRs) for the equity component of WACC and Gordon's Wealth Growth Model are 0.05 and 0.05, respectively. The IQR for CAPM is 0.01, which is significantly different from

Table I

Cost of equity and the mean squared error for Anglo American Platinum

Year	Equity component of WACC	CAPM (%)	Difference (%)	MSE	Gordon's Wealth Growth Model	Difference (%)	MSE
2002	8.65	13.13	4.48	0.002	17.25	8.60	0.007
2003	7.11	15.95	8.84	0.008	16.39	9.28	0.009
2004	7.93	17.18	9.25	0.009	13.01	5.08	0.003
2005	12.53	17.16	4.63	0.002	11.38	-1.15	0.000
2006	10.99	16.36	5.37	0.003	11.13	0.14	0.000
2007	12.75	16.60	3.85	0.001	15.37	2.62	0.001
2008	15.01	16.47	1.46	0.000	20.63	5.62	0.003
2009	13.62	16.32	2.70	0.001	14.46	0.84	0.000
2010	23.78	16.15	-7.63	0.006	9.62	-14.16	0.020
2011	14.28	15.84	1.56	0.000	11.03	-3.25	0.001
2012	10.13	15.61	5.48	0.003	11.34	1.21	0.000
Average	12.43	16.07	3.64	0.003	13.78	1.35	0.004

Table II

Descriptive statistics for Anglo American Platinum

Measure	Equity component of WACC	CAPM	Gordon's Wealth Growth Model
Mean	0.124	0.161	0.138
Standard deviation	0.046	0.011	0.034
Range	0.167	0.041	0.110
Correlation coefficient	1.000	0.191	0.309
Sum	1.368	1.768	1.516

that of the equity component of WACC; this is expected because the spread of the CAPM data is narrower (see Table I).

Summary for Anglo American Platinum

The cost of equity estimates using the CAPM and Gordon's Wealth Growth Model failed to predict the actual discount rates for Anglo American Platinum. Descriptive statistics were used to check for similarity in the data for CAPM and Gordon's Wealth Growth Model and the equity component of WACC rates. The statistical measures show that the data for Gordon's Wealth Growth Model and the equity component of WACC are similar, with only MSE results favouring CAPM when looking at the means.

Summary for all six companies

The same methodology used to estimate the cost of equity and compare it to the equity component of WACC was applied in analysing the other companies. The cost of equity values estimated using Gordon's Wealth Growth Model produced a similar trend for all platinum companies, and the same applies for all gold companies used in the study. On the other hand, the CAPM produced different trends for different companies and this can be attributed to the individual risk profiles of the companies. The correlation coefficient measure was used to summarize the findings of this study. The rating system used to analyse the ability of CAPM and Gordon's Wealth Growth Model to estimate the cost of equity for mining companies is shown in Table III.

The colour classifications in Table IV were assigned values in order to calculate an overall rating score for CAPM and Gordon's Wealth Growth Model. A summary of the findings of this study using the correlation coefficient are shown in Table IV.

The Gordon's Wealth Growth Model has a higher overall rating compared to CAPM. Hence, Gordon's Wealth Growth Model was chosen as the better model to estimate the cost of equity for mining company projects.

Conclusions and recommendations

Since the GFC in mid-2008, it has been difficult for mining projects to access capital as mining companies have to compete with other sectors for this scarce resource. Therefore, it is vital to determine reliable project values. A widely applied valuation technique is DCF analysis, which uses appropriate discount rates to discount future cash flows to present values. There are numerous methods that can be applied to determine discount rates; this study considered Gordon's Wealth Growth Model and the Capital Asset Pricing Model (CAPM). These methods were chosen because of their simplicity and availability of the parameters required to

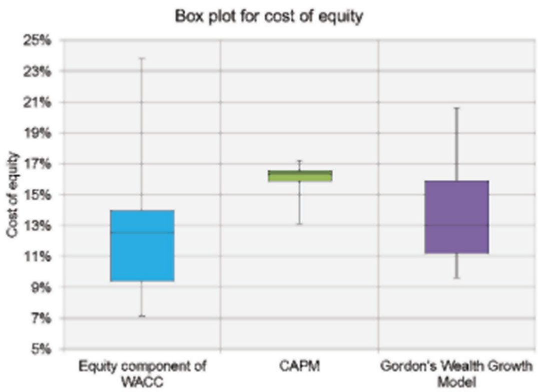

Figure 3—Box and whisker plot for Anglo American Platinum cost of equity

Table III

Rating system for models used to estimate the cost of equity

Coefficient of correlation	0–0.4	0.4–0.6	0.6–1.0
Rating	Poor	Moderate	Good
Colour	Red	Yellow	Green
Rating value	1	2	3

Table IV

Summary of findings based on correlation coefficient

Company	CAPM		Gordon's Wealth Growth Model	
	Correlation coefficient	Rating	Correlation coefficient	Rating
Anglo American Platinum	0.191	1	0.309	1
Impala Platinum	0.421	2	0.246	1
Lonmin	0.640	3	0.640	3
AngloGold Ashanti	0.024	1	0.899	3
Gold Fields	0.355	1	0.490	2
Harmony	0.017	1	0.767	3
Overall rating		9		13

estimate the discount rates. Based on market capitalization as quoted on the JSE over the period January 1998 to December 2012, the top three platinum and top three gold mining companies were selected for the study. Data used for empirical analysis was obtained from I-Net Bridge, McGregor BFA, and Bloomberg databases. The CAPM closely captured the actual cost of equity throughout the period under study, while Gordon's Wealth Growth Model captured the trend for the actual equity component of Weighted Average Cost of Capital (WACC) values. However, Gordon's Wealth Growth Model consistently underestimated the cost of equity. This can be attributed to the fact that economic market movements do not immediately affect mining companies.

The data-set for Gordon's Wealth Growth Model was similar to that of the equity component of WACC for both gold and platinum mining companies. There was a high correlation between Gordon's Wealth Growth Model and the equity component of WACC for gold mining companies, whereas a weak correlation between the equity component of WACC and both CAPM and Gordon's Wealth Growth Model was identified for platinum mining companies. However, CAPM had a narrower spread than Gordon's Wealth Growth Model throughout, suggesting that it is a better model for estimating cost of equity. It is therefore recommended that investors, practitioners, and/or researchers consider using Gordon's Wealth Growth Model when estimating cost of equity for mining companies under depressed economic market conditions, given that this method uses *ex-ante* data to estimate *ex-ante* returns.

Acknowledgement

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Estimating mine planning software utilization for decision-making strategies in the South African coal mining sector

by B. Genc*, C. Musingwini*, and T. Celik†

Synopsis

Mine planning software continues to be an important factor when it comes to the development of the South African mining industry. To contribute to this development, a new methodology to define and measure mine planning software utilization in the South African coal mining sector within an evolving data-set framework was developed. An initial data-set showing the mine planning software providers, their corresponding software solutions, as well as the software capabilities and information on the number of licences was collected and compiled in 2012 in an online database for software utilized in the South African mining industry. Details of the database development and implementation were published in the *Journal of the Southern African Institute of Mining and Metallurgy* in 2013. In 2014 the data-set was updated with additional and new information.

In this paper, using the 2012 and 2014 time-stamps, a methodology for estimating the software utilization is developed. In this methodology, three variables: commodity, functionality, and time factor are used to define and measure the software utilization in order to ultimately inform decision-making strategies for software utilization. Utilization in the coal sector was measured according to six different functionalities, namely Geological Data Management, Geological Modelling and Resource Estimation, Design and Layout, Scheduling, Financial Valuation and Optimization. The methodology is useful for stakeholders reviewing existing software combinations or intending to purchase new software in the near future and wanting to estimate the comparative attractiveness of a certain software package. These stakeholders include mining companies, consulting companies, educational institutions, and software providers. The work presented in this paper is part of a PhD research study in the School of Mining Engineering at the University of the Witwatersrand.

Keywords

coal mining, coal sector software utilization, database, South African mining industry.

Introduction

This paper outlines the development of a new methodology to define and measure mine planning software utilization in the South African coal mining sector. Although the calculations can be done for any commodity, in this paper calculations were done only done only for coal. Coal, which is used to generate electricity, accounted for almost 26% of South Africa's mining income during 2013 (Statistics South Africa, 2014).

An initial data-set showing the mine planning software providers, their corresponding software solutions, as well as the software capabilities and information on the number of licences was collected and compiled in 2012 in an online database. The database development and implementation was published in the *Journal of the Southern African Institute of Mining and Metallurgy* in 2013 (Katakwa, Musingwini, and Genc, 2013). In 2014 the data-set was updated with additional and new information. Using the updated data-set, a methodology was developed to measure mine planning software utilization in the coal sector in order to ultimately inform decision-making strategies for utilization of the coal sector's software.

Software utilization

Utilization is an important factor as it is often associated with the level of productivity in the South African mining industry. According to the Oxford English Dictionaries (2014), the root of the word 'utilization' comes from the word 'utilize', meaning 'make practical and effective use of'. Hence software utilization can be defined as the effective use of mine planning software, but in general, utilization is associated with the overall equipment effectiveness, which is one of the key performance-based metrics. Overall equipment effectiveness (OEE) is not only one of the most widely used metrics to determine performance against capability of the equipment, but is also commonly used as a key performance indicator (KPI) in Total Productive Maintenance (TPM) and Lean Manufacturing programmes for measuring production efficiency (Vorne Industries, 2008). Detailed information

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regarding overall equipment effectiveness was documented by Genc, Musingwini, and Celik (2015).

OEE is designed for measuring equipment utilization, or hardware utilization, and there is a need for a definition of software utilization, which can be used to establish a framework towards defining strategic mine planning software utilization.

Some researchers have tried to define software utilization using the number of techniques available, such as system-user interaction data to understand how often the software is being used as well as to what degree it is being used. (El-Ramly and Stroulia, 2004). However, this approach cannot be used to measure mine planning software utilization, considering the size of the whole South African mining sector and user privacy. Due to these reasons, a methodology was developed in such a way that utilization of the various mine planning software solutions that are available could be measured. The next section defines this measurement framework.

Utilization framework

Software utilization can be defined by associating many-to-many, one-to-many, and many-to-one relationships between entity types. In this association, the relationship between software vendors, commodity, functionality, and time factor were used to develop the following terminology:

$$\{C_i, F_l\} \rightarrow S_{k=\{i,l\}}$$

where C_i denotes commodity (i) and F_l denotes functionality (l). Furthermore, S_k is the software that performs tasks on commodity (i) according to functionality (l). In the market there is usually more than one software solution specifically designed for commodity (i) and functionality (l). Genc, Musingwini, and Celik (2015) gave a detailed explanation about the terminology which defined utilization: $u^{(m)}_{i,l}$ is the utilization of the software that performs task on commodity (i) and functionality (l) by using software (m). Although there is no rigid definition of software utilization, it may be defined as a numeric value that falls in to the range between 0 and 1 inclusive, *i.e.*

$$u^{(m)}_{i,l} \in [0,1]$$

thus enabling further analysis of software utilization.

Furthermore, the utilization formula can be extended by considering the time factor (t):

$$u^{(m,t)}_{i,l} = f^{(m,t)}_{i,l} . w^{(m,t)}_{i,l}$$

where $f^{(m,t)}_{i,l}$ is a quantity factor that relates to the software that performs a specific task on commodity (i) and functionality (l) using software (m) at a specific time (t), and $w^{(m,t)}_{i,l}$ is the weighting factor, which will handle the missing data-related issues and/or other factors such as market capitalization of the companies. For instance, $f^{(m,t)}_{i,l}$ can be defined as the total number of sites. For example, if the market capitalization of the software companies X and Y are US$1 million and US$100 million respectively, but both companies have a software solution with the same functionality, then the weighting factor of the small company will be higher than the other software company. Furthermore, the price of the mine planning software as well as support availability plays an important role when considering the weighting factor.

Although software utilization is already defined in a generic way, it can also be defined in a specific way, *i.e.* the relative utilization (r). Relative utilization can be considered as a weighted software utilization, and can be formulated as:

$$r^{(m,t)}_{i,l} = \frac{u^{(m,t)}_{i,l}}{\sum_{n=1}^{M} u^{(n,t)}_{i,l}}$$

where

$$\sum_{n=1}^{M} u^{(n,t)}_{i,l}$$

is total utilization of all software, and is used for normalization.

Calculating relative utilization leads to the weighted market impact of software utilization. In the calculation of relative utilization, three variables were used to generate the results, namely:

- Commodity (i)
- Functionality (l)
- Time factor (t).

The following results were calculated for only one commodity (coal) using six different functionalities (Katakwa, Musingwini, and Genc, 2013):

- Geological data management
- Geological modelling and resource estimation
- Design and layout
- Scheduling
- Financial valuation
- Optimization.

These six functionalities originated from the Open Group's Business Reference Model, which categorizes not only the functionalities of mine planning software, but also mine value chain stages and mining methods (The Open Group, 2010). The Business Reference Model illustrates how the various software solutions interact with each other, although this classification can be debateable. For example, Mine 2-4D software, which is used in mine scheduling, is often used in conjunction with Enhanced Production Scheduler (EPS) as it cannot produce a schedule without the use of EPS. Figure 1 shows the names of available mine planning software solutions and their functionalities along the mining value chain.

The time factor (t) has two timestamp indicators showing different data collection dates:

- September 2012, t=1
- April 2014, t=2.

By using all three variables, the weighted software utilization, and hence the market impact of each participating mine planning software solution, was calculated. The data-set was extracted from the updated database and the programming language GNU Octave was used for the data analysis and calculation of the software utilization per functionality using the two different time-stamps.

It is important to note that if $f^{(m,t)}_{i,l}$ is zero, the subject software either does not support the specific functionality or does not support the specific commodity. Furthermore, when calculating $u^{(m)}_{i,l}$ and $w^{(m,t)}_{i,l}$, the value is set to unity, as at this stage of calculation it was decided that the weighted software utilization did not have any impact on the calculation of the relative software utilization.

	Geological Data Management	Geological Modelling and Resource Estimation	Design and Layout	Scheduling	Financial Valuation	Optimisation
Bentley Evaluation				■	■	
Bentley Scheduler				■		
BLOCK AGG	■			■		
CADSMine		■	■	■		
Carbon 14 Mine Scheduler				■		
Carbon Economics					■	■
Carbon Micro Scheduler				■		
Carbon Performance Manager					■	
Carbon Processing				■	■	
Carbon Risk					■	■
Carbon V						■
Chronos				■		
Dragsim				■		■
Enhanced Production Scheduler (CAE)				■		■
EPS (MineRP)				■		
EPS Viz (Visualizer)				■		
EPS-PCBC Interface				■		
EPSOT (EPS Schedule Optimization Tool)				■		■
GEMS	■	■	■			
Geological Data Management Solution	■					
HAULNET				■		
Interactive Short Term Scheduler			■	■		
LoM Economics					■	■
Maptek I-Site	■					
Maxipit					■	■
Mine 2-4D			■	■		
Mine Scenario Planning			■	■		
Mineable Layout Optimizer			■			
Mineable Reserves Optimizer (CAE)			■	■		
Mineable Shape Optimizer			■	■		■
mineCAD			■			
mineCAVE			■			
mineHAUL			■			
mineMARKUP	■					
Mineral Beneficiation					■	■
MineSched				■		
mineSERV			■			
mineSTRUCTURE		■	■			
Minex	■	■	■			
MKP (Mining Knowledge Platform)	■	■				
MRM	■	■	■	■	■	■
NPV Scheduler (CAE)				■	■	■
NPV Scheduler (MineRP)				■	■	
Open Pit Metals				■		
PCBC				■		
Pegs Lite	■					
Performance Diagnostics						■
Portfolio Modelling					■	
Qerent Modeller					■	■
Sable Data Warehouse	■					
Services and Logistics			■	■		■
Sirovision	■	■				
Strat 3D		■				
Studio 3 - Engineering			■			
Studio 3 - Geology		■				■
Studio 5D Planner			■	■		
Studio3 - Basics						■
Surpac	■	■	■			
Talpac			■			
Underground Coal				■		
Ventsim Visual (Advanced)			■			
Vulcan	■	■	■			
Whittle					■	■
Workforce Planning					■	■
Xact				■		
Xeras					■	■
Xpac				■		

Figure 1 – Available mine planning software and functionalities along the mining value chain

Results

Six functionalities (l) with two time-stamps (t) were used for the calculations, and the results for each functionality with two time-stamps are presented as tables and figures. Accordingly, a total of {6(l) x 2(t) = 12} tables were created. According to the functionality list provided earlier, the first functionality, 'Geological Data Management' was used with two different time-stamps to produce the first sets of two tables. After generating the tables, pie charts were created for each table for easy interpretation of the results. Consequently, using the functionality list, the remaining tables and figures were created in a similar manner.

The following software providers participated in this study: Geovia, MineRP Solutions, Sable, RungePincockMinarco, Maptek, Cyest Technology and CAE Mining. Note that data on CAE Mining was made available only in the April 2014 data-set. The results presented here do not distinguish between either the mining methods or the type of mine (surface or underground operation).

Geological Data Management functionality

Table I shows the market share of the individual software solutions for coal using the functionality Geological Data Management, as at September 2012. Figure 2 is a graphical representation of Table I. (The column headings $f^{(m,t)}_{i,l}$, $w^{(m,t)}_{i,l}$, $u^{(m)}_{i,l}$ and $r^{(m)}_{i,l}$ in Tables I to IX were defined in the section 'Utilization framework'.)

The results for the September 2012 and April 2014 time-stamps are identical, indicating that there were no changes between the two different data-sets. This is the reason why there is only a single table showing results for both time-stamps, and similarly, Figure 2 represents both time-stamps.

Minex software clearly is the market leader and the most utilized mine planning software for Geological Data Management. Minex is followed by Sable Data Warehouse, with a 22% market share for the commodity coal.

Geological Modelling and Resource Estimation functionality

The results for the Geological Modelling and Resource Estimation functionality are very similar to those for Geological Data Management, in terms of both time-stamps being identical. Hence there is only one table (Table II) showing results for Geological Modelling and Resource Estimation. Figure 3 is a graphical representation of Table II.

Minex software is again the market leader, with a 82% share, followed by Surpac with 22%.

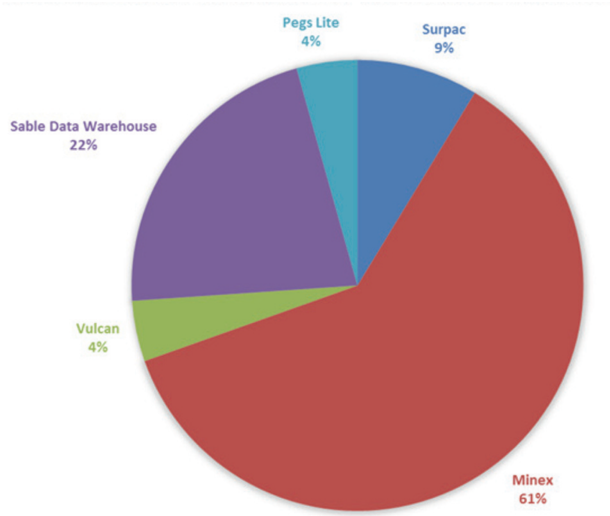

Figure 2 – Geological Data Management functionality: software utilization for coal

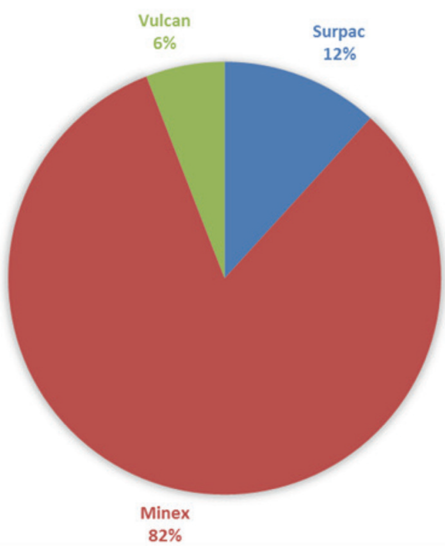

Figure 3 – Geological Modelling and Resource Estimation functionality: software utilization for coal

Table I

Geological Data Management functionality: software utilization for coal

m	Software	$f_{i,l}^{(m,t)}$	$w_{i,l}^{(m,t)}$	$u_{i,l}^{(m)}$	$r_{i,l}^{(m)}$
1	BLOCK AGG	0	1	0	0
2	GEMS	0	1	0	0
3	Geological Data Management	0	1	0	0
4	Maptek I-Site	0	1	0	0
5	mineMARKUP	0	1	0	0
6	Minex	14	1	14	0.6087
7	MKP (Mining Knowledge Platform)	0	1	0	0
8	MRM	0	1	0	0
9	Pegs Lite	1	1	1	0.0435
10	Sable Data Warehouse	5	1	5	0.2174
11	Sirovision	0	1	0	0
12	Surpac	2	1	2	0.087
13	Vulcan	1	1	1	0.0435

Table II

Geological Modelling and Resource Estimation functionality: software utilization for coal

m	Software	$f_{i,l}^{(m,t)}$	$w_{i,l}^{(m,t)}$	$u_{i,l}^{(m)}$	$r_{i,l}^{(m)}$
1	CADSMine	0	1	0	0
2	GEMS	0	1	0	0
3	Minex	14	1	14	0.8235
4	MKP (Mining Knowledge Platform)	0	1	0	0
5	MRM	0	1	0	0
6	Sirovision	0	1	0	0
7	Strat 3D	0	1	0	0
8	Studio 3 - Geology	0	1	0	0
9	Surpac	2	1	2	0.1176
10	Vulcan	1	1	1	0.0588

Design and Layout functionality

Table III shows the Design and Layout software results as at September 2012, while Table IV shows results as at April 2014. Figure 4 is a graphical representation of both tables.

The pie charts in Figure 4 show that the only difference between September 2012 and April 2014 is the inclusion of the Studio 5D Planner, from CAE Mining, in the 2014 data-set. Studio 5D Planner has a 5% market share in the April 2014 time-stamp. Nevertheless, Minex continues its dominance in the coal mining sector with a 64% market share in April 2014 compared to 67% in September 2012, followed by Talpac software, with 19% and 18% respectively for Design and Layout functionality.

Scheduling functionality

Table V shows the Scheduling functionality results as at September 2012 for coal, while Table VI shows the results using the second time-stamp, April 2014. Figure 5 is a graphical representation of both tables.

Similar to the results for Design and Layout, the entry of CAE Mining software once again visible in the April 2014 time-stamp, with Studio 5D Planner and Enhanced Production Scheduler software both having a 2% market share. However, although it lost 3% of the market share between September 2012 and April 2014, Xpac is still the leader in the coal mining sector when it comes to Scheduling functionality software. Xpac is followed by Dragsim software,

Table III

Design and Layout functionality: software utilization for coal as at September 2012

m	Software	$f^{(m,t)}_{i,l}$	$w^{(m,t)}_{i,l}$	$u^{(m)}_{i,l}$	$r^{(m)}_{i,l}$
1	CADSMine	0	1	0	0
2	GEMS	0	1	0	0
3	Interactive Short Term Scheduler	0	1	0	0
4	Mine 2-4D	0	1	0	0
5	Mine Scenario Planning	0	1	0	0
6	Mineable Layout Optimizer	0	1	0	0
7	Mineable Reserves Optimizer (CAE)	0	1	0	0
8	Mineable Shape Optimizer	0	1	0	0
9	mineCAD	0	1	0	0
10	mineSERV	0	1	0	0
11	Minex	14	1	14	0.6667
12	MRM	0	1	0	0
13	Services and Logistics	0	1	0	0
14	Studio 3 - Engineering	0	1	0	0
15	Studio 5D Planner	0	1	0	0
16	Surpac	2	1	2	0.0952
17	Talpac	4	1	4	0.1905
18	Vulcan	1	1	1	0.0476

Table IV

Design and Layout functionality: software utilization for coal as at April 2014

m	Software	$f^{(m,t)}_{i,l}$	$w^{(m,t)}_{i,l}$	$u^{(m)}_{i,l}$	$r^{(m)}_{i,l}$
1	CADSMine	0	1	0	0
2	GEMS	0	1	0	0
3	Interactive Short Term Scheduler	0	1	0	0
4	Mine 2-4D	0	1	0	0
5	Mine Scenario Planning	0	1	0	0
6	Mineable Layout Optimizer	0	1	0	0
7	Mineable Reserves Optimizer (CAE)	0	1	0	0
8	Mineable Shape Optimizer	0	1	0	0
9	mineCAD	0	1	0	0
10	mineSERV	0	1	0	0
11	Minex	14	1	14	0.6364
12	MRM	0	1	0	0
13	Services and Logistics	0	1	0	0
14	Studio 3 - Engineering	0	1	0	0
15	Studio 5D Planner	1	1	1	0.0455
16	Surpac	2	1	2	0.0909
17	Talpac	4	1	4	0.1818
18	Vulcan	1	1	1	0.0455

with an 11% market share on both the 2012 and 2014 time-stamps.

Financial Valuation functionality

Table VII shows the Financial Valuation functionality results. The results for both time-stamps were found to be identical, hence there is only one table. Figure 6 is a graphical representation of Table VII.

It can be clearly seen that Xeras is the market leader in Financial Valuation software, with a 92% share. Xeras is followed by Carbon Economics software, with an 8% market share.

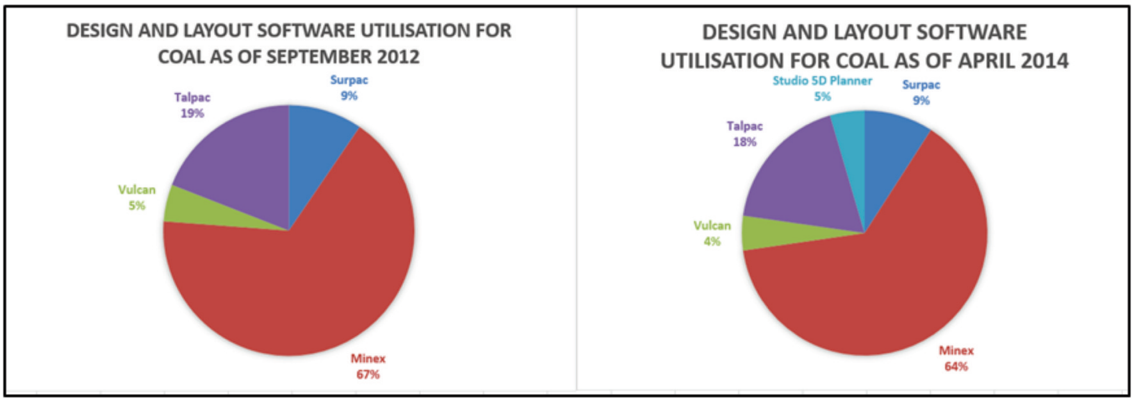

Figure 4 – Design and Layout functionality: software utilization for coal

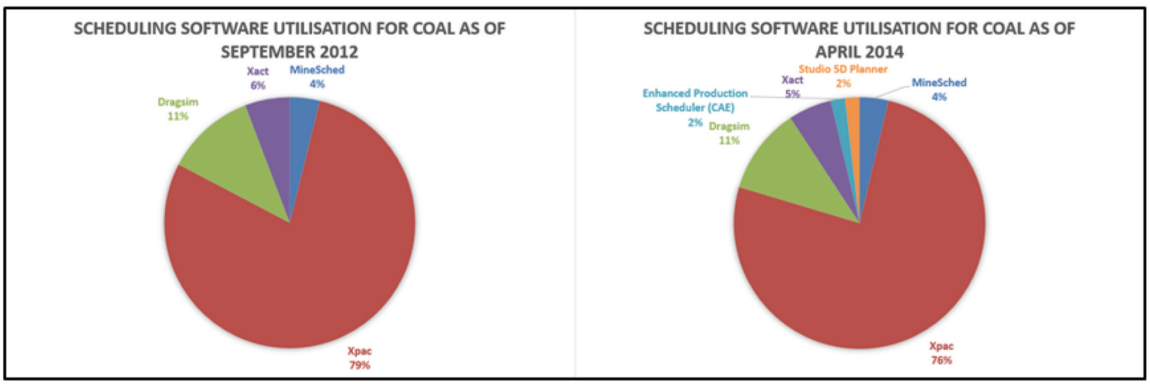

Figure 5 – Scheduling functionality: software utilization for coal

Table V

Scheduling functionality software: utilization for coal as at September 2012

m	Software	$f_{i,l}^{(m,t)}$	$w_{i,l}^{(m,t)}$	$u_{i,l}^{(m)}$	$r_{i,l}^{(m)}$
1	Bentley Evaluation	0	1	0	0
2	Bentley Scheduler	0	1	0	0
3	BLOCK AGG	0	1	0	0
4	CADSMine	0	1	0	0
5	Carbon 14 Mine Scheduler	0	1	0	0
6	Carbon Micro Scheduler	0	1	0	0
7	Carbon Processing	0	1	0	0
8	Chronos	0	1	0	0
9	Dragsim	6	1	6	0.1154
10	Enhanced Production Scheduler (CAE)	0	1	0	0
11	EPS (MineRP)	0	1	0	0
12	EPS Viz (Visualizer)	0	1	0	0
13	EPS-PCBC Interface	0	1	0	0
14	HAULNET	0	1	0	0
15	Interactive Short Term Scheduler	0	1	0	0
16	Mine 2-4D	0	1	0	0
17	Mine Scenario Planning	0	1	0	0
18	Mineable Reserves Optimizer (CAE)	0	1	0	0
19	Mineable Shape Optimizer	0	1	0	0
20	MineSched	2	1	2	0.0385
21	MRM	0	1	0	0
22	NPV Scheduler (CAE)	0	1	0	0
23	Open Pit Metals	0	1	0	0
24	PCBC	0	1	0	0
25	Services and Logistics	0	1	0	0
26	Studio 5D Planner	0	1	0	0
27	Underground Coal	0	1	0	0
28	Xact	3	1	3	0.0577
29	Xpac	41	1	41	0.7885

Table VI

Scheduling functionality: software utilization for coal as at April 2014

m	Software	$f_{i,l}^{(m,t)}$	$w_{i,l}^{(m,t)}$	$u_{i,l}^{(m)}$	$r_{i,l}^{(m)}$
1	Bentley Evaluation	0	1	0	0
2	Bentley Scheduler	0	1	0	0
3	BLOCK AGG	0	1	0	0
4	CADSMine	0	1	0	0
5	Carbon 14 Mine Scheduler	0	1	0	0
6	Carbon Micro Scheduler	0	1	0	0
7	Carbon Processing	0	1	0	0
8	Chronos	0	1	0	0
9	Dragsim	6	1	6	0.1111
10	Enhanced Production Scheduler (CAE)	1	1	1	0.0185
11	EPS (MineRP)	0	1	0	0
12	EPS Viz (Visualizer)	0	1	0	0
13	EPS-PCBC Interface	0	1	0	0
14	HAULNET	0	1	0	0
15	Interactive Short Term Scheduler	0	1	0	0
16	Mine 2-4D	0	1	0	0
17	Mine Scenario Planning	0	1	0	0
18	Mineable Reserves Optimizer (CAE)	0	1	0	0
19	Mineable Shape Optimizer	0	1	0	0
20	MineSched	2	1	2	0.037
21	MRM	0	1	0	0
22	NPV Scheduler (CAE)	0	1	0	0
23	Open Pit Metals	0	1	0	0
24	PCBC	0	1	0	0
25	Services and Logistics	0	1	0	0
26	Studio 5D Planner	1	1	1	0.0185
27	Underground Coal	0	1	0	0
28	Xact	3	1	3	0.0556
29	Xpac	41	1	41	0.7593

Table VII

Financial Valuation functionality: software utilization for coal

m	Software	$f_{i,l}^{(m,t)}$	$w_{i,l}^{(m,t)}$	$u_{i,l}^{(m)}$	$r_{i,l}^{(m)}$
1	Bentley Evaluation	0	1	0	0
2	Carbon Economics	1	1	1	0.0833
3	Carbon Performance Manager	0	1	0	0
4	Carbon Processing	0	1	0	0
5	LoM Economics	0	1	0	0
6	Maxipit	0	1	0	0
7	Mineral Beneficiation	0	1	0	0
8	MRM	0	1	0	0
9	NPV Scheduler (CAE)	0	1	0	0
10	Portfolio Modelling	0	1	0	0
11	Qerent Modeller	0	1	0	0
12	Whittle	0	1	0	0
13	Xeras	11	1	11	0.9167

Optimization functionality

Table VII shows the Optimization software results as at September 2012, while Table IX shows results as at April 2014. Figure 7 is a graphical representation of both tables.

The only difference between the two pie charts is the presence of the Enhanced Production Scheduler (CAE) software in the April 2014 chart, with a 5% market share. However, Xeras (58%) and Dragsim (32%) continued their dominance in 2014.

Conclusion

A methodology for the evaluation of mine planning software utilization in the South African coal mining sector has been developed. In this framework, three variables, namely, commodity (i), functionality (l), and time factor (t) were used to calculate the results. Although the calculations can be done for any commodity in a similar manner, this paper deals only with utilization in the coal sector. Six functionalities, namely Geological Data Management, Geological Modelling and Resource Estimation, Design and Layout, Scheduling, Financial Valuation, and Optimization were applied using two different time-stamps (September 2012 and April 2014).

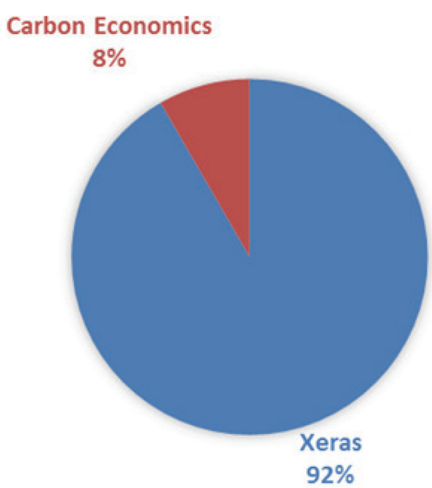

Figure 6 – Financial Valuation functionality: software utilization for coal

Table VIII

Optimization functionality software: utilization for coal as at September 2012

m	Software	$f_{i,l}^{(m,t)}$	$w_{i,l}^{(m,t)}$	$u_{i,l}^{(m)}$	$r_{i,l}^{(m)}$
1	Carbon Economics	1	1	1	0.0556
2	Carbon V	0	1	0	0
3	Dragsim	6	1	6	0.3333
4	Enhanced Production Scheduler (CAE)	0	1	0	0
5	LoM Economics	0	1	0	0
6	Maxipit	0	1	0	0
7	Mineable Shape Optimizer	0	1	0	0
8	Mineral Beneficiation	0	1	0	0
9	MRM	0	1	0	0
10	NPV Scheduler (CAE)	0	1	0	0
11	Performance Diagnostics	0	1	0	0
12	Qerent Modeller	0	1	0	0
13	Services and Logistics	0	1	0	0
14	Studio 3 - Geology	0	1	0	0
15	Studio3 - Basics	0	1	0	0
16	Whittle	0	1	0	0
17	Xeras	11	1	11	0.6111

Table IX

Optimization functionality: software utilization for coal as at April 2014

m	Software	$f_{i,l}^{(m,t)}$	$w_{i,l}^{(m,t)}$	$u_{i,l}^{(m)}$	$r_{i,l}^{(m)}$
1	Carbon Economics	1	1	1	0.0526
2	Carbon V	0	1	0	0
3	Dragsim	6	1	6	0.3158
4	Enhanced Production Scheduler) (CAE	1	1	1	0.0526
5	LoM Economics	0	1	0	0
6	Maxipit	0	1	0	0
7	Mineable Shape Optimizer	0	1	0	0
8	Mineral Beneficiation	0	1	0	0
9	MRM	0	1	0	0
10	NPV Scheduler (CAE)	0	1	0	0
11	Performance Diagnostics	0	1	0	0
12	Qerent Modeller	0	1	0	0
13	Services and Logistics	0	1	0	0
14	Studio 3 - Geology	0	1	0	0
15	Studio3 - Basics	0	1	0	0
16	Whittle	0	1	0	0
17	Xeras	11	1	11	0.5789

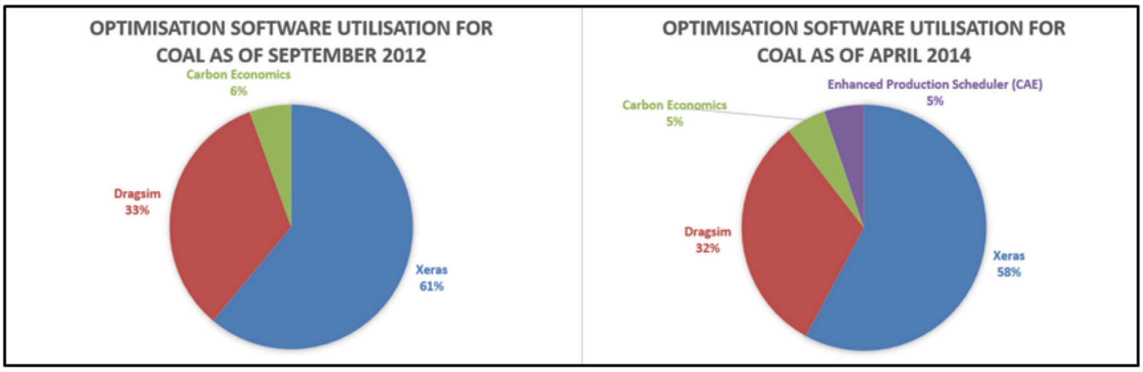

Figure 7 – Optimization functionality: software utilization for coal

Data on CAE Mining software was made available only in the April 2014 data-set, nevertheless the CAE Mining market impact is minimal in the coal sector.

By using this newly developed framework, utilization of the various mine planning software solutions was measured. This methodology provides an opportunity for software users to review existing software combinations, or those intending to purchase new software with a tool for estimating the comparative attractiveness of certain software packages. For example, mining companies can position themselves better by acquiring combinations of mine planning software; consulting companies can advise their clients more effectively to make the right choices of software solution; tertiary education institutions offering mining-related qualifications can choose which software to expose their students to; and software providers can strategically position themselves in the mine planning software market.

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Determination of value at risk for long-term production planning in open pit mines in the presence of price uncertainty

by M. Rahmanpour* and M. Osanloo*

Synopsis

Mine planning is a multidisciplinary procedure that aims to guarantee the profitability of a mining operation in changing and uncertain conditions. Mine plans are normally classified as long-term, intermediate-term, and short-term plans, and many factors affect the preciseness of these plans and cause deviations in reaching the objectives. Commodity price is the heart of mine planning, but it has a changing and uncertain nature. Therefore, the determination of mine plans in the presence of uncertain mineral price is a challenge. A robust mine plan reduces the risk of early mine closure. A procedure is presented to determine the value at risk (VaR) in any possible mine planning alternative. VaR is considered together with downside risk and upside potential in order to select the most profitable and least risky plan. The model is tested on a small iron ore deposit.

Keywords
mine planning, value at risk, downside risk, upside potential, price uncertainty.

Introduction

Mining projects are normally evaluated with the objective of maximizing economic value, as measured by the net present value (NPV). The aim of mine planning is to develop a yearly extraction plan that guides the mining operation to the highest NPV (Dagdelen, 2007; Yarmuch and Ortiz, 2011). This plan considers the technical constraints such as blending requirements, block sequencing, and pit slope (Caccetta and Hill 2003). Thus, the mine plan affects the economics and the payback period of mining operations.

Mine planning is a procedure based on which the profitability of the mining operation is guaranteed in changing and uncertain conditions (McCarter, 1992). However, conventional mine planning ignores the volatility and uncertainty of future commodity prices and does not allow for managerial flexibility. It is common practice to assume a deterministic and constant commodity price through the mine life. The problem with this approach is that the commodity price is the heart of the mine planning procedure and governs the profitability and feasibility of the operation; thus ignoring price volatility will result in a sub-optimal mine plan.

Note that the NPV of a mining project is highly sensitive to the future commodity price estimates, therefore any decision in the context of mining should be take price uncertainty into consideration. The question here is whether a mine plan that is optimal for an estimated future commodity price will also be reasonably optimal when price uncertainty is accounted for. To answer the question, this paper presents an approach to select a mine design under mineral price uncertainty.

The essential data required for mine planning comprises geological, geo-mechanical, hydrological, economic, environmental, and infrastructural information, and knowledge of mining engineering. These factors contribute to a greater or lesser degree to the mine planning procedure and mine plans. The characteristics of future mining activities, and also inherited uncertainty in the data (such as commodity price, ore grade, mining costs, and recoveries), highlights the importance of mine planning under conditions of uncertainty. As shown in Figure 1, mineral prices have a changing and volatile nature. In this figure, the price changes for iron ore and copper are given from the year 2000.

Open pit mine production planning is a multi-period precedence-constraint knapsack problem, and it normally fits into the mixed integer linear programming (MILP) framework (Osanloo *et al.*, 2008; Newman *et al.*, 2010). This problem is defined on a spatial representation of the mining area called a

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block model (Hustrulid *et al.*, 2013). Considering the exploration drilling pattern, geological conditions, and the size of mining equipment to be used, the size of the blocks in a block model will be determined (Figure 2). After determining the block dimensions, the geological characteristics of each block are assigned using any available estimation techniques (Journel and Kyriakidis, 2004). By considering the economic parameters such as commodity selling price, operating costs, and overall recovery, one can calculate the economic value of each block. The aim of the mine planner is to determine whether a given set of blocks should be mined or not; if so, at which time it should be mined, and once it is mined, how it should be processed to maximize the NPV of the operation (Dagdelen, 2007). Once a block is earmarked for mining, different destinations or processing methods can be determined for that block. The process of determining the time and extraction sequence of blocks through the mine life is referred to as mine production planning (Johnson, 1968). This procedure is represented in Figure 2.

The problem of production planning in open pit mines has been studied by many researchers (Dowd and Onur, 1992; Tolwinski and Underwood, 1996; Denby and Schofield, 1996; Kumral, 2003; Ramazan and Dimitrakopoulos, 2004; Menabde *et al.*, 2007; Boland *et al.*, 2008; Dimitrakopoulos and Ramazan, 2008; Elkington and Durham, 2009; Sattarvand, 2009; Bley *et al.*, 2010; Kumral, 2010; Groeneveld and Topal, 2011; Murakami *et al.*, 2011; Gholamnejad and Moosavi, 2012; Lamghari and Dimitrakopoulos, 2012; Moosavi *et al.*, 2014). These studies aimed to maximize the NPV of the operation while satisfying all the operational constraints.

There are two optimization approaches to solve the production planning problem in open pit mines, named deterministic and uncertainty-based approaches (Osanloo *et al.*, 2008). In deterministic models, all the inputs are assumed to have fixed known real values (*e.g.* Tan and Ramani, 1992; Akaike and Dagdelen, 1999; Zhang, 2006; Elkington and Durham, 2009; Cullenbine *et al.*, 2011). However, this assumption is not always realistic and some data such as ore grades, future product demand, future product price, and production costs can vary throughout the mine life. Researchers (Albach, 1976; Rovenscroft, 1992; Smith, 2001; Godoy and Dimitrakopoulos, 2004; Zhang *et al.*, 2007, Abdel Sabour *et al.*, 2008; Abdel Sabour and Dimitrakopoulos, 2011; to name a few) have developed various models based on MILP, meta-heuristics, heuristics and simulation, chance constraint linear programming, and stochastic programming to determine the mine plan in uncertain conditions.

In this paper price uncertainty and its affect on mine planning is studied. To do so, a simple integer linear programming model is formulated to determine the production planning and mining sequence of blocks.

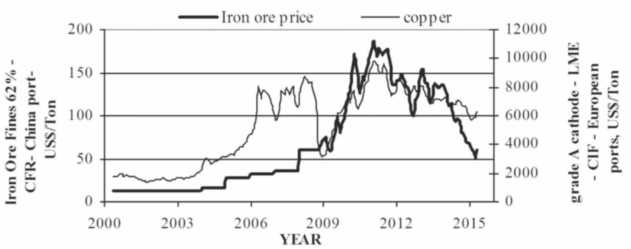

Figure 1—Iron ore and copper price changes 2000–2015

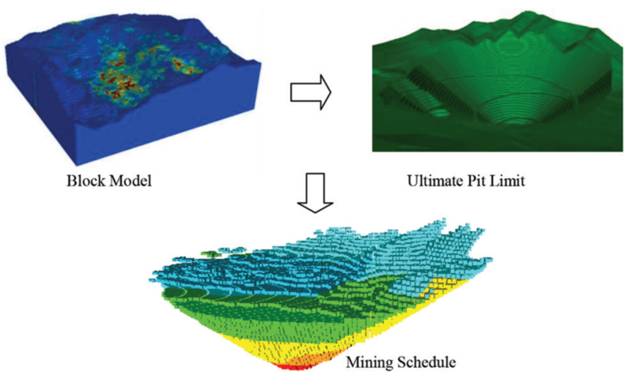

Figure 2— A representation of open pit mine planning

Method description

The mineral or commodity price affects the mining cut-off grade, mineable reserve, mine size, and mine life. Any long-term change in commodity price will affect the cut-off grade and amount of mineable reserve. For example, in the case of a price fall, the ultimate pit limit shrinks, cut-off grade increases, low-grade ore blocks will be classified as waste, and the mineable reserve decreases. All these factors will change the long-term and short-term plans of the mine. Short-term changes in commodity price will affect the short-term plans (Tulp, 1999). In these conditions, short-term plans must be changed cautiously because short-term gains may cause other problems in long-term plans (Hall, 2009a).

This paper studies the effects of price uncertainty on mine planning and aims to minimize the risks from price changes throughout the mine life. Figure 3 shows the procedure of the proposed algorithm. Based on historical price data, scenarios are generated. Then in the third step of the procedure, mine plans are optimized based on each price scenario. The plans are analysed to check their behaviour with respect to different price forecasts. In the fifth step of the procedure, the downside risk and upside potential of each mine plan are determined. These two criteria will indicate the optimum mining schedule among the different mine plans.

This approach leads to the selection of a mining schedule that captures the upside potential and minimizes the downside risk of the mining operation associated with price uncertainty. For that purpose, the available price data, its volatility, and its simulated values are integrated into the process. The practical aspects of the approach are illustrated in a simple example.

Bootstrapping

According to Figure 3, in the first step the historical changes in mineral price are collected in order to simulate future possible price forecasts. There are various methods to generate different price paths, including bootstrapping and geometric Brownian motion (GBM). In the second step of the procedure, the bootstrapping method is applied to determine

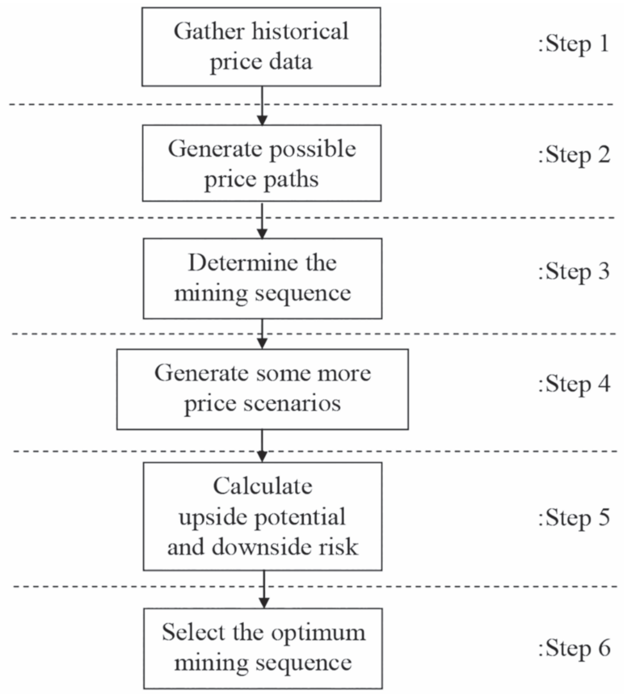

Figure 3—Procedure for determining the optimum mining sequence

some scenarios to model future price trends and changes. Generally, bootstrapping is a sampling method. Assuming a given data-set of historical mineral prices, the yearly or monthly returns (*i.e.* rate of monthly/yearly price change in each period) are calculated. The return of a mineral price is equal to the natural logarithm of the division of two consecutive prices (Equation [1]).

$$\mu = Ln\left(\frac{S_t}{S_{t-1}}\right) \qquad [1]$$

where μ is the return, and S_t, S_{t-1} represent the mineral price in periods t and t-1, respectively.

In order to predict future mineral prices or to generate a new sequence of price path using bootstrapping, the individual returns should be calculated for each period. It should be noted that to generate a price path for the next 10 years, the returns for the previous 10 years should be calculated. When the historical returns are calculated, then by selecting the returns arbitrarily from the historical data and putting them together, many price paths can be generated.

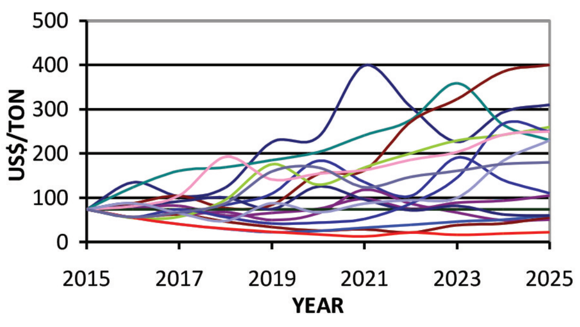

Figure 4—Bootstrapped iron ore price for the next 10 years

The main advantage of this method is that it does not need any assumptions about the distribution of the returns, and the empirical distribution of the returns is inherited in the sampling procedure. Using the historical data in Figure 1 and applying the bootstrapping method, several scenarios are generated for the iron ore price (Figure 4). In the traditional approach, the mineral price is assumed constant through the mine life. In order to have a benchmark to compare the results with the traditional approaches, another scenario with the assumption of fixed price throughout the mine life is generated.

Mathematical model of block sequencing

According to the procedure, in the third step, the optimum block sequencing should be determined for each price scenario. For that purpose, production planning and block sequencing in open pit mines is formulated in an integer linear programming model (Equation [2]). This model is capable of determining the time of extraction of each block. It also determines the destination or processing method of each block such that it maximizes the total NPV. The notation used in the model in Equation [2] is as follows:

- B — The set of all blocks in the block model
- P_b — The set of blocks that cover or precede block b
- t,t' — The time indices
- T — The mine life or number of the periods to be planned
- M — The number of possible destinations or processing alternatives
- c_{bmt} — The discounted economic value of block b mined at time t and sent to destination m
- x_{bmt} — The decision variable, which is equal to unity if block b is mined at time t and sent to destination m; otherwise it is equal to zero
- MC_t, $\overline{MC_t}$ — The minimum and maximum mining rate at time t, respectively
- X_b — The amount or tonnage of rock in block b
- g_b — The grade of commodity in block b (normally expressed as a percentage of the total block tonnage)
- $G^{t,m}{}_{\min}$, $G^{t,m}{}_{\max}$ — The minimum and maximum acceptable grades at destination m at time t, respectively
- PC_{mt}, $\overline{PC_{mt}}$ — The minimum and maximum processing capacities at destination m at time t, respectively

Objective function:

$$\text{Max} \sum_{b \in B} \sum_{t \in T} \sum_{m \in M} c_{bmt}\, x_{bmt} \qquad [2a]$$

subject to:

$$\sum_{t \in T} \sum_{m \in M} x_{bmt} \le 1, \quad \forall b \qquad [2b]$$

$$MC_t \le \sum_{b \in B} \sum_{m \in M} X_b\, x_{bmt} \le \overline{MC_t}, \quad \forall t \qquad [2c]$$

$$PC_{mt} \le \sum_{b \in B} X_b\, x_{bmt} \le \overline{PC_{mt}}, \quad \forall t, m \qquad [2d]$$

$$\sum_{b\in B}(g_b - G_{\max}^{t,m})x_{bmt} \le 0, \forall t, m \quad [2e]$$

$$\sum_{b\in B}(g_b - G_{\min}^{t,m})x_{bmt} \ge 0, \forall t, m \quad [2f]$$

$$\sum_{t=1}^{t'}\sum_{b'\in P_b}(x_{bmt} - x_{b'mt}) \le 0, \ \forall b\in B, \forall t' \in \{1,\ldots,T\} \quad [2g]$$

$$x_{bmt} = 0 \, or \, 1, \ \forall b, t, m \quad [2h]$$

In Equation [2a], the objective function of the model is defined as the maximization of the total discounted economic value or the NPV of the mining operation. Constraint [2b] ensures that if block b is to be mined then it could only be mined once and sent to destination m at time t. Constraints [2c] and [2d] ensure that the total amount of rock mined and processed at time t does not exceed the prescribed lower and upper bounds on mining and processing capacities, respectively. Constraints [2e] and [2f] ensure that the average grade of material sent to each destination is within the prescribed lower and upper bounds. Constraint [2g] is known as the slope or proceeding constraint. This constraint ensures that wall slope restrictions are obeyed and block b can be mined only if all the overlying or covering blocks are removed beforehand. The model in Equation [2] is programmed in Visual Studio version.2012 and solved using the CPLEX 12.5 solver.

The term c_{bmt} in the objective function is a function of mining time and the mineral price in that period. The scenarios in Figure 4 are used to calculate the block values in different periods. In this figure, 16 price paths are generated. Using these price paths and applying the model in Equation [2], one can optimize the production plans for each price path. At the end of this step, different options for mine plans are generated for the ore deposit. The block sequencings in these plans differ from each other due to different and variable mineral prices in each period. The next step aims to select the optimum block sequencing for the deposit by applying a simulation-based approach.

Optimizations of the mine plan for each of the price paths are generated based on the parameters given in Table I.

These parameters are the same for each mine plan, and the only difference is the iron ore price trend. The price trend in each mine plan is taken from the generated price paths in Figure 4.

Table I

Technical and economic parameters considered for optimizing mine plans

Parameter	Value
Pit slope (degree)	45
Mining cost (US$ per ton)	1
Processing cost (US$ per ton)	8
Recovery (%)	80
Annual discount rate (%)	10
Mine life (years)	10
Iron ore price (US$ per ton)	From Figure 4

The ultimate pit limit in all the cases is the same, and only the block sequencing differs due to the embedded price scenario. As mentioned, production planning and block sequencing for each of the simulated price paths are optimized using the model in Equation [2]. The planning period is 10 years for all the cases. Differences in the block sequencing and the schedules result in significant variations in expected cash flow returns. In the example presented here, 16 price paths are used to illustrate different aspects of the suggested approach. The experiments show that nine different mine plans could be generated in this case study. The question is: which of these mine plans is the optimum mining schedule? To answer this question, a simulation-based approach is conducted.

Geometric Brownian motion

In order to select the optimal mining schedule, the generated mine plans in the previous section are analysed to check their behaviour with respect to different price forecasts. For this purpose, geometric Brownian motion is applied to define some more scenarios based on which the behaviour of each mine plan should be determined. Geometric Brownian motion (GBM) is one of the most common methods for price modelling and scenario generation (Erlwein *et al.*, 2012; Hall 2009b). In this method, the stock or the mineral price is modelled as a Wiener process and is a function of return and volatility of the mineral price (Equation [3]).

$$\frac{dS}{S} = \alpha dt + \sigma dz \quad => \quad S_t = S_0 \exp\left(\left(\mu - \frac{\sigma^2}{2}\right) + \sigma W_t\right) \quad [3]$$

where S_0 and S_t are the current mineral price and the mineral price in period t, and μ is the return of the mineral price and is calculated using Equation [1]. The terms α and dz are the expected growth rate and an increment in a standard Weiner process, respectively. Also, σ is the volatility of the mineral price, and the Wiener process W_t is discretized using Equation [4].

$$W_t = z_t\sqrt{t}, \ z_t = N(0,1) \quad [4]$$

The price volatility is calculated from the historical data using Equation [5].

$$\left.\begin{array}{l} \mu_i = Ln\left(\dfrac{S_i}{S_{i-1}}\right) \quad for\, i = 1^{st}\ to\ n^{th}\ observation \\ s = \sqrt{\dfrac{1}{n-1}\displaystyle\sum_{i=1}^{n}\mu_i^2 - \dfrac{1}{n(n-1)}\left(\displaystyle\sum_{i=1}^{n}\mu_i\right)^2} \end{array}\right\} => \sigma = \frac{s}{\sqrt{t}} \quad [5]$$

where n is the number of historical data, and t is the time length for which the volatility is calculated. When the volatility of the mineral price is calculated, then it could be estimated for the coming periods using the generalized autoregressive conditional heteroskedasticity (GARCH) method. In this method, the volatility of the mineral price is a function of its previous conditions, and is calculated using Equation [6].

$$\begin{array}{l} \sigma_t^2 = \gamma V_L + \alpha u_{t-1}^2 + \beta \sigma_{t-1}^2 \\ \gamma + \alpha + \beta = 1 \end{array} \quad [6]$$

In this equation, V_L is the long-term volatility of the mineral price. The parameters γ, α, β are defined such that they maximize the model in Equation [7].

$$\sum_{i=1}^{t}\left(-\ln\left(\sigma_i^2\right)-\frac{u_i^2}{\sigma_i^2}\right) \qquad [7]$$

Then, the GARCH model can be applied to estimate the future volatility of the mineral price (Equation [8]).

$$E\left(\sigma_t^2\right)=V_L+\left(\alpha+\beta\right)^t\left(\sigma_0^2-V_L\right) \qquad [8]$$

Using the model in Equation [8], the volatility of the iron ore price in the coming years is estimated. Combining the result of the GARCH model and the GBM method, it is possible to simulate more scenarios for future prices. The result of 50 simulations is shown in Figure 5.

This simulation-based risk analysis discussed can be used to choose the best mine design from the available designs. The best design is the one that minimizes the potential for losses while maximizing the possibility of better financial performance (Dimitrakopoulos *et al.*, 2007; Azimi *et al.*, 2013). For the iron ore case considered in this study, the key project indicator is NPV. For a given mine design, a distribution of NPV is calculated using the 200 price paths generated by the GBM method. The cumulative probability distributions of NPV for the two of the mine plans (plan 1 and plan 2) are represented in Figure 6.

Downside risk, upside potential, and value at risk

In the fifth step of the procedure, the downside risk and upside potential of each mine plan should be determined (Figure 7). These two criteria will indicate the optimum mining schedule. The selected mining schedule maximizes

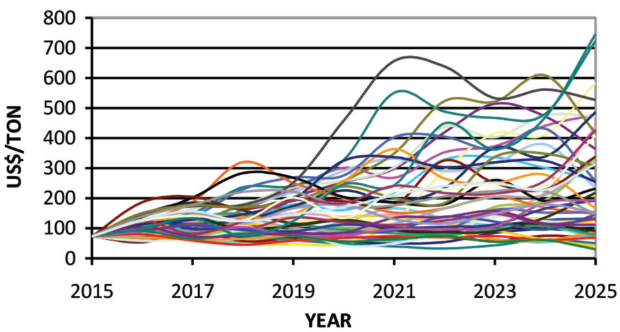

Figure 5—Simulated iron ore price using GBM

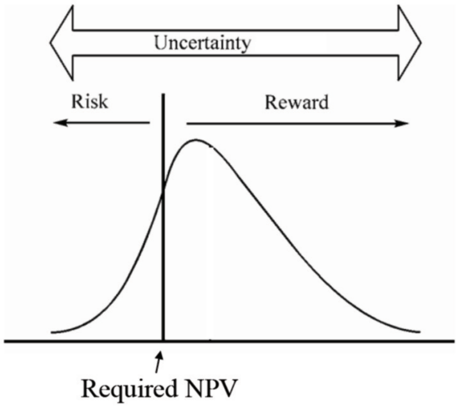

Figure 7—Distribution of NPV, reward or upside potential, and downside risk

the upside potential and minimizes the downside risk of the mining operation associated with price uncertainty.

Assessing price uncertainty and the risks associated with it suggests that there is a probability that a given mine plan may perform better in other scenarios as well. Figure 7 elucidates the concept of maximum upside and minimum downside based on the risk associated with price (Dimitrakopoulos *et al.*, 2007; Azimi *et al.*, 2013). The figure shows the distribution of NPV for a mine plan that can be generated from simulated price paths. With a defined point of reference such as the minimum acceptable NPV, the distribution that minimizes risk and maximizes the reward or upside potential leads to the selection of a suitable mine plan. The upside potential (UP_{it}) and the downside risk (DR_{it}) during period t are formulated in Equations [9] and [10].

$$UP_{it}=\sum_j\left(V_{ijt}^{+}-C_t\right)P_j \qquad [9]$$

$$DR_{it}=\sum_j\left(C_t-V_{ijt}^{-}\right)P_j \qquad [10]$$

where the term C_t is the accepted NPV of the mine plan during a period of t years, and V_{ijt}^{+} and V_{ijt}^{-} represent the total net present value of ith mine plan generated for jth simulated price paths during period t, where $V_{ijt}^{+} \geq C_t$ and $V_{jt}^{-} \leq C_t$, respectively. Using Equations [9] and [10], one can calculate the upside potential and the downside risk criteria.

The other useful criterion is the value at risk, VaR. This is a widely used risk management tool, and is a measure of the worst expected loss under normal market conditions that is

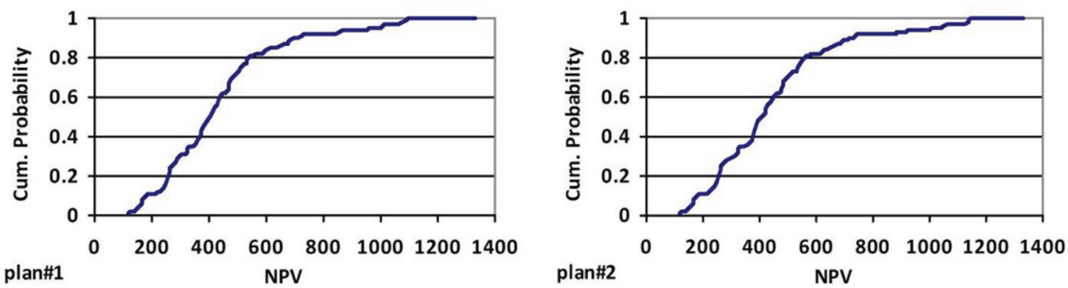

Figure 6—Probability distribution of the NPV for plan 1 and plan 2

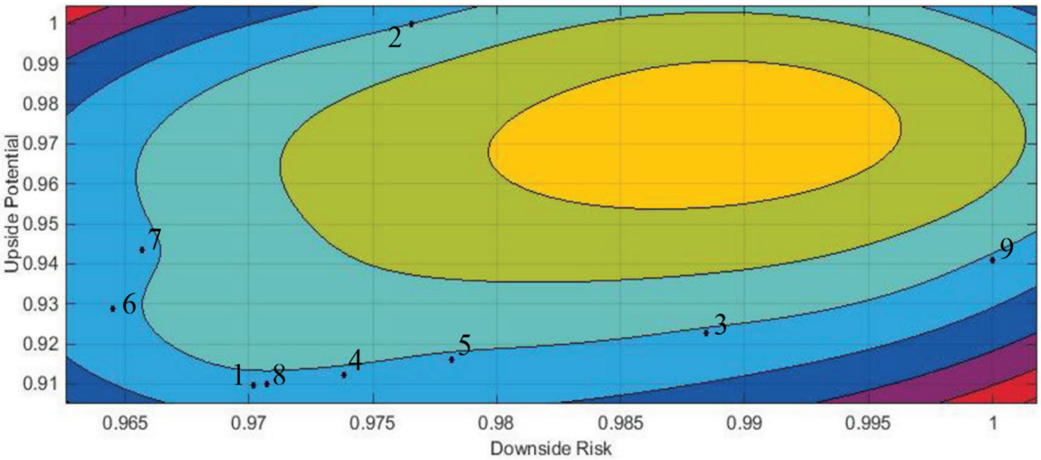

Figure 8—Contour map of normalized VaR, upside potential, and downside risk

calculated for a specific time period at a given confidence level. VaR answers the questions like: with a probability of x%, how much can I lose over a pre-set horizon?' (Benninga, 2008). This means that VaR (1%) is the 1% quintile point of the probability distribution function of mine value. VaR provides the answer of two questions: (1) what is the distribution of the mine value at the end of its life; and (2) with a probability of 1 %, what is the maximum loss at the end of mine life? The main drawback of VaR is that it does not consider the tail of distributions, thus the result may be misleading. Therefore, it should be complemented with other tools.

Now the decision-maker has to choose among the available options with regard to the three criteria. The three criteria of downside risk, upside potential, and value at risk are used to rank the mine plans. The result is given in Table II. According to the results, plan 2 has the maximum upside potential, plan 6 has the minimum downside risk, and plan 1 and plan 8 have the lowest VaR. The VaR in case of plan 1 and plan 8 is about 10.8% of the initial capital cost. Also, plan 2 has the highest expected NPV. In this step, selection of the optimal mine plan is converted into a multi-criteria decision-making problem. The selection procedure depends on the decision-maker and his risk-taking behaviour. The risk-taking behaviour of the decision-maker is called the 'utility function' and is beyond the scope of the current paper.

Table II

Ranking of mine plans based on upside potential, downside risk, and VaR

Plan ID	Upside potential	Downside risk	VaR
Plan 1	9	3	1
Plan 2	1	6	9
Plan 3	5	8	7
Plan 4	7	5	5
Plan 5	6	7	4
Plan 6	4	1	3
Plan 7	2	2	6
Plan 8	8	4	2
Plan 9	3	9	8

In this example there are two negative criteria (*i.e.* VaR and downside risk) and one positive criterion (*i.e.* upside potential). Assuming that the importance weights of these positive and negative criteria are identical, these weights can be calculated. In this example the weights of VaR, upside potential, and downside risk are 25%, 50%, and 25%, respectively. Finally, plan 6 seems to be the ideal mine plan. The plan has the minimum downside risk. Moreover, the rank of plan 6 is among the top five considering the other criteria. Therefore, plan 6 is suggested as the optimal mining schedule in this case study.

Discussion and conclusions

Financial uncertainty has a significant impact on the value of mining projects. In this paper a new framework was proposed to optimize mine schedules based on price uncertainty. The procedure is an iterative implementation of block sequencing optimization under varied price forecasts and market conditions. The approach is based on criteria that include maximum upside potential, minimum downside risk, and value at risk. The approach provides a set of mining scheduling options for the mine planner. Selecting the optimal solution is a multi-criteria decision-making problem.

The main weakness of the approach is that it is hard to implement, particularly in the case of larger ore deposits where the number of block is very large. This suggests the application of blocks aggregation methods that reduce the size of the problem, such as the fundamental tree algorithm. The other limitation of the procedure is the difficulties in determination of price scenarios. The generated price scenarios must be a representative sample of future price outcomes. Also, the number of price paths must be low in order to reduce the complexity of the mine scheduling models.

Considering the bootstrapped price scenarios, nine options for mining plan are generated. Then, using a combination of GARCH and GBM methods several (200 scenarios) price scenarios are generated to analyse the behaviour of each individual mine plan through the mine life. The result of this analysis represents the probability distribution function of NPV for each mine plan. These mine schedules have some important characteristics. Firstly, an optimal mine schedule based on a given simulated price path is not necessarily optimal for other price scenarios. Secondly, besides the fact that the price scenarios are equally probable, the corresponding mine plans are not equally probable. According to the optimization results, 16 price scenarios are considered but only nine different mine plans are generated. Careful analysis of the mine plans shows that plan 1 and plan 8 are approximately the same, with a difference of less than 5%. In these two plans, the block sequencing has changed in the 8th year. This is true also for the case of plan 4 and plan 5, where the difference is also less than 5%. This is confirmed by the results in Table II, where it can be seen that the rankings of these plans are virtually the same.

The method aims to determine the optimum mining sequence with minimal risk. In that regard, three attributes are calculated for each mining scenarios, namely 'upside potential', 'downside risk', and 'VaR'. First, the plans are evaluated based on VaR. The contour map of normalized VaR, upside potential, and downside risk shows the relationship among these three criteria (Figure 8). The data is normalized with respect to the maximum values. In Figure 8, the x-axis is the downside risk and the y-axis is the upside potential; the contour lines represent VaR. According to the results, plan 6 is suggested as the optimum mining sequence in this case. Analysing the price scenario on which plan 6 is optimized, it can be seen that, in this scheduling scenario, the mineral price has a falling trend compared with the other scenarios. Under a falling price trend, high-grade ore blocks will be mined earlier in order to gain the maximum possible NPV. Conclusively, those mine plans that are determined based on the falling price scenarios always have the lowest value at risk compared with other plans.

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Impact of the South African mineral resource royalty on cut-off grades for narrow, tabular Witwatersrand gold deposits

by C. Birch*

Synopsis

A mineral resource royalty is payment to the holder of mineral rights for the utilization of the mineral resource. In South Africa, this payment is made to the State as holder of the mineral rights (The Mineral and Petroleum Resources Royalty Act of 2008). The principle purpose of this research paper is to identify if the State is benefitting from the mineral resource royalty by considering its impact on seven individual Witwatersrand gold mines.

For this study, a simple financial optimiser model was created in Microsoft Excel that links the ore flow, block listing and the cash flow (excluding or including the cost of the mineral resource royalty). Mixed integer linear programing (the Excel Solver function) is utilised to optimise either profit or NPV (at 9% and 12%) by adjusting the cut-off grade.

The impact on each of the seven mines mine was different but overall R7.9 billion is estimated to be paid in mineral resource royalty over their expected remaining lives. Due to the cost of the mineral resource royalty and increasing the cut-off grades, the total revenue decreases by R10 billion. A significant portion of this lost revenue would have been paid to the State in the forms of other taxation including company income tax which decreases by R2.8 billion. It is recommended that an industry wide investigation be conducted to determine if the resource royalty is adding to the State's revenue, or destroying value including premature job losses.

Keywords

Mineral and Petroleum Resources Royalty Act of 2008, cut-off grade, optimization, net present value (NPV), mixed integer linear programing, excel solver.

Introduction

Taxation of mining companies has become a very topical subject in the context of the post-apartheid South African economy. There have been increasing calls for equitable re-distribution of mineral asset wealth for the benefit of all South African citizens and not just to an elite few. As part of the Mineral and Petroleum Resources Development Act (MRDPA) of 2002, the Mineral and Petroleum Resources Royalty Act of 2008 was introduced and came into effect in 2010 (South African Revenue Services, 2008). A mineral royalty is payment to the holder of mineral rights for the utilization of the mineral resource. In the case where the holder of the mineral right is the State, then this payment is made to the State (Cawood, 1999). Studies on the potential impact of the mineral resource royalty on the minerals industry have been conducted during the formulation of the Act and subesquently. These include Cawood and Macfarlane (2003), who suggested a cap of no more than 3% (Cawood and Macfarlane, 2003), as well as Cawood (2011), who predicted rising cut-off grades and thus reducing mineral reserves. In 2013, Grobler completed a Masters in the Faculty of Economic and Management Sciences at the University of Pretoria and concluded that the impact of the mineral resource royalty on the economy is positive. She stated that the royalty has caused operating expenses to increase, but this does not appear to be significant. Furthermore, she stated that the royalty has not affected the way the mining industry operates (Grobler, 2014).

If the mineral resource royalty was levied only on profit, then it could be ignored for cut-off grade purposes, as is the case for income tax. However, due to the fact it is levied on total sales, it should be considered a cost and be taken into account in determining the cut-off grade (Hall, 2014). Mines use the cut-off grade to indicate what is considered ore and what is waste. An increase in cut-off grade due to an increase in costs (including the mineral resource royalty) leads to a reduction of available mineral reserves above the calculated cut-off grade. The purpose of this study is to consider how the cost of the mineral resource royalty has increased cut-off grades at a selection of mines, and determine the impact on overall revenue generated for the State. This study differs from previous studies found during the literature review in that a selection of individual mines is

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considered, and the impact on their individual orebodies is determined. Each orebody has a unique grade distribution and thus it is expected that this will result in differences in how the economics of the orebody are impacted by the additional costs associated with the mineral resource royalty.

The impact of income tax (gold tax formula) and royalty payments on the life-of-mine and how much of the resource is converted into reserve as a function of the economic benefit will be considered. There is an increasing demand for mining companies to address the social and cultural needs of the communities within which they operate. All stakeholders benefit if mines utilize the full potential of the orebody. In the State Intervention into the Mining Sector (SIMS) document, the ANC indicated that the State should be allowed a greater portion of the profits ('supertax or windfall tax') (ANC, 2012) and indicated that the resource royalties tax in its current form is not achieving this. The SIMS document (released in June 2012) called for reviews of how the State receives revenue from the mining sector. There could thus be changes in the future as to how the mineral resource royalty is structured. However, this study is focused on the tax in its current form, and how it affects mine planning and investment decisions.

There are two forms of taxation that will be considered for this study: income tax (in the form of gold tax for gold mines) (South African Revenue Services, 2011) and the mineral resource royalty (Republic of South Africa, 2008). The gold taxes, as well as the mineral resource royalty, both have sliding scales based on the profitability of the operation. In periods with higher profitability, the structure of these taxes result in a higher percentage of the revenue generated being paid to the State. The difference in the two forms of tax is on what they are levied. Income tax is paid as a percentage of the profit, while the mineral resource royalty is paid on total gold sales revenue. These two taxes address the issue that the State has raised concerning not receiving a higher portion of the revenue in times when commodity prices are high. The standard rate of income tax for other commodities does not allow for this, or protect the operations in times of low commodity prices.

Research background and context

The mining companies that the study focused on operate mines that exploit the narrow, tabular mineral deposits that characterize the gold mines of South Africa. These gold operations include AngloGold Ashanti (the Carletonville and Klerksdorp regions), Sibanye Gold (Carletonville and Free State operations), Harmony Gold (Carletonville, Klerksdorp, and Free State operations) and Gold One (Evander, Randfontein, and Far East Rand operations). For this study, a number of individual mine block listings were selected, together with their corresponding ore flow and financial planning figures.

Mining companies calculate a cut-off grade to determine the portion of the mineral deposit that can be mined economically. This cut-off grade takes into account the forecast price of the commodity, the expected mine recovery factor, the cost to mine the ore and extract the commodity, as well as the fixed costs for the mine. By using the planned extraction rate, expected recovery factor, and total mineral extraction and sales costs, the variable factor in the break-even grade calculation then becomes the *in situ* grade of the material being sold. As long as the grade is higher than the break-even grade in a particular block being mined, the block will be mined profitably. The mining companies that were investigated apply these basic calculations in different forms, but generally use their chosen method to optimize the profit from their operations.

The starting point for the model created for this study is the block listing. This is created from the geological model and is a list of all the potential mining areas, their estimated grades, and volumes. Figure 1 shows a typical tabular gold deposit grade-tonnage curve based on a block model for Mine A used in the study.

Mining companies are approaching the question of mineral resource royalty costs and its impact on the cut-off grade in six possible ways. These are:

- Ignore the implications and continue calculating the cut-off grade in the way it was done prior to the introduction of the tax
- Use the minimum rate (0.5%)

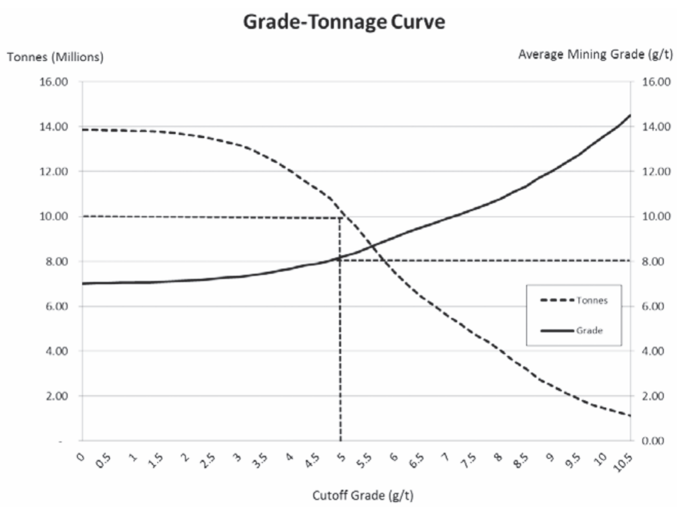

Figure 1—Typical grade-tonnage curve with a cut-off of 5 g/t. The resultant tons above the cut-off are 10.2 Mt, and the average mining grade is 8.1 g/t

- Estimate the expected rate that will be applied by looking at the historical rate
- Estimate the expected rate by modelling the optimized cash flow based on the break-even grade, and then determine the expected rate and use that as an additional cost for break-even grade calculations
- Assume that the highest rate will be applied, depending on whether the mine is applying the refined (5%) or unrefined (7%) rate
- Use some sort of profitability optimization equation that applies a variable rate depending on the profitability, and determine if there is a higher profit to be made even if the profitability rate is lower.

Cut-off grade calculations

According to the SAMREC Code, Mineral Reserves are that portion of the mineral resource that is valuable, and legally, economically, and technically feasible to extract (SAMREC, 2009). The commonly accepted method for determining if material forms part of the Mineral Reserve is thus to calculate the cut-off grade at the current economic conditions. Only material above this grade is then considered to have economic value and included in the mine plan. The material with a grade lower than the cut-off remains in the Resource; although with rising commodity prices and/or lower mining costs it may at a later time be included in the Mineral Reserve. The basics of the cut-off grade theory are described in Lane's '*The Economic Definition of Ore*' and most researchers use this book as the basis for their models (Lane, 1988). Lane describes the economic principles of how cut-off grades are derived and also how cut-off grades can be optimized at various stages of a mine's life. Minnitt (2003) looked at how Lane's cut-off grade calculations were being adapted to Wits-type gold mines in, and found that the application of the net present value (NPV) criterion for determining and optimizing value in mining operations was limited. He considered NPVs at various points in the value chain (mining, processing, and marketing) to determine a balanced cut-off grade. Both Lane and Minnitt consider the NPV calculated over the life-of-mine rather than short-term profitability as the primary measure of value. The model created for this study simplifies these dynamic and balanced cut-off grades into a single process using the block listing as the starting point.

'*Cut-off Grades and Optimising the Strategic Mine Plan*' by Hall (2014) is a comprehensive study of the various techniques currently used in the mining industry. It includes various measures of value including optimizing the discounted cash flow DCF and NPV.

The cut-off grade calculation is essentially very simple. It determines the grade required for a unit of ore to return a profit. It is essentially a break-even volume calculation where the volume is known (usually limited due to shaft capacity, mill capacity, or some other physical constraint), and the unknown is the *in situ* grade of the commodity. The other parameters required are total fixed cost and unit variable cost. From these the total unit cost can be obtained (typically expressed in rands per ton). The other factor required for the cut-off grade calculation is the mine recovery factor (MRF), which is the mine call factor (MCF), multiplied by the plant call factor (PCF). The commodity price in rands per gram is obtained from the commodity price in US dollars (usually quoted per troy ounce for gold and platinum) and the exchange rate. These are all estimates and subject to variation throughout the period in which the cut-off grade is to be used, and thus add to the financial risk to the investors if they change significantly. This can be expressed as follows:

$$\text{Unit total cost (R/ton)} = \frac{\text{(Total fixed cost (R/ton)}}{\text{Volume (tons)}}$$
$$+ \text{ Unit variable cost (R/ton)}$$

$$i.e.\ \text{UTC} = \frac{\text{(TFC)}}{X} + \text{UVC}$$

Unit revenue (R/gram) = Grade (g/t) * Mine recovery factor (%) * Price (R/gram)

$$i.e.\ UR = Grade * MRF * Price$$

Thus

$$\text{Grade} * \text{MRF} * \text{Price} = \frac{\text{(TFC)}}{X} + \text{UVC}$$

(since unit revenue = unit total cost)

$$\text{Grade} = \frac{(\frac{\text{(TFC)}}{X} + \text{UVC})}{(\text{Price} * \text{MRF})}$$

The costs that are included in the cut-off grade calculation are subject to much debate and often change through the life of the project. While a company is still recovering the initial capital costs, a budget cut-off grade can be used. This will include the costs as well as an additional percentage to recover the initial capital costs quickly. In the final stages of the mine, development costs are minimal and certain areas can be mined that were previously considered below cut-off grade. This is called a marginal cut-off grade (Lane, 1988).

Companies can use some sort of 'optimizer' program that utilizes the block listing, as well as the basic inputs, to calculate the cut-off grade. The grade-tonnage curve is then automatically generated – indicating how much material is available above the cut-off grade. The average mining grade (AMG) is also then obtained. This is the average grade of the material above the cut-off grade and becomes the planning grade.

Harmony Gold Mining Company uses an in-house 'optimizer' for calculating the cut-off grade for all their narrow tabular orebodies (Harmony Gold Mining Company, 2001). This optimizer was also created in Visual Basic and Excel. The author is familiar with this optimizer and has used it extensively. The optimizer requires the following as inputs to calculate the cut-off grade:

- The block listing with the measured resource blocks (per shaft section and per reef being evaluated)
- An assumed gold price
- Planned production rates
- Planned working costs (rands per ton)

- The mine recovery factor (MRF), which is equivalent to the mine call factor multiplied by the plant recovery factor.

Harmony has been using their optimizer program for approximately 15 years, and it is based on work done previously by one of their geologists in classifying the orebody and determining a profit-optimized cut-off grade.

Optimizing on NPV can drive decision-making to focus on the short-term gains at the expense of the longer term, and thus companies often use the total undiscounted cash flow (Hall, 2014). The discount rate used for the calculation of the DCF and resultant NPV is critical to the cut-off grade calculation. For this study, costs and income parameters were kept constant and a real discount rate of 12% was used for the NPV optimizer model as this is near the upper limit for typical South African mining companies (Smith *et al.*, 2007). This figure was chosen to emphasize the expected differences between the non-discounted cut-off grade and the NPV-optimized cut-off grades and resultant mineable reserves.

Resource royalty and income tax calculations

Mineral resource royalty

The introduction of the mineral resource royalty has increased costs as it is payable on the total revenue from the sale of the commodity, not just on the profits generated by the sales. This leads to increased mining risk, because even if the mine makes a loss, there is still a payment due to the State. The calculation of the mineral resource royalty is as follows:

Refined Royalty Rate Y(r)

$$Y(r) = 0.5 + X/12.5$$

where X is Earnings Before Interest and Tax (EBIT)/(aggregate gross sales)

Unrefined Royalty Rate Y(u)

$$Y(u) = 0.5 + X/9.0$$

where X is EBIT/(aggregate gross sales).

The difference between the refined and unrefined rates was introduced to encourage companies to beneficiate the commodities. These formulae take the profitability of the company into account when calculating the rate payable, with highly profitable companies paying a higher rate (Republic of South Africa, 2008). There is thus scope for optimizing the cut-off grade by considering how these formulae vary with changes in profitability.

Gold tax formula

Gold mining companies pay income tax based on the gold tax formula with the following formula:

$$Y(\%) = 34\text{-}170/X + 10\% \text{ dividends tax}$$

where X is the profit / revenue ratio expressed as a percentage.

The gold tax is only paid on the taxable profits of the mining company and excludes non-mining income (South African Revenue Services, 2011) and is generally well understood by the various mining companies. The tax also has an element of variation due to the profitability, and this was considered together with the mineral resource royalty when optimizing the cut-off calculations.

A good tax should not disturb economic behaviour and thus should be neutral (Mangondo, 2006). Income tax is calculated on profits and therefore does not alter economic behaviour as making a profit is considered a primary objective of any mining business. The mineral resource royalty is not considered neutral because the implementation of the tax alters the cut-off grade calculation and influences the portion of the orebody that can be considered economically viable.

Modelling exercise

For this study, a model has been developed that calculates the break-even cut-off grade value of the orebody. This is then run through an Excel optimizer model excluding the cost of the mineral resource royalty. The resultant profitability percentage (earnings before interest and tax/ divided by total revenue), dictates the rate applicable for the mineral resource royalty. The cost of the mineral resource royalty is determined from the applicable formula and converted into a cost per ton. This is then added onto the direct mining cost and the revised cut-off grade is determined. The resultant differences in the two cash flows can be compared to determine the impact of the mineral resource royalty on overall mine income, income tax, and costs, which could lead to further sources of revenue generation for the State (including direct taxes like employee income tax as well as secondary taxes like value-added tax on supplies, customs duties, dividend taxes, exercise duties, Social and Labour Plan contributions, skills development levy, and fuel levies). It has been estimated that in 2009, 25% of tax revenue came from direct company income taxes, 30% from personal income tax, and the balance from these other revenue streams (Cawood, 2011). The model was then re-run to optimize using the DCF and resultant NPV in the cut-off grade calculation. A 12% cost of capital was used for this exercise and the results obtained as in the previous method, which optimized on basic profit not considering the time value of money.

Break-even grade calculation

The break-even grade calculation is very simplistic. It just considers direct mining costs. The MRF is utilized and consideration is given to the additional waste tons that are milled (expected dilution like gullies, historical discrepancies, and waste development hoisted as reef). It is possible to depict a grade-profit curve. Where the profit peaks, the grade at this point is the cut-off grade. This relationship is shown in Figure 2.

For the case study, a gold price of R420 per gram was used (US$1136 per ounce with R11.5 per US dollar. The total costs for the mines (including corporate costs) as well the direct mining costs are the based on figures supplied by the mines and used in the 2015 planning cycle for break-even grade determination. The mining costs are based on mining/milling figures and are assumed to represent an optimal rate of mining considering constraints for each individual operation included in the study. The MRF used is based on the previous 12 months' ore flow figures, as well as the historical PCF. This break-even grade is calculated independent of the block listing.

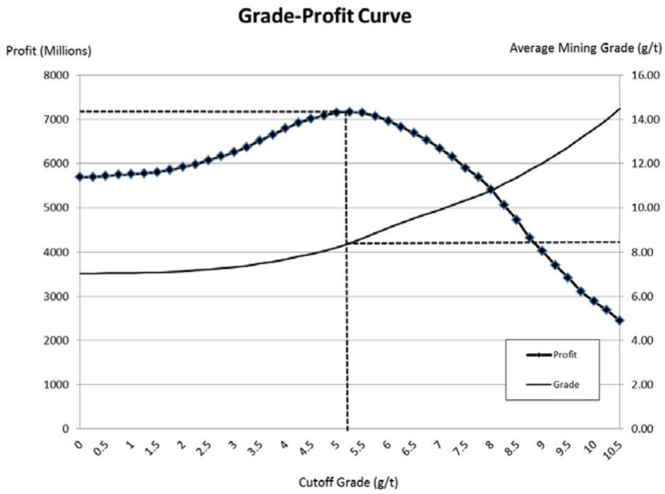

Figure 2—Typical grade-profit curve. The peak of profit is at a cut-off grade of 5.25 g/t. The total profit is R7 173 million and the average mining grade is 8.4 g/t

The break-even grade was calculated excluding as well as including the estimated cost of the mineral resource royalty. The cost of the mineral resource royalty was estimated by inserting the break-even cut-off grade into the model and determining the expected rate. This was then applied to the total sales value and thus the cost per ton of the mineral resource royalty was determined.

Optimizing models

A more advanced method for determining the cut-off grade is to use the block listing for each individual orebody. The block listing contains gold grades in grams per ton (g/t) or centimetre-grams per ton (cmgt), the channel width, stoping width, and area of the blocks. From the area and specific gravity, the tons can be determined for each individual block. The ratio of tonnage from stope faces compared to all the tonnage milled is determined from a simple ore flow. This ore flow considers face tonnage, gully dilution and other sources of dilution, historical discrepancies, and how much development waste will be hoisted and milled with the ore. The ore flow also uses the historical MCF and PRF to calculate the planned MRF for use in the financial model. Revenue is derived from the recovered gold, the planned gold price, and the expected exchange rates. The mining costs can be estimated considering the fixed and variable costs for the mine and the expected production rate, and the resultant profit for each block can be determined.

Two methods were considered for cut-off grade optimization to maximize the profit as well as the NPV. They both utilize the Solver function built into Microsoft Excel (Meissner and Nguyen, 2014). The variable for the Solver function is the cut-off grade, and the Solver function is set up to optimize the cut-off grade to maximize either the resultant profit or NPV from the cash flow. The financial model was limited to a maximum of 20 years. Due to the discounting, income after 20 years has very little impact on the overall NPV.

For the profit model, the Excel Solver function is used to determine the optimal cut-off grade by varying the cut-off grade to maximize the profit. Only the blocks above this resultant cut-off grade are mined. The AMG is determined and this grade is then used in the financial planning. The cut-offs obtained using this method are almost identical to those obtained using the break-even method. Using this method, there is no consideration given for the time value of money, cost of capital, discount rates, or NPV. The model was run both including and excluding the costs determined from the break-even calculation for the mineral royalty resource tax. This allowed comparisons to be made for total revenue, total costs, mineral resource royalty and income tax (gold formula) payments, profit, and life of mine.

To calculate a DCF, a discount rate is required. This discount rate is essentially the cost of capital and it is usually calculated by the weighted average cost of capital (WACC). This essentially considers all the sources of capital required for a project (equity and debt), the portion of the total each source comprises, and its cost. The various sources are then weighted by their proportion and an average is calculated. A risk factor can be added to the WACC to account for uncertainties in the plan and ensure a positive NPV, even if the plan is not achieved.

According to Smith *et al.* (2007), real discount rates of 9–12% for mining projects are appropriate for South African mining projects. This is equivalent to 14.5–17.6% at a 5% annual inflation rate for WACC in nominal terms. For this study, 12% was used for the real cash flows.

Like the profit model, the NPV model considers the total face tonnage available as well as the AMG for the blocks above cut-off grade. The total planned milled tonnage is considered a fixed amount determined considering the constraints on the shaft. The other assumption is that the mix of mining areas will be the AMG, and thus the mined grade for the financial model is this grade.

The NPV is calculated by applying the selected discount rate to obtain the DCF for each year, and adding these together. The model then runs through a series of cut-off grades, with the resultant AMG and available tons being determined for each cut-off. Solver then selects the cut-off grade that results in the highest NPV.

Results

Two exercises were conducted. The first exercise involved a single mine and determining the impact on the cut-off grade and resultant AMG, total income and costs, income tax, and mineral resource royalty. Just the profit optimizer model was used as the exercise was run to determine the impact of profitability on the above measures. The second exercise followed the same methodology, but different mines were selected with their current planning parameters and financial forecasts.

Exercise 1

Figure 3 shows the difference between the break-even grades as the profitability decreases, resulting in the mineral resource royalty rate to decreasing from 5% to 0.5%.

As can be observed for this particular mine, the difference between the break-even grade when royalties are added compared to when they are ignored is approximately 0.4 g/t when profitability is high, and 0 g/t when profitability is low.

Figure 4 shows the differences in revenue the State will receive in direct taxes from the mine with or without mineral resource royalty included as the profitability and resultant mineral resource royalty varies.

From Figure 4, the inclusion of the mineral resource royalty lowers the income tax as it is a tax-deductible cost. The gold tax formula is based on profitability, and the rate for the tax also changes (unlike for other commodities, which are paid at the constant corporate rate of 28%). The difference in State revenue when mineral resource royalty is included is highest when the mine is very profitable. For this mine, the difference in overall direct tax is R730 million, dropping to less than R250 million as the mine becomes very marginal.

Figure 5 shows the differences in gross sales (total revenue) for the mine as the profitability varies.

From Figure 5, if the mine is very profitable or very marginal, there is little difference in total production of gold. However, if the mine is being taxed between 1.5% and 3.0% mineral resource royalty, there is a significant drop in total revenue. This varies between R790 million and R1 220 million for this range.

Exercise 2

Figures 6, 7, and 8 show the differences in break-even grade, profit-optimized cut-off grade, and NPV-optimized cut-off grade for seven different Witwatersrand-type gold mines. The discount rate used in the NPV model is 12%.

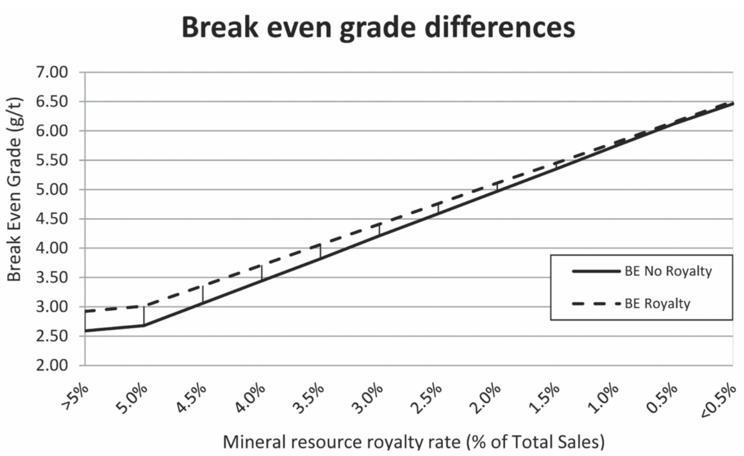

Figure 3—Changes in the break-even grade as the profitability of a mine varies

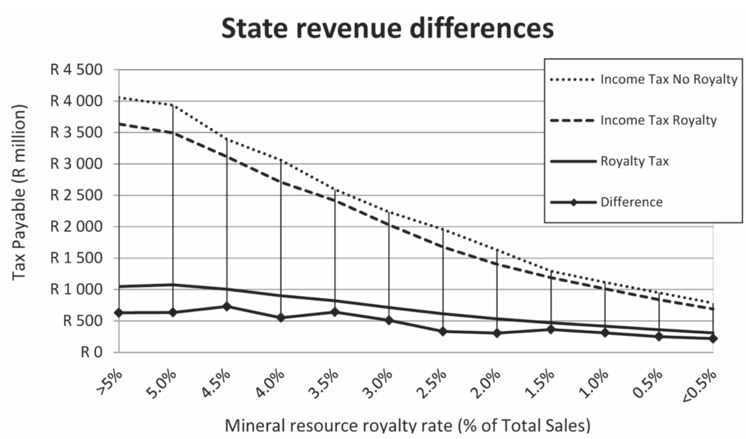

Figure 4—Differences in State revenue

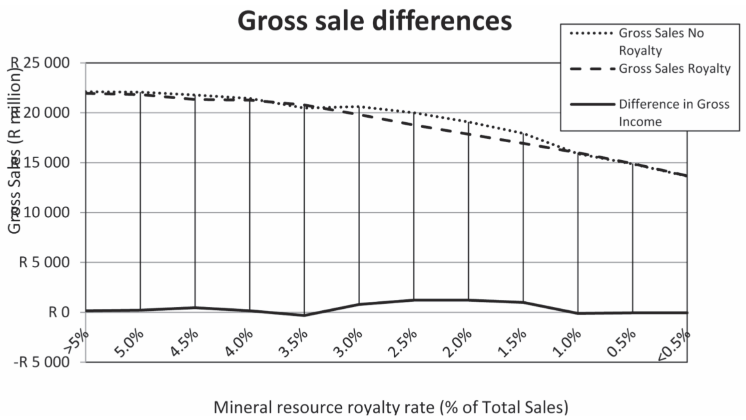

Figure 5—Differences in gross sales with different profitability

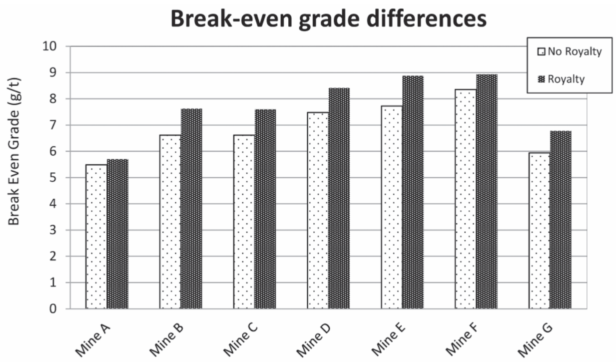

Figure 6—Differences in break-even grade due to mineral resource royalty

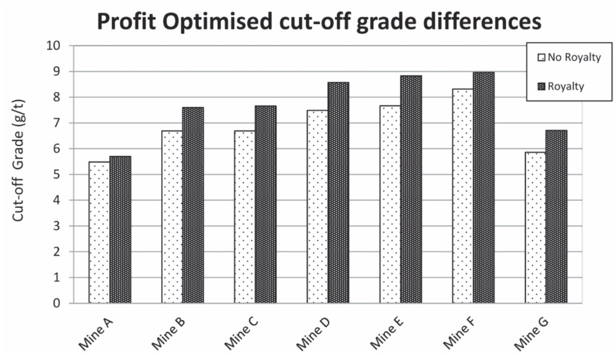

Figure 7—Differences in profit-optimized cut-off grade due to mineral resource royalty

From Figures 6 and 7 the break-even grade and profit-optimized cut-off grades are essentially the same. There is generally approximately a 1 g/t difference between the cut-off grade including mineral resource royalty and excluding mineral resource royalty. For Mine A, however, the difference is just 0.2 g/t, while for Mine D this is 1.2 g/t. There is generally a far closer match between the cut-off grades for the NPV-optimized model. However, it can be observed that for Mine D there is a difference of over 2 g/t in the two grades.

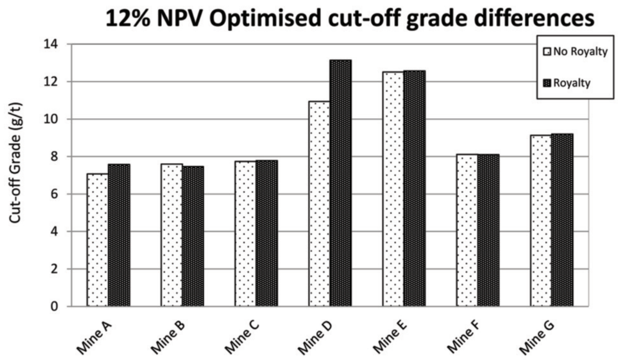

Figure 8—Differences in NPV-optimized cut-off grade due to mineral resource royalty

The impact that modelling for NPV has on cut-off grades generally is noticeable when comparing the break-even and profit-optimized model grades to those obtained with the NPV model. This is due to the model selecting higher grades to increase income in the early years of the cash flow, where the discount effect is less pronounced.

Tables I and II show the taxation differences when the mineral resource royalty is included or excluded using the profit and NPV models respectively.

It can be observed that excluding the mineral resource royalty generally results in less revenue to the State if only direct taxes are considered. The difference is, however, less than just the removal of the mineral resource royalty amounts. This is a function of the formula for gold tax, which is also variable on profitability. The mineral resource royalty figure is considered an income tax-deductible amount. When the mineral resource royalty is included, this results in a lower rate being applied, not a fixed rate like with other mines and companies. One noticeable exception in these sets of figures is for Mine F. When modelled using the profit optimizing model, the result is that excluding mineral resource royalty actually leads to an increase in revenue payable to the State in direct taxes.

Tables III and IV compare the gross sales, costs, and life-of-mine for the mines using both the profit- and NPV-optimized models.

Table I

Profit, income tax, and mineral resource royalty obtained from the profit model, including and excluding the mineral resource royalty (all amounts in R million)

Profit-optimized	Mine A	Mine B	Mine C	Mine D	Mine E	Mine F	Mine G
Profit no royalty	R3 180	R2 870	R21 275	R12 805	R22 462	R445	R601
Income tax no royalty	R1 200	R1 281	R9 561	R 5 625	R10 038	R175	R264
Profit with royalty	R2 866	R 2 400	R 18 362	R10 579	R22 071	R303	R502
Income tax royalty	R1 068	R1 332	R8 198	R4 633	R9 775	R115	R223
Royalty	R428	R413	R2 449	R1 495	R3 096	R47	R70
Difference	-R296	-R 464	-R1 086	-R503	-R2 833	R13	-R29

Table II

Profit, income tax, and mineral resource royalty obtained from the NPV model, including and excluding the mineral resource royalty (all amounts in R million)

NPV-optimized	Mine A	Mine B	Mine C	Mine D	Mine E	Mine F	Mine G
NPV no royalty	R2 072	R2 619	R8 568	R0 061	R10 204	R432	R602
Income tax no royalty	R1 315	R1 472	R9 103	R5 061	R9 881	R206	R324
NPV royalty	R1 870	R2 369	R7 877	R5 622	R9 412	R394	R556
Income tax with royalty	R1 133	R1 313	R8 267	R4 157	R9 010	R183	R297
Royalty	R399	R410	R2 460	R1 196	R2 562	R68	R82
Difference	-R217	-R251	-R1 624	-R292	-R1 691	-R45	-R55

Table III

Gross sales, costs, and life-of-mine obtained from the profit-optimized model, including and excluding the mineral resource royalty (amounts in R million)

Profit-optimized	Mine A	Mine B	Mine C	Mine D	Mine E	Mine F	Mine G
Gross sales no royalty	R17 020	R7 674	R54 323	R37 725	R59 489	R2 110	R1 771
Gross sales royalty	R15 848	R8 266	R48 973	R31 715	R61 932	R1 955	R1 398
Difference in gross income	R1 172	-R592	R5 350	R6 010	-R2 443	R155	R373
Taxable costs no royalties	R13 840	R4 804	R33 048	R24 920	R37 027	R1 665	R1 170
Taxable costs royalties	R12 982	R5 866	R30 611	R21 136	R39 861	R1 652	R896
Difference	R858	-R1 062	R2 437	R3 784	-R2 834	R13	R274
LOM no royalty	11	4	20	18	20	2	2
LOM royalty	10	3	17	14	20	2	2

Table IV

Gross sales, costs, and life-of-mine obtained from the NPV model, including and excluding the mineral resource royalty (amounts in R million)

NPV-optimized	Mine A	Mine B	Mine C	Mine D	Mine E	Mine F	Mine G
Gross sales no royalties	R12 524	R8 266	R49 199	R28 028	R51 898	R2 209	R1 640
Gross sales royalties	R11 174	R8 206	R49 199	R24 286	R51 898	R2 209	R1 640
Difference in gross income	R1 350	R60	R 0	R3 742	R0	R 0	R0
LOM no royalty	7	4	17	11	15	2	2
LOM royalty	6	3	17	9	15	2	2

These tables display how including mineral resource royalties generally leads to a drop in total gold revenue. This is more pronounced when optimizing for profit than when optimizing for NPV. The differences in total costs is where the State loses the opportunity to levy further revenue, including personal income tax on mine employees, value-added tax on supplies, customs duties, dividend taxes, exercise duties, Social and Labour Plan contributions, skills development levy, and fuel levies. Considering that only 25% of total State revenue comes from direct company taxes, the loss of income generation opportunities to the State due to the reduction in mine spending could far outweigh the income from the mineral resource royalty.

The differences in the life-of-mine figure, including or excluding mineral resource royalty, are more pronounced with the profit-optimized model. Using the NPV-optimized model lowers the life-of-mine compared with the profit-optimized model, and adding the costs of the mineral resource royalty has a less of an impact.

The reductions noted in life-of-mine, gross sales, costs, profit, and NPV between the different mines as shown in Tables I to IV are not consistent. Each mine is affected by the mineral resource royalty in a different way, even though the same methodology for the modelling has been kept consistent for the exercises. This is due to differences in grade distributions in the orebodies, direct mining costs, indirect costs, and variances in the ore flows. Figures 9 and 10 show grade-tonnage curves for Mines C and G.

Mines C and G are neighbouring mines with similar depths and mining costs. The break-even grades, excluding and including the mineral resource royalty respectively, are 6.62 g/t and 7.63 g/t for Mine C and 5.94 g/t and 6.78 g/t for Mine G. The cost of the mineral resource royalty has a greater impact on the break-even grade for Mine C (1.01 g/t vs 0.84 g/t). It can be seen by comparing the two graphs that the impact of including the costs for the mineral resource royalty has a far greater impact on the AMG, as well as tons above cut-off grade, for Mine G than for Mine C. The these different grade-tonnage curve shapes demonstrate that adding the costs of the mineral resource royalty into the cut-off grade for Mine C results in a drop of 10% in gross sales, while for Mine G this drop is 21%.

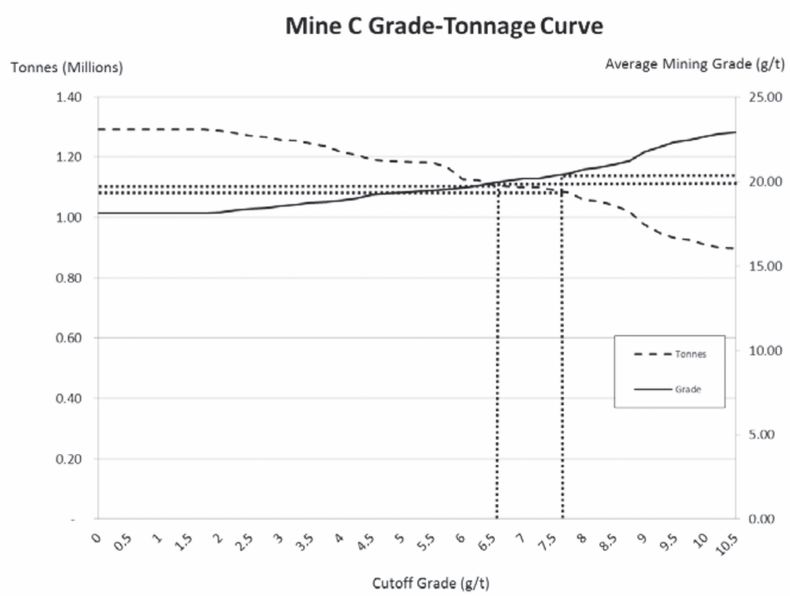

Figure 9—Grade-tonnage curve for Mine C, showing the AMG and corresponding tonnage for the cut-offs calculated including and excluding the cost of the mineral resource royalty

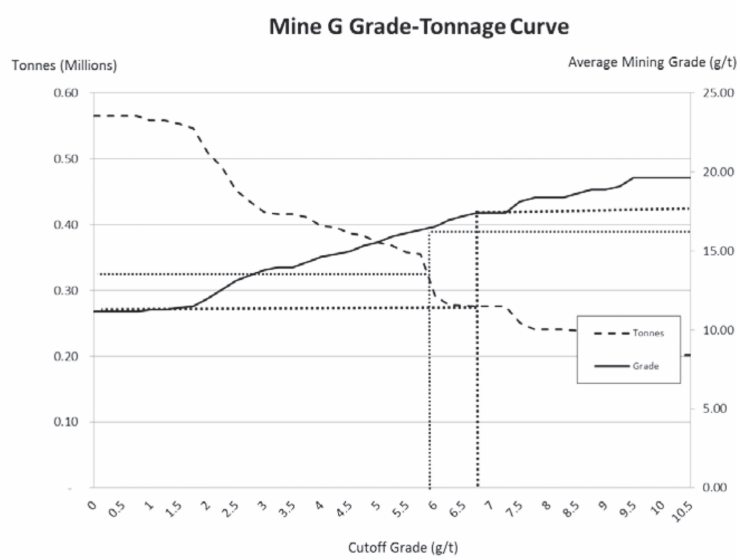

Figure 10—Grade-tonnage curve for Mine G, showing the AMG and corresponding tonnage for the cut-offs calculated including and excluding the cost of the mineral resource royalty

Conclusions

The mineral resource royalty introduced in 2010 has an effect on mineral reserves. Since the royalty is paid on total mineral sale revenue, it is considered as an additional cost. This then causes the break-even grade to increase, and reduces the tons above this grade that can be economically mined.

In the first part of the study, a single mine was selected and the block list run through an optimizer model developed for this purpose. The optimizer uses the Solver function in Excel to vary the cut-off grade and determine the optimal cut-off grade to ensure either maximum profit or maximum NPV. The result of this exercise showed that the impact of the mineral resource royalty is greater on mines that have higher profitability if just considering the differences in cut-off grade. The largest impact on gross sales (due to reducing the life-of-mine), however, was noted when the profitability of the mine was less, resulting in mineral resource royalty rates between 1.5–3%. For very marginal mines, the sliding scale of the mineral resource royalty results in little change to the overall financials of the mine as it is a constant 0.5% of gross sales revenue.

The second exercise compared seven currently operating gold mines. The mines have different grade-tonnage curves and the effect of mineral resource royalty differs for each one. Changes in cut-off grade varied from 0.2 g/t to 1.2 g/t. The increased direct revenue payable to the State in most cases was largely offset by reduced taxation opportunities due to lower mineable reserves and reduced mine life. The impact was generally greater when the mines were optimized for profit rather than for NPV, which already reduces the life-of-mine significantly due to high-grading the orebody.

This study has shown that although the tax is meant to protect marginal mines from early closure, mines with higher profitability are negatively affected and the additional costs can reduce the life significantly. The expectations of the State to receive additional revenue from the mines are satisfied in the short term. However, when considering the income generation capacity from other taxation forms over the life of the mine, this revenue is likely to be less than expected. Considering the total impact for these seven mines, the profit-optimized model indicates that R7.9 billion is likely to be paid in mineral resource royalty. Income tax decreases by R2.8 billion, and thus the State gains R5.1 billion in direct tax. The difference in total gross sales decreases by R10 billion. Additional tax revenue to the State decreases by R3.5 billion. A significant portion of this amount would have been paid to the State in the form of other taxation. On deep-level gold mines, labour costs represent more than 50% of total costs. The loss of earnings to mine employees due to the mineral resource royalty for these seven mines is estimated to be R1.75 billion.

Recommendations

A complete study should be conducted across all the mines to establish the impact of the introduction of the mineral resource royalty on the industry. The income tax on profits of gold mines is calculated differently from mines producing other commodities, and this reduces the impact of the mineral resource royalty on the mineral reserves compared with taxation on the fixed 28% rate.

Each orebody is unique, and studies to establish the industry-wide impact of the introduction of the mineral resource royalty should take this into account, rather than looking at just industry-wide averages. The impact on some mines could be minimal, while the impact on others could be very significant. The results of a full study should be considered in the light of declining mining revenues and expected continuing job losses. This could then be used to guide future policy changes as proposed in the SIMS document (ANC, 2012).

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Rock Strength and Geometallurgical Modelling, Mogalakwena Mine

by J. P. Germiquet* and R.C.A. Minnitt†

Synopsis

Rock properties have a material impact on mining processes, including drilling performance. An investigation using point loaded index (PLI) data converted to uniaxial compressive strength (UCS) has revealed that a direct relationship between grain size and UCS exists at Mogalakwena Mine. This correlation is best seen in unaltered rock, with lower correlations for altered rock types. Measurements from the new RockMa system installed on drill rigs can be used to obtain rock strength data to validate the current rock strength domains and create additional data for the next benches below. An investigation of penetration rates in different lithologies shows that rock composition plays an important role in determining drill performance. Additionally, there is an inverse relationship between rock strength and drilling penetration rate – a measure of how efficiently a hole is drilled.

The domaining of grain-size-adjusted UCS at Mogalakwena Mine will allow more accurate scheduling of drill rigs through increased knowledge of rock strength in various areas. Successful rock strength domaining has the potential to be incorporated into blast indexing and predicting crushing/milling performance.

Keywords
geometallurgy, rock strength, point load, comminution, drilling rate of penetration, crushing.

Introduction

The optimization of mining and processing through geometallurgical characterization of ore are key drivers of value realization (Deutch, 2013). Rock strength is one such property and is affected by, among other variables, composition, texture, and grain size (Ozturk, Nasuf, and Kahraman, 2014). Mogalakwena Mine is exploiting the intrusive Platreef PGE (platinum group element)-bearing orebody which forms part of the northern limb of the Bushveld Complex. This complex orebody exhibits areas of metasomatic alteration, metamorphism, and sporadic mineralization of the footwall and hangingwall (Nex *et al.*, 2006). Understanding this variability is paramount due to the implications for rock strength and hardness, which in turn affect crushing, milling, and liberation efficiency. These variables must be known prior to blasting in order to optimize mine and plant processes using *in-situ* variables. This necessitates predictive modelling (Deutch, 2013). Geological and geotechnical data was sourced from the Overysel (OY) exploration database for the study. Point load testing was done for each borehole and the results converted to uniaxial compressive strength (UCS) (Akram and Bakar, 2007). The relationship of lithology, texture, and grain size with UCS revealed an inverse relationship between grain size and UCS in unaltered rock. The relationship was not present in altered rock types. A comparison of mean strength and hardness results indicated that Bond work index and UCS may be related. Drilling performance is affected by UCS (Bourgoyne *et al.*, 1986), and this was proven using recent measurements obtained during drilling. Lastly, grain-size-adjusted UCS values were applied to a 3D model for the OY property.

The case for understanding rock properties

Rock properties have a material impact on mining processes, including slope design, drilling, blasting comminution, and flotation. An investigation into penetration rates in different lithologies will show that rock properties play an important role in determining drill performance. There is an inverse relationship between rock strength and drilling penetration rate – a measure of how efficiently a blast-hole is drilled (Bourgoyne *et al.*, 1986). UCS is used widely in the mining industry to quantify rock strength. An investigation using point load index (PLI) data converted to UCS has revealed a direct relationship between grain size and UCS at Mogalakwena. This correlation is best seen in

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unaltered rock, with lower correlations for altered rock types. Drilling data from the newly installed RockMa system on drill rigs can be used to obtain rock strength data to validate the current rock strength models and create additional data for benches below. The domaining of grain-size-adjusted UCS at Mogalakwena Mine will allow for more accurate drill rig scheduling and blast indexing, and open up the potential for predicting the comminution characteristics of ore batches.

Lithological and textural investigation

Data from 754 exploration boreholes located on the OY property shows that 97 rock types and 34 textural descriptions are used in the Mogalakwena database. Although complexity of the nomenclature allows for greater naming accuracy, it reduces both standardization and usefulness while inadvertently encouraging subjectivity. The analysis revealed that only seven rock types and four textural descriptions are sufficient to represent 97% of the coded lithological entries for the reef stratigraphy (Figures 1 and 2).

The major reef lithology is feldspathic pyroxenite, which conforms with studies done at other farms within the Mogalakwena Mine property (Hayes and Piper, 2003) (Figure 1). Feldspathic pyroxenite and pyroxenite comprise 80% of the lithological entries in the database. Entries reveal the less frequent occurrence of serpentinization of the reef at OY compared to the Zwartfontein and Sandsloot farms.

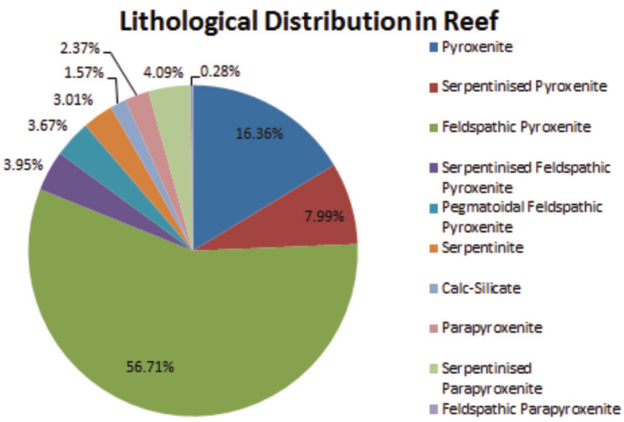

Figure 1—Lithological descriptions used for reef stratigraphic unit

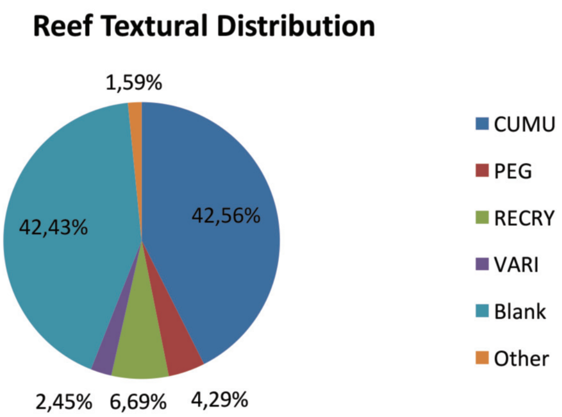

Figure 2—Textural descriptions used in the reef stratigraphic unit for cumulus (CUMU), pegmatitic (PEG), recrystallized (RECRY), and varied (VARI) units

Texturally, the unaltered reef lithologies (feldspathic pyroxenite and pyroxenite) are dominated by a cumulate textures, while altered reef lithologies are dominated by a recrystallized texture (Figure 2).

Relationship between rock strength and texture

A number of compositional and textural factors influence rock strength. Grain size specifically has a significant influence on rock strength (Akram and Bakar, 2007). Generally, rock strength ranges at Mogalakwena Mine would be classified as high to very high, and fall within a rather narrow range. The relationship between grain size and UCS for various rock types is depicted in Figure 3.

The results show that rock composition and mineral content influence the range of UCS for specific rock types. Feldspathic pyroxenite (FPYX) has a higher feldspar content than pyroxenite (PYX), resulting in a higher mean UCS. The curves in Figure 3 indicate an inverse relationship between grain size and rock strength for unaltered rocks. This relationship was confirmed with a set of 41 feldspathic pyroxenite samples from exploration core. The variability in UCS between rock types shows that major rock types differ sufficiently in mean UCS to allow for rudimentary geometallurgical rock strength characterization.

Correlation between rock strength and hardness

Expanding the research into the relationship between rock strength and comminution-specific tests would strengthen the use of *in-situ* strength parameters in process optimization. Studies conclude that there is a fair correlation between UCS and drop weight test (DWT) values, which allows for the use of UCS as a proxy for crushing efficiency (Bye, 2005). Multiple studies confirm little or no correlation between UCS and Bond Work index (BWi) values, indicating that UCS cannot be used as an indirect proxy for milling performance. BWi is an industry-accepted test to describe milling behaviour (Doll, Barrat, and Wood, 2003). Average values of UCS and BWi for rock samples taken at Mogalakwena confirm some correlation (Figure 4).

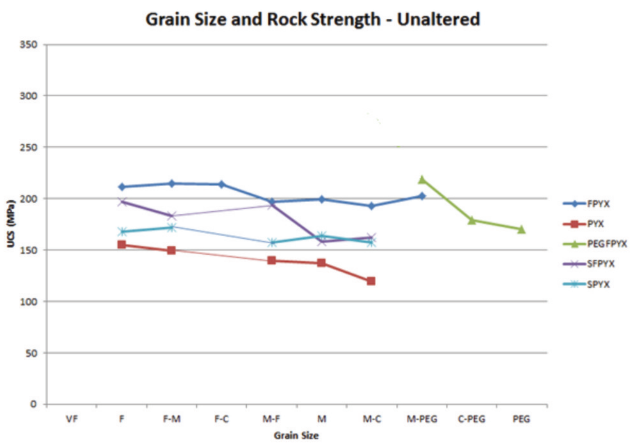

Figure 3—Inverse correlation between rock strength and grain size. Rock nomenclature is as follows: feldspathic pyroxenite (FPYX), pyroxenite (PYX), pegmatoidal feldspathic pyroxenite (PEGFPYX)

Drilling penetration rates and rock strength

The penetration rate of production drill rigs is affected by rock strength (Bourgoyne *et al.*, 1986). A comparison of rock strength and penetration rate shows an inverse relationship between penetration rate and rock strength. Mafic-rich reef material (pyroxenite in particular) has the highest penetration rate, with hangingwall and footwall rocks showing slower rates (Figure 5). More accurate scheduling of drilling rigs can be achieved by utilizing accurate penetration rates for different rock types or stratigraphic units.

Measurements while drilling (MWD)

MWD technology has enhanced the ability to measure drilling performance variables. The results are calculated using the performance variables (but are not calibrated at time of writing). Currently, qualitative rock strength measurements gathered by the RockMa drilling system displays rock strength colour signatures revealing different strengths in different stratigraphic units. The data can be loaded for an entire bench to show strength changes along strike and dip of the orebody. The system was applied to Cut 8 at Mogalakwena, where the hangingwall norite on bench 8 was unaltered, while the reef and footwall were more weathered and oxidized (Figure 6). Plotted blue holes indicate stronger rock than the yellow holes. The area between the blue and red lines is the estimated position of the oxidized reef.

The qualitative data can be used to identify the different stratigraphic units and localized changes in reef strike, which may not be known prior to production drilling. This rock strength information can be calibrated and used to update a UCS model for patterns before blasting.

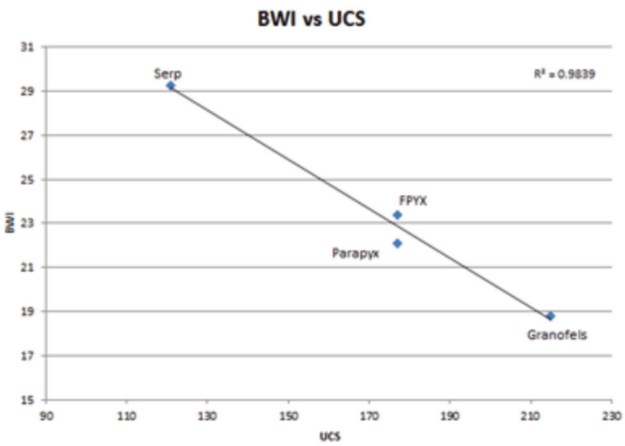

Figure 4—Mean BWi plotted against UCS for major rock types

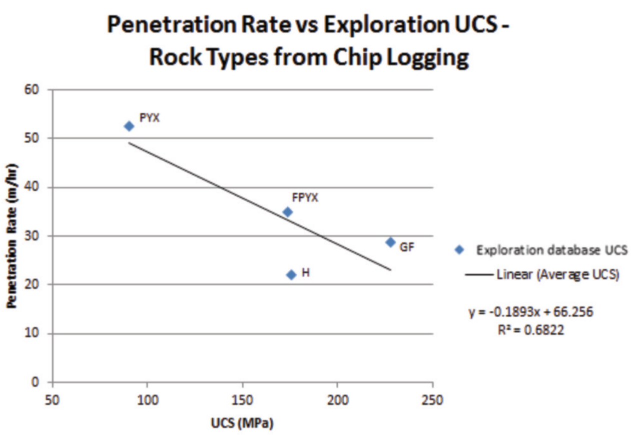

Figure 5—Penetration rate compared to UCS for four rock types. Rock nomenclature is as follows: pyroxenite (PYX), feldspathic pyroxenite (FPYX), granofels (GF), and hybrid norite (H)

Rock strength modelling

A geometallurgical model requires accurate sampling of ore variables (Deutch, 2013). Rock strength is one such variable. Usually, rock strength modelling is based on a calculated average for the rock type which disregards textural differences (Hayes and Piper, 2003). Textual changes were incorporated into the data-set to represent more accurate rock strength changes across the orebody (Figure 7).

The model can be updated at smaller grid spacing using strength data generated by calibrated MWD technology. Potentially, mining blocks can be delineated within the model for predicting and simulating drilling, blasting, crushing, or milling performance. Predictive tools are vital for the optimization of mining and plant processes.

Financial benefits

The South Plant concentrator at Mogalakwena Mine utilizes autogenous milling, which is heavily dependent on two populations of feed material – coarse (strong) and fine (weak) material. The two feed material types are separated after primary crushing by means of a grizzly and stockpiled

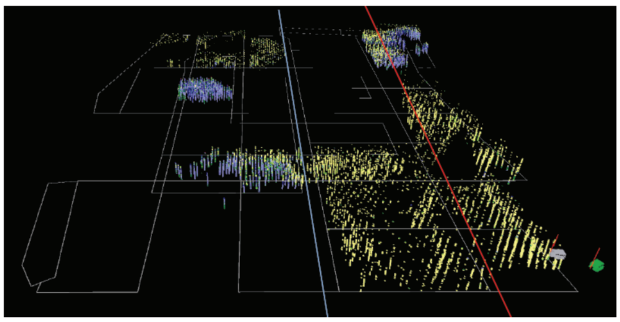

Figure 6—Qualitative rock strength data displayed in the RockMa software package for Cut 8

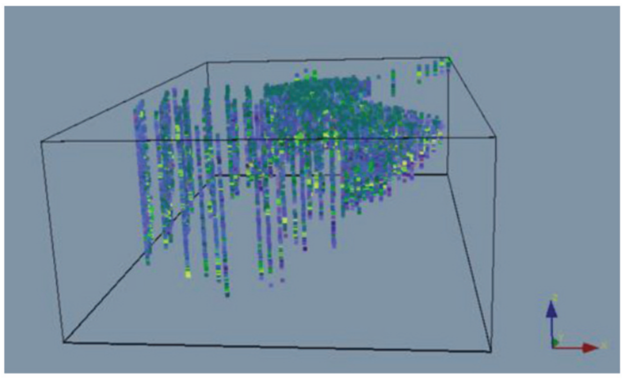

Figure 7—Rock strength domaining using grain-size-adjusted UCS

separately. The plant has suffered from stoppages related to the depletion of the coarse material, resulting in a total of 226.02 hours of lost milling in 2014. At an average milling rate of 380.74 t/h, the lost output is over 86 kt for 2014. Had the stoppages been avoided, total output for 2014 from South Plant would have increased by 2.58%. At mean PGE grades, recoveries, and commodity prices, the lost revenue amounts to R51.35 million.

The findings show that *in-situ* ore strength characterization would allow a stable feed supply, based on the required rock strength proportions in the feed to South Plant. Two additional stockpiles based on rock strength would stabilize the ore feed to the requirements of South Plant (Figure 9).

Discussion

The performance of mining and plant equipment is directly affected by rock properties. Simplified, fit-for-purpose lithological and textural nomenclature allows for useful rock type descriptions in a large-scale opencast mining environment. Lithological and textural descriptions will need to be suited for each pit area in order to represent changes in reef character between farms (such as lower occurrence of altered ores). Additionally, the research shows that identifying pyroxenite and feldspathic pyroxenite within the reef would categorize 80% of the ore. Focusing efforts on characterizing strength changes within these two lithological units would be more beneficial for modelling. The differences in strength of the two rock types alone would indicate probable differences in plant response. Rock strength values obtained through cost-efficient point load testing are inversely correlated withgrain size. This realization allows for more accurate rock strength domaining and modelling. Accurate modelling will also be useful for scheduling drill rigs and averting extended standing times. Mineralogical or geochemical variables can be added to fine-tune the understanding of rock strength and rock type for more advanced geometallurgical modelling. Ultimately, rock strength and hardness domaining will guide plant feed, with a positive effect on plant throughput as shown by the potential financial benefits at South Plant.

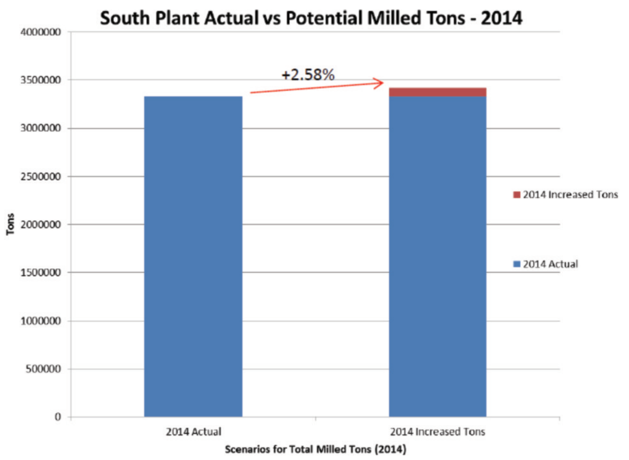

Figure 8—Actual milled tons vs. potential milled tons without stoppages

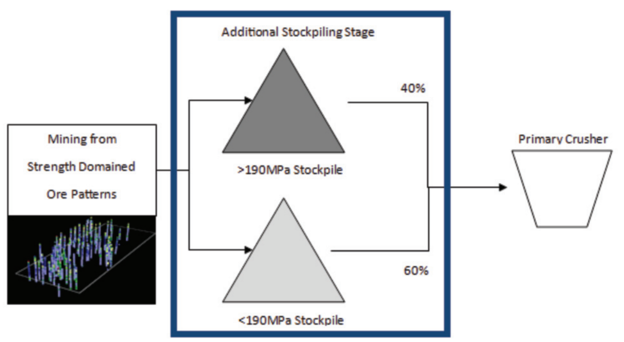

Figure 9—Additional stockpiling stage before primary crushing to stabilize grade and rock strength

Conclusion

The research established a source of relevant data for a geometallurgical programme, which includes rock strength variability based on rock type. Rock strength has a material effect on drilling performance and scheduling can be optimized. The project revealed that ore categories can be tailored to represent variables that impact mining performance, including the potential to predict plant performance. The results increase understanding of geometallurgical variability in day-to-day mining operations, enabling optimization of the downstream recovery process.

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Microscopic analyses of Bushveld Complex rocks under the influence of high temperatures

by G.O. Oniyide* and H. Yilmaz*

Synopsis

The South African platinum mines in the Bushveld Complex (BC) have unique features that distinguish them from the gold mines. They have a higher horizontal stress closer to the surface, lower occurrence of seismic activity (caused mainly by pillar failure), and are sited in an area of high geothermal gradient. One of the future challenges of platinum mining is the increasing temperatures as the mining depth increases. High temperatures invariably bring about higher ventilation costs and require different approaches to human factors and ergonomics. Excavation stability would also be another area of concern due to increasing stresses. In this investigation, the effect of temperature on the physical and chemical properties of selected Bushveld rocks (chromitite, pyroxenite, norite, leuconorite, gabbronorite, mottled anorthosite, varitextured anorthosite, granite, and granofels) was studied by means of optical microscopy and scanning electron microscopy using energy-dispersive X-ray spectrometry (SEM-EDX). The rock specimens were heated in a temperature-controlled oven a rate of 2°C /min to 50°C, 100°C, and 140°C and kept at temperature for five consecutive days. Samples were allowed to cool to ambient temperature (approximately 20°C) before image capturing. Micrographs of the specimens were taken before and after heat treatment. All the SEM samples were also subjected to heating and cooling on alternate days for ten days in order to observe the effect of repeated heating and cooling on the rocks. The results of the optical microscopy analyses showed minor physical changes in the rocks. The SEM images revealed that cracks initiating at lower temperatures extend with increasing temperature. The chemical analyses showed that the temperature range considered in this research is not high enough to induce changes in the chemical composition of rock samples.

Keywords

Bushveld Complex, microscopic analyses, heat treatment of rock.

Introduction

Clauser (2011) observed that the thermal regime of the Earth is governed by its heat sources and sinks, transport processes and heat storage, and their equivalent physical properties. Clauser and Huenges (1995) explained that the interior heat of the Earth is transmitted to its surface by conduction, radiation, and advection. They further stated that the thermal conductivity of igneous rocks is isotropic, while the thermal conductivity of sedimentary and metamorphic rocks displays anisotropy.

Geothermal heat and heat from autocompression, which are sources of underground heat, exist in all mining situations to varying degrees, depending on the geothermal gradient and surface temperature. Other sources of underground heat are blasting, mining equipment, and groundwater, as shown in Figure 1. From the figure, it can be seen that autocompression – that is, compression of air by its own weight – makes the highest heat contribution, followed by heat flow from the rock mass (Brake and Bates, 2000; Payne and Mitra, 2008).

Payne and Mitra (2008) also stated that the geothermal gradient of the upper crust is between about 15°C/km and 40°C/km. A comparison is made between the geothermal gradient of Australia and South Africa. Brake and Bates (2000) asserted that Australia is the world's hottest and driest continent. They explained that the geothermal gradient of Australia is between 23°C/km and 30°C/km. Biffi *et al.* (2007) reported the average geothermal gradient for the South African gold and platinum mines as 13°C/km and 23°C/km respectively. Figures 2 to 5 show the heat flow pattern in Australia and South Africa.

Biffi *et al.* (2007) reported that the geothermal gradient of the South African platinum mines is about twice that of the gold mines. They also pointed out that the rock mass in the platinum mines, mainly norite, cools faster than quartzite, which is the major host rock in the Witwatersrand Basin. However, Witwatersrand quartzite has a higher thermal conductivity and lower heat capacity than Bushveld Complex rocks (Jones, 2003, 2013, pers. Com.). The thermal conductivity and heat capacity of quartzite and norite are 6.37 and 2.32 $Wm^{-1}K^{-1}$ and 810 and 840 $Jkg^{-1}K^{-1}$ respectively. This implies that

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quartzite will absorb and release heat faster than norite. The heating and cooling of rocks may lead to micro-factures in rocks. This is one of the subjects of this investigation.

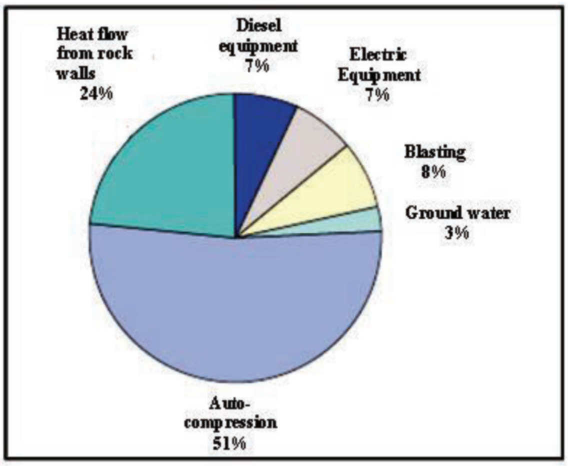


Figure 1—Contributions to underground heat at Mount Isa (after Payne and Mitra, 2008)

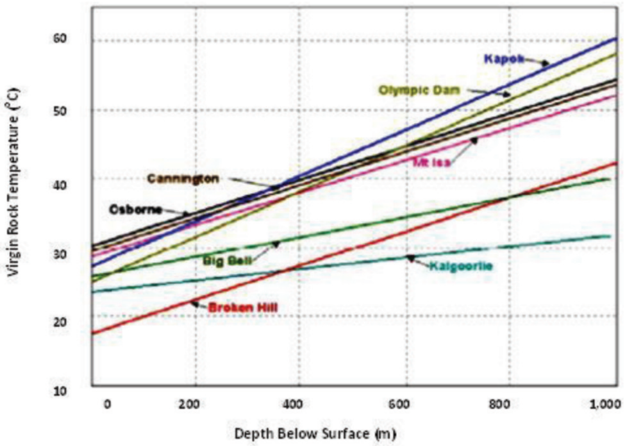


Figure 2—Geothermal gradients at several mining locations in Australia (after Payne and Mitra, 2008)

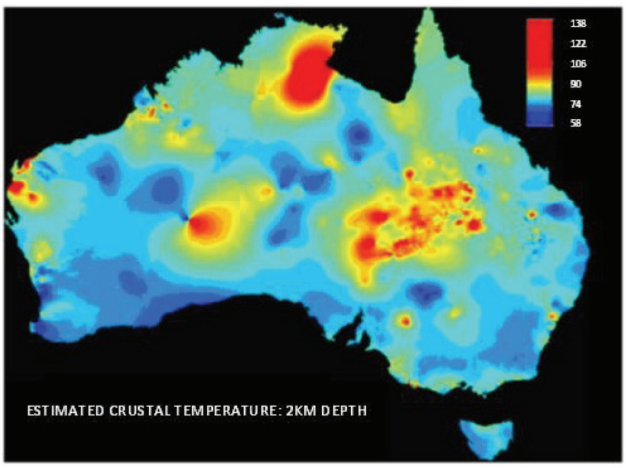


Figure 3—Rock temperatures in Australia at 2 km depth (after Payne and Mitra, 2008)

Comparing Figures 2 and 5, it can be seen that apart from Broken Hill, Kalgoorille, and Big Bell mines, which have similar a geothermal gradient to the platinum mines, all the other Australian mines shown in Figure 2 have higher gradients. Mt Isa is Australia's deepest mine, which started operations in 1924 and is now mining at 1900 m below surface. At the current depth of mining, the virgin rock temperature at Mt Isa mine is approximately 80°C. This temperature will be reached at a depth of 2200 m below surface at Bafokeg Rasimone platinum mine. This means that the investigation into the effect of temperature on the behaviour of rocks in mines will be beneficial not only to South African mines, but also to all other mines across the globe that are located in high geothermal gradient areas. Most of the previous researchers, such as Biffi *et al.* (2007), Brake and Bates (2000), and Payne and Mitra (2008), focused on strategies of improving ventilation or cooling of the mines and safeguarding the health of the underground workers.

The aim of the present research is to examine the effect of heat on rock texture and composition, in terms of initiation or extension of micro-cracks and changes in chemical composition. In the mining context, an attempt was made understand the effect of temperature drop on the surfaces of

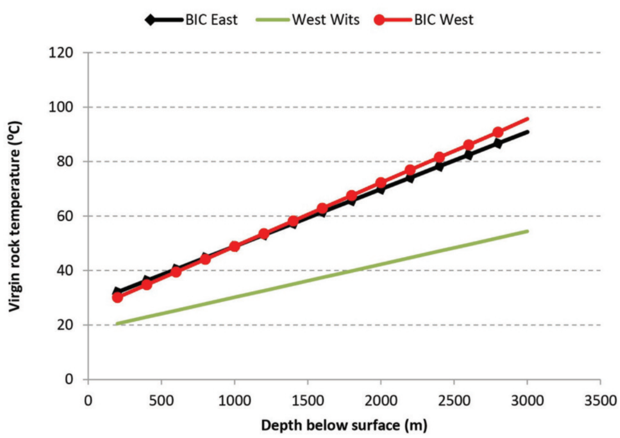


Figure 4—Virgin rock temperatures in South African gold and platinum mines (after Biffi *et al.*, 2007)

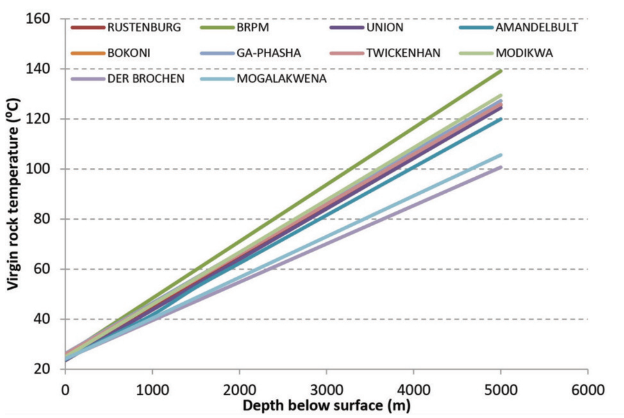


Figure 5—Variation of virgin rock temperature with depth in South African platinum mines

underground openings due to ventilation by the help of microscopic analysis. Planar cracks, which are perpendicular to the mineral grain boundaries, have been reported to form during the cooling of igneous rocks. These cracks were attributed to a mismatch of physical properties across grain boundaries during cooling (Richter and Simmons, 1974; Fonseka *et al.*, 1985). The micro-cracks affect the compressive and tensile properties, sonic velocity, and thermal conductivity of the rocks.

Microscopy

Image were captured by means of two types of microscope:

➤ Optical microscope (OM). The OM used is an Olympus BX61 (Figure 6a). The advantage of the OM is that the colour of the minerals that make up the rock samples can be seen, since surface coating of the specimens is not required. The OM uses visible light and a system of lenses to magnify images of small samples. Using the Olympus BX61, the images were captured in both bright field (sample contrast is due to absorbance of light in the sample) and dark field illumination (sample contrast is due to light scattered by the sample)

➤ Scanning electron microscope (SEM). The SEM used is FEI Nova 600 Nanolab, (Figure 6b). The SEM has higher resolution than the OM, but the specimens were gold-palladium coated in order to allow for good quality image. The SEM produces image by scanning the sample with a focused beam of electrons. The interaction of the electrons from the beam and the atoms in the samples produces signals that contain information about the surface topography and composition.

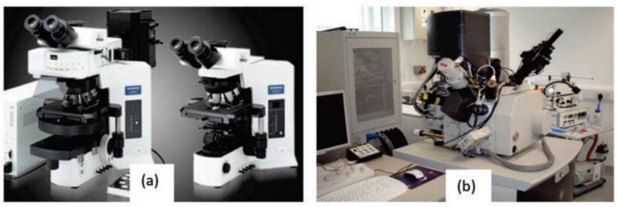


Figure 6—(a) Olympus BX 61 optical microscope, (b) FEI Nova 600 Nanolab scanning electron microscope

Rock samples and location

The rock samples were prepared from geological core samples obtained from mines in the western and northern Bushveld Complex (BC). Chromitite, pyroxenite, norite, and leuconorite were prepared from cores samples from Bafokeng Rasimone Platinum Mine (BRPM) in the western BC; gabbronorite, mottled anorthosite, and varitextured anorthosite from Khomanani Mine (a subsidiary of Anglo American Platinum) in the western BC; while granite and granofels were from Mogalakwena Mine (also a subsidiary of Anglo American Platinum) in the northern BC.

The rock specimens were cut to a thickness of 2.5 mm and the surfaces to be examined were smoothed with the aid of a lapping machine. The surfaces of the SEM specimens were coated with a thin layer of gold-palladium alloy. The layer, which is typically 10–20 nm thick, is necessary to make the specimens conductive. Figures 7 (a) and (b) show the samples prepared for the optical and scanning electron microscope respectively.

Heat treatment

The thermal treatment of the rock specimens was done in a temperature-regulated oven. The samples were heated to 50°C, 100°C, and 140°C at a rate of 2°C /min and kept at constant temperature for five consecutive days. These temperatures are approximately the same as the temperatures in the mines at depths of 1073 m, 3276 m, and 5038 m below surface respectively. Samples were allowed to cool to ambient temperature (approximately 20°C) before image capturing. Micrographs of specimens were taken before and after heat treatment. For SEM analyses, a single specimen from each rock type was subjected the specified temperature range, that is 50°C, 100°C, and 140°C, while different specimens were heat-treated for OM analyses. All the SEM samples were also subjected to heating and cooling on alternate days for ten days in order to observe the effect of repeated heating and cooling of the rocks in the mine face.

Result of optical image capturing

The essence of the optical image capturing was to examine whether heat treatment of the rock sample will have a significant effect of the physical properties in terms of the colours of the minerals. All the samples were imaged using the BX 61 Olympus optical microscope under bright field illumination. Figures 8 to 10 show the micrographs of

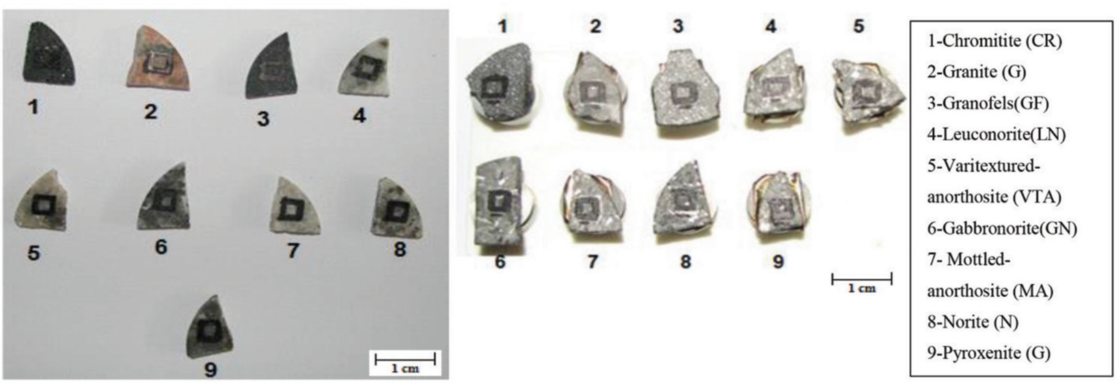


Figure 7—Samples for (a) OM and (b) SEM image capturing

chromitite before and after heat treatment at different temperatures.

Voordouw, Gutzmer, and Beukes (2009) reported the average modal mineral abundance for the UG1 and UG2 seams (Table I). Schouwstra, Kinloch, and Lee (2000) stated that UG2 consists predominantly of chromite (60 to 90 vol.%), with lesser silicate minerals (5 to 30% pyroxene and 1 to 10% plagioclase), while other minerals are present in minor concentrations.

The colour of plagioclase (pl) ranges from white to colorless, cream, gray, yellow, orange, and pink; chromite (chr) is mostly black, while pyroxene (px) ranges from dark green to black. Plagioclase is a silicate of sodium, calcium, and aluminium; pyroxene is a silicate of magnesium and iron; while chromite is an oxide of chromium, iron, and magnesium (Schouwstra, Kinloch, and Lee, 2000).

The major chromitite minerals are indicated in Figures 8 in red. Figures 8(a) and (b) show the micrographs of chromitite before heat treatment and when cooled to ambient temperature after heat treatment at 50°C. The physical changes observed (red circles) are in the chromite-rich areas. In Figure 8(b), heat treatment exposed yellow-looking minerals (plagioclase, rutile or serpentine) and whitish minerals (quartz, talc, or plagioclase) – the upper two circles. The lower circle shows the disappearance of a whitish mineral, which may be talc or plagioclase, that flaked off from its original position due to heating and cooling of the specimen.

In Figure 9 (a) and (b), it can be seen that heat treatment at 100°C does not cause significant physical change in the specimen. However, some spots of a white mineral appeared after the heat treatment (red circles in Figure 9). A major difference is observed after heat treatment at 140°C (Figure 10). Some of the dark minerals, such as chromite and pyroxene, were displaced by quartz or plagioclase.

In general, the heating and cooling of rocks is seen to cause changes in the physical characteristics of the minerals.

Result of scanning electron microscopy

The SEM was used to examine the effects of heat treatment on crack extension/widening. The changes in the elemental composition of the rock samples were also determined using energy-dispersive X-ray spectrometry (EDX) equipment (known as INCA) attached to the SEM. Figures 11 and 12 show the SEM micrographs of chromitite before and after heat treatment and repeated heating and cooling.

Of all the samples heat-treated, chromitite was the most affected by the heat treatment, followed by pyroxenite. Only few cracks were observed in mottled anorthosite, leuconorite,

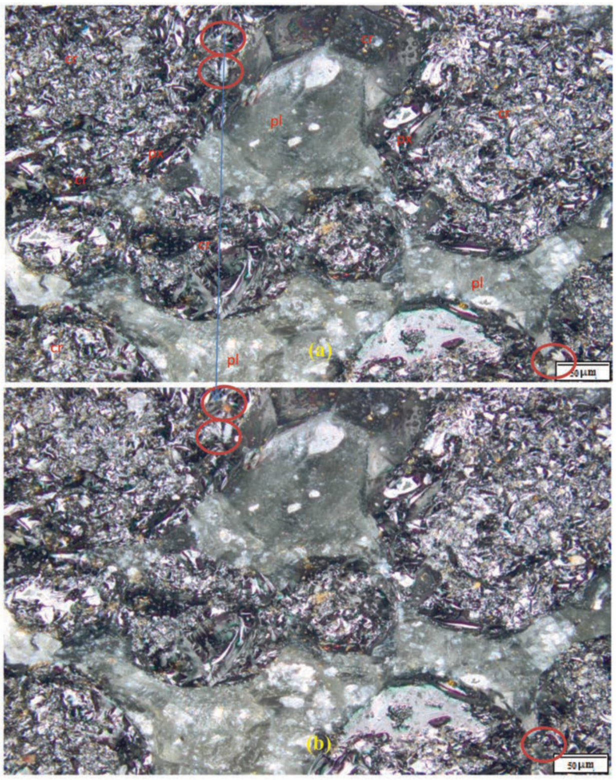


Figure 8—Micrographs of chromitite (a) before and (b) after heat treatment at 50°C

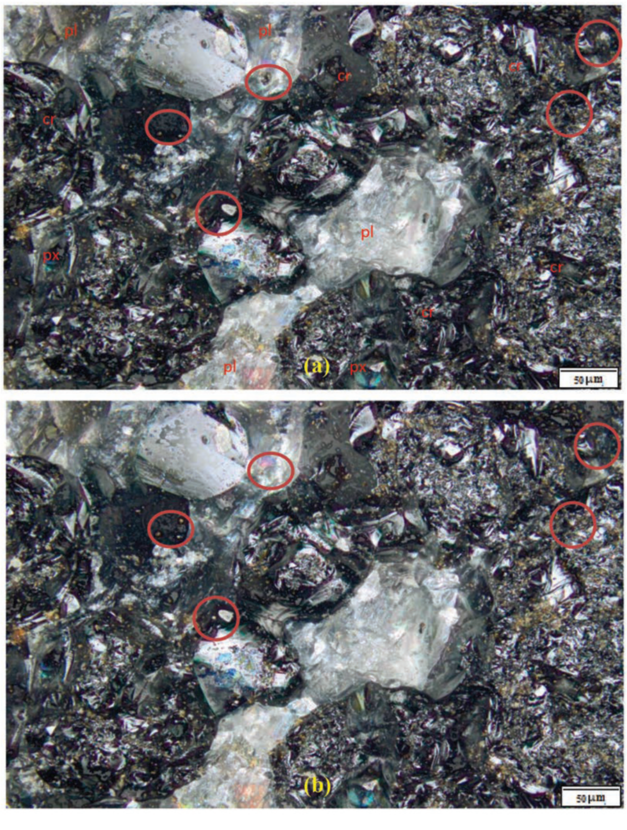


Figure 9—Micrograph of chromitite (a) before and (b) after heat treatment at 100°C

Table I

Average modal mineral abundance of UG1 and UG2 seams, vol.% (Voordouw, Gutzmer, and Beukes, 2009)

Lithology	pl	px	cr	Other minerals	Total
UG1 chromitite	31	15	53	1	100
UG2 chromitite	10	8	62	19	99

norite, gabbronorite, granofels, and varitextured anorthosite. No visible cracks were observed in granite (Figure 13) when heated from ambient temperature through 140°C, but when subjected to repeated heating and cooling, a clearly visible macro-crack extending from the lower to the upper end of the sample was observed, as shown in Figure 14.

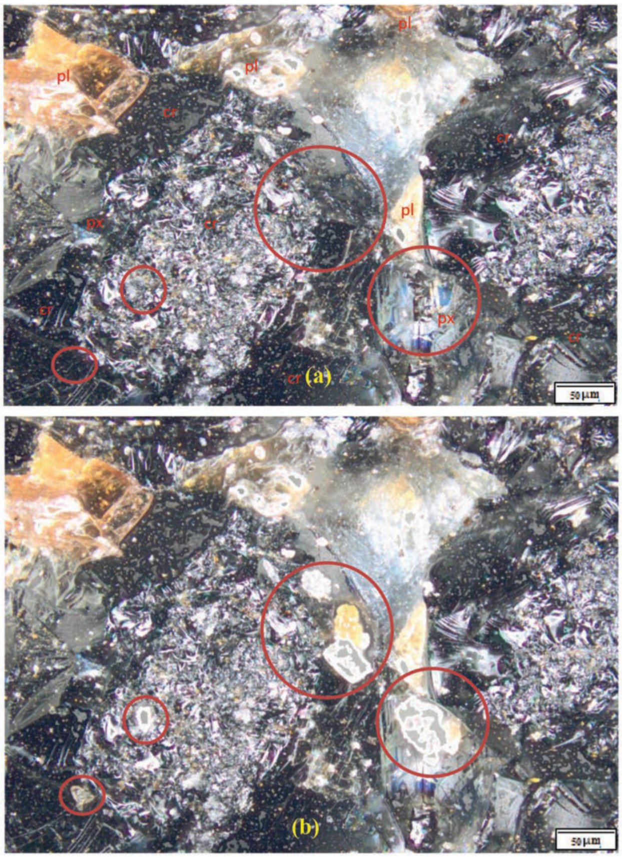


Figure 10—Micrographs of chromitite (a) before and (b) after heat treatment at 140°C

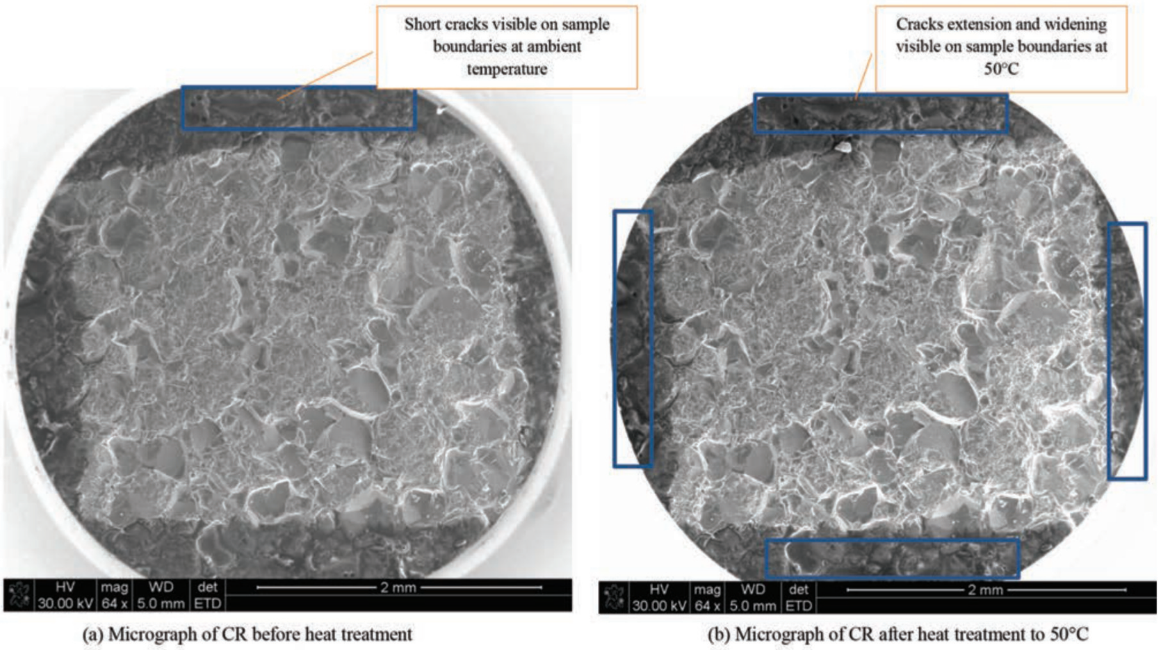
Result of EDX

The elemental composition of all the samples were determine for each temperature range, that is at ambient, 50°C, 100°C, and 140°C. The results of the chemical characterization of the samples are presented in in Figure 15 and Table II.

An element map is an image showing the spatial distribution of elements in a sample. In Figure 15, the letters ‘K’ and ‘L’ following the chemical symbols refer to the first and second electron shells respectively, while ’a’ represents alpha X-rays. The bright portions in the maps indicate the dominance of elements in the group, and the maps that look similar indicate a strong correlation between the elements. It should be noted that in Table II and Figure 15, oxygen is the most abundant element. The reason is that oxygen reacts with almost all the other elements to form silicates and oxides. A careful examination of all the elemental compositions in Table II shows that there is not much difference in the mineral compositions of the specimen when heat treated from ambient temperature up to 140ºC. However, temperature caused some elements to be displaced while others resurface. For example, the element maps and Table II reveal that zinc appears only in ambient conditions. The sudden increase in the values for gold and palladium in the sample that underwent repeated heating and cooling results from sputter coating.

Conclusion

The results of the optical microscope analyses show that physical changes take place in the rocks subjected to heat treatment. However, the observed changes are not so significant as to affect the physical behaviour of the rocks.



Figures 11—SEM micrographs of chromitite (a) before and (b) after heat treatment at 50°C

Microscopic analyses of Bushveld Complex rocks under the influence of high temperatures

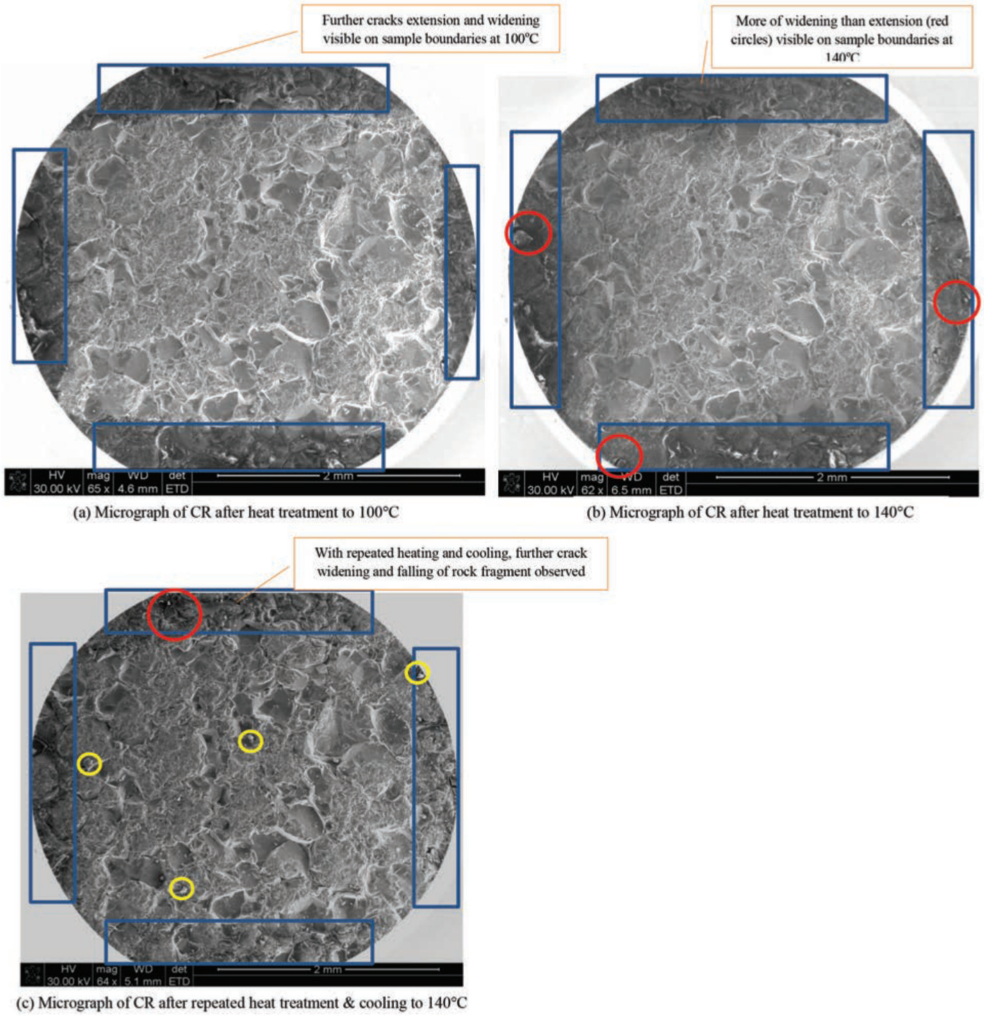


Figure 12—SEM micrographs of chromitite after heat treatment at (a) 140°C, (b) 140°C, and (c) 140°C after repeated heating and cooling

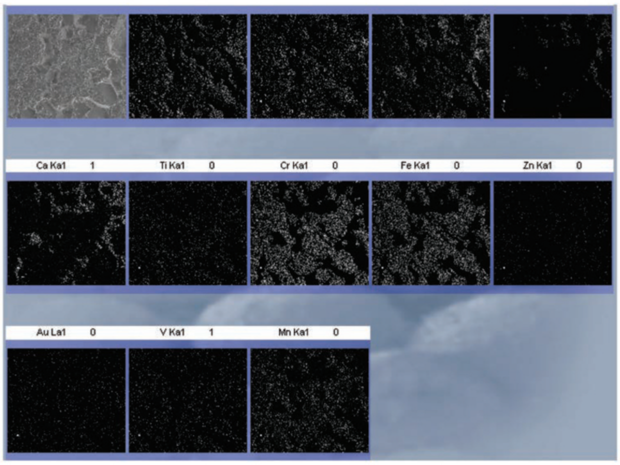


Figure 15—Element map for chromitite

Table II

Summary of elemental composition (% weight) of chromitite for all temperatures

	Ambient	50°C	100°C	140°C	Repeated heating/ cooling
Mg	4.47	4.50	4.46	4.62	4.20
Al	10.70	10.86	10.59	10.76	10.22
Si	6.27	6.45	6.54	6.11	6.48
Ca	1.72	1.73	1.77	1.64	1.80
Ti	0.39	0.37	0.40	0.41	0.36
V	0.25	0.24	0.23	0.24	0.25
Cr	23.02	22.24	22.32	22.79	21.36
Fe	14.16	13.71	13.84	14.10	13.27
Au	2.66	2.53	2.49	2.55	4.60
O	35.89	36.04	35.99	35.88	35.19
Na	0.00	0.90	0.96	0.90	0.90
Mn	0.41	0.44	0.41	0.00	0.39
Zn	0.05	0.00	0.00	0.00	0.00
Pd	0.00	0.00	0.00	0.00	0.98

The scanning electron microscope images revealed that crack initiation starts at lower temperature and the cracks extend with increasing temperature. This is an indication that temperature would influence the texture and strength of rocks in underground mines with high geothermal gradients.

The chemical analyses of the samples subjected to different temperatures show that the temperature range used in this research is not high enough to induce noteworthy chemical changes in the samples.

Appendix I

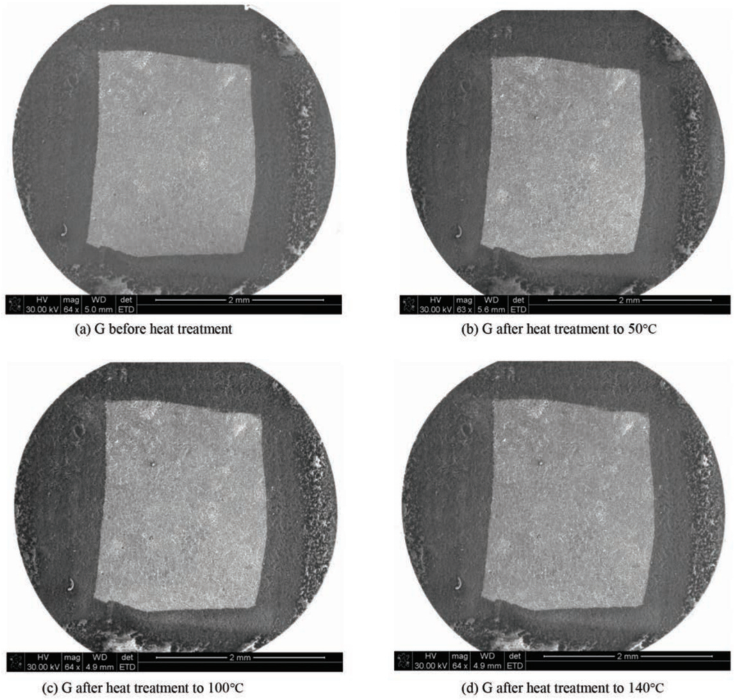


Figure 13—Micrographs of granite (a) before and after heat treatment at (b) 50°C, (c) 100°C, (d) 140°C

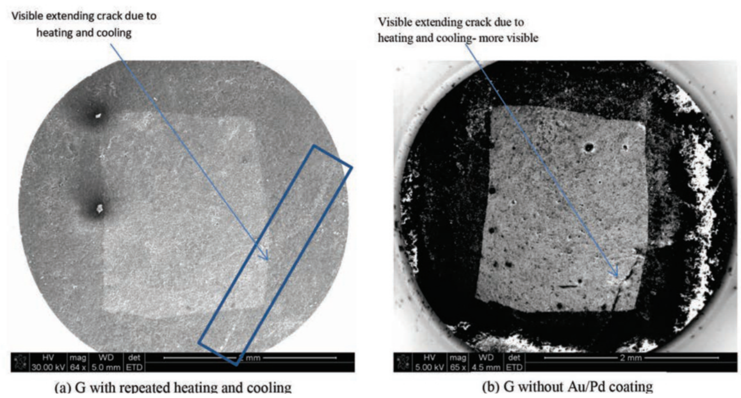


Figure 14—Heat treatment of granite subjected to repeated heating and cooling: (a) sputter-coated, (b) not sputter-coated

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The effect of geological uncertainty on achieving short-term targets: a quantitative approach using stochastic process simulation

by M. Soleymani Shishvan* and J. Benndorf*

Synopsis

Continuous mining systems containing multiple excavators producing multiple products of raw materials are highly complex, exhibiting strong interdependency between constituents. Furthermore, random variables govern the system, which causes uncertainty in the supply of raw materials: uncertainty in knowledge about the reserve, the quantity demanded by the customers, and the breakdown of equipment. This paper presents a stochastic-based mine process simulator capturing different sources of uncertainties. It aims to quantify the effect of geological uncertainty and its impacts on the ability to deliver contractually defined quantities and qualities of coal, and on the system efficiency in terms of utilization of major equipment. Two different areas of research are combined: geostatistical simulation for capturing geological uncertainty, and stochastic process simulation to predict the performance and reliability of a large continuous mining system.

The process of modelling and simulation in this specific production environment is discussed in detail. Problem specification and a new integrated simulation approach are presented. A case study in a large coal mine is used to demonstrate the impacts and evaluate the results in terms of reaching optimal production control decisions to increase average equipment utilization and control coal quality and quantity. The new approach is expected to lead to more robust decisions, improved efficiencies, and better coal quality management.

Keywords

continuous mining, scheduling, stochastic process simulation, geological uncertainty.

Introduction

Continuous coal mining systems containing multiple excavators producing multiple products of raw materials are highly complex and exhibit strong interdependencies between constituents. A network of conveyor belts is used for transportation of the extracted materials to different waste dumps or the coal stockpile.

Optimal decision-making in short-term planning and production control are impacted mainly by geological uncertainty associated with incomplete knowledge of the coal deposit represented in the reserve block model. These uncertainties cause deviations from expected process performance. For more robust decision-making, understanding the impacts of these stochastic elements plays a key role.

Techniques of stochastic process simulation, whether discrete, continuous, or combined (Kelton and Law 2000), provide a powerful tool for measuring the performance indicators of a complex system. In the past few years there has been significant development in the applications of process simulation in the mining industry. Panagiotou (1983) described the application of the simulation program SIMPTOL for opencast lignite mines that use bucket wheel excavators (BWEs), conveyors, and stackers. The main objective was to select and match the equipment to fit material characteristics while meeting production requirements and mine profiles. Michalakopoulos *et al.* (2005) presented a simulation model of an excavation system at a multi-level terrace mine using the GPSS/H simulation language. The principal model output variables are production and arrival rate of product and waste at the transfer point. Michalakopoulos *et al.* (2015) utilized Arena simulation software for the simulation of the Kardia Field mine in Greece. Validation of the results illustrates an acceptable agreement with the actual data. Fioroni *et al.* (2007) used discrete tools for simulation of continuous behaviour for modelling the conveyor belt network of a large steelmaking company. The authors proposed a modelling approach to the flow process that uses portions of materials and treats them as discrete entities in simulation modelling. The results demonstrated that this technique was valid and successful. Salama *et al.* (2014) used a combination of discrete event simulation and mixed integer programming (MIP) as a tool to improve decision-making in underground mining. The proposed method uses the simulation approach to evaluate the operating costs of different haulage system scenarios and obtains the cash flows for input into the MIP model.

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The effect of geological uncertainty on achieving short-term targets

The reviewed literature demonstrates that the stochastic process simulation is a potent method for measuring the key performance indicators (KPIs) in continuous mining systems. However, the investigation of the impacts of geological uncertainty in the performance of continuous mining systems is still seen as a major gap. This paper presents a stochastic-based mine process simulator focusing on the effects of geological uncertainty to predict the mine process performance and reliability. The following section specifies the problem of stochastic simulation of continuous mining systems. Thereafter, the procedure for a new integrated simulation approach is discussed. In this study, a discrete-continuous methodology is proposed and Arena simulation software (Rockwell Automation Technologies Inc. 2012) is used for modelling. As a case study, a completely known data-set is analysed and the results presented.

Problem description

In general, continuous coal mining systems contain parallel production lines that begin with excavators, followed by material transport by conveyor belts and distribution at the mass distribution centre where material is divided into two destinations, namely, the coal bunker and the waste materials dump. Waste materials are dumped by spreaders at the dump while lignite is stacked by the stacker in the stockpile yard. The reclaimer and a network of conveyor belts are used for loading lignite into railway wagons. Finally, the product is despatched to customers (mostly power plants) based on their daily demands.

The problem considered here is to quantify the effect of geological uncertainty and its impact on the ability to deliver contractually defined coal quantities and qualities, and on the system efficiency in terms of utilization of major equipment. The KPIs of the system are defined as the ability to meet coal quality and quantity targets and the utilization of the system. The parameters of the KPIs to be evaluated for each simulation replication are presented in Table I. More details about the mathematical formulation of evaluation function and parameters are discussed in Shishvan and Benndorf (2014).

As an example, Figure 1 shows a scatter plot that illustrates the relationship between two ash contents measured on the same samples. It appears from this figure that as the variable on the vertical scale (ash content measured in the laboratory) changes, the variable on the horizontal scale (ash content based on the estimated model) seems to vary randomly within a relatively small range without tending to increase or decrease significantly. It can be seen that the reality (laboratory measurements) shows a significant higher fluctuation compared to the estimated model. There is a weak relationship between two variables, with a correlation coefficient of 0.182 in the scatter plot. These observations give rise the question 'where do these fluctuations come from?' which is investigated in this paper.

Integrated simulation approach

When using interpolated reserve models as a basis for mine planning, a smoothing effect is to be expected. This is due to the nature of spatial interpolators, which are often designed to minimize the estimation errors. Alternatively, conditional simulation methods in geostatistics have been developed to quantify variability and uncertainty associated with the geology (Chiles and Delfiner, 2012). These techniques result in a set of equally probable scenarios defining the spatial distribution of attributes within a deposit (realizations), which capture in-situ variability as found in the data. Local differences between these realizations can be used for mapping uncertainty. The applicability to multi-seam coal deposits was demonstrated by Benndorf (2013a, 2013b).

On the other hand, the process simulation model of the continuous mining system is intended to reproduce the operational behaviour in a real opencast coal mine. The extraction and conveying of lignite and waste are emulated in a combined discrete-continuous stochastic environment. This allows incorporation of uncertainty associated with the geological block model. It allows the re-creation of the

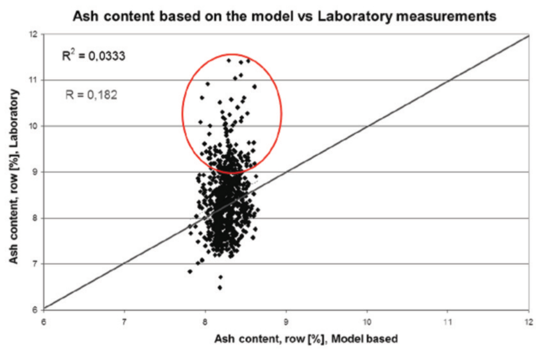

Figure 1—Scatter plot of ash content of product delivered to the power plants

Table I

The parameters of evaluation function of the KPIs

Objective	Evaluation of aggregated KPIs of each simulation replication based on the pre-defined short-term planning targets
KPIs	• J_1 - coal quality: should be between defined lower and upper limits, otherwise penalties should be applied • J_2 - coal quantity: should be between defined lower and upper limits, otherwise penalties should be applied • J_3 - utilization: average utilization of the system can be derived from the average utilization of excavators
Decision variables	• Task schedules: different alternatives for short-term plans (daily/weekly/monthly) • Extraction sequences: sequence of extracting mining blocks for each excavator • Extraction rate of excavators in the different time spans • Stockpile management: the quality and the quantity of coal needed in the stock yard influences other decision variables
Constraints	• Each block can be mined just once • The conveyor belt can be moved further along only if all the blocks in one pass are mined out

deterministic and/or random occurrences of events such as operating stoppages caused by unavailability of spreaders or conveyor belts, equipment failures, and preventive and corrective maintenance activities.

Dowd and Dare-Bryan (2005) explored the general concepts of the integration of the geostatistical simulation within the entire design and production cycle. The authors illustrated these concepts with particular reference to blast modelling. This paper aims to combine the two simulation concepts, namely geostatistical simulation for capturing geological uncertainty and stochastic process simulation, to predict the performance and reliability of a large continuous mining system. Figure 2 shows the integrated simulation approach.

In this approach, realizations based on conditional simulation and an interpolated model using kriging are considered as input for the mine process simulator. The kriged model is used for comparison. The software selected to implement the integrated simulation approach is Rockwell ARENA 14.5, which permits close reproduction of the behaviour of complex real systems with complicated decision logic. The software offers intuitive flowcharting support to the modelling, control over the flow of entities in the system, custom statistics, user-defined expressions, and interfacing with external databases and spreadsheets (Kelton and Law 2000). The output of the simulator is the set of values for each KPI. At this stage, penalties are applied when deviating from production targets. The KPIs are summarized in an evaluation function, which results in a probability distribution when multiple replications are evaluated (Figure 2).

Case study and implementation aspects

System description

The objective of this study is to illustrate the effect of geological uncertainty on the performance of a complex continuous mining system. To analyse the performance of the proposed approach, the case study is presented in a completely known and fully controllable environment. In this regard, the Walker Lake data-set as a completely known environment is chosen (Isaaks and Srivastava 1989). The real value block model (complete Walker Lake data-set), an average-type estimated block model using ordinary kriging, and 20 conditionally simulated realizations using sequential Gaussian simulation of the deposit are used as different replications for building the simulation experiments.

Figure 3 schematically shows a typical opencast mine. The mining operation uses six BWEs at six benches. Extracted material is transported by conveyor belts to the mass distribution centre. Here, destinations are determined based on the type of materials and the excavator from which they were derived. Finally, waste materials are conveyed to spreader no. 1 or 2 and the coal to the stockpile yard.

The block model is divided into six equal areas and each area is assigned to one excavator (Figure 4).

The major steps in simulation modelling are as follows:

➤ The first step is to define appropriate entities. Entities are the block portions to be extracted in each period
➤ The second step is to assign block attributes. As an entity enters the system its attributes, consisting of block coordinates (x, y, and z), block tonnage, block

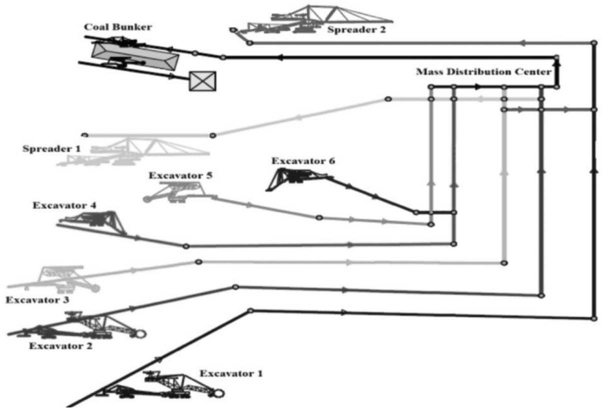

Figure 3—Schematic view of the problem (opencast mine)

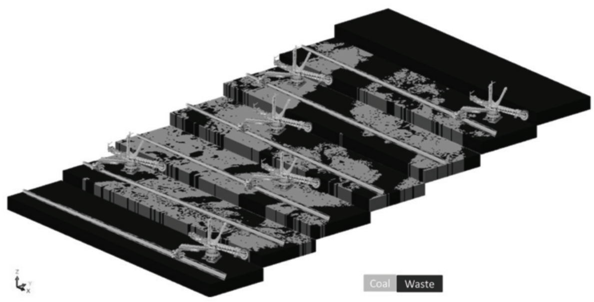

Figure 4—Block model, assigned area for the excavators

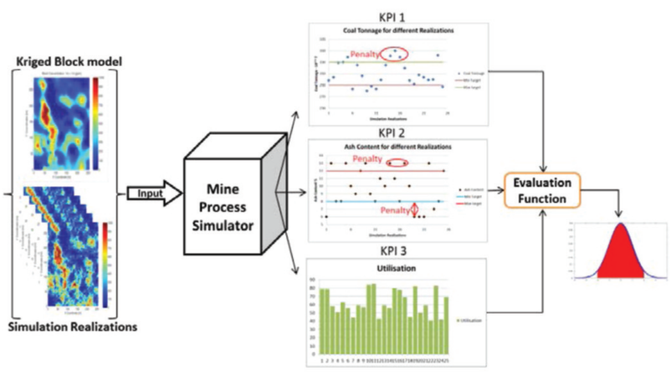

Figure 2—Integrated simulation approach

type, quality parameters, and destination, are assigned. These attributes are read from the geological block model
- Subsequently, the entity is placed in a queue for extraction by the excavator as a resource module
- Each entity has a delay based on operating time, after which it is released
- At the final step, variables such as total waste tonnage and ore tonnage entering the system are calculated.

A capacity constraint is implemented to prevent overflow of loose material on the conveyor belt that is connected to the coal bunker. Based on the maximum amount of coal that can be on the belt, a constraint of 6000 m3/h is considered for coal. When the production rate exceeds these limits, the model starts to identify the excavators that are producing coal. The excavator that corresponds to the minimum production rate is set to standby.

Decision variables of this case consist of:

- *Task schedule*—working schedule for a time horizon of 7 days is given in Table II. This mine operates 24 hours per day in three working shifts. As an example, in Table II, the number 110 shows that the corresponding equipment is available for first and second shifts and is not available for the third shift
- *Extraction sequence*—in this case, considered to be a constant (from one side of bench to the other side) without any movement of excavators during the excavation
- *Extraction rate of excavators*—in this case, assumed to be equal to the theoretical capacity of excavators (Table III)
- *Stockpile management*—in this case, if the stockpile for a specific coal type is full, the excavator(s) that produces that type of coal should be idled until stockpile space is available.

Tables III summarize the general information and technical parameters that are used for the simulation model building.

Results and discussion

The results of 22 different block models (simulation replications, *r*): real, estimated, and 20 realizations are analysed in the specified time horizon (one week, in this case, *i.e.*, 168 hours). The total extracted coal tonnages for the different scenarios are presented in Figure 5. The system simulation based on the estimated model shows significantly less coal (10%) than the average of the 20 realizations. On the other hand, the real model shows a very similar value to the average value of the 20 realizations. Clearly, the estimated model underestimates coal production for the defined schedule. This is due mainly to ignoring *in situ* variability and geological uncertainty. The capability of conditional simulation to quantify geological uncertainty improves the prediction of system performance.

Note that the application of average-type estimated models does not always lead to underestimation. Depending on local geological conditions, these techniques may also lead to an overestimation.

Figure 6 presents the average ash content of extracted coal for the real model (light grey), the estimated model (black), the realizations (dark grey), and the average of simulation (red line). The ash contents of all realizations substantially exceed the value predicted by the estimated deposit model. The average is again similar to the real value. Relying on the estimated model would indicate a biased and over-optimistic ash content.

In this study three KPIs are measured and penalties are applied for not meeting the coal quality target (Figure 7a), quantity targets (Figure 7b), and equipment utilization target (Figure 8a). The values in Figure 7 are calculated based on parameters specified in Table IV. The costs of deviation from the targets (the penalties) in this study are one unit for one ton of coal. Hence, these penalties can be interpreted as percentage and tonnage of deviation from the targets.

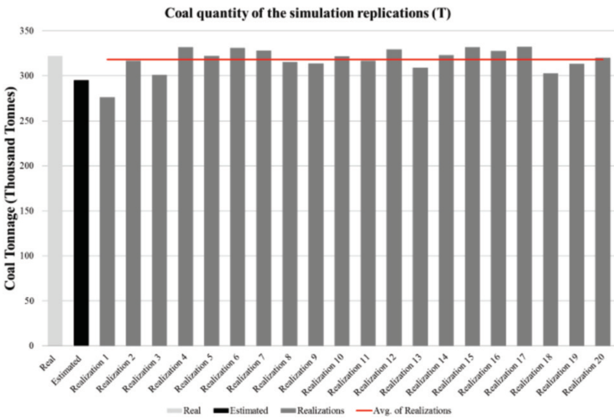

Figure 5—Illustrative results of different scenarios, coal tonnage

Table II

Working schedule of the equipment as a decision variable

Type of equipment	Days						
	1	2	3	4	5	6	7
Excavator 1	111	001	111	111	111	111	111
Excavator 2	111	111	111	001	111	111	111
Excavator 3	111	011	111	001	111	111	111
Excavator 4	111	111	111	111	111	111	111
Excavator 5	111	111	001	111	111	111	111
Excavator 6	111	110	110	111	111	111	111
Spreader 1	111	111	111	001	111	111	111
Spreader 2	111	001	111	111	111	111	111
Conveyor belts	111	111	111	111	111	111	111

Table III

General information on equipment

Type	Theoretical capacity (loose, m3/h)	Scheduled time $T_{scheduled}$ (h)
Excavator 1	4 900	152
Excavator 2	4 900	152
Excavator 3	3 770	144
Excavator 4	1 400	168
Excavator 5	3 770	152
Excavator 6	740	152
Spreader 1	10 000	152
Spreader 2	10 000	152
Conveyor belts	6 000	168

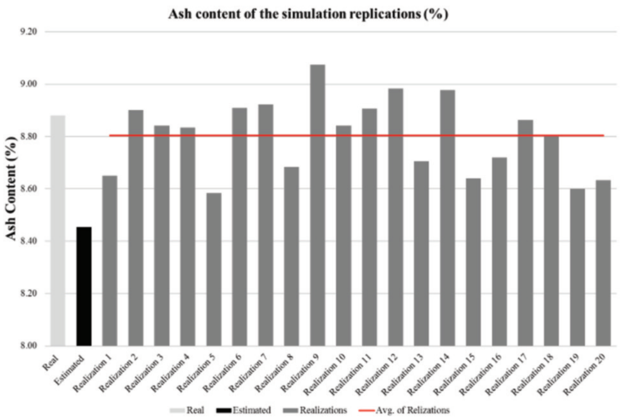

Figure 6—Illustrative results of different scenarios, ash content

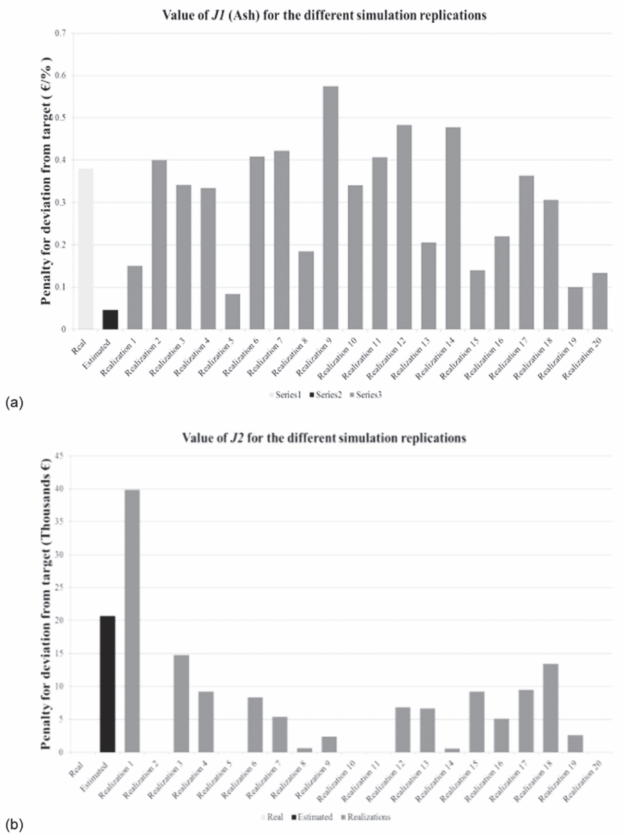

Figure 7—(a) Coal quality KPI, (b) coal quantity KPI

Figure 7b also indicates that, for example, realization 1 will lead to an underproduction of 40 kt of coal. To account for this uncertainty the stockpile inventory should be at least 40 kt before the start of the week to accommodate potential deviations from target and secure a sufficient supply to customers. Figure 8 demonstrates the average utilization of the system for different realizations and shows box plots of the utilization of each excavator. Evidently, geological uncertainty and variability have a significant impact on the measured KPIs.

Figure 9 shows the ash content of a week's production to be delivered by rail to the power plants. The results reveal that predictions based on the estimated model (black line) and the reality (dark grey line) are not well correlated. This means that the prediction based on an interpolated model has limits. When considering the conditional simulation model, there are 20 realizations (light grey cloud) and the average of the realizations (red line), which is the stochastic prediction. Comparing these with the reality, the red line generally follows the true ash content very well. Deviations are in the expected range of deviations, which are mapped by the shadow range (realization cloud).

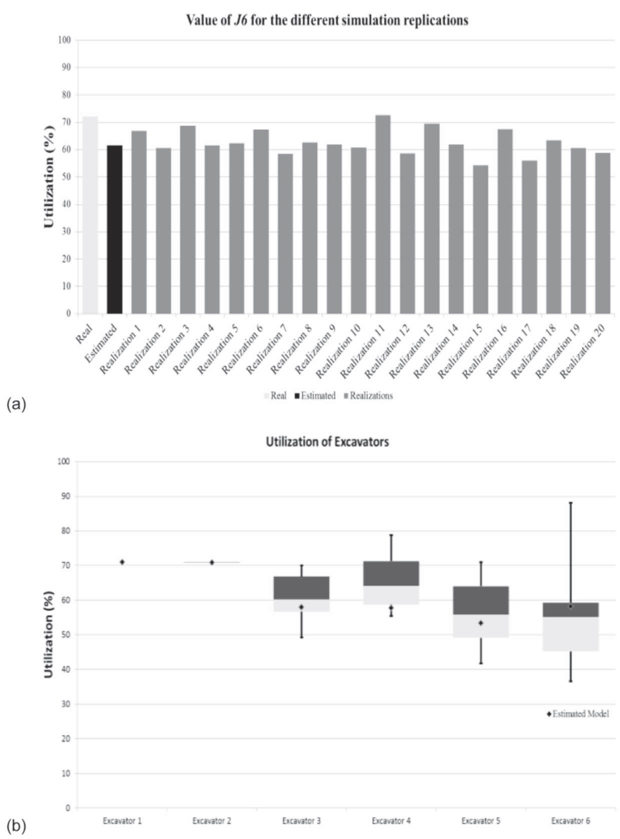

Figure 8—(a) Utilization KPI, (b) box plots of utilization of excavators

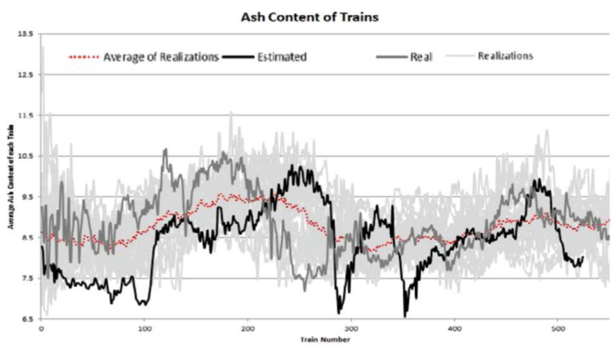

Figure 9—Average ash content in a week's production

Previous examples illustrated that stochastic system simulation is a valid and powerful tool for exploring the effect of geological uncertainty on the expected performance of complex continuous mining systems. It provides the mine planning engineer with a valuable tool to foresee critical situations affecting the continuous supply of raw material to customers and the system performance.

Conclusions

Continuous mining systems require large investments and have high operational costs. Decisions in daily scheduling are impacted by uncertainties such as incomplete knowledge about the deposit, which can have a significant impact of actual production performance. This contribution has proposed a simulation-based framework where the method of geostatistical simulation has been integrated with mine system simulation to account for the effects of geological uncertainty. Results show that such an approach provides the mine planning engineer with a valuable tool to foresee critical situations affecting the continuous supply of raw material to customers and the system performance.

Future research will be carried out to extend the system simulation to also to capture stochastic downtime behaviours and stochastic demand. For optimal control decisions, the simulation approach will be integrated to a simulation-based framework for optimization of short-term mine planning and operations control (Benndorf, 2014; Benndorf *et al.*, 2014).

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The importance of people in the process of converting a narrow tabular hard-rock mine to mechanization

by D. Vogt* and T. Hattingh*

Synopsis

This paper argues that the technology change to mechanization is also going to require a change in people. It presents a model for technological progress and adapts it to the mining industry. It then goes on to motivate the need for mines to become learning organizations in order to achieve maximum value from their people as they become less labour-intensive. In an important sense, mechanization is as much about knowledge as it is about technology.

The change when a mine introduces mechanization or a level of automation is not simply one of technology but also a stage in the development of mining from 'art' to 'science'. 'Art' describes a state of technology characterized by tacit knowledge, an understanding that comes only from experience, and has no formal procedures and little structure. In contrast, 'science' represents a state of technology where all the component processes are understood in detail, all knowledge is explicit, and processes and structures are formal.

Studies of other industries, including metal part manufacture and aviation, show that each stage in the progression from art to science changes the nature of the organization and requires a different mix of skills from the workforce. Perhaps the most important change is the increase in knowledge and decision-making required as the technology gets closer to science.

There is anecdotal evidence that South African underground hard-rock mines are not learning organizations. Mines by their nature are capital-intensive with long lead times from investment to returns, so there is reluctance to change from the original plans or to encourage staff to think independently. Particularly as mines move from art to science, this reluctance must be overcome.

The important lesson for engineers involved in introducing mechanization is to understand that success or failure will be determined by the people involved, and not solely by the technology. The process will almost certainly require a culture change on the mine. We recommend that the mechanization team includes an expert in human and organizational behaviour to ensure that a receptive workforce and management are in place to accept the new technology when it arrives..

Keywords
levels of technology, mechanization, underground mining, productivity, change management.

Introduction

All mineral commodity prices are currently at historically low levels. The South African mining industry is under pressure to reduce costs in an attempt to become profitable again. All the chief executives of the major narrow-stope, hard-rock mining companies in South Africa announced their intent to mechanize more widely during the first quarter of 2014 and it is now accepted that mining companies in this field will not open new shafts that are not mechanized.

The underground South African coal industry mechanized successfully during the 1970s–1990s from predominantly drill-and-blast operations to continuous miners and longwalls. There are now very few drill-and-blast sections remaining in the country. Surface mining in South Africa has also followed international best practice, and is highly mechanized.

However, the low stoping heights, modest grades, and difficult dips have slowed the introduction of mechanization in gold and platinum mines, the focus of this paper. Attempts to mechanize in South Africa have met with mixed success. Several gold mines proposed or adopted some level of mechanization in the late 1980s (Association of Mine Managers, 1988), including Joel, Vaal Reefs South, Randfontein Estates, Western Areas, and Freddies. None of those early efforts in gold are today producing by primarily using mechanized methods. Later mechanization efforts concentrated on thicker orebodies at mines like Target and South Deep, and have been more successful.

Mechanization has fared somewhat better in platinum. The lower dip allows the use of rubber-tyred vehicles, and mines have been successful in mechanizing both using mechanized bord-and-pillar and extra-low-profile methods and by hybrid mining – a combination of conventional layout with machines (Nong, 2010).

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Anecdotally, many mechanization efforts fit the model of technology implementation in factories developed by Chew *et al.* (1991) and illustrated in Figure 1. Planning for the implementation of a new process allows for a delay after the process starts up before the expected improved performance level is reached. In reality, performance actually starts to deteriorate before the change is made, as the factory prepares for the change. Anywhere from weeks to years later, the actual performance returns to its previous level and, with luck, exceeds it. In fact, in many cases, the costs involved in the adjustment period cause the factory to stop the implementation (Chew *et al.*, 1991). The same has happened in many underground narrow-reef hard-rock mines that have attempted to implement mechanization in South Africa.

Chew *et al.* define the challenge as one of knowledge. If managers knew exactly what problems were going to occur they could anticipate them in advance, but there is a lack of knowledge of the new process, because the experience of current employees is not relevant – this is particularly true when mechanizing a mine. Chew *et al.* also suggest that managers and engineers do not know in detail what is going on, and that the scientific method is not employed in most manufacturing plants. We argue that the same is true of mining: there are not enough experiments and those that do occur are often not recorded.

In this paper, an existing model for documenting the transition from art- or craft-based activities to those based on science is presented and a variation of the model is proposed for mining. The model highlights how the workplace changes as mining becomes more automated, and therefore how people need to change. It is suggested that South African mines are not learning organizations, and that to succeed in mechanizing, they need to introduce a culture of learning.

Technology development from 'art' to 'science'

In this section, a general model is described to characterize the level of technology used in an industry. Descriptions of two industries in terms of the model provide the basis for applying the model to mining.

Bohn (2005) describes how technologies develop by invoking the metaphor of transforming from art to science. The metaphor is long established and widely used, for example in the title of an 1802 book on surveying, '*Art without Science, or, The Art of Surveying Unshackled with the Terms and Science of Mathematics, Designed for Farmers' boys*' (Sedger, 1802). Today, art and science are seen as ends of a spectrum: 'art' implies a measure of craftsmanship and tacit knowledge that has been passed down from master to apprentice over generations. Each 'artwork' is individual. 'Science' implies a process that uses explicit mathematics and physics to produce predictable, repeatable products.

The maturity of any technology can be described along a number of dimensions. Bohn chose three: how work is done; the quality of results achieved; and how well the technology is understood (Bohn, 2005), tabulated in Table I.

Bohn applied his description of moving from art to science in an analysis of the development of flying (Bohn, 2010). He described how flying started out as a *craft*, with pilots flying using knowledge laboriously gained, and passed from person to person. By about the 1930s, aircraft had become sufficiently complex that *rules and instruments* started to be used: knowledge started to become codified. In the 1940s, aircraft became so complex that a level of standardization was required to lower pilot workload, and *standard procedures* were implemented.

As the science of flying became better understood, it became possible to *automate* aspects of flying. Today, automated systems are connected together to operate the aircraft, with supervision by the pilot in *computer-integrated flight*. The key point is that at each stage, the role and nature of the work of the pilot changed until the pilot eventually became a 'manager' of the process of flying.

There is a parallel with the development of the technology of mining: it started as a craft, become subject to rules, and is currently deeply invested in standard operating procedures. As mining becomes more mechanized, the role of the miner becomes similar to that of a modern pilot: managing the process of mining.

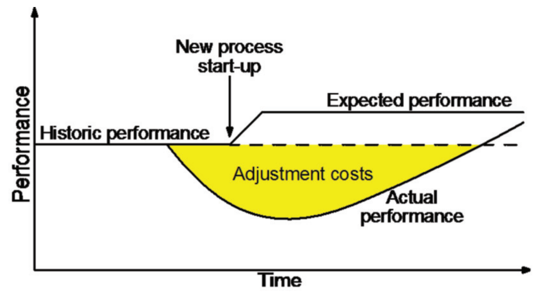

Figure 1—Murphy's Curve. While the details of the curve change, the shape is remarkably consistent (Chew *et al.*, 1991)

Table I

Characterizing technology on the spectrum from 'art' to 'science' (Bohn, 2005)

	Embryonic technology: 'art'	Ideal technology: 'science'
How activities are executed	Zero procedure; idiosyncratic	Fully specified procedure
What results are achieved	Each one different, mostly poor	Consistent and excellent
Characteristics of knowledge		
How knowledge is specified	Tacit	Codified
What is knowledge about	Purely know-how	Also know-why
Extent of knowledge	Minimal, can distinguish good from bad results, but little more	Complete

Bohn's dimensions of art and technology in Table I describe some measures that can be used to qualitatively investigate the maturity of the technology in a particular mine or mining method, but they leave much to be interpreted. It is also difficult to draw a parallel with flying, given the small number of pilots on aircraft and the large number of miners in a mine. A more explicit, though not direct comparison for mining can be made by comparing it with a well-characterized existing process. In the next section, a study of metal parts manufacturing is summarized as a framework to suggest an analogy for the mining industry.

A case study of technology development: gun manufacture

Beretta manufactures guns in northern Italy. It has operated since 1492, has a continuous record of its activities dating back to 1526, has been owned by the same family for 16 generations, and has survived six revolutionary transformations in metal part manufacturing. Beretta is an excellent case study in manufacturing technology because its product, guns, has remained constant for more than five hundred years, while the process of manufacturing the product has changed dramatically (Jaikumar, 2005).

Jaikumar's study (2005) is summarized here for the insights it can offer into the similar technological change occurring in mining. His paper is well worth reading for its complete description of many of the aspects of technology change and manufacturing, only some of which are summarized here.

The Craft System

When Beretta was established, guns were manufactured using the Craft System. Craftsmen made each part of the lock of a gun using a forge and files, and by continuously comparing the part being made with a reference model kept in the workshop. Each part was then adapted to fit with the parts that had already been manufactured, until the lock was complete. The people in the lock shop were product-focused, with everyone involved in all processes. The degree of skill of the worker had a huge effect on productivity, with the most skilled workers being up to four times more productive than the least skilled (Jaikumar, 2005). A master managed a workshop of about eight people and passed his craft on to journeymen, who in turn aspired to become masters through a process of learning. Everyone in the shop was skilled and adaptable. Adaptability was required because of the inability of the process to create accurate, precise, and repeatable parts (Jaikumar, 2005).

The English System

In about 1810, Beretta introduced the English System. From about the 1790s, machine tools were being introduced in Britain. Two innovations allowed the English System to become dominant: the machine lathe and the micrometer (Jaikumar, 2005). These two tools allowed gun part manufacture to become less labour-intensive and, for the first time, they allowed for parts to be made according to an engineering drawing. The master in the workshop was no longer required to provide guidance or approval: drawings gave guidance and a micrometer ensured that a part complied with its specification. Workers were no longer product-specific, but became experts on particular tools.

In this paper, productivity is taken to be the number of guns made per employee and the minimum effective scale is the smallest group that can economically produce a particular set of parts.

The English System improved productivity over the Craft System about fourfold, but at the same time, it increased the minimum scale of a workgroup from eight craftsmen to about 40 people using three machines. In the English System, all workers were still directly involved in making products (Jaikumar, 2005) and there was a formal system of learning, though now of trades rather than components.

The American System

The availability of skilled operators became a constraint on growth. The American System was designed to overcome this limit. It was introduced at Beretta in 1860 through the purchase of a factory from Pratt and Whitney (Jaikumar, 2005). The American System was developed to provide precise and interchangeable parts. In the English System, parts were made to fit with one another as closely as possible, and were not interchangeable. In the American System, parts could be interchanged easily by being designed for clearance (Jaikumar, 2005). Charles Babbage was the first person to characterize the advance of the American System over the English by using the word 'manufacture' rather than 'make' to describe the process (Balconi, 2002).

In practice, the American System introduced the idea of a factory as a large machine for making a product. The American System divided the manufacture of each part into a number of processes, and assigned each process to one operator using one machine. Each machine was designed only to apply a single operation to the part. Go/no-go gauges determined if a part met specification.

The American System started the shift towards unskilled labour on the production line. Operations were codified so that new operators could be trained with an absolute minimum of effort. It was also during the introduction of the American System that the workforce stopped being 100% devoted to production. Now, there were workers who built, maintained, and improved machines, and others made parts by the thousand. The minimum scale of a factory increased to about 150 people, where 20 of those were not directly working in production. Productivity per person increased by about threefold over the English System (Jaikumar, 2005). It is also likely that the labour cost per gun produced went down due to the lower number of skilled workers.

The Taylor System

If the factory was the machine, the next step in technological advancement was to apply the principles of science to the workers. Frederick Taylor realized that workers were not making maximum use of machines, and productivity could be improved if human activity was measured, analysed, and then controlled in the manner that machine work was controlled in the American System (Jaikumar, 2005).

Taylor broke each job down into its smallest elements, and measured the effort required for each. He also separated activities: for example, a lathe operator might find that his cutting tool needed sharpening. In earlier times, the lathe operator would have sharpened the tool himself. Taylor considered sharpening to be a different activity to cutting and separated the tasks. He measured how long each task took through time study and he standardized the machines to achieve high utilization.

At Beretta, where it was introduced in 1928, the Taylor System resulted in a threefold increase in productivity over the American System, but increased the minimum scale of a factory to 300 people operating 150 machines. Of the workforce, 60 were involved in maintaining the factory and in quality control (Jaikumar, 2005). As in the American System, the bulk of workers were unskilled, with a small group of highly skilled workers who maintained the factory, planned new products, and developed the procedures used by the rest of the staff. The Taylor System represents the lowest point of worker involvement with the whole process as the workers on the line had no concept of the product beyond their own small tasks.

Statistical process control

In 1950, Beretta started producing its own machine tools as a result of a contract to produce a new rifle, the Garand M1, for NATO. The contract specified tolerances an order of magnitude tighter than they had previously achieved, and interchangeability of components had to be 100% (Jaikumar, 2005).

In order to achieve high accuracy and precision, the variability in factory processes had to be accounted for and removed. This was done through the introduction of statistical process control (SPC). While Taylorism is based on the idea of a ‘right’ or ‘best’ way to conduct a machining operation, machines do not perform consistently. The Taylor way will not produce identical parts, for example, as a lathe tool loses its sharpness.

To counter variations in the process, Beretta started visually recording key parameters of parts on charts as they were manufactured. The charts allowed the operator to identify systematic error and compensate for it. Day-to-day management now focused on the process, rather than the products (Jaikumar, 2005). At the same time, automation reduced the labour required for manufacture, which freed operator time to monitor processes and identify and solve problems.

Under SPC, quality control became an integral part of manufacture. The staff function increased from 60 to 100 people, while people with a line function decreased from 240 to 200. Overall, the minimum scale of 300 people and 150 machines remained.

SPC reduced rework from 25% to 8%, but productivity increased only by between 25% and 50%. So was SPC a revolution? Jaikumar argues that it was, as the nature of work changed. Operators became managers of the process, with a requirement for learning, discretion, and control of work that had been removed by the American and Taylor systems (Jaikumar, 2005).

Numeric control

SPC was the first process that recognized the difference between information about the process parameters and information about the physical process itself. This realization was taken further with the introduction of numeric control (NC), later to become computer numeric control or CNC.

NC allowed a general-purpose machine to undertake a number of operations on a single part. The major difference between NC and previous semi-automated machines was the ability to easily alter the operations by changing the programming on NC machines.

For NC to work at high production rates, every element of the machining operation has to be made explicit. Knowledge held by machinists, for example about adjusting tools as they wear, now has to become codified.

CNC greatly reduces the manual set-up time so a single operator can now look after more than one NC machine. One operator and a group of machines becomes a cell, leading to a cellular plant layout (Jaikumar, 2005).

An NC operator now becomes concerned not with a process, but with a procedure. NC at Beretta improved productivity threefold over SPC, but significantly, it dropped the minimum scale from 300 people running 150 machines to 100 people running 50 machines. Only half of the people were directly involved in production, although the tasks undertaken on the line blurred the definition of ‘line’ versus ‘staff’. NC greatly improved the flexibility of the line too, allowing a single line to produce up to 100 different products with just a 2% rework rate (Jaikumar, 2005).

The change to NC was significant and difficult. Jaikumar (2005) reports ‘“It was,” averred Ugo Beretta, “the biggest change in the culture of the plant that I have ever seen.’” But it added a flexibility and standardization that was of huge benefit. In 1985, Beretta won the contract to supply the standard pistol for the US military, but the weapon had to be manufactured in the USA. The reproducibility and transportability of NC methods meant that Beretta could build a new factory in the USA in less than eighteen months, and make money despite their bid being less than half that of the next cheapest bidder.

Computer-integrated manufacturing

For Beretta, the next move after NC was to computerize the entire manufacturing process, including design and factory operation, in what became known as computer-integrated manufacturing (CIM). CIM at Beretta started with a flexible manufacturing system (FMS) that consisted of a number of NC machines serviced by an automatic belt and robot arms for loading and unloading. In a typical line at Beretta, three machines perform all the operations to manufacture a receiver (part of a gun) and are managed by a single worker.

The flexibility reduces the minimum viable scale to just 30 people, with 20 of them in staff functions, and 10 operators managing 30 machines. CIM offers a threefold improvement in productivity over NC, and a remarkably low rework rate of just 0.5% (Jaikumar, 2005). The flexibility also offers the potential of manufacturing any product that can be designed, one day potentially leading to production runs of just a single custom gun.

Jaikumar (2005) makes the point that CIM started off as a productivity enhancement tool, but it quickly became apparent that it is also a knowledge enhancement or learning tool. It encapsulates knowledge about processes, and adds to that knowledge with each new product. It also provides a level of system intelligence by integrated problem-solving over different areas in the factory, rather than concentrating on single functional or product areas.

Summary

Jaikumar's study discussed the evolution of manufacturing using at least sixteen different factors. Four key factors are summarized here: productivity, minimum effective scale, the ratio of line to staff functions, and decision-making freedom on the line.

In Figure 2, the productivity improvement as a result of each technology change is plotted. Other data (Jaikumar, 2005) shows that each technology change resulted in a step change in productivity, and that between step changes there was little organic improvement in productivity.

In Figure 3, the blue graph plots the number of people required to achieve a workable scale. In the craft era, it was eight. In a modern CIM factory, it is 30. In between, it went up as high as 300 in the time of the Taylor System and SPC.

The red line in Figure 3 illustrates the percentage of people working in the factory actually making parts. In the Craft and English systems, everyone was making parts. With time, the overhead for maintenance, quality, and planning increased, until in the CIM age, only one-third of employees were directly operating machines.

Although the percentage of workers on the line decreases, Figure 4 shows that modern line workers have similar levels of flexibility in decision-making as those in the Craft System. The figure also implies that the need for learning in the workforce has increased since the Taylor era.

What mining can learn from gun manufacture

At a time when mining companies are desperately searching for improvement in productivity, experience at Beretta suggests that large increases are not going to come from organic improvements in productivity, but through a step change in technology.

As technology changes, the nature of work in mining is going to change. As gunmaking moved from art to science, the number of people employed first rose, then fell dramatically. We saw the rise in worker numbers in the early years of the South African gold mines. As mechanization is implemented, numbers will fall.

At the same time, the skills mix will change. At the science end of the Beretta example (Figure 3), the fraction of operators in the floor as a percentage of overall staff has fallen. As mechanization is implemented, while operator numbers will fall, there will be an increasing demand for skilled professionals in indirect roles such as mine design, operations planning, ventilation, and rock mechanics, to support the rapid rate of mining enabled by mechanization.

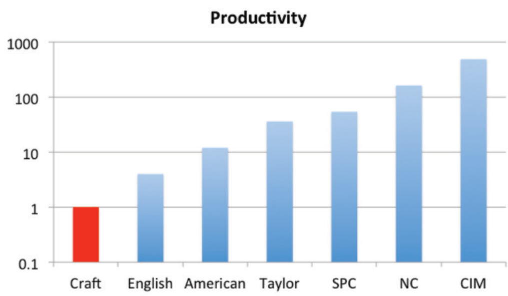

Figure 2—Productivity improvements by technology change

At the same time, the operators themselves will have greater flexibility of decision-making (Figure 4), and will be expected to use this flexibility. The current culture in mining in South Africa does not promote a high level of autonomy in operators, or more generally. A culture change will be required.

The level of change management required should also not be underestimated. Beretta's comment that the change to NC was the biggest change to the culture of the plant that he had ever seen is equally applicable to the change to mechanization in narrow-reef mining. Moxham (2004) cited change management as an unresolved issue for mechanization, and it remains so today.

Mapping mining from 'art' to 'science'

By analogy to the manufacturing example, it is possible to map mining on a scale from art to science. There are different ways of marking out epochs in mining, for example Karmis *et al.*, (2010) describe how mining engineering education has changed with time. Another typical mapping is of the technologies involved, moving from conventional to fully automated. Here a mapping is developed that aligns with the epochs described for manufacturing in Jaikumar (2005), focusing on the nature of the work experience: the autonomy of the operators, the knowledge required to do the job, and the need for learning,

The focus on the operators and on their knowledge requirements shows the importance of people, learning, and knowledge for the introduction of mechanization.

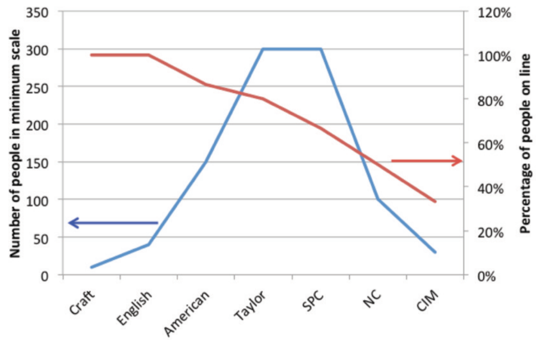

Figure 3—Number of people in minimum effective scale, and percentage of workers engaged in line functions

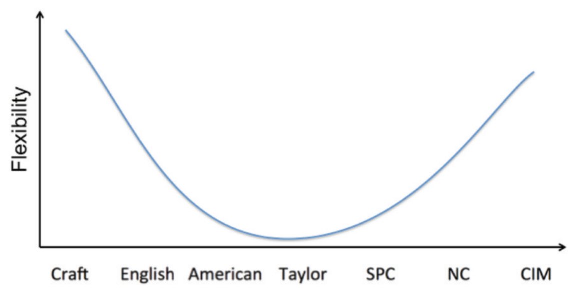

Figure 4—Schematic of decision-making flexibility on the line in the various technological epochs

The craft epoch

The craft epoch of mining is epitomized by the descriptions in '*De Re Metallica*', the famous book on the state-of-the-art as it was in the mid-16th century, written by Georg Agricola (Hoover and Hoover, 1950).

At the time, metal implements were used to break rock, typically hammers and chisels; or fire was used to flake off pieces of rock, today known as thermal spalling. Agricola paints a picture of relatively small groups of people working together. Of their leader, the miner, he says '*For a miner must have the greatest skill in his work*.' Agricola goes on to list the skills required of a miner, including knowledge of geology, mine design, assaying, metallurgy, 'philosophy' (what we would call the scientific method), medicine, astronomy, surveying, arithmetic, architecture ('*that he himself may construct the various machines and timber work required underground*'), drawing, and law (Hoover and Hoover, 1950, bk. 1). The list is not that different from a modern mining curriculum.

According to Bohn's description in Table I, mining of this period is typical of the craft state of knowledge. Miners had extensive knowledge of 'what', but not of 'why' in the sense that they did not understand much of the underlying science of what they did. They understood the entire process and they were open to learning. In the same way as in Jaikumar's craft epoch in industry, they also passed knowledge from person to person and knowledge was largely tacit. The miners were responsible for their own performance in the sense that they knew how to follow veins, how to mine them safely, and how to extract the metal for profit.

Two key technologies came to the craft epoch in the second half of the nineteenth century: dynamite and drilling machines. Early attempts to use gunpowder for rock-breaking quickly led to placing it in a hole drilled in the face. The pneumatic drill, patented by Ingersoll in 1871 (Mahmud, 2012) greatly increased the speed of drilling, and with the introduction of dynamite in 1867 the technology was in place to allow high productivity within a craft environment. The technology itself did not change the nature of the work: craft miners still understood the whole process and still valued learning.

There is evidence of the craft skills of Cornish miners in early South African gold mines: the quality of early excavations shows deep understanding of mining techniques. The Cornish were prized as miners precisely because of their skills, passed down from father to son, or through fellow workers (Hovis and Mouat, 1996).

But the large scale and low grades of the Witwatersrand orebodies led to a need for mining on a massive scale and with it, the demise of the craft miner.

The labour-intensive epoch

When a mining operation has low grades and high capital costs, it is economical to mine only on a large scale. The early Witwatersrand goldfields were an example. Sinking shafts was expensive, and once the early shallow claims had been mined out, mines had to increase their production to cover the costs of their shafts and other development. They did this by greatly increasing their scale, exactly as gun-making had done in the move to the American System not many years before.

By 1910, there were already more than 200 000 people employed on Witwatersrand gold mines, of whom 20 000 were white (Wilson, 1972). Because of colour-based job reservation (Wilson, 1972) the ratio of white to black workers at this time is a proxy for the ratio of skilled to unskilled workers.

The epoch of labour intensity is also one of lower skills, matching the American System in the gun industry described earlier. Mines started work-study activities, and went on to implement Taylorist management. A study of mining in the western USA (Hovis and Mouat, 1996) at around the same time as the early Witwatersrand (1896–1930) shows that those mines were following a similar trend, becoming much larger, controlled by mining engineers and metallurgists, with a lesser role played by skilled miners. The picture is of many unskilled workers in a large undertaking, managed by experts. Each individual worker is unskilled, and does not see or understand the whole process of mining.

Labour intensity led to high costs associated with labour, which caused industrial relations problems on several occasions on the Witwatersrand, most famously in 1922 and in 1989, but more recently in 2014 when most major South African platinum producers suffered a five-month strike caused by a demand for higher wages. One way around the impasse of high wage costs and low commodity prices is to mechanize.

Mechanization

Replacing human muscles with machines as a source of power increases the power available and hence the extraction rate. Nowhere in mining is this better demonstrated than the longwall mining system in a colliery, where just a handful of operators could already mine nearly a million tons of coal per month in the late 1990s (Kingshott and Graham 1998). Machines can also remove people from areas of highest risk, making mines safer.

The required change in the nature of work in the move to mechanization underground is comparable to the changes that occurred in the statistical process control and NC epochs in manufacturing mentioned above:

- Workers become more productive
- For each machine, there is an increasing group of support people. A typical continuous miner may require one artisan for two machines, with further artisans employed in maintenance, and professionals employed to plan and manage operations
- A miner has more freedom to make decisions about how to execute work, although still within a framework of standard operating procedures (SOPs)
- The decisions taken by an individual miner start to have a material influence on the performance of the whole mine.

The epoch of mechanization is characterized by a move toward science: knowledge is becoming increasingly codified, procedures are becoming specified, and results can be consistent and excellent. The extent of knowledge has grown.

The largest area of discrepancy between many mechanized operations and a 'science' as defined by Bohn is in the nature of knowledge: there is often still a gap in the knowledge of operators about why they are undertaking certain operations. There is also a gap in valuing knowledge and consciously making use of experience and experiments to learn and gain new knowledge.

Automation

Automation is defined here as mechanized equipment operating independent of human control at some level. There is limited automation in mining already, particularly in trucks. In high-wage countries with mines in remote locations, an automated truck can offer a considerable cost saving over a truck that requires drivers. In 2013, Rio Tinto announced that its automated trucks had moved 100 Mt of rock in Australia (Validakis, 2013).

There are also automated systems running underground. Among others, De Beers implemented a truck loop on its Finsch mine in 2005 (Burger, 2006), and plans automated tramming in the block cave level of its Venetia mine. An advanced automated system is operating at Kiruna in Sweden (Paraszczak and Planeta, 2001).

The level of automation varies from driver assistance, through remote tele-operation with driver assistance, to full automation. Apart from removing the driver, automation extends the benefits of mechanization and improves productivity, improves the quality of the excavation, and reduces operating and maintenance costs because the machines are always operated within their design limits (Brouillette *et al.*, 2003; Paraszczak and Planeta, 2001; Swart *et al.*, 2002).

Possibly the biggest saving to be gained through automation is by making roadways narrower. Automated trucks can navigate so accurately that they can safely pass each other with less space between them, so roads can be narrower and correspondingly cheaper to build (Bellamy and Pravica, 2011).

Automation occurs at a high level on the art–science continuum. In addition to the requirements for an operator in a mechanized system, operators of automated systems require knowledge of why operations occur as well as how. Much of the knowledge is already codified within the computer programs that operate the machines, much as it is in automated flying or NC control in factories.

While independent machines have been automated, the next phase, of automating fleets of equipment, is just starting to be investigated.

Computer-integrated mining

Beyond automation, there is much to be gained by optimizing the entire process of mining, from exploration to product (Miller, 2009). Mines are traditionally managed using compartmentalized systems, with each activity optimizing its own operations, independent of the needs of the larger system. Modern information technology systems are making it possible to monitor and control the entire value chain within the mine to optimize chosen metrics such as cost, productivity, or capital expenditure.

A fully automated mine with computer integration would have few operators in the traditional sense. Those operators would be knowledgeable about their own tasks and the broader context of the entire process. The majority of people employed on the mine would be involved in maintaining the machinery (artisans), and in planning and executing the construction and operation of the mine (geologists, mining engineers, electrical, mechanical, and industrial engineers, *etc.*).

Implications for mining

The five stages of mining listed here describe the type of people required in each phase, and can be used as a guideline to what will be required from the workforce when mechanization is implemented.

The literature from other sectors that have mechanized is useful because it can prepare miners for the inevitable steps and challenges at higher levels of technology. While solutions for some steps are self-evident, such as the need for more skilled artisans, thinking about how jobs change can speed the transition. It is clear that workers in automated factories are expected to think differently, and have a more holistic view of the operation. We argue that this expectation is still missing in most thinking about mine mechanization.

The need for a learning organization in mining

The changes in the level of discretion or autonomy, required knowledge, and individual productivity as mining moves from art to science require new thinking from everyone working on the mine. The step from massive labour-intensive mining to mechanized mining is particularly challenging. Changed thinking is intimately associated with learning. Learning organizations, discussed here, provide environments that enable the learning necessary to adapt to new techniques like mechanization.

Comparisons of mining with technological epochs in other industries show the need for workers who are able to learn and organizations that encourage and facilitate learning. Are mines currently learning organizations?

There are many lines of evidence that South African narrow tabular hard-rock mines are not learning organizations. As one senior technical consultant to the industry recounted, 'When I started on the mines I learnt two things: how to lie and how to keep out of sight.' The statement typifies the culture: results-driven and authoritarian.

The discussion in this section is anecdotal, based on informal discussions over many years with mining professionals and others who work underground.

Student thinking

Initial evidence of the constraints on thinking comes from students. The University of the Witwatersrand offers courses as one of the components of a Master's degree in mining. On one 2015 course, the student attendance was split roughly 50:50 between students who had come straight from their undergraduate studies and students who were working in mining and studying part time.

There was a notable division between the two groups of students because of the constraints on creativity demonstrated by the students with work experience. While any change to improve mining productivity ultimately has to be practical, the work experience of these students had limited their ability to even consider approaches outside of the current norm.

Change in Bushveld Complex mines

Miners in the platinum sector have often commented on the relative ease of introducing new technology on the eastern limb of the Bushveld Complex compared with the western limb. Miners on the eastern limb are usually from local communities, with no previous history of mining. Mine managers are able to implement novel practices, because the miners don't know any other way of doing things.

Miners on the western limb often come from the Witwatersrand gold mines, or have long histories in mining. Attempts to change practices on the western limb have run into problems, with miners simply reverting to previous practices.

The unwillingness to learn on the western limb is indicative of the lack of a learning culture: the miners have preconceived ideas and have no desire to learn new ones. Miners on the eastern limb have no choice but to learn, as they have no prior experience. Companies can introduce new techniques because the workforce is willing to learn. The evidence does not indicate that eastern limb workers have a learning culture and will retain it in the future, but does explain part of why mechanization has been more successful on the eastern limb.

Information flow

Compliance with the results-driven and authoritarian culture is exacerbated by a lack of timely management information about the state of operations underground. On deep gold and platinum mines, many key operating parameters such as face advance are measured only once a month. The low sampling rate makes it possible to get away with a lie like, '*Yes sir, I did blast today*'. The miner hopes to compensate for the lie with greater productivity before the measurement date and the manager is not easily able to check the truth of the statement.

This lack of timely information, combined with a relatively simple layout and very large workforce, has been managed through a strongly hierarchical system. The system has applied typically bureaucratic methods, including standard operating procedures. It is seldom that a mine dynamically adapts its SOPs or plans to cater for changing geological conditions. Miners underground are not asked to learn and are discouraged from challenging their superiors, the current processes, or practices. Miners are also expected to function only within their own trade, such as drilling, and not to understand or even ask questions about the broader priorities of the workplace.

Mining priorities

Production is the key priority in mines. Even with the recent focus on safety, it is widely understood that production is still the priority. Mineworkers in South Africa earn a large percentage of their packages through production bonuses. They regard any threat to production as an imminent threat to their livelihoods. It is challenging to introduce a new technology when there might be a short-term drop in production prior to a long-term increase – people are not interested in the delay that will be caused by the learning process.

The learning organization

Peter Senge promoted the idea of a learning organization in his book 'The Fifth Discipline', subtitled 'The art and practice of the learning organisation', (Senge, 2006). Senge defines learning organizations as '*organizations where people continually expand their capacity to create the results they truly desire, where new and expansive patterns of thinking are nurtured, where collective aspiration is set free, and where people are continually learning to see the whole together*'. Senge believes that 'in the long run, the only sustainable source of competitive edge is your organisation's ability to learn faster than its competitors.' (Senge, 2006).

The idea of the learning organization is also intrinsic to one successful school of thought about manufacturing. Toyota Motor Corporation emerged to become the largest car manufacturer in the world on the basis of its Toyota Production System (TPS), also known as Lean Manufacturing or just Lean (Womack and Jones, 2003). Ballé *et al.*, (2006) argue that four frames pervade the TPS: 'performance mindset, problem awareness, solving problems the "right" way, and developing people through problem-solving'. All but the first frame are associated with creating an environment that allows and encourages learning in order to solve problems.

Within a context of core capabilities, Leonard-Barton (1992) discusses how she observed four responses to proposed technological change in manufacturing plants. All are a consequence of people: abandonment; recidivism; reorientation; and isolation. The first two lead to failure, the last two are routes to success:

- In the case of mining, *abandonment and recidivism* both lead to mechanization not being applied: either the new technology is abandoned, or the workers go back to their old ways – in other words, it might be a mechanized mine, but it looks and operates like a labour-intensive mine. Abandonment of mechanization is often a consequence of a lack of preparedness (Nong and Musingwini, 2011), which may stem from insufficient emphasis on change management
- *Reorientation* refers to relocating a change project into a group without the same core capabilities as the original group. In mining, it could refer to using a new team to implement mechanization that does not share key core capabilities, particularly in managerial systems and values, with the original team
- Isolation occurs in new product development when development is undertaken in a new plant. Isolation is related to reorientation, but refers to adopting a new technology in a different place, rather than with different people. In mining, it suggests that mechanizing a new mine is easier than mechanizing an existing one, not least because the people employed on a new mine would not share the managerial systems and values of an existing mine.

What we can learn for mechanization

Returning to Murphy's Curve, the case study that started this paper (Figure 1), Chew *et al.* (1991) offer some recommendations on how to make a successful technological transition. The recommendations are both people- and process-related and are adapted here for a mechanized mining context.

- Think of implementation as research and development. If mechanization is seen as something that can just be purchased and put into operation, then the implementation programme will be planned in that way. Instead, the introduction of mechanization should be seen as an experiment. There will be successive phases of data gathering and learning through experiment as the implementation moves through the scoping, pre-feasibility, and feasibility stages before being implemented. The goal of experiments should be to address technical *and organizational* uncertainty. Miners need to be part of the team, working as co-developers, rather than having the system dumped on them

➤ Ask 'what made it hard', not 'how well did it work'? Miners do look at other mines before implementing mechanization, but the conversation is often about the success. To learn more about the challenges, it is necessary to dig into them with questions like 'how did you make this work? What had to be changed? What would you do differently?' (Chew *et al.*, 1991). Miners should also look at parallel experiences in other industries. As this paper makes clear, the manufacturing industry has a lot of knowledge about mechanization available for miners to consider

➤ Learn in many ways at once. There are generally four methods that a mine could use:
  - Vicarious learning – watch what others are doing and learn from them
  - Simulation – construct synthetic models and use them for experimentation and learning
  - Prototype – build and operate the proposed system on a small scale and in a controlled space
  - Production, or on-line, learning – learn while trialling or operating the final product (Chew *et al.*, 1991).

  As we move down the list, the cost goes up and the scope for experimentation goes down. According to Chew *et al.* (1991) managers prefer the perfect fidelity of the operating system (by definition it matches reality perfectly), failing to recognize that learning is not all or nothing – there is value in learning about single key issues using tools with less fidelity, which leads naturally to:

➤ Simulate and prototype *everything*. If production learning is expensive and restrictive, and vicarious learning is limited, simulation and prototyping are critical. Simulation can vary from spreadsheets to physical models of a mine. It allows for easy investigation of complex systems, where individual components are understood, but the way they interact is not. It is easy and cheap to run many simulations to investigate each component of a proposed mining system.

  In manufacturing, a prototype is a physical simulation of a process. For mining, a prototype is a trial area. For best value, the trial should be conducted with the purpose of improving the process, rather than just as a method of training people to comply with the process

➤ 'Everything' includes the organization. If the rule is 'simulate everything', then everything must include the organization. The physical processes of a new mining layout, for example, are often simulated (Valicek *et al.*, 2012). There is much to be gained by also simulating the organizational structure and culture of the proposed new system.

➤ Be prepared, but not too prepared. As the military strategist von Moltke said 'No battle plan survives first contact with the enemy.' Implementation failures, particularly in mining, are often blamed on poor planning. Plans are often too specific and end up being unable to deal with reality. It is better to have a clear goal, and prepare for contingencies by having resources and good people. The plan should focus on what to look out for and think about, rather than what to do (Chew *et al.*, 1991)

➤ Produce two outputs: ore and *knowledge*. Once a new mining method is up and running, problems will still arise. If the need to produce knowledge is explicitly recognised as part of production, that knowledge will lead to faster problem-solving. This recommendation from Chew *et al.* follows the philosophy of Lean (Ballé *et al.*, 2006). The most expensive time to learn is once the mine is running – but it is still better in the long run to take the short-term cost of learning in exchange for the long-term gain in operational performance.

Chew *et al.* argue for the need to plan and manage directed learning, '*Anything you don't learn about early will hurt you later*.' They criticize typical investment analysis that sees learning as a pure cost and recommend budgeting for learning throughout the project schedule. In particular, there needs to be budget for learning on the operating system in the first few months of operation, over and above the planned lower output during start-up. If the earlier learning stages of vicarious learning, simulation, and prototyping have been thoroughly used, 10% of production time is realistic (Chew *et al.*, 1991).

Conclusion and recommendations

Mechanizing a mine requires a change in technology, but it also creates a change in the nature of work for the miner. Experience in manufacturing shows that there is high resistance to changes in the nature of work, particularly when the change requires people to accept higher levels of decision-making and responsibility. There is also a need for employees to better understand the overall process as mining moves from art to science.

It is much easier to implement change in a learning organization. Anecdotal evidence shows that mining companies in South Africa are not learning organizations, but are still organized along Taylorist principles. Taylor's thinking has been superseded with the recognition that the circumstances of an operation change with time.

In an environment that is as subject to variability as the rock in an underground mine, a mechanized mining approach requires a management system that allows for higher levels of personal initiative. Such a learning system would improve conventional mining operations, but it is essential for a successful shift to mechanization.

Specific recommendations

Given that engineers drive most mechanization projects, we recommend that mechanization teams incorporate experts in human and organizational behaviour to ensure that the people issues discussed here are not neglected during the planning and execution of the project.

Consider approaches to mechanization that allow for learning, such as trial sites, mechanizing on new mines, or using teams from different backgrounds to the conventional miners on your mine. Allow reorientation and isolation to work for you in the change process.

Use simulation. It is much cheaper than physical trials, it is getting rapidly more comprehensive, and it can help ask and answer questions about organizational issues that won't occur during small-scale prototype tests.

Think about implementing mechanization as a quest for knowledge as well as rock, and budget for experiments. If generating knowledge is incorporated into the planning from the start, it will not come as a surprise when experiments need to be conducted to determine parameters that weren't planned for.

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Improvement in the overall efficiency of mining equipment: a case study

by H. Fourie*

Synopsis

In mechanized mining, poor equipment efficiency (availability, utilization, productivity, and quality) can endanger the success of the operation. This case study will show how an initiative to improve equipment performance developed into a comprehensive turnaround plan for the mine that placed it in the forefront of performance achievement.

As part of a company-wide review process, poor overall equipment effectiveness (OEE) was identified as a major reason for the mine not achieving its targets. A project to improve the OEE identified eight improvement areas (elements) that contributed significantly to the poor performances. Measurement metrics were determined for these elements, followed by determination of baseline and target (improved) key performance indicators (KPIs). Cost savings associated with the improved efficiencies were calculated and tracked throughout the project. The mine team determined the specific actions required to achieve the target KPI in each element. These were individually developed and managed like mini-projects with allocated responsibilities for delivery

The paper will indicate how this OEE improvement initiative triggered an improvement in almost all sections of the mine. Soon after launch, the initiative gathered momentum as the KPIs starting to improve. A visible tracking system exists at the mine and each employee can see the improvements and feel the success. The original eight elements were extended by five more, and the mini-projects grew as participants saw the success of the initiative.

This paper concludes that through management and worker involvement, visible measurement and controls, and carefully chosen improvement elements, the mine was turned around. It is now achieving and exceeding its targets, and employee relations and motivation as well as safety have improved considerably. All of these achievements are reflected in the bottom line.

Keywords

overall equipment effectiveness, heavy mining equipment, productivity improvement, efficiency improvement, management support, cost reduction, motivated workforce, sustainability of success, key performance Indicators, action plans.

Introduction

As a result of the 2008/09 economic downturn, Mogalakwena platinum mine reduced its run-of-mine production in 2009 from 100 Mt/a to 35 Mt/a. The recovery to a budgeted 85 Mt/a in the following years proved to be a challenge, and was not achieved until 2014.

Mogalakwena is a Tier 1 operation in Anglo American Platinum (AAP) and crucial to the success of the company, especially in recent years with the restructuring that has been widely implemented in the industry. Under-target performance from Mogalakwena impacted negatively on the company's ability to deliver on its promises to shareholders and other stakeholders.

In 2012 AAP launched a company-wide review ('the Platinum Review') with one of the main objectives being to identify opportunities for cost reduction and productivity improvement.

One of these initiatives was aimed at equipment performance at Mogalakwena. Overall equipment effectiveness, also referred to as OEE, is a hierarchy of metrics that focus on how effectively equipment is utilized. The OEE of a machine is the product of its operational availability, utilization of the availability, its productivity, and quality of operation.

$$OEE = Av \times UoAv \times Prod \times Q$$

where
OEE = Overall equipment effectiveness [%]
Av = Availability of machine or fleet of machines
UoAv = Use of availability of a machine or fleet of machines
Prod = productivity (expressed as operating tempo) of a machine or fleet of machines
Q = Quality of work done by a machine or fleet of machines.

Hypothesis for improvement

As part of the Platinum Review, consulting company KPMG was contracted to investigate opportunities for cost reduction and productivity improvement. One of these opportunities was for improvement of the OEE at Mogalakwena, and was formulated as follows:

* *Anglo American Platinum.*

Improvement in the overall efficiency of mining equipment: a case study

'There is an opportunity to enhance operating hours and bring down operating cost by improving the overall equipment effectiveness (OEE) for key mining equipment at Mogalakwena mine.'

KPMG further stated that:

- ➤ Analysis and management interviews showed that certain equipment types may not be operating at optimum efficiency levels
- ➤ Comparator data provided further evidence that equipment may not be operating at optimal utilization and availability; currently, levels were below the benchmarks for similar equipment.

KPMG obtained data from the mine's fleet management system (FMS), Modular Dispatch, during 2011 and 2012 and compiled infographics showing the downtime distribution for each equipment type. Figures 1 and 2 show the downtime for shovels. Similar analyses were done for haul trucks and drill rigs.

A preliminary assessment showed that the major reasons for shovels downtime were:

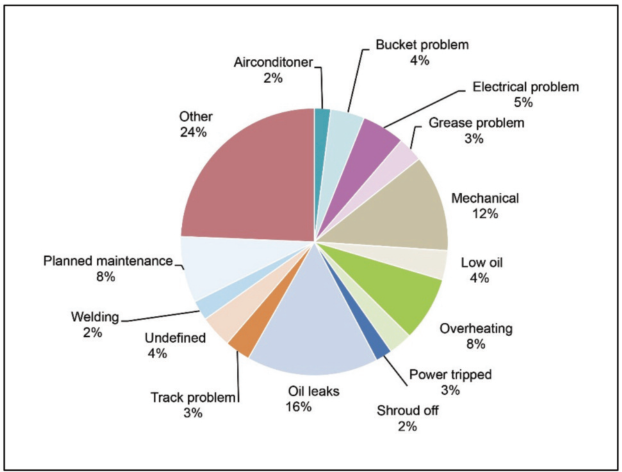

Figure 1—Machine downtime for hydraulic shovels (Anglo American Platinum, 2013b)

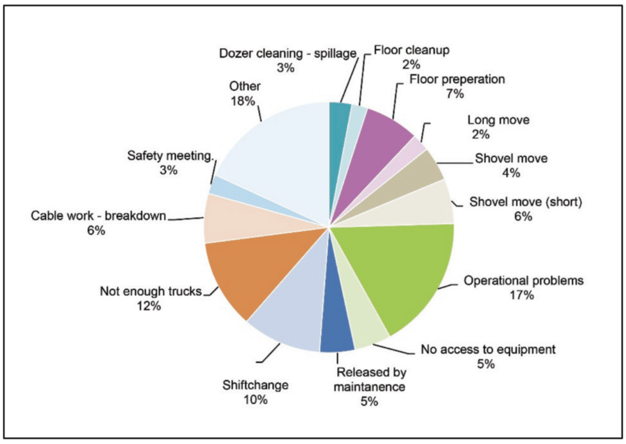

Figure 2—Lost time and operational downtime for hydraulic shovels (Anglo American Platinum, 2013b)

- ➤ Hydraulic oil leaks (16%)
- ➤ Mechanical downtime (12%) – a generic failure code that covers a variety of failures and is indicative of the reliability of the machine
- ➤ Overheating (8%)
- ➤ Bucket problems (15% for RH 400) caused by severe rock conditions resulting from poor blasting fragmentation
- ➤ Operational problems (17%) – a generic downtime code mostly attributable to poor blasting fragmentation
- ➤ Shift change (10%)
- ➤ Not enough haul trucks (12%)
- ➤ Shovel moves (12%).

Similar analyses were done for all the equipment in operation at the mine.

Development of improvement projects

Project initiation

During a workshop at the mine attended by the management team and key operational personnel, the analyses from the KPMG hypothesis together with the mine's own assessments were discussed and the following issues were identified as the main contributors to the poor equipment performance:

Availability

- – Accident damage
- – Power/cables
- – Machine reliability
- – Tyres
- – Ground engaging tools
- – Supply chain
- – Maintenance by OEM/contractor
- – Sources and facilities
- – Maintenance scheduling.

Utilization

- – Blasting delays
- – Housekeeping
- – Safety stoppages
- – Equipment moves
- – Meetings and travel to workplace
- – Shift change
- – Attendance / absenteeism / no operator
- – Dust
- – Dewatering
- – Rain
- – Equipment performing wrong jobs
- – Road construction.

Productivity

- – Quality (fragmentation, floors, water, MTBF, etc.)
- – Tool selection (right machine for the job)
- – Equipment size selection
- – Short-term planning and work scheduling
- – Haul profiles
- – Operator competency.

The mine's team realized that a number of these issues were interrelated and that an effective improvement programme should focus on a small number of major issues. A list of eight issues (termed elements) to be addressed was then defined, as indicated in Table I.

Table I

List of initial eight elements included in the improvement project

Lever		Element
Availability	1	Equipment damage
	2	Cable management
Utilization	3	Shift change
	4	Attendance
	5	Blasting delays
Productivity	6	Truck payload management
	7	Shovel loading rates
Quality	8	Maintenance reliability

Measurement metrics and key performance indicators (KPI)

The metrics to be used to measure each element were determined as per Table II. Each element had to be measurable through existing systems on the mine, and preferably a history of data should exist in order to determine the baseline KPI.

Although it had to be upgraded and optimized, the FMS provided all the data required. From its historical data the baseline KPIs were determined.

The mine decided to include all of its primary production equipment, namely shovels, trucks, and drill rigs, in the initiative.

Three methods were used to determine the target KPI for each element:

- *Benchmarking* – information on the machine type and size used at Mogalakwena was not readily available within the group and from other mining operations. Where information was obtained, it was often generic and the definitions for the KPI quoted were unclear, making the benchmarking exercise of limited value
- *Best demonstrated performance* – after removing clear outliers, these were good indicators of what is achievable at the mine
- *Using 'optimistic' OEE* – the project team decided on optimistic factors for the OEE levers and calculated the achievable KPI taking cognisance of the technical and physical issues at the mine.

Cost saving calculations

Increase in operating time (improved availability and utilisation)

When a machine achieves more operating hours as a result of improvements, how much cost is saved? This is not merely the hourly operating cost, because the machine did not incur variable costs while it was standing, yet when it operates it incurs that cost. It was determined that the cost saving is the fixed cost portion of the machine's hourly cost. This can be explained by the fact that if a machine is on breakdown or is not operating, it continues to incur fixed cost. In order to do the work, another machine with the same variable and fixed cost has to operate. The total hourly cost is therefore double the fixed cost plus the variable cost, hence when a machine reduces its downtime, the cost saving is its fixed hourly cost.

Increase in productivity

When productivity improves the machine produces more output for the same input. The additional output units (*e.g.* more tones per hour) therefore do not incur additional cost. The cost saving for productivity improvement is therefore the cost per unit (*e.g.* rands per ton) multiplied by the additional units produced.

Improvement in quality

Most of the KPIs used to measure quality improvement cannot be converted into monetary value, *e.g.* MTBF, which is a measure of the quality of maintenance, cannot be converted directly into a cost saving. Another KPI that is closely linked to the quality KPI must be identified to enable the cost saving to be calculated.

For the quality element of Maintenance Reliability in this project, the reduction in hours lost due to equipment breakdowns is used as the KPI for cost-saving calculations.

Table II

Measurement metrics and KPI

Lever	Element	Metric	Equipment	KPI baseline	KPI target
Availability	Equipment damage	Operating hours lost per month due to equipment damage [hours per month]	Shovels	15	12
			Trucks	1 831	900
			Drills	11	8
	Cable management	Drill and shovel operating hours lost due to cable work [hours per month]	Shovels	300	150
			Drills	240	220
Utilization	Shift change	Tons loaded and metres drilled in first and last hour of shift [tons or metres]	Loading [t]	4 940	6 000
			Drilling [m]	76	90
	Attendance	Operating hours lost due to no operator [hours per month]		2 160	1 200
	Blasting delays	Shovel and drill operating hours lost due to blasting delays [hours per blast]		102	45
Productivity	Truck payload management	Average tons per truck load [tons per load]	Kom930E	254	265
			TerexMT4400	175	185
			TerexMT3700	132	150
	Shovel loading rate	Instantaneous average shovel loading rate [tons per hour]	RH400	2 100	2 320
			RH340	1 500	1 800
Quality	Maintenance reliability	Mean time between failure (MTBF) [hours]	Shovels	3	8
			Drills	3.5	8
			Trucks	12	18

Action plans and cost of implementation

The improvements required in this exercise were facilitated by a range of action plans. Although some root cause analyses were done to identify the appropriate actions required, most of the actions had previously been identified by the mine's technical teams.

The action plans for each element were listed. Where an action plan supported two or more elements, it was allocated to the element where it would have the largest impact. Table III lists the action plans per element.

The details of the 42 action plans are not discussed in this paper. It must be noted that some of the action plans are large projects in their own right, *e.g.* 'Rebuild all overdue heavy mining equipment' under Maintenance Reliability. Similarly, the impacts of some of these action plans are significant and span improvement over a number of the elements, *e.g.* 'Blast fragmentation' under Shovel Loading Rate.

The costs for implementing the action plans were then estimated. These were divided into ongoing operating costs and once-off or capital cost items. Some of the costs were already included in the mine's budget; but for others, either application was made for over-expenditure or the item was included in the next budgeting cycle.

Schedules for action plan implementation

The project was planned to run for 3½ years until the end of 2016. A monthly schedule was then compiled for the implementation of the action plans, depending on:

- ➤ Expected impact on improvement
- ➤ The complexity (or ease) of implementation
- ➤ Availability of funds
- ➤ Long lead-time items
- ➤ Nature of the action plan (*e.g.* does it involve new technology?)
- ➤ Where applicable, the required sequence of implementation.

With the improvement plans scheduled out, the associated cost savings schedule was determined by estimating realistic KPI improvement targets over time. With

Table III

Action plans for each element

Lever	Element		Actions
Availability	**Equipment damage**	1	Compile and Implement fatigue management plan (FMP)
		2	Secure radio network coverage of all operating areas
		3	Various main haul road improvements
		4	Make training video of heavy mining equipment (HME)
		5	Install Guardvant (or similar system) on all HME
		6	Redesign and construct access to crusher tipping areas
		7	Install propel inhibiting system on dump trucks
		8	Install vehicle and people detection system on all light vehicles
	Cable management	1	Appoint cable management contractor
		2	Buy 1 cable reeler and 1 stationary reeler
		3	Move cable yard to central area
		4	Convert to vulcanized joints
Utilization	**Shift change**	1	Do shift change 'on-bus'
		2	Overlap shifts during change
		3	Automate assignment of operators
		4	Redesign hard park
		5	Automate refuelling
		6	Do daily equipment checks with refuelling
	Attendance	1	Psychological intervention to change work culture
		2	Recruitment and redeployment of operators
		3	Strict implementation of new absent and sick abuse procedures
		4	Monitor individual operator actual operating hours
		5	Increase HR department capacity deal with poor attendance
		6	Training and multi-licencing of operators
	Blasting delays	1	Change blasting time to mid-shift
		2	Reduce blasting to once per week
		3	Facilitate quick re-start after blast
Productivity	**Truck payload management**	1	Install payload management systems on all trucks with communications to dispatch
		2	Truck bucket standardisation for Komatsu 930E trucks
		3	Optimise Dispatch efficiency
		4	Install load metrics on all shovels
	Shovel Loading rates	1	Introduce double-sided loading
		2	Introduce boulder breaker per loading area
		3	Measure fragmentation and blast movement
		4	Maintain productive loading area
		5	Acquire large front end loader for remnant loading
Quality	**Maintenance reliability**	1	Rebuild all overdue heavy mining equipment
		2	Fill all vacancies and upskill the artisans
		3	Stock and manning the equipment spare stores
		4	Provide vehicles for all artisans
		5	Implement MineCare software
		6	Implement planned maintenance strategy

this, the costs and expected cost savings per month were calculated on the basis determined earlier. The quantity of the improvement metric per time unit (*e.g.* reduced hours lost per month due to a reduction in shovel damage) was multiplied by the cost per time unit (*e.g.* fixed cost portion of the operating cost per hour for shovels) to determine the cost saving during the time period.

Recording and controls

The final step in the project development was to formalize the measurements of KPI and to ensure controls are in place to provide a fully auditable process.

The FMS was the main source of data, although establishing some of its measurements was part of the action items, *e.g.* the payload measurement on the older Terex trucks. Some of the measurements were verified through the month-end surveys as well as records kept by the engineering planning department.

Controls took a wide variety of forms. The Business Improvement (BI) department uses the measurements for statistical analysis and presents the results in the form of control charts and other graphical displays. Project tracking and controls were put in place to track the implementation of the action plans. The KPI achieved are also reported to the corporate office on a monthly basis, with the cost as well as the savings recorded.

A Project Management Office (PMO) was established at corporate office to ensure a holistic, enterprise-wide approach and appropriate support, focus, coordination, tracking, and delivery of intended benefits. Regular auditing of the project takes place.

Project implementation and progress tracking

Project leaders for each element were appointed. These appointments were crucial to the success of the project and the leaders were carefully selected for their knowledge, experience, seniority, and passion for the job.

The project leaders generated detailed project schedules for the implementation and execution of the action plans for their elements. This led to verification of, and sometimes modifications to, the initial action plans. Detailed costing was also done and adjustments made to the costing schedules where required.

Initially progress meetings were held every second week, chaired by the general manager (GM) and attended by all heads of department on the mine. This gave prominence to the project, provided cross-functional coordination and support, and an opportunity to resolve any implementation hiccups. These meetings continued for several months until the project yielded the expected results.

Some of the action plans were budgeted for and would have been implemented anyway, but most were new in the sense that they were 'launched' by making all role players aware of the plans, the methods of implementation, and the benefits for them as well as the mine as a whole.

Another method used was a process called 'Rapid Results Projects'. Some of the people on the mine were familiar with it as it had been used with success in the past. The method consists of involving a selected team from all levels in the area of the project, who seeks their own solution to the issue, set their own targets, and drive it themselves under the leadership of a supervisor. The results are measured over a 100-day period, by which time the targets set by the team should have been met.

These results were publicized around the mine with visual displays on large colourful graphs (see Figure 3) and pictures on noticeboards, both electronic and in hard copy.

Results

The project yielded results soon after launch and by the third month it was clear that the mine was on its way to major improvements. In addition to the OEE improvements and reduction in unit costs, the largest benefit was in the behavioural change of the workforce. The previous negative and sometimes destructive attitude changed relative quickly into one of positive self-motivation, and the personnel began to reduce wastage on their own initiative. This was further fuelled by the decent bonuses that people started to earn. The improved morale of the workforce was evident in the reduction in absenteeism, reduced shift change stoppages, and reduction in equipment damage.

The cost savings were positive from day one and grew continuously as the KPI targets were attained and surpassed. By month 16 (September 2014) the savings targeted for the full project period of 36 months were already achieved!

Figures 4 to 7 show some of the results.

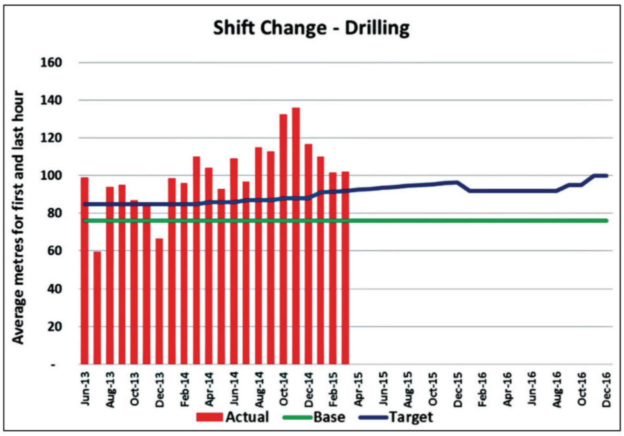

Figure 3—Example of visual display of results on noticeboards (Mogalakwena Mine, 2015d)

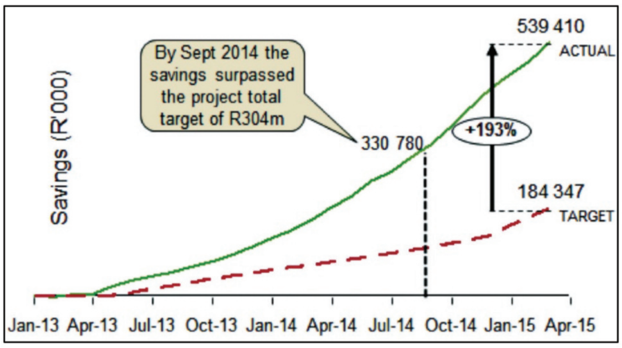

Figure 4—Cumulative cost savings (Mogalakwena Mine, 2015c)

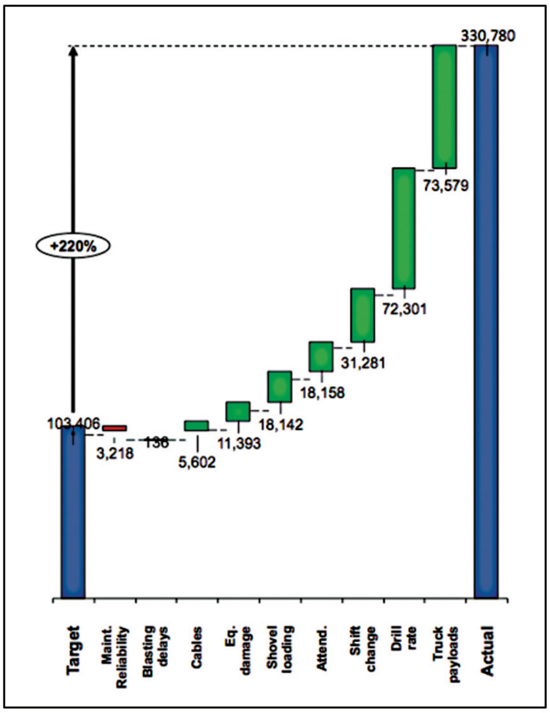

Figure 5—Cost saving per element above target (Mogalakwena Mine, 2014a)

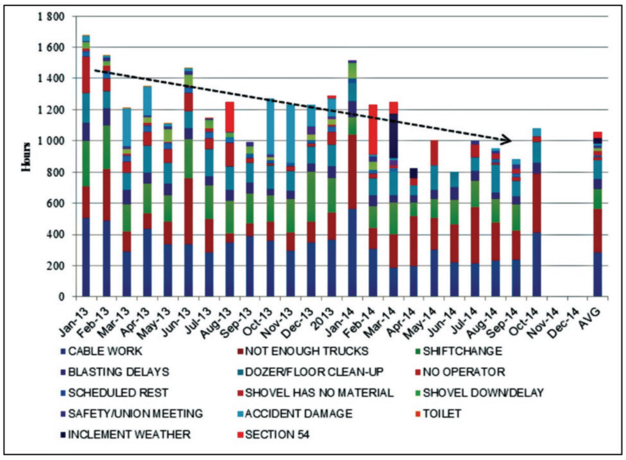

Figure 6—Production hours lost on shovels (Mogalakwena Mine. 2015a)

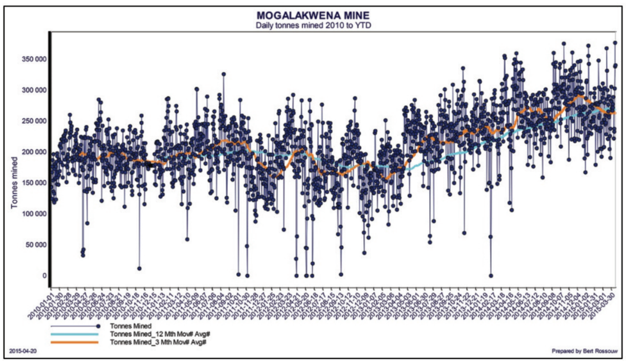

Figure 7—Daily tons mined since 2010 (Mogalakwena Mine, 2015b)

Project growth

During the implementation of the project, Anglo American conducted a technical review of all its major mines, including Mogalakwena. The improvement actions from that Asset Review included all eight elements of the Platinum Review and five more elements were added, all enablers for improvement (Figure 8). These were Support Equipment OEE, Supervisor Competencies, Quality Control on Drilling, Blasting, and Loading, and Resource to Market Initiatives.

The project name was also changed to the Asset Review Benefit Realisation, or ARBR.

As the project gathered momentum, the supervisors and workers who did not benefit directly from the original eight elements motivated further growth. Drill penetration rate, rope shovel loading rate, and tyre life improvement were all added to the original scope.

The improvement project originally aimed at improving OEE had by then become the all-inclusive improvement drive on the mine, and was the reason for the performance turn-around since Q2, 2013.

Sustainability

To ensure that the improvements obtained in this project are sustainable, the following measures were taken:

- ➤ A number of the action plans were directed at improving the measuring and reporting systems at the mine, *e.g.* the payload measurement and recording system on trucks. The drill quality measurement and recording through the Rockma system is another example. These types of management actions are not dependent on a motivated champion to maintain their existence, but are part of the management and reporting system on the mine
- ➤ Management support of the initiative, both at the mine and in the corporate offices, ensured continuous focus on the project

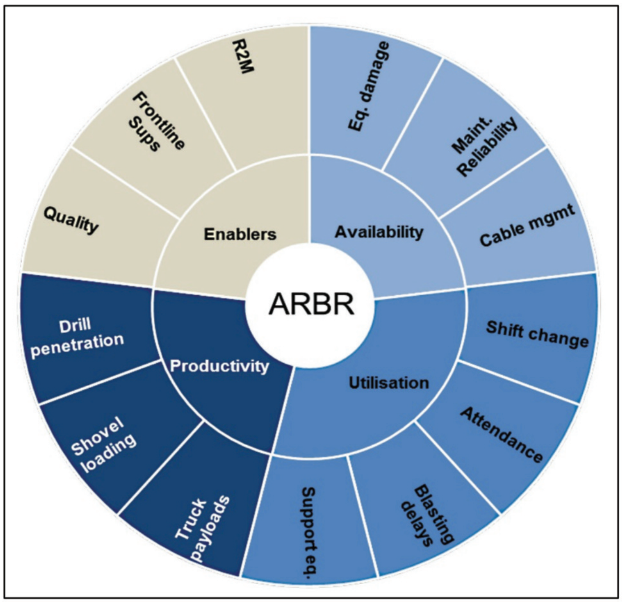

Figure 8—Thirteen elements of the ARBR (Mogalakwena Mine, 2015f)

- ➤ The monthly reporting and reviews of the progress of the project. This provides a forum to discuss and address any problems or restrictions that arise during the implementation and development of the action plans
- ➤ The visual presentation of results at the mine, updated daily, coupled with the bonus payments, certainly maintains the momentum created amongst the workforce
- ➤ The evolving of the scope of the project by growing it into other worthy issues keeps the interest alive and helps the identification of further improvement opportunities
- ➤ The process allows for continual monitoring of elements that have achieved their final targets for a couple of months, with the focus turned onto those where it is more difficult to achieve the targets. However, when the performance of any of these elements falls below target, they are brought back into the focus area to analyse and rectify the cause of the down-dip. This has happened with blasting delays when the delays increased again after additional precautionary measures were taken following the blasting incident at the mine in March 2015.

Success factors

The improvement initiative was formalized in the company with full management commitment and support on all levels. Management actively engaged in the process to ensure any constraints and obstacles were addressed as soon as practical.

The fact that the Platinum Review was a company-wide formal process, with audits of the savings and improvements reported and regular reporting to the highest level, further supported the success of the project.

The strategy when setting the initial targets was not to beat the world benchmarks or to achieve world best practice in one step, but rather to set achievable targets. There was also a clear understanding that achieving the initial targets was not the end, but growing beyond that would align the mine with its vision 'To be the best PGM mine in the world'.

The success of the improvement, supported by the visual displays of the results around the mine, provided for a positive workforce as the self-esteem of individuals increased together with the recognition they received. The resultant healthy bonuses further supported worker motivation.

Figure 9 is a pictorial of the improvements on all fronts of the mining operation.

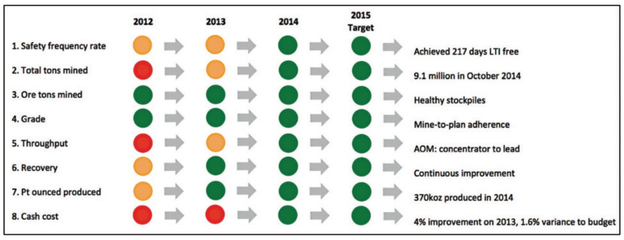

Figure 9—Improvements in all areas (Mogalakwena Mine, 2015e)

Conclusion

The OEE improvement project as part of the Platinum Review paved the way for a turn-around in overall performance at Mogalakwena mine. The project focus was not on cost savings, but on OEE improvement, which is fully aligned with the Anglo Operating Model (AOM) by doing the right work at the right time in the right way.

During the project, the mine progressed from an inefficient to an effective operation, and is on its way to becoming efficient in most aspects.

Success breeds success. As the improvements gathered momentum, more people wanted to be part of the process and this resulted in the change from an unmotivated workforce to a highly motivated workforce, as is evident by the fact that the mine did not participate in the five-month industry strike of 2014.

As the mine turned around as a result of the OEE project, the pursuit of improvement caused a ripple effect that encompassed other activities not necessarily related to OEE's. It became the vehicle for success at the mine.

Acknowledgements

The major role that the General Manager and management team of Mogalakwena Mine played in the success of this project is acknowledged.

Mogalakwena mine is also thanked for allowing figures and tables from their management report to be used in this paper, as well as for their assistance in compiling this paper.

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Trends in productivity in the South African gold mining industry

by P.N. Neingo* and T. Tholana*

Synopsis

Mining companies globally are currently facing severe economic and financial challenges. In addition to global challenges, the South African mining industry has to face other operational challenges that are unique to the country and which threaten the survival and competitiveness of the industry. Profit margins are being squeezed by rising costs and decreasing commodity prices, while labour productivity is greatly affected by intermittent labour unrest. This paper analyses how the South African gold mining industry has performed pre-, during, and post the global financial crisis of 2008. The competitiveness of the industry in terms of labour productivity and industry cost curve position is analysed for the period 2006–2013 to assess the impacts of both the global financial crisis and labour unrest. An analysis of the South African gold mining industry is presented at company as well as mine level. Productivity measure in this paper is limited to labour productivity, in line with limited reporting on productivity. All the data analysed was obtained from the public domain.

Keywords

gold mining sector, competitiveness, productivity, industry cost curve, labour availability, labour utilization.

Introduction

South Africa dominated the world as the number one gold producer until 2009 when China took that position, and today South Africa ranks fifth after China, Australia, Russia, and the USA. According to Statistics South Africa (2009), South Africa still had about 30 years of production in the gold sector, a forecast that fluctuates based on the day's modifying factors. Although South Africa has a comparative advantage in terms of minerals endowment, there are challenges faced by the mining industry that hinder translation of the comparative advantage into a competitive advantage. The industry continually seeks to address these challenges to remain competitive in global markets while addressing national and community needs.

Neingo and Cawood (2014) argued that stakeholders in mining are continually concerned with mine management's ability to translate use inputs effectively into quality outputs. They further suggested close monitoring of effective utilization of people, materials, and money to obtain maximum production of commodities under given conditions. Productivity is measured in various ways, including unit cost, output per employee, and output per unit capital equipment.

The impacts of various challenges, including escalating costs, labour availability, and labour utilization on the mining industry, and the gold sector in particular, are quite evident. This is confirmed by EY (2014), who state that productivity on cost and volume bases has been declining since 2000, and various companies have engaged in different cost-cutting exercises. EY (2014) emphasized that the mining industry needs to regain ground lost over the supercycle; innovate to recover/gain competitive advantage; and counteract rising wages, hence the need to boost productivity. This paper discusses trends in productivity for South African gold mines and gold mining companies between 2006 and 2014. Productivity reporting is not yet fully practiced and where done, it is done in silos. This paper aims to provide a guideline to where the South African gold sector stands in terms of productivity, over a time period that captures the impacts of (1) the global financial crisis and (2) labour unrest. The choice of the gold sector was in line with its labour-intensive nature as well as to provide boundaries to the paper.

Due to variation in reporting, the paper discusses unit costs for AngloGold Ashanti, DRD Gold, Gold Fields, Harmony Gold, Pan Africa Resources, Sibanye Gold, and Village Main Reef. These were selected because they are listed companies, therefore their data is easily obtained from the public domain. The companies were also taken as a proxy for the South African gold mining industry due to their level of market capitalization. However,

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only AngloGold Ashanti and Harmony Gold reported labour productivity between 2010 and 2014. Therefore, trends in labour productivity were presented for the period reported. This practice in reporting is not strange to the mining industry, as Neingo and Cawood (2014), citing Strassman (2004), argued that the traditional financial reporting leans towards the interests of financiers.

Challenges facing the South African gold sector

Despite the historical contribution of the South African gold mining sector to both the global and national economy, the sector is faced with various technical, economic, social, and operational challenges that are seeing South Africa slowly losing its competitiveness in the global gold mining industry. In addition to the global challenges, the South African gold mining industry has to face challenges that are unique to the country.

Gold price volatility

Deloitte (2014) stated that even though mining companies are not strangers to commodity price volatility, the impact on the mining industry has recently been severe owing to declining economic growth in China. This volatility due to global economic conditions, which are beyond the control of South African companies, makes strategic planning challenging. This is because the gold price is a major value driver and price volatility has an adverse effect on revenue, cash flows, profitability, and mineral asset values. In addition to price volatility the gold price has been steadily declining. At the time of writing this paper the gold price was US$1189.95, 65% lower than its 10-year peak of US$1826.80 per ounce in September 2011, as shown in Figure 1.

The gold price volatility that is evident in Figure 1 and the current price decline is not sustainable for gold mines and if this situation is prolonged for longer, most operations will be mothballed. In South Africa, the declining gold price in US dollars is somewhat offset by a weakening rand, resulting in a relative increase in the rand price. However, despite this, real value may not be realized because of increasing costs and other constraints to productivity.

Escalating costs of production

Meadows *et al.* (1992), as cited in Müller and Frimmel (2010), mentioned that in mining the relationship between ore grade and energy consumption in mineral processing is exponential. Mudd (2007), cited in the same paper, proved that this general exponential relationship also holds for gold mining. This means that the lower the ore grade, the greater the consumption of energy, reagents, water, and other consumables per unit of gold produced. This relationship is also proved on most typical industry cost curves: assuming everything else being equal, mines producing a higher head grade run-of-mine (ROM) are usually positioned in the lower quartile of the cost curve. This increased consumption, coupled with a general increase in input costs (and electricity in particular for South African mining operations), exponentially increases gold production costs. Figure 2 shows Eskom's historical tariff increases from 2007 to 2017.

The increasing electricity cost has recently been compounded by the current erratic supply, which further threatens the productivity and viability of the energy-intensive gold mining sector. In addition to energy costs, the costs of other major inputs to the mining process have been escalating.

Declining gold resource grade

Gold mining in South Africa started more than a century ago in the Witwatersrand goldfields. Most of the high-grade gold deposits have been exhausted and mines now exploit lower grades. This means that optimization is required to derive optimal value from the remaining low-grade and deep-lying deposits (Musingwini, 2014). Figure 3 shows the arithmetic mean of the ore grade of the main gold-mining countries.

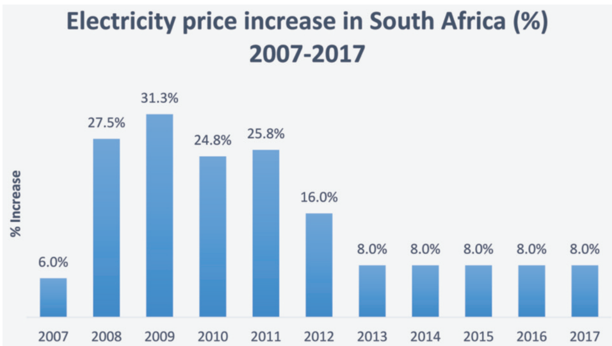

Figure 2—ESKOM electricity price (Eskom Holdings, 2013)

Figure 1—Historicl gold price (adapted from GoldPrice, 2015)

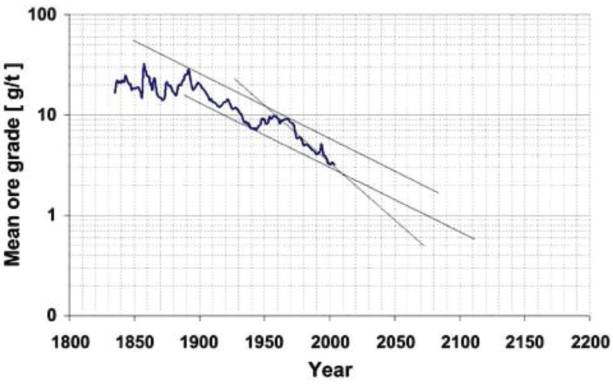

Figure 3—Arithmetic mean of the gold grade in ores mined in South Africa, Australia, Canada, Brazil, and the USA, 1830–2004 (Müller and Frimmel, 2010)

Figure 3 indicates that gold ore grades have constantly declined over the past 80 years. Müller and Frimmel (2010) predicted that if only the decrease in gold grade since 1968 is plotted and extrapolated into the future the average gold grade will be as low as 0.9 g/t by 2050. This means that most current gold operations will be marginal. Musingwini (2014) mentioned that the average grade of South African gold mines has declined from approximately 12 g/t in the 1970s to approximately 5 g/t currently. Figure 4 shows the average recovered gold grade for South African mines from 2004 to 2013.

The steady decline in gold grades is evident in Figure 4. The graph shows a decrease from 5.15 g/t in 2005 to 2.91 g/t in 2013. This combination of declining resource grades, declining gold price, and increasing cost – the key value drivers – severely impacts the competitiveness of the South African gold mining industry. These challenges are further compounded by increasing depth of the mines, which further escalates the cost of production.

Depth and mining method

Over 95% of primary gold production in South Africa comes from underground mines. The orebodies in these mines are narrow and further characterized by geological discontinuities, preventing the application of mechanization and automation. South Africa hosts the worlds' deepest mines, which reach depths of close to 4 km. The Chamber of Mines of South Africa (2015) attributes the ability to mine to greater depth in South Africa to a lower geothermal gradient compared to other parts of the world. However, temperatures of up to 60°C have been experienced at the stope faces in some of the South African gold mines, necessitating the extensive use of ventilation and refrigeration to cool the working environment. At these depths, the cost of ventilation, coupled with the cost of rock support necessary to stabilize the working environment, are among the cost drivers. Seismicity also affects productivity in South Africa's deep gold mines. According to Mining Technology (2015), 'production at TauTona fell to 409 000 ounces in 2007, down from 474 000 ounces in 2006, due to increased seismic activity'

The South African gold mining sector is currently locked into existing mine designs and methods, which are producing at near full capacity. The existing mining methods are cyclic, non-continuous, and rely on equipment, such as scraper winches, that has limited capacity and efficiency. Arguably, productivity in the South African gold sector may be improved in the short to medium term with the existing mining methods. Therefore, the sector requires extensive research to review mine planning, design, and optimization as well as alternative mining methods to achieve sustainable productivity. In addition to the depth, most South African gold deposits are narrow reefs which makes mechanization difficult, hence the sector is highly labour-intensive.

Labour issues

The South African gold industry is characterized by inherent technical constraints such as the stoping width (about 1 m). Equipment is selected to suit the stoping width, causing the industry to continue being highly reliant on labour. Labour availability and utilization are becoming increasingly important, given the unionized nature of the mining industry. Tying in with the conventional mining methods used, depth, and low application of technology, labour productivity is affected by travelling time to the workplace (stope face), operator efficiency, mechanical availability of equipment, and non-input factors such as technological progress (Syed and Grafton, 2011). In some deep-level gold mines it takes up to an hour to travel from surface to the working place.

Several authors, including Roussos (1996) and Aljuhani (2002), classified human factors affecting productivity as:

➤ Labour availability and utilization
➤ Labour/management relations
➤ Labour/labour relations
➤ The crew quality.

Labour utilization is further dependent on the availability of equipment. However, while equipment availability is often fairly accurately predicted, based on some measured/real data, labour utilization is often not measured and estimations are made. Further issues affecting South Africa such as HIV and AIDS compound the availability of labour in the gold mining sector. Pan Africa Resources (2013) has shared concern about absenteeism, stating that 'at Barberton Mines, the management of absenteeism and sick leave remains a

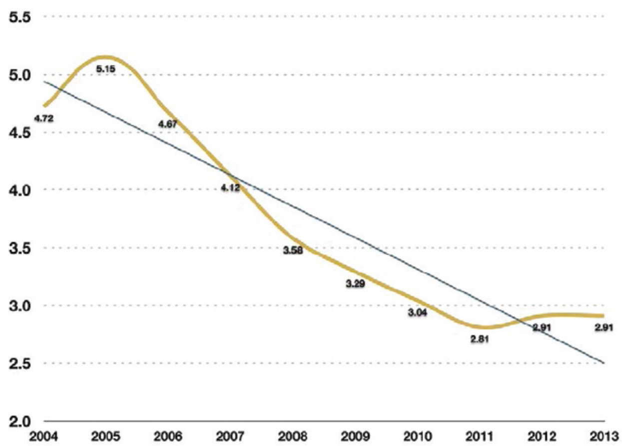

Figure 4—South African average gold grades (Y-axis) against production year (X-axis) (Chamber of Mines of South Africa, 2014)

time-consuming function, necessary to ensure continuous, safe mining. Approximately 91 employees (3.6% of the workforce) were either sick or absent each working day.'

In 2011, industrial action took place in the form of strikes in the coal and gold sectors. However, labour representatives and the Chamber of Mines handled the wage negotiation process smoothly, and in both cases two-year wage agreements were signed (Chamber of Mines of South Africa, 2010/2011). In 2012, there was widespread labour unrest in the mining industry, especially in hard-rock mining, and which was more pronounced in platinum mining.

Without oversimplifying or ignoring the root causes of these incidents of unrest, they are an indication of the lack of trust between management and employees. With continued union pressures and labour issues, lack of trust extends further back among employees, which has led to some union members losing trust in union representatives. Therefore, there is a need to achieve employee and organizational goal alignment. While the root causes of this labour unrest remain largely unknown, it is important for mining companies to establish and report effective measures of productivity that *inter alia* take cognisance of such human factors.

In cases where agreements between unions and companies were reached to increase salaries, these agreements have proved to be beneficial, but only in the short to medium term. Companies invested in, and succeeded with, the adult basic education and training (ABET) programmes. However, it is clear that the recent labour unrest is also an indication of the conflict between companies' goals and expectations of employees, which are perhaps amplified by lack of education. Roussos (1996) stated that lack of education among workers is an important barrier to productivity because poorly educated employees have:

➤ Low literacy rates
➤ A low skill base
➤ Lack of understanding of business principles
➤ Lack of understanding of how workers fit into a productive workplace, or why productivity is important.

Roussos (1996) further argued that some cultures/languages/values can also hinder productivity. These include, but are not limited to:

➤ The lack of common values
➤ Different attitudes/cultures to/ of work
➤ Different political/ideological values and beliefs
➤ Linguistic barriers to effective communication.

Overall, most mines experience high variances in the time workers are paid for and the actual productive time. As observed at several mines visited, shorter effective shift time is caused by longer meetings than planned (and late starts thereof), long travelling time to the work stations, unauthorized breaks due to lack of self-discipline, and early work stoppages, to mention but a few.

Political, social, and environmental issues

The legal safety-related actions taken by the Department of Mineral Resources (DMR) against noncompliance with regulations such as Section 54 and/or Section 55 of the Mine Health and Safety Act 29 of 1996 also affect productivity. Therefore, some delays such as safety meetings cannot be avoided, but time spent can be improved, for example by giving incentives which incorporate health and safety, productivity, as well as degree of plan compliance, provided the plan is good. While these losses in production translate into fewer tons per employee, they also affect unit costs because during closure fixed costs such as labour have to be paid while no revenue is generated.

In 2011, the Youth League of the African National Congress called for nationalization of the mines, which caused concern among foreign investors (Chamber of Mines of South Africa, 2010/2011). In 2009 and more recently in 2015, South Africa experienced 'xenophobic attacks'. The short- to medium-term impacts of these actions include damage to socio-economic relations with other countries, leading to a weaker rand. However, in the long term these could affect investors' confidence. While discussions are ongoing to resolve these issues, there is a need for a more sustainable solution.

Productivity trends

Unit costs

Jaguar Mining (2010), as cited in Tholana *et al.* (2013), stated that industry cost curves are a useful analysis tool because they provide a trend in costs as a mine matures. The report also mentioned that industry cost curves provide an internal benchmark of performance to allow for comparison of mines/companies in the same industry. Given the maturity of most South African gold mines, a cost curve is useful in analysing the competiveness of different mines in the sector. Industry cost curve were constructed using an algorithm in Tholana *et al.* (2013). As mentioned previously, only listed companies with operations in South Africa were considered because of the availability of data and the appropriateness as proxy for the South African gold mining industry. A list of producing mines was compiled, from which data was collected. In constructing the cost curves, two data-sets for each mine were collected from the respective companies' annual reports; production in ounces and cash cost per ounce produced.

Gold mine industry cost curves

Figures 5 and 6 show the industry cash cost curves for the gold mines for the period 2006 to 2009 and 2010 to 2014, respectively. On each cost curve, different colours are used to distinguish mines from the same company, and the average gold prices published by Kitco (2015) for the different years are indicated by the red dotted lines.

The lower half of most of the cost curves is dominated by large gold producers such as AngloGold's Mponeng and Great Noligwa, and Gold Field's Driefontein and Kloof, while the small gold producers dominate the upper half of the curve. This is because of the principle of economies of scale, which states that the more the production, the lower the unit cost of production. This is because of a high fixed cost component associated with the business of mining, which means that whether a mine produces or not there are high fixed costs that are still incurred, including ventilation, pumping costs, and in some instances salaries. In the middle of the cost curve there tends to be a mixture of low- and high-volume producers. This is because the small producers have high-grade deposits while the large producers have low-grade

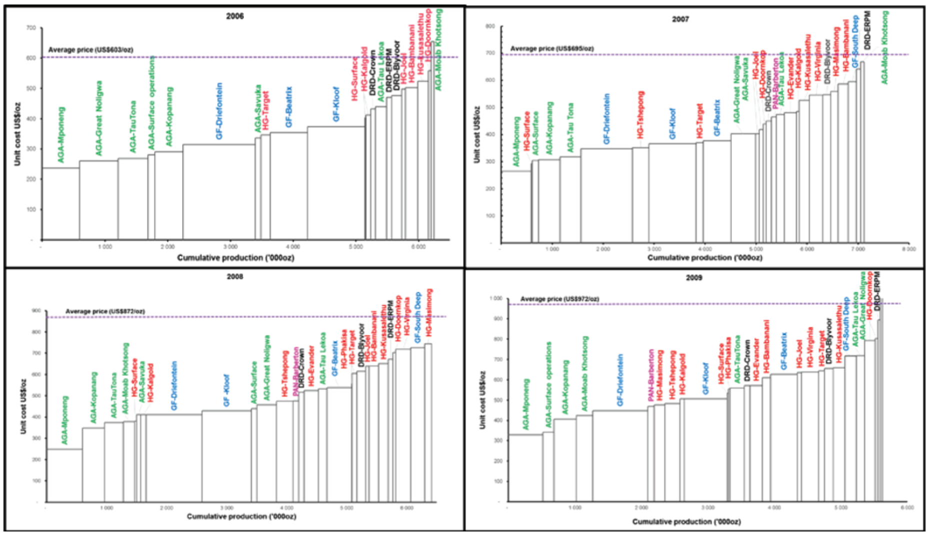

Figure 5—Cost curves for gold mines for 2006–2009

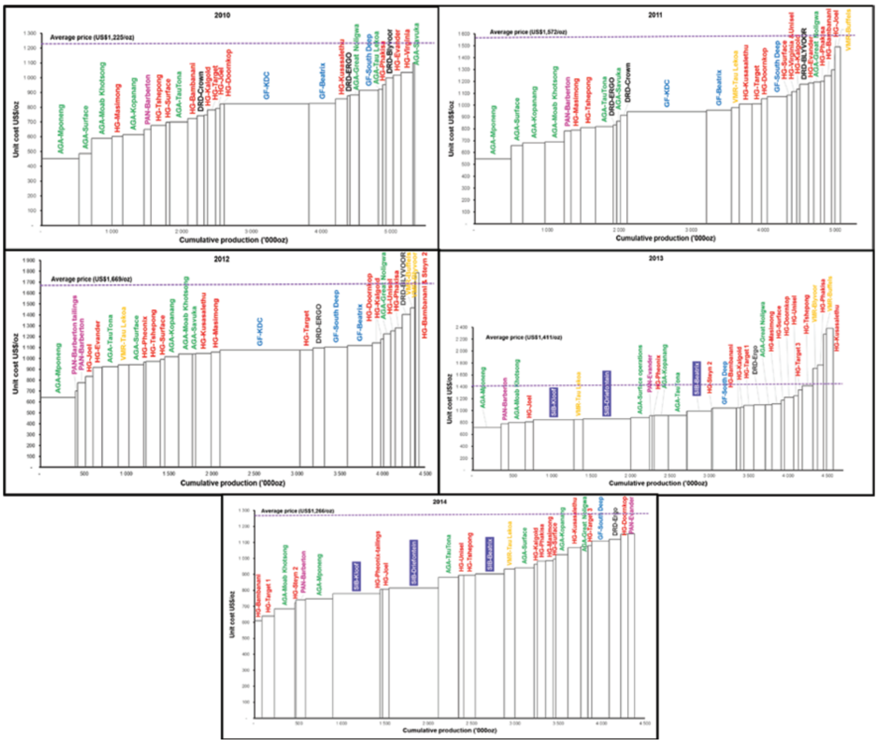

Figure 6—Cost curves for gold mines for 2010–2014

deposits, hence ultimately they would have comparable unit cash costs of production. Factors that have been identified to determine the position of a mine on an industry cost curve are:

- Mponeng, Moab Kotshong, and Kopanong are relatively low unit cost producers because of their relatively high-grade deposits. Moab Khotshong is also a relatively new operation compared to the rest of the gold mines in South Africa
- Barberton is also a lower unit cost producer because of its relatively high head grade
- Tailings treatment operations such as Harmony Gold's surface/Phoenix operations and AngloGold's surface operations are lower unit cost operations because of the lower mining and transportation costs associated with these operations compared to underground mines.

Gold mining companies' industry cost curves

The cost curves for gold mining companies were constructed from total production and weighted averages unit cash costs. A particular company's production was calculated from the total production of the individual mines in the analysis, while the unit cash costs were weighted averages calculated as:

$$\text{Unit cash cost} = \frac{\sum(\text{mine 1's production} \times \text{mine1's unit cost, mine 2's production} \times \text{mine 2's unit cost} \ldots)}{\sum(\text{mine 1's production, mine 2's production} \ldots)}$$

The consolidated production and cash cost data were not used because some gold companies have operations outside South Africa whose contribution is included in their consolidated annual reports, so using data from consolidated reports would not be representative of the South African gold mining sector. Figures 7 and 8 show the cash cost curves for the gold companies for the period 2006–2009 and 2010–2014, respectively. The average gold prices published by Kitco (2015) for the different years are indicated on each cost curve by the red dotted lines.

Gold production

Figure 9 shows the production by companies included in the analysis.

Figure 9 shows that the challenges discussed earlier have caused gold production to steadily declining from 2007 to 2014.

Labour productivity

Gold production is affected by various factors along the mining value chain, including accuracy of drilling and blasting, precision of short-term planning input factors, grade control, and quantification and minimization of mining losses from the resource to the final product. Labour productivity in the context of this paper refers to the metric ounces (oz) per total employee costed (TEC). According to the Chamber of Mines of South Africa (2014), the mining industry has generally seen a decline in productivity over the last 10 years.

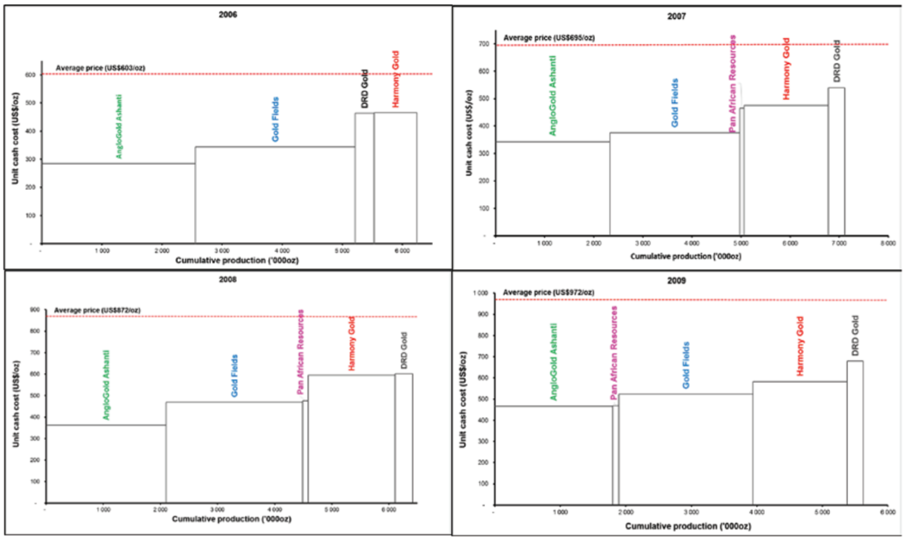

Figure 7—Cost curves for gold companies for the period 2006–2009

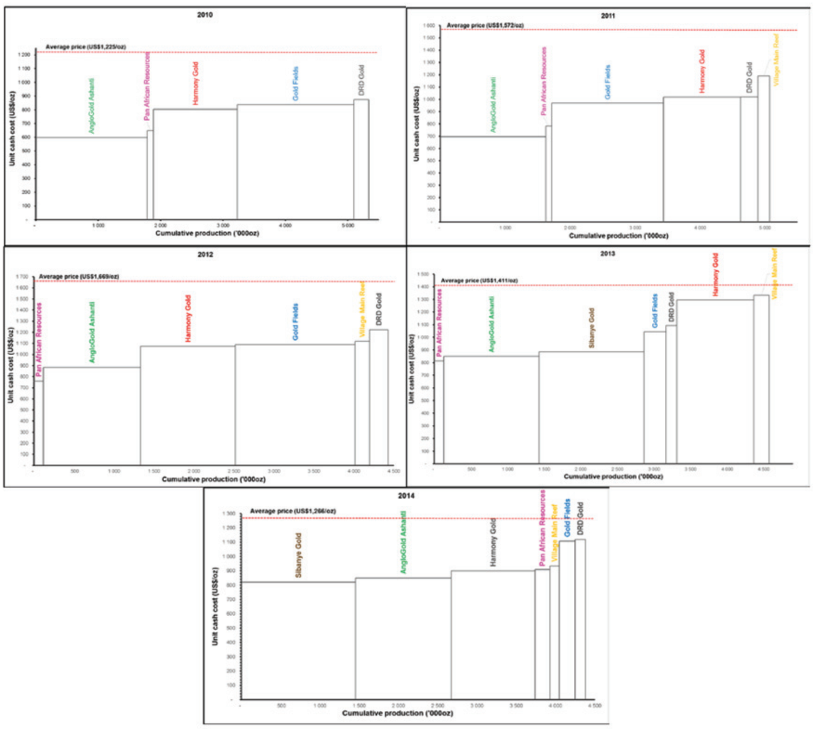

Figure 8—Cost curves for gold companies for the period 2010–2014

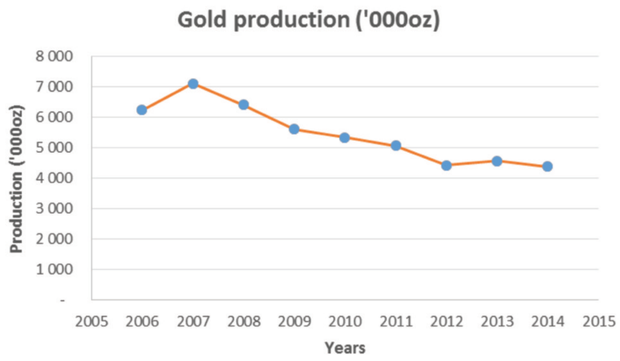

Figure 9—Cost curves for gold companies for the period 2006–2009

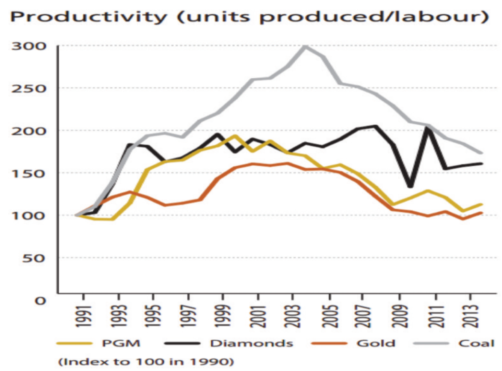

Figure 10—Labour productivity in the South African mining industry (COMSA, 2014)

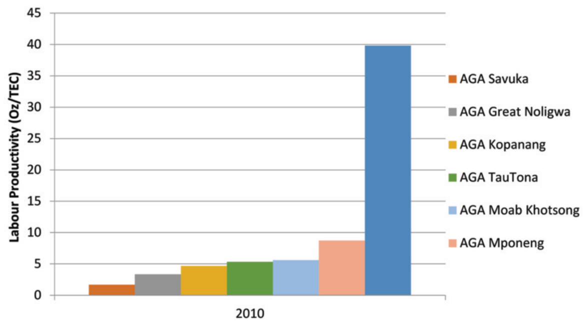

Figure 11—Labour productivity 2010 (various annual reports)

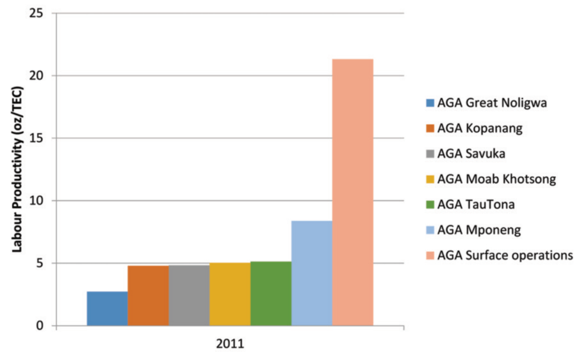

Figure 12—Labour productivity 2011 (various annual reports)

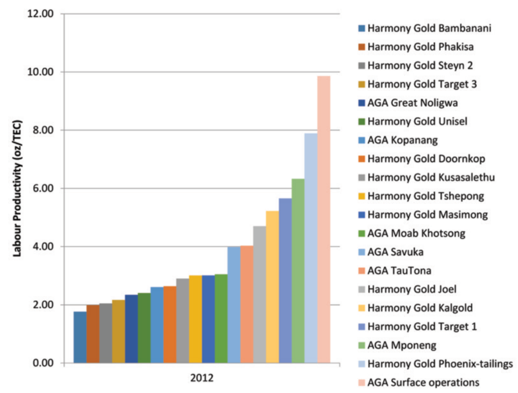

Figure 13—Labour productivity 2012 (various annual reports)

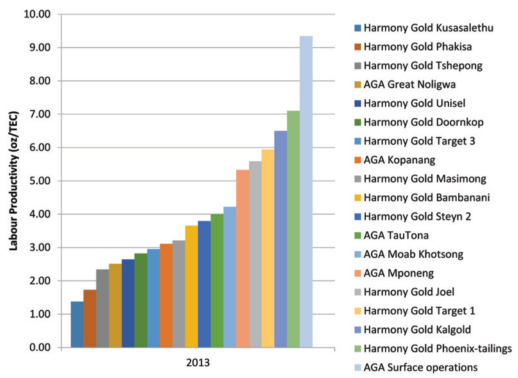

Figure 14—Labour productivity 2013 (various annual reports)

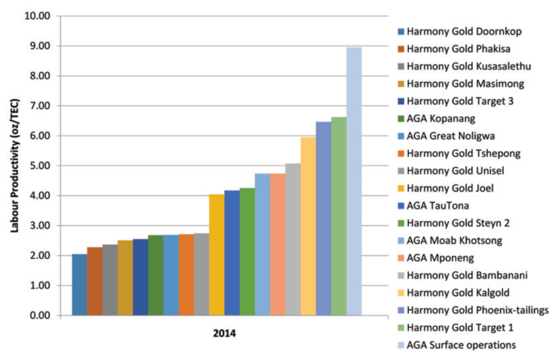

Figure 15—Labour productivity 2014 (various annual reports)

Figure 10 shows the trend in productivity for the major commodities produced in South Africa, with gold recording the lowest productivity.

The over-reliance on labour in gold mines, due to lack of advanced technology that can be integrated into the existing mining layouts, cannot be over-emphasized. While some companies analyse their company and operations productivity periodically, it is important to establish how these mines compare with other mines. Figures 11–15 show the positions of mines in terms of ounces produced per total employee costed.

During the period 2010–2014, Anglo Gold Ashanti recorded major drops in productivity of 78%, 46%, and 43%

at Surface Operations, Mponeng, and Kopanang, respectively. Among other drivers of declining productivity noted by Anglo Gold Ashanti (2014) included safety-related stoppages, unscheduled shaft maintenance at Mponeng, the 5.3 magnitude earthquake affecting Vaal River mines, as well as the decreasing grade of material sourced from marginal ore dumps for Surface Operations.

During the same period, Harmony Gold experienced both increments and declines in productivity at their operations. The highest increments noted included 187%, 108%, and 17% at Mbambanani, Steyn 2, and Target 1 mines, respectively. Declines of 23%, 17%, and 14% were experienced at Doornkop, Masimong, and Kalgold, respectively. Kalgold was noted to have experienced a decrease in productivity of 28% between 2013 and 2014. Reasons noted by Harmony Gold (2014) for the major decline at Kalgold were the delay in accessing higher-grade ore blocks and the fall in milling grades following heavy rains. Overall, Harmony Gold (2014) noted approximately 40 000 ounces lost due to safety-related stoppages, infrastructure and equipment failure, as well as insufficient mining flexibility as key challenges affecting production and productivity.

Conclusions and recommendations

Gold mining continues to be a major contributor to the South African economy. The country has a great comparative advantage in terms of gold mineral resource endowment, but the gold mining sector is faced with several challenges that prevent South Africa from turning this comparative advantage into a competitive advantage. Due to globalization, South Africa is not immune to global economic forces. The 2008 global financial crisis and the volatility and declining gold prices, declining grades of gold deposits, and access to capital are some of the global challenges faced by the South African gold sector. In addition to the global challenges, South African operations are also faced with challenges unique to the country that severely impact the sector's productivity, and hence profitability and sustainability. Such challenges include industrial action, political, social, and environmental issues, high electricity cost and erratic supply, the Department of Mineral Resources' Section 54 safety-related stoppages, and technical challenges associated with deep-level mining. All these global and local challenges have led to operating cost escalation, resulting in profit margin squeezes, decreased productivity, and possible future mothballing of some gold mines.

South Africa requires gold mining companies to develop innovative extraction strategies to ensure the long-term competitiveness and viability of the sector. Mining companies need to move away from traditional mine planning practices and start to incorporate optimization techniques into their mine plans. This includes shifting from the deterministic way of mine planning to probabilistic mine planning techniques.

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Critique of the South African squat coal pillar strength formula

by M. Mathey* and J.N. van der Merwe*

Synopsis

The South African squat coal pillar strength formula, developed by researchers from the Chamber of Mines Research Organization in the 1980s, predicts an exponential increase in coal pillar strength once a critical width-to-height ratio of 5 is exceeded. The arguments that have been proposed in favour of this formula are discussed critically in this paper.

Field experience with squat pillars in the USA and evidence from published physical, analytical, and numerical model pillar studies corroborate the fact that a squat effect is unlikely to occur in coal pillars at width-to-height ratios less than 10. Furthermore, an exponential increase in peak strength of coal pillars does not exist. An alternative design criterion for squat pillars in South Africa is therefore suggested.

Keywords

squat pillars, coal pillar strength, bord-and-pillar mining.

Introduction

Five decades of coal pillar research in South Africa have produced a wealth of information on the mechanisms contributing to the strength and failure of *in-situ* coal pillars and accompanying phenomena. The relevant knowledge was sourced from, and for, a South African coal mining environment that remained largely unchanged in the second half of the 20th century. The depth of mining seldom exceeded 250 m, with mined seam thicknesses averaging 3 m and typical pillar width-to-height ratios (*w/h*) of 3–4.

It is therefore not surprising that comparatively little research has been directed at the prediction of coal pillar performance in deeper deposits, where squat pillars of larger width-to-height ratios will be required to support the overlying strata. However, a thorough understanding of squat pillar performance will be necessary should the remaining deep coal deposits of the country be extracted in future, for instance in the Waterberg or Ermelo coalfields.

A squat coal pillar strength formula has already been proposed in South Africa. The origin of this formula will be discussed in some detail in the following sections and its appropriateness will be critically reviewed against a great amount of evidence from relevant squat coal pillar research conducted internationally.

Squat pillar strength in South Africa

The background to the South African squat coal pillar strength formula is provided by the original work of Salamon and Munro (1967), who predicted the strength of slender pillars of width-to-height ratios of up to 3.6 to be:

$$\text{Strength} = 7.2\ w^{0.46}/h^{0.66}\ \text{MPa} \qquad [1]$$

The formula implies that for a constant pillar height h and increasing pillar width w (*i.e.* increasing width-to-height ratio), the strength of pillars increases regressively.

Salamon and Oravecz (1973) commented that Equation [1] 'underestimates the strength of pillars when [the width-to-height ratio] is greater than 5 or 6. There is some evidence that when pillars have a width-height ratio exceeding, say, 10–12, they do not fail under any practically possible load'. Salamon (1982) proposed a separate strength formula for squat pillars, which allowed for a rapid increase in pillar strength once a critical width-to-height ratio is exceeded:

$$\text{Strength} = 7.2\,\frac{R_0^{0.5933}}{V^{0.0667}}\left\{\frac{0.5933}{\epsilon}\left[\left(\frac{R}{R_0}\right)^{\epsilon}-1\right]+1\right\}\ \text{MPa,} \qquad [2]$$

where R_0 is the critical width-to-height ratio for the onset of squat pillar strength, R is the width-to-height ratio of the squat pillar, V the pillar volume, and ϵ a parameter which controls the rate of strength increase with increasing R in squat pillars.

Salamon and Wagner (1985) subsequently published further explanations as to the background of Equation [2]. They thought to find evidence in theoretical and experimental studies of cohesionless, granular materials, as

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well as in compression tests on sandstone model pillars, 'that the strength of squat pillars increases very rapidly, perhaps even exponentially, with increasing width-to-height ratio, once a certain value of the ratio has been exceeded' (Salamon and Wagner, 1985). It was suggested that $R_0 = 5$ would perhaps be a reasonable estimate of the critical width-to-height ratio for the onset of squat pillar strength, and a rate of strength increase of $\epsilon = 2.5$ was selected to design experimental panels in collieries.

Equation [2] predicts significant increases in the strength of squat coal pillars as compared to the original Salamon and Munro (1967) strength formula, which is demonstrated in Figure 1. Consequently, it improves the extraction rates for bord-and-pillar mining layouts at greater depths.

Madden (1990) undertook research to substantiate Equation [2] with further evidence. He reported on an extensive laboratory testing programme, in which five sets of sandstone model pillars with sizes ranging between 24–100 mm and width-to-height ratios of 1–8 were loaded in compression between steel platens. The results were such that up to $w/h = 5$ or 6, the strength of the model pillars increased approximately linearly, and thereafter more rapidly with further increasing width-to-height ratio. Madden's laboratory test results will be discussed in further detail in a later section.

Madden (1990, 1991) also investigated the performance of squat coal pillars at Piet Retief and Longridge collieries in KwaZulu-Natal. By means of boreholes drilled horizontally through the pillars, he established the maximum depth to which stress-related fractures penetrated into the pillars. He concluded that the fracture penetration depth was limited to the pillar skin only, and the large intact cores of those pillars suggested that the overall strength of the pillar would be significantly higher than predicted by the formula of Salamon and Munro (1967). This was thought to substantiate the validity of the squat pillar strength formula. Madden (1991) reported further that extensive field trials at Hlobane and Piet Retief collieries had shown that the squat pillar formula gives stable pillar dimensions. However, more accurate estimates of the critical parameters R_0 and ϵ could not be established from this research.

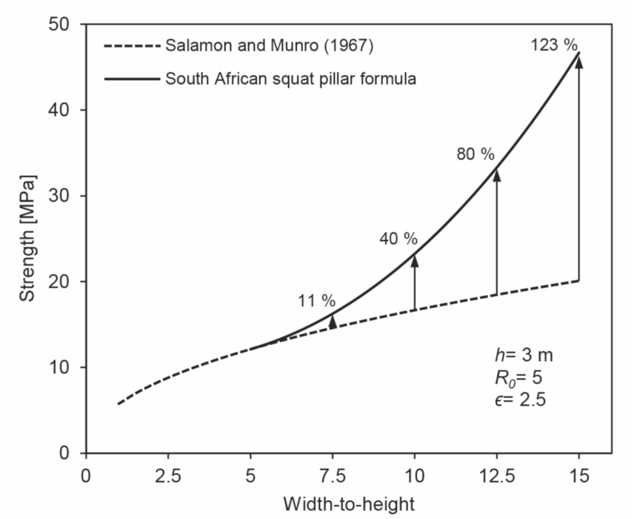

Figure 1—Prediction of coal pillar strength according to the squat pillar formula

During the past 30 years since the introduction of the squat pillar strength formula, no pillar with $w/h \geq 5$ has ever been observed to collapse in South Africa. The largest collapsed width-to-height ratios reported in the literature are 3.6 and 4.3 in coalfields of *normal* and *weak* coal pillar strength respectively (Van der Merwe and Mathey, 2013b).

However, the absence of squat pillar collapses does not necessarily confirm the validity of the squat pillar formula. A great number of squat pillars in South African collieries are simply overdesigned and do not fail due to the very high safety factors employed. Figure 2 plots the safety factors of all 84 squat pillar cases stored within the South African databases (Van der Merwe and Mathey, 2013b). The strength has been conservatively calculated with Equation [1] and would be even greater if calculated according to Equation [2]. Note also that none of the squat pillars were loaded beyond 11 MPa.

The strength of coal pillars revisited

The assumption of an exponential increase in strength of squat pillars with $w/h > 5$ is certainly unique and not free from reasonable doubt. To the contrary, there is substantial evidence available from field observations on squat pillars *in-situ* and from model pillar studies that argue against an exponential strength increase. This evidence will be presented in the following sections of this paper.

Beforehand, it may be advantageous to begin the critique of the South African squat pillar formula with a detailed review of the arguments that have been proposed in its favour. These have been outlined in the previous section and may be summarised as follows. Firstly, it has been said that the strength of squat pillars in collieries of KwaZulu-Natal appears to be higher than predicted by the original Salamon and Munro (1967) formula. Secondly, studies of model pillars consisting of cohesionless materials or intact rock materials *other than coal* have indicated a rapid strength increase above a width-to-height ratio of 5. And thirdly, pillars with $w/h = 10$ or 12 have been observed to withstand any practical load, *i.e.* they do not fail. The following sub-sections

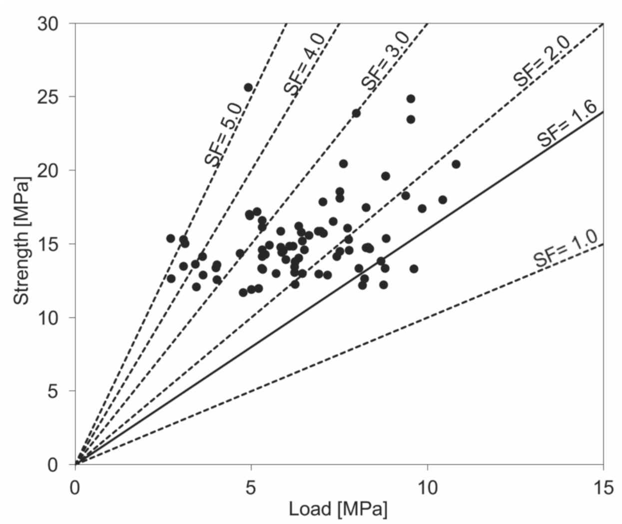

Figure 2—Safety factors of squat pillars ($w/h \geq 5$) in the database of stable cases, calculated with the Salamon and Munro (1967) strength formula

will provide alternative explanations for these observations, from which it will be seen that an exponential strength increase in squat coal pillars is not necessarily evident.

The strength of squat pillars in KwaZulu-Natal appears to be greater than predicted by the Salamon and Munro (1967) formula

The Salamon and Munro (1967) pillar strength formula, and any subsequent update of the same (Madden, 1991; Van der Merwe, 2003; Salamon, Canbulat, and Ryder, 2006; Van der Merwe and Mathey, 2013c), do not represent the actual strength of coal pillars in South Africa. They merely predict the average strength of a specific group of coal pillars, namely those that have collapsed. The advantage of these empirical formulae is that they are very useful as design guidelines to prevent further collapses. However, they do not necessarily predict the actual strength of any given coal pillar.

There is sufficient evidence to assume that the actual strength of coal pillars may be highly variable. Differences in pillar strength between coalfields (Van der Merwe and Mathey, 2013b, 2013c) and groups of seams (Salamon, Canbulat, and Ryder, 2006) have already been proposed, based on the statistical characteristics of collapse cases. But even within areas with the same failure characteristics, it is estimated that only about 10 % of all pillars with a predicted safety factor of unity actually collapse (Van der Merwe and Mathey, 2013a). This in turn suggests that the average strength of coal pillars may be higher than that assumed by the Salamon and Munro (1967) formula.

A good idea of the possible variation in the actual coal pillar strength can be gained from the compression tests on large-scale model pillars conducted at Witbank, Usutu, and New Largo collieries (Bieniawski, 1968a; Wagner, 1974; Van Heerden, 1975). The corresponding pillar strength equations are reproduced in Figure 3. Van Heerden (1975) in particular undertook great efforts to produce a stress environment in his model pillars, which resembled those of full-size mine pillars. The result of his tests at New Largo colliery (Figure 3) suggest that the site-specific strength of coal is about three times higher than predicted by the back-analysis of collapse cases (Equation [1]).

In light of the above, it is not surprising that the inferred strength of partially fractured squat pillars in collieries of KwaZulu-Natal appeared to be higher than the strength predicted by the Salamon and Munro (1967) formula. The obvious conclusion from this observation, however, is not that there must have been some kind of squat effect that accounts for the discrepancy. It is rather that the site-specific strength of the investigated coal pillars may simply be different from the prediction made by a formula that is derived from a different coal mining environment. Madden's observations on sidewall fracturing in squat pillars could have been an argument for an increased squat pillar strength only if the validity of the Salamon and Munro formula (1967) for the relevant collieries had first been established for fracture observations in slender pillars.

Evidence from compression tests on model pillars designed from rock materials other than coal

One may distinguish two different failure modes in pillars (Figure 4). Firstly, *brittle* failure, which has been observed for mine pillars in South Africa up to width-to-height ratios of at least 4. Brittle pillars exhibit a distinct peak strength σ_p and a subsequent strength drop to a residual level σ_r, if additional strain is imposed on the pillar. The failure process is associated with an abrupt or gradual loss of cohesion of the pillar material.

The second category is a *quasi-ductile* failure mode. Here, the pillar may also lose its cohesion entirely within the failure process, but it maintains or even increases its load-bearing capacity with increasing strain (Figure 4). This failure mode can occur only in pillars with sufficiently large width-to-height ratios, which allow high lateral confinement stresses to be generated within the fractured pillar. While the load-bearing capacity of brittle pillars is dominated by the cohesive strength of the material, pseudo-ductile pillars

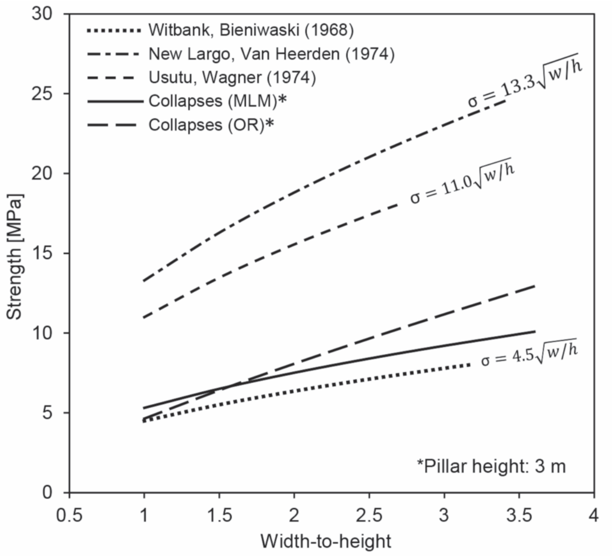

Figure 3—Coal pillar strength tested and back-calculated in South Africa

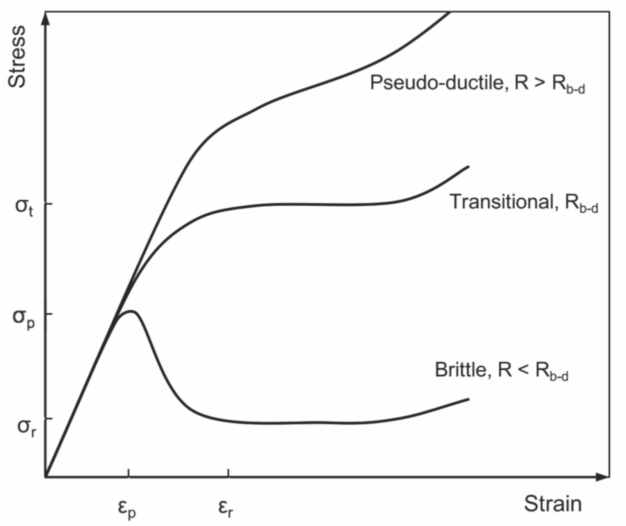

Figure 4—Stress-strain relationship for pillars of varying width-to-height ratios

obtain their seemingly unlimited load-bearing capacity from the frictional shear resistance of the fractured material.

The concept of two different failure modes implies that a critical width-to-height ratio R_{b-d} exists, at which brittle failure transits into pseudo-ductile failure. The stress-strain behaviour of such a transitional pillar is depicted in Figure 4.

It is obvious that a peak pillar strength criterion such as Equations [1] and [2] can be meaningful only for pillars that fail in a brittle manner. Pillars that perform in a quasi-ductile manner do not exhibit a peak strength, nor do they pose the risk of abrupt failures, load-shedding, and pillar runs in panels. However, they may allow undue deformations to take place in the surrounding strata, which contributes to the deterioration of ground conditions. A strain-based design method, such as the ground-response curves for pillar-strata systems (Esterhuizen *et al.*, 2010b) appears to be most suitable for such mining environments.

However, for the development of appropriate peak strength design criteria for brittle pillars it is necessary to identify the critical width-to-height ratio R_{b-d} at which brittle-ductile transition occurs in pillars, and how the peak strength develops in brittle pillars over the full range of $1 \leq R \leq R_{b-d}$.

Model pillars that consist of cohesionless material (*e.g.* a pile of gravel) have been used by Salamon and Wagner (1985) to argue in favour of the exponentially increasing strength of squat pillars. Such cohesionless models are, however, of little use in this regard, as they resemble mine pillars that are already fully crushed. Therefore, they can only simulate the performance of pillars that are in a state of residual strength. From a practical perspective, such crushed pillars would already have undergone substantial deformation and the entries would already been lost.

In the past, a large number of laboratory compression tests have been conducted to observe failure modes and strength trends in model pillars. These tests highlight significant differences between model pillars designed from rock materials (which have been used to argue in favour of the current South African squat coal pillar formula) and those designed from intact coal materials.

Laboratory tests on rock model pillars

Bieniawski (1968b) published the results of compression tests on sandstone model pillars which showed that the relationship between strength and the width-to-height ratio increases linearly up to $w/h = 5$. Further tests at $w/h = 10$ demonstrated that a model pillar of that size could not be broken even at very high loads.

Cruise (1969) also reported on a testing programme involving sandstone model pillars, for which he observed an upward curving peak strength trend for pillar width-to-height ratios between 1 and 6.7. He found that the trend could be best described by a polynomial curve.

Bieniawski and Van Heerden (1975) presented the results of tests on sandstone model pillars, which again showed a linear relationship between the strength and width-to-height ratios up to $w/h = 4$. At width-to-height ratios of 5, 6, and 7.5, the strength of specimens was markedly higher than predicted by the trend for the more slender pillars.

The tests on sandstone model pillars conducted by Madden (1990) showed a linear strength increase for width-to-height ratios of up to 5 or 6, at which point brittle-ductile transition occurred in the specimens. The rapid increase in strength for width-to-height ratios greater than 6, which has been reported from Madden's tests, therefore does not refer to an increase in (brittle) peak strength of the model pillars, but to a pseudo-ductile behaviour of the pillars. The onset of brittle-ductile transition at 5 or 6 may also explain why Bieniawski (1968b) was not able to crush a sandstone model pillar of width-to-height of 10. Yet this observation is in conflict with the peak strength values reported for sandstone model pillars by Bieniawski and Van Heerden (1975).

York *et al.* (1998) reported on compression tests on Merensky Reef model pillars with width-to-height ratios of up to 10. The strength of pillars in the range of 1–6 was found to increase linearly with increasing width-to-height ratio. The test results for higher width-to-height ratios were not reported.

The reviewed tests on rock model pillars agree that the relationship between the strength of pillars and width-to-height ratio for w/h values of up to 5 or 6 may be expressed by a single trend. This trend is linear in most cases. The finding of a critical width-to-height ratio above which pillars start behaving markedly differently may indeed highlight the need for a separate squat pillar design criterion for rock materials. However, the results are inconclusive for the question as to whether squat rock pillars experience a brittle-ductile transition or a more rapid increase in brittle peak strength.

Laboratory tests on coal model pillars

The performance of coal model pillars has been studied in numerous laboratory investigations. Holland (1942) reported on an extensive laboratory testing programme in which coal specimens from different coal seams in West Virginia, USA, were tested at w/h ratios between 1 and 12. He described the results to be very erratic in general, but demonstrated that a linear or regressive increase in specimen strength up to a w/h ratio of 8 fitted the average data well.

However, Holland also noted that some specimens did not fail abruptly. He stated that for these outliers 'there was no point during the application of the load when it could be definitely stated that failure occurred. The bending of tool-steel bearing plates, denting of soft-steel bearing plates, and the distinct imprint of the round hole on the coal specimen suggest that the coal was forced into the plastic state' (Holland, 1942). Yet, upon examination of those specimens it was discovered that the specimens were very fragile and exhibited shear surfaces when broken.

The phenomena described by Holland are exactly those that are associated with the brittle-ductile transition in pillars. In Holland's tests they occurred only for some specimens at w/h ratios between 5 and 12. Holland reasoned that the critical w/h ratio for the transition from abrupt failure to failure through flow or squeezing may be influenced by the generic strength of the coals. Even though not fully conclusive, his tests indicated that the stronger the coal, the higher the critical w/h ratio for the occurrence of flow in the model pillars. In conclusion, Holland therefore commented that 'if these results can be applied to coal-mine pillars, the possibility is indicated that the stronger the coal is on a mine pillar, the wider the pillar must be relative to its height to prevent abrupt failure' (Holland, 1942).

Meikle and Holland (1965) investigated the influence of the contact friction angle between the coal model pillars and the loading platens on the strength of pillars. The study focused on specimens with w/h ratios between 4 and 8. In general it was found that the stronger the interface friction, the higher the strength of pillars at a given width-to-height ratio. However, it was observed that the relationship between specimen strength and the width-to-height ratio increased only regressively, irrespective of the interface friction angle.

The results of Meikle and Holland's tests, together with the results from other published coal model pillar experiments discussed in the following paragraphs, are plotted in Figure 5. It should be noted that for the sake of comparison, all different test results have been normalized to give the same strength for a pillar of w/h = 5. The individual strength trends, which may indicate possible squat effects in the tests, remain unaffected by the normalization. In the study of Meikle and Holland (1965), a squat effect in the form of an exponential strength increase or the brittle-ductile transition was not observed for model pillars with $w/h \leq 8$.

A similar laboratory testing programme was subsequently carried out by Khair (1994) in the USA, who found that the relationship between the strength and shape of coal model pillars with w/h = 4–8 could be expressed by a linear equation for each selected contact friction angle. Again, a squat effect was not observed.

Kroeger, Roethe and Li (2004) also performed compression tests on model coal pillars from different seams in Illinois, USA. They found that the relationship between strength and width-to-height ratio of the specimens can be expressed as a regressively increasing curve for the entire range of w/h = 1–12. The trend for the Murphysboro seam is reproduced in Figure 5. A squat effect was not observed.

In India, Das (1986) tested coal specimens from five different seams at w/h ratios of up to 13.5. He observed that brittle-ductile transition occurred in specimens at very large w/h ratios of around 10. A squat effect in the form of a progressive strength increase is not discernible from his data. The average strength *versus* width-to-height relationship of specimens from all seams follows a linear trend for w/h rations between 1 and 10, as shown in Figure 5.

In South Africa, Madden and Canbulat (1995) conducted a comprehensive testing programme with more than 900

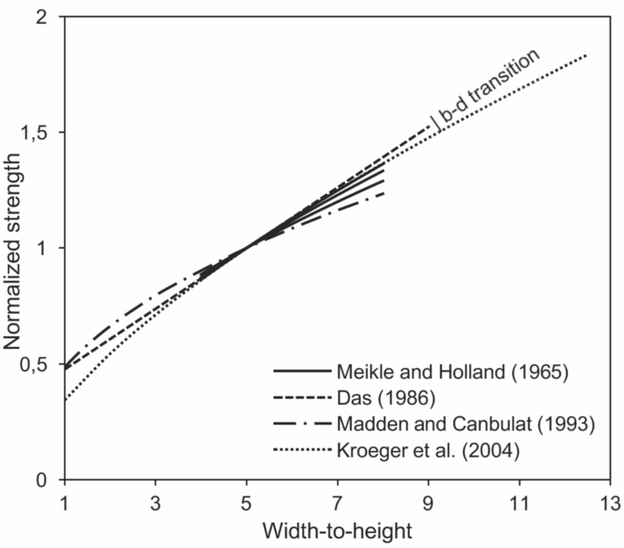

Figure 5—Normalized strength of laboratory coal model pillars

model pillars of different sizes between 25–300 mm and with w/h ratios between 1–8. Coal samples were sourced from 11 collieries in different coalfields. The strengths of all the model pillars were analysed statistically, and it was found that the following equation provided an adequate fit to the data:

$$\text{Strength} = k w^{0.139} / h^{0.449} \text{ MPa} \qquad [3]$$

where k is a strength coefficient, and w and h the specimen width and height in metres. Equation [3] is plotted in Figure 5 in its empirical range, using the average strength factor k = 15.83 MPa for all coals used in the study of Madden and Canbulat (1995), and a specimen width of 100 mm. It should be noted that Equation [3] describes a regressively increasing relationship between strength and the width-to-height ratio of coal model pillars. Neither an exponential increase in strength nor strain-hardening could be identified conclusively in the tested specimens up to w/h = 8.

It is evident from this summary that coal model pillars behave very differently to rock model pillars. The relationship between strength and shape follows a single linearly or regressively increasing trend up to width-to-height ratios of at least 8, in some cases even up to 12, before the brittle-ductile transition can occur. An exponential strength increase has also not been observed for coal model pillars. This raises considerable doubt as to the validity of the South African squat coal pillar formula (Equation [2]). The adoption of the critical width-to-height ratio of 5 for squat effects in coal pillars appears to have been based solely on experience with rock materials, with the assumption that coal behaves similarly. However, the evidence indicates that coal does not behave like the rock samples tested.

Pillars with width-to-height ratios of 10 or 12 do not fail under any practical load

Salamon and Oravecz (1973) stated that coal pillars with very large width-to-height ratios of 10 or 12 do not fail under any practical load. This observation, which has also been made by other researchers (without defining what this practical load would be), initially sparked the idea that squat pillars increase their strength more rapidly than the more slender coal pillars. However, the evidence provided in the previous sub-section strongly suggests that the phenomenon of an apparently unlimited load-bearing capacity of very squat pillars is linked to the brittle-ductile transition in pillars rather than to an exponentially increasing peak strength.

It has thus been demonstrated that not a single piece of evidence exists that unambiguously substantiates the correctness of the South African squat coal pillar formula. The following section will describe different experiences with squat coal pillar strength prediction and design.

Experience and predictions of squat pillar strength internationally

The questions of squat coal pillar strength and coal pillar design at depth have been addressed by various researchers internationally. In Australia, where the strength of slender coal pillars has also been determined from back-calculation of stable and collapsed cases (Salamon *et al.*, 1996), Salamon's suggestion of an exponential increase in squat pillar strength has been adopted without further investigations. Equation [2] has only been modified (Galvin and Hebblewhite, 1995) to meet with the empirical strength equation for slender pillars in Australia at the critical width-to-height ratio of R_0 =

5. The exponential rate of strength increase is the same as in South Africa with $\epsilon = 2.5$.

A unique approach to pillar design at depth was pursued by Sheorey *et al.* (1987) in India, in that a strength formula was proposed that accounts for the influence of virgin stress conditions on the load-bearing capacity of the pillar. The fundamental assumption behind coal pillar strength at depth was that a pillar may be able to retain some of the original horizontal confining stress existing in the coal prior to excavation, depending on its width-to-height ratio and its contact conditions with the surrounding strata. Therefore, with increasing depth, the pillar confinement and vertical load-bearing capacity should increase. The latest available update of the formula, Equation [4], was provided by Sheorey (1992):

$$\text{Strength} = 0.27\sigma_c h^{-0.36} + \frac{H}{150}\left(0.6 + \frac{150}{H}\right)\left(\frac{w}{h} - 1\right)\ \text{MPa} \qquad [4]$$

where σ_c is the compressive strength of 2.5 cm cubes of coal in the laboratory, and all other dimensions (w, h, and H) are in metres. Evidence from failed and stable cases in India suggested that Equation [4] was suitable for design of pillars with width-to-height ratios up to 6.7.

In the USA, more detailed insights into the strength and stability of coal pillars with dimensions far in excess of w/h = 5 have been gained. These will be discussed in the following sections.

Evidence from in situ experience

Bieniawski (1992) reviewed a number of empirical coal pillar strength equations from different researchers and compared their performance to the *in situ* strength of pillars in the USA. He emphasized the suitability of an empirical pillar strength equation that he had derived from the *in-situ* compression tests in South Africa (Bieniawski and Van Heerden, 1975):

$$\text{Strength} = \sigma_c\left(0.64 + 0.36\frac{w}{h}\right)\ \text{MPa} \qquad [5]$$

Bieniawski (1992) explains that 'although the original *in situ* test data ... were based on pillar width-to-height ratios up to 3.4, when applied to full-size coal pillars ... the pillar strength formula given by Equation [5] was found applicable even for pillar width-to-height ratios of up to 12'. The strength trend predicted by Equation [4] for the Pittsburgh seam in the USA is reproduced in Figure 6. It should be noted that Equation [5] predicts a very similar rate of strength increase for pillars as the formula derived by Van der Merwe and Mathey (2013c) for collapsed coal pillars in South Africa, based on the overlap reduction (OR) technique.

Mark (2000) also reviewed his experience with coal pillars in the USA and came to the conclusion that for pillar w/h ratios up to 8, empirical strength formulae were 'reasonably accurate' (Mark, 2000). A further important observation was that abrupt failure occurred only for pillars with w/h up to 3, while 'squeezing' was the predominant failure mode in the interval of $4 < w/h < 8$. Squat pillars were those 'which can carry very large loads and are strain-hardening, and which are dominated by entry failure (roof, rib and floor) and by coal bumps' (Mark, 2000). This typically occurred at pillar w/h ratios greater than 10. Mark's observation on *in situ* squat pillars appear to confirm the trends observed in laboratory coal model pillar tests.

Maleki (1992) reported that in the USA he had observed squat pillars with width-to-height ratios up to 15 to fail under load. In back-calculating the average peak vertical stresses on collapsed pillars from seven coal seams and eight collieries by means of empirical and numerical analysis as well as stress measurements, he established strength *versus* width-to-height curves for coal pillars far into the squat range. In his analysis of pillar failures, he distinguished the confinement-controlled pillar collapses in competent geological environments from those where the failure mechanism appeared to be structurally controlled, *i.e.* where failure was aided by persistent cleats and in-seam contact planes. He proposed two strength equations:

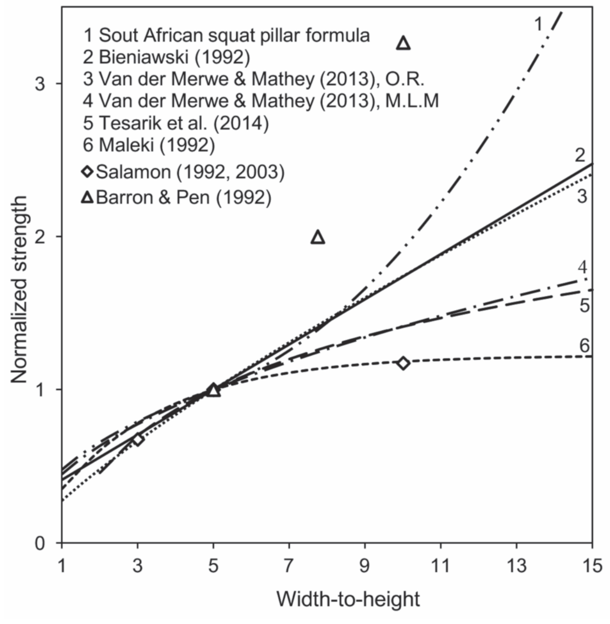

Figure 6—Predictions of squat coal pillar strength

$$\text{Strength} = 32\left(1 - \text{Exp}\left(-\frac{0.339\,w}{h}\right)\right)\ \text{MPa} \quad \text{(in confinement control)} \qquad [6]$$

$$\text{Strength} = 26\left(1 - \text{Exp}\left(-\frac{0.264\,w}{h}\right)\right)\ \text{MPa} \quad \text{(in structural control)} \qquad [7]$$

It should be noted that these equations describe a regressive strength increase with increasing width-to-height ratio, which levels off at maximum limiting strength values of 32 MPa and 26 MPa respectively. These maximum values are approached for pillars with width-to-height ratios between 10 and 15. Maleki (1992) states that above these limits, stability problems may occur as a result of failures in roof, seam, and floor. Equation [6] is reproduced in Figure 6.

Indications from analytical models

A number of analytical pillar models have been proposed in the past decades, *e.g.* (Wilson, 1981; Barron and Pen, 1992; Salamon, 1992; Napier and Malan, 2007). In two cases, the developers have applied their models to the prediction of squat coal pillar performance.

Barron and Pen (1992) developed a model that uses the Hoek-Brown failure criterion to describe the intact and residual strengths of the coal. The criterion is modified to allow brittle-ductile transition to occur in zones of high lateral confinement within the model pillar. The strata surrounding the pillar are assumed to be elastic and infinitely stiff.

Barron and Pen (1992) applied this model to the prediction of squat pillar strength in the Witbank coalfield. It should be noted that the results did not show an exponential increase in strength, but rather suggested an overall regressive increase in pillar strength between width-to-height ratios of 5 and 20. However, for width-to-height ratios of less than 12, the predicted strength was higher than that predicted by the South African squat coal pillar formula (Figure 6).

Salamon (1992) proposed a very comprehensive analytical coal pillar model in which the deformation of a laminated rock mass surrounding the pillar is explicitly taken into account. Salamon *et al.* (2003) applied this model to the prediction of the strength of three coal pillars with width-to-height ratios of 3, 5, and 10. The results are reproduced in Figure 6, from which it will be seen that only a regressively increasing trend between strength and the width-to-height ratio of coal pillars is indicated.

The magnitude of strength and the rates of strength increase are predicted to be significantly lower than those in the Barron and Pen model. However, both models agree that an exponential increase in peak pillar strength does not occur.

Indications from numerical models

Substantial progress has been made with regard to an understanding of the various factors that contribute to the strength and stability of coal pillars, due to the application of numerical modelling techniques. Some of these findings are particularly relevant for squat coal pillars:

It has been found that the adverse influence of joints on the strength of coal pillars may vanish for pillar width-to-height ratios greater than 6 or 7 (Esterhuizen, 2000). Rock partings in the coal seam can have either a strengthening or weakening influence on pillars, depending on the competence of the parting (Su and Hasenfus, 1999). The seam strength itself, however, may have a negligible impact on the performance of pillars with squat dimensions. The competence of the surrounding rock mass appears to be more important: weak floor strata, for instance, can decrease the ultimate pillar strength by as much as 50 % (Su and Hasenfus, 1999). Also, the cohesive strength of the pillar-strata interface may considerably influence the critical width-to-height ratio for occurrence of brittle-ductile transition in pillars (Lu *et al.*, 2008).

Results from numerical models can give only qualitative insight into the role of different factors that influence pillar strength, unless some form of calibration of the model is conducted. However, the choice of criteria for the model calibration may influence the results dramatically.

For instance, *et al.* (2010) calibrated coal pillar models against the empirical peak strength criterion for the combined South African and Australian coal pillar databases (Galvin, Hebblewhite, and Salamon, 1999). The extrapolation of the models into the squat range predicted that brittle-ductile transition occurred in pillars at *w/h* ratios of between 5 and 6.7.

Esterhuizen *et al.* (2010a) calibrated numerical coal pillar models against Bieniawski's linear strength formula (Equation [4]) and measured stress profiles for *in-situ* pillar ribs. The extrapolation of the models to greater width-to-height ratios predicted that brittle-ductile transition occurred in pillars at around *w/h* = 8. A progressive strength increase was not observed.

Tesarik *et al.* (2013) calibrated models against one pillar stress-strain curve that was obtained by Van Heerden (1975) in his *in situ* testing programme. The idea behind this calibration procedure was to incorporate information on peak strength, residual strength, and the gradual softening of pillars after fracturing. The calibrated model was extrapolated to larger width-to-height ratios, assuming that the strata surrounding the pillars remained elastic. Tesarik *et al.* (2013) observed that the strength of their coal model pillars increased regressively up to *w/h* = 16, when brittle-ductile transition finally occurred. The strength trend is plotted in Figure 6 and it will be seen that there is good agreement with the trend predicted by the Van der Merwe and Mathey (2013c) formula, which was derived from collapsed and stable cases based on the maximum likelihood method (MLM).

It is obvious from the above that the numerical prediction of squat pillar performance is very sensitive towards the selected calibration procedure. There is no agreement on the point of brittle-ductile transition in pillars. However, all presented coal pillar models agree on the fact that a progressive increase in peak strength cannot occur.

Conclusions

The South African squat pillar strength formula predicts an exponential increase in strength for pillar width-to-height ratios of greater than 5. Three major arguments for the South African squat coal pillar strength formula were proposed by Salamon and Wagner (1985):

- ➤ The observation that pillars of *w/h* = 10-12 do not collapse under any practical load
- ➤ Theoretical and experimental studies on model pillars consisting of materials other than intact coal had indicated a rapid strength increase above *w/h* = 5
- ➤ *In situ* observations in collieries in KwaZulu-Natal suggested that squat pillars performed better than expected from the original Salamon and Munro (1967) formula.

It has been argued in this paper that none of the above observations is actually capable of substantiating the exponential nature of the South African squat pillar formula:

Firstly, the very high load-bearing capacity of pillars with *w/h* ratios of 10 or 12 can be explained more plausibly by the brittle-ductile transition in pillars, and not by an exponentially increased peak strength.

Secondly, laboratory tests on coal model pillars unambiguously agree that the peak strength increases only linearly or regressively for this material until the brittle-ductile transition occurs. There is a great amount of evidence that brittle-ductile transition does not occur in coal specimens with *w/h* ratios smaller than 8. Also, the brittle-ductile transition does not necessarily occur in coal specimens up to *w/h* = 13, even though it has been observed in tests on Indian coals to occur at *w/h* = 10. The phenomena accompanying strength and failure in coal model pillars are therefore appreciably different from those occurring in rock model pillars, for which higher rates of strength increase and an early brittle-ductile transition have indeed been observed by some researchers.

Thirdly, it has been argued that empirical pillar strength formulae that are based on collapse cannot reliably predict the actual strength of any given coal pillar. Therefore a divergent strength of pillars in the squat range does not necessarily justify a squat effect.

To date, no squat pillar has ever been observed to collapse in South Africa. This is because squat coal pillars are designed with very high safety factors. The absence of collapse cases complicates the task of finding a more appropriate design criterion for such structures in the local coalfields. However, a wealth of information is already available from international field experience and analytical and numerical models, which can give qualitative insight into the expected behaviour of squat pillars.

Experience with failed squat pillars in the USA suggests that empirical design formulae, which are derived from large-scale compression tests or from back-calculation of collapsed slender pillars ($w/h < 5$), may still be reasonably accurate when extrapolated to larger width-to-height ratios of up to 8 or 12. There is even some evidence that a maximum limiting strength may exist for coal pillars, which strongly opposes the South African assumption of an exponential strength increase.

Likewise, the majority of published analytical and numerical coal pillar models agree on the point that the peak strength of coal pillars increases only regressively with increasing width-to-height ratio, until the brittle-ductile occurs. This phenomenon, however, is likely to occur only at very large pillar width-to-height ratios of around 10.

The evidence presented from field experience, laboratory tests, and numerical and analytical models therefore corroborates the conclusion that a progressive increase in coal pillar strength for width-to-height ratios greater than 5 does not exist. Consequently, the South African squat coal pillar formula (Equation [2]) is misleading from a mechanical point of view and must be replaced by a more appropriate design criterion.

Implications for squat pillar design in South Africa

The determination of a more appropriate design criterion is complicated by the fact that no experience with failed squat pillars in South Africa is available so far. Therefore we can currently only estimate a more suitable design criterion based on the international experience outlined in this paper. The following deductions may be made in this regard:

- The empirical coal pillar strength formulae, derived from back-analysis of collapse cases or from *in situ* compression tests in South Africa, may be extrapolated into the squat pillar range up to $w/h = 10$. Pillars with width-to-height ratios greater than 10 may perform in a pseudo-ductile manner without exhibiting an ultimate peak strength. For these pseudo-ductile pillars, a strain-based failure criterion is required to limit undue deformations of the pillar-strata-systems
- Since it has become customary in South Africa to base pillar design on experience with failed cases rather than on *in situ* compression tests, it is suggested that either one of the two formulae provided by Van der Merwe and Mathey (2013c) are used to design squat coal pillars:

  Strength = $6.61\ w^{0.5}/h^{0.7}$ MPa (maximum likelihood method - MLM) [8]

  Strength = $5.47\ w^{0.8}/h$ MPa (overlap reduction technique - OR) [9]

  It will be noted from Figure 7 that Equations [8] and [9] predict a markedly different strength for squat pillars. According to Equation [9], a 3 m high pillar is predicted to have 50 % higher strength at $w/h = 6$ and 75 % higher strength at $w/h = 10$ as compared to Equation [8]. Therefore, designing bord-and-pillar layouts based on Equation [9] will result in improved extraction of coal, which even exceeds the economic advantages of the South African squat coal pillar formula. However, it should be noted that even with Equation [9] the extraction rates of a bord-and-pillar panel at depth exceeding 300 m would remain less than 40%. This may render the bord-and-pillar method uneconomic for coal mining at greater depth
- From a safety point of view, no experience is available to date that may argue against the use of any of these two formulae. The positive performance of squat pillars in the South African database of stable cases can be explained by both Equations [8] and [9]. Both formulae also perform equally well when tested against the database of failed slender pillars ($w/h < 5$) in South Africa (*i.e.* both formulae predict an average safety factor of failed cases close to SF = 1 with an appreciably small standard deviation). However, Equation [9] has the additional advantage that it distinguishes between failed and stable cases with improved reliability
- An additional argument in favour of Equation [9] is that it predicts a very similar rate of strength increase as Equation [5], which was developed by Bieniawski from the three large-scale *in situ* compression tests in South Africa. Figure 7 provides a plot of Equation [5] for a seam strength value of $\sigma_c = 6.4$ MPa (Pittsburgh coal), which Bieniawski found suitable for pillar design up to $w/h = 12$.

It is strongly recommended that any squat pillar design based on Equation [9] should be accompanied by an appropriate pillar monitoring programme.

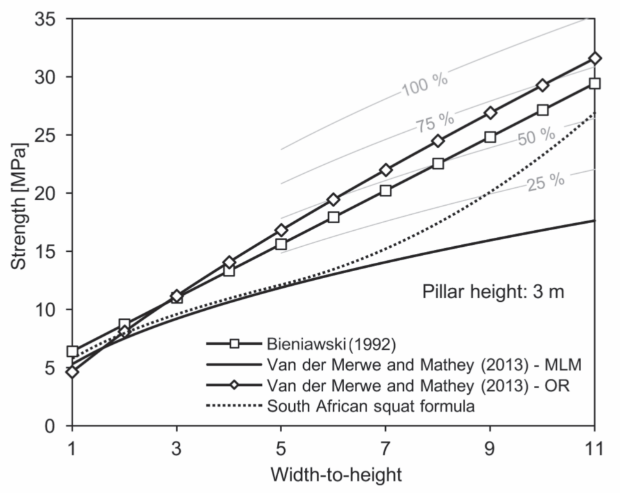

Figure 7—Estimations for squat pillar strength in South Africa

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INTERNATIONAL ACTIVITIES

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2–7 October 2017 —AfriRock 2017: ISRM International Symposium 'Rock Mechanics for Africa'
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BSC Resources
CAE Mining (Pty) Limited
Caledonia Mining Corporation
CDM Group
CGG Services SA
Chamber of Mines
Concor Mining
Concor Technicrete
Council for Geoscience Library
CSIR-Natural Resources and the Environment
Department of Water Affairs and Forestry
Digby Wells and Associates
Downer EDI Mining
DRA Mineral Projects (Pty) Ltd
DTP Mining
Duraset

Elbroc Mining Products (Pty) Ltd
Engineering and Project Company Ltd
eThekwini Municipality
Exxaro Coal (Pty) Ltd
Exxaro Resources Limited
Fasken Martineau
FLSmidth Minerals (Pty) Ltd
Fluor Daniel SA (Pty) Ltd
Franki Africa (Pty) Ltd Johannesburg
Fraser Alexander Group
Geobrugg Southern Africa
Glencore
Goba (Pty) Ltd
Hall Core Drilling (Pty) Ltd
Hatch (Pty) Ltd
Herrenknecht AG
HPE Hydro Power Equipment (Pty) Ltd
Impala Platinum Limited
IMS Engineering (Pty) Ltd
Ivanhoe Mines SA
JENNMAR South Africa
Joy Global Inc. (Africa)
Kadumane Manganese Resources
Leco Africa (Pty) Limited
Longyear South Africa (Pty) Ltd
Lonmin Plc
Lull Storm Trading (PTY)Ltd T/A Wekaba Engineering
Magnetech (Pty) Ltd
Magotteaux(PTY) LTD
MBE Minerals SA Pty Ltd
MCC Contracts (Pty) Ltd
MDM Technical Africa (Pty) Ltd
Metalock Industrial Services Africa (Pty)Ltd
Metorex Limited
Metso Minerals (South Africa) (Pty) Ltd
Minerals Operations Executive (Pty) Ltd
MineRP Holding (Pty) Ltd
Mintek
MIP Process Technologies
Modular Mining Systems Africa (Pty) Ltd
MSA Group (Pty) Ltd
Multotec (Pty) Ltd
Murray and Roberts Cementation

Nalco Africa (Pty) Ltd
Namakwa Sands (Pty) Ltd
New Concept Mining (Pty) Limited
Northam Platinum Ltd - Zondereinde
Outotec (RSA) (Proprietary) Limited
PANalytical (Pty) Ltd
Paterson and Cooke Consulting Engineers (Pty) Ltd
Polysius A Division Of Thyssenkrupp Industrial Solutions (Pty) Ltd
Precious Metals Refiners
Rand Refinery Limited
Redpath Mining (South Africa) (Pty) Ltd
Rosond (Pty) Ltd
Royal Bafokeng Platinum
Roymec Tecvhnologies (Pty) Ltd
Runge Pincock Minarco Limited
Rustenburg Platinum Mines Limited
SAIEG
Salene Mining (Pty) Ltd
Sandvik Mining and Construction Delmas (Pty) Ltd
Sandvik Mining and Construction RSA(Pty) Ltd
SANIRE
Sasol Mining(Pty) Ltd
Sebilo Resources (Pty) Ltd
SENET
Senmin International (Pty) Ltd
Shaft Sinkers (Pty) Limited
Sibanye Gold (Pty) Ltd
Smec SA
SMS Siemag South Africa (Pty) Ltd
Sound Mining Solutions (Pty) Ltd
South 32
SRK Consulting SA (Pty) Ltd
Technology Innovation Agency
Time Mining and Processing (Pty) Ltd
Tomra Sorting Solutions Mining (Pty) Ltd
Ukwazi Mining Solutions (Pty) Ltd
Umgeni Water
VBKOM Consulting Engineers
Webber Wentzel
Weir Minerals Africa
WorleyParsons (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 120 years, the Southern African Institute of Mining and Metallurgy, has promoted technical excellence in the minerals industry. We strive to continuously stay at the cutting edge of new developments in the mining and metallurgy industry. The SAIMM acts as the corporate voice for the mining and metallurgy industry in the South African economy. We actively encourage contact and networking between members and the strengthening of ties. The SAIMM offers a variety of conferences that are designed to bring you technical knowledge and information of interest for the good of the industry. Here is a glimpse of the events we have lined up for 2016. Visit our website for more information.

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Fax: (011) 833-8156 or (011) 838-5923
E-mail: raymond@saimm.co.za

SAIMM DIARY

2016

- CONFERENCE
  The SAMREC/SAMVAL Companion Volume Conference
  17–18 May 2016
  Emperors Palace, Johannesburg
- COLLOQUIUM
  New technology and innovation in the Minerals Industry Colloquium
  9–10 June 2016
  Emperors Palace, Johannesburg
- SCHOOL
  The 2nd School on Manganese Ferroalloy Production
  Johannesburg
  27–28 June 2016
- CONFERENCE
  Resilience in the Mining Industry Conference 2016
  University of Pretoria
  13–14 July 2016
- CONFERENCE
  Innovations in Mining Conference 2016
  Holiday Inn Bulawayo
  19–20 July 2016
- SCHOOL
  Production of Clean Steel
  Mintek, Randburg
  25–26 July 2016
- CONFERENCE
  Hydrometallurgy Conference 2016 'Sustainability and the Environment' in collaboration with MinProc and the Western Cape Branch
  Belmont Mount Nelson Hotel, Cape Town
  1–3 August 2016
- CONFERENCE
  The Tenth International Heavy Minerals Conference 'Expanding the horizon'
  Sun City, South Africa
  16–18 August 2016
- CONFERENCE
  MINESafe Conference Striving for Zero Harm
  Emperors Palce, Hotel Casino Convention Resort
  31 August–2 September 2016
- CONFERENCE
  Mining for the Future 2016 'The Future for Mining starts Now'
  Electra Mining, Nasrec, Johannesburg
  12–13 September 2016
- CONFERENCE
  AMI Ferrous and Base Metals Development Network Conference 2016
  Southern Sun Elangeni Maharani, KwaZulu-Natal
  19–21 October 2016
- COLLOQUIUM
  The Young Professionals Week
  14th Annual Student Colloquium
  25 October 2016
  Mintek, Randburg

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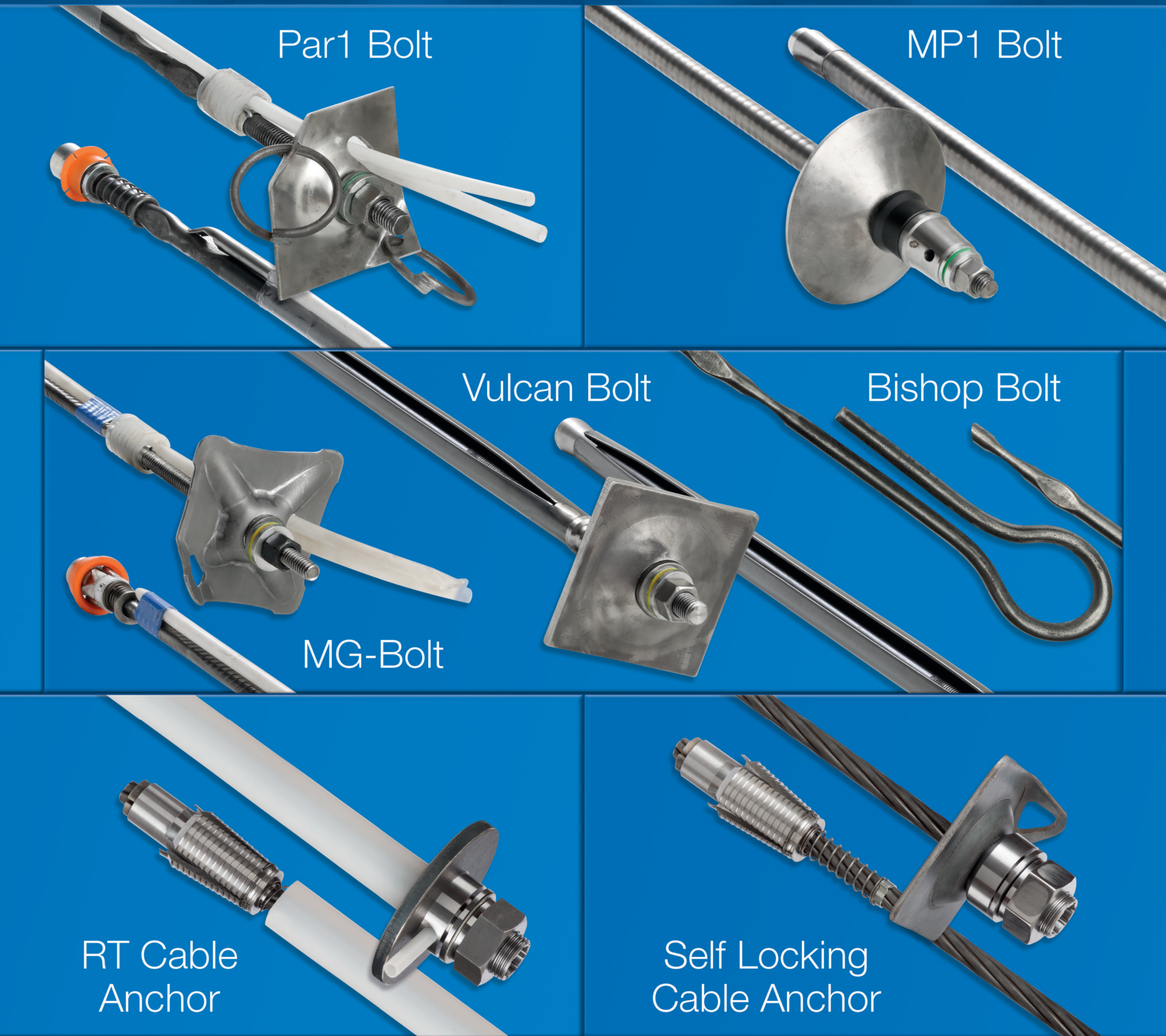

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