PREFACE

A total of 254 students enrolled in the Mining Research Course and completed their honours research project in 2014. This is a record student number for Mining Education Australia (MEA) in this final year course of the Mining Engineering program with a corresponding record number of 231 MEA graduates who finished their studies in 2014.

The papers in the fourth volume of the Journal of Research Projects Review were prepared jointly by students, their academic supervisor and in some instances the student’s industry supervisor. All the papers in this volume were peer-reviewed by at least two academics who were from partner institutions of MEA.

The MEA program is an education initiative between the four major universities in Australia that offer a degree program in Mining Engineering: Curtin University through the Western Australian School of Mines (WASM), University of Adelaide (UoA), UNSW Australia (UNSW) and University of Queensland (UQ). The program, which was launched in 2007, was collaboratively developed and is continually being updated by academic staff from each of the partner institutions with advice from education consultants to ensure the latest in teaching and learning practices are integrated into the program. The program is supported by the Minerals Council of Australia (MCA) who through its various industry partners has helped ensure MEA Graduates will be able to meet the current and future challenges of the global minerals industry.

The 2014 MEA Student Conference was hosted by UoA on 27 October with 20 students from the four MEA institutions who presented a paper on their research project; a copy of the program is shown in Appendix 1. The student conference reflects the high calibre of research undertaken by students in the MEA program. It bodes well for the future of the global mining industry in terms of the quality of graduates who will drive the improvements necessary to ensure the sustainability of the industry. A full list of presenters at this and past conferences is shown in Appendix 2.

The prize winners for the 2014 MEA Student Conference include:

- First Prize – Ashton INGERSON (UoA)
- First Runner ups – Kelsey ROBERTS (WASM) and Holly KIELY (WASM)
- Second Runner up – Jack BUTLER (UQ)
- Third Runner ups – Cameron WHITE (UQ) and Daniel HTWE (WASM)
- Fourth Runner ups – Kelly WILLIAMS (UNSW), Zoe UREN (UQ), Miguel BACHILLER (WASM) and Audie TRUTWEIN (WASM)

The contributions made by the mining industry and especially those persons who assisted the students with their research projects are acknowledged. Many of the projects were formulated during the period of vacation employment at mine sites, which reinforces the important contribution that work experience makes to the mining engineering program. It is important that industry continues to recognise the essential role that work experience plays in contributing to student education ensuring the continual supply of high quality graduate engineers to the mining industry.

Paul Hagan
MEA Mining Research Project Course Leader
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Investigation into the Applicability of the Different Failure Criteria to Conventional Triaxial Data

R Keaveney¹ and H Masoumi²

ABSTRACT
This research assesses the applicability of 16 different failure criteria to the broad range of conventional triaxial data and compares the functionality of their equations through a logical non-dimensionalisation process. The failure criteria and the triaxial data were obtained from literature. Triaxial data was then grouped based on geological formation (sedimentary, metamorphic and igneous) and the geographical region in which they originated (United States of America, Japan and Europe). All criteria and triaxial data were then transformed into mean effective and deviatoric stresses to allow fitting parameters extraction. This process was unique to each criterion and allowed for simplification during the calibration process. Once the data was imported into the program, each criterion was individually fitted to each specific data set. The coefficient of determination ($R^2$) was used to identify criteria that produced the most accurate representation of each data set. Only non-linear criteria have been assessed as Pariseau (2007) confirmed that these criteria can better represent the behaviour of intact rocks compared to the linear criteria. Consequently the linear type criteria such as Mohr–Coulomb and Drucker-Prager have been omitted from the investigation.

The results from this study showed that there is not a unified criterion that provides the best fit to all data sets. There is a variety of criteria that can be applied to each data set to produce the most accurate representation. Notably, there was a trend in criteria that produced a coefficient of determination consistently greater than 0.8. These criteria included: Mogi (1971), Bieniawski (1974), Franklin (1977), Ottosen (1977), Johnston (1985) and Modified Wiebols and Cook (Zhou, 1994) criteria. There was also a trend of criteria that consistently produced a coefficient of determination greater than 0.6. This included the Ramamurthy (Ramamurthy and Arora, 1993), Christensen (1997) and Hoek and Brown (1980) failure criteria. The Shima and Oyane (1976), Gurson (1977), Sheorey (Sheorey, Biswas and Choubey, 1989) and spatially mobilised plane (Matsuoka and Nakai, 1974) criteria did not produce a consistent result as the coefficient of determination varied significantly between each data set.

The Plane Griffith crack theory (Griffith, 1921), Hobbs criteria (1970) and Yoshida (Yoshida, Morgenstern and Chan, 1990) failure criteria provided poor representations of all data sets as these returned coefficients of determinations of 0 for the majority of investigations.

INTRODUCTION
Failure criteria are used extensively in the mining industry for many applications such as the design of underground excavations and predicting slope stability. The Hoek and Brown (1980) criterion is currently the industry standard, however research by Carter, Duncan and Lajtai (1991) suggested that the Johnston (1985) criterion provides a superior representation of rock failure compared to that proposed by Hoek and Brown (1980). Further, Colmenares and Zoback (2002) conducted a larger scale investigation using the Hoek and Brown (1980) criterion and six other criteria. The results from this investigation indicated that the Mogi (1971) and Modified Wiebols and Cook (Zhou, 1994) criteria produced the more accurate representation of rock failure compared to that proposed by Hoek and Brown (1980). Previous research has been limited to a small amount of failure criteria and triaxial data sets. This research has incorporated 16 failure criteria that have been applied to 22 triaxial data sets. Only non-linear criteria have been assessed as Pariseau (2007) confirmed that these criteria can better represent the behaviour of intact rocks compared to the linear criteria. Consequently the linear type criteria such as Mohr–Coulomb and Drucker-Prager have been omitted from the investigation.

In this study, it is aimed to identify which criteria can provide the better representation of the triaxial data sets. The grouping system used here is an extension of that applied by Hoek and Brown (1980) in which a number of different

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sandstones from various geographical conditions such as Vosges (France), Fontainebleau (France), Bentheim (Germany) and Berea (North America) with varying strengths were grouped together to suggest a range of values for $m$, which is one of the Hoek and Brown (1980) failure criterion parameters. This happened through a logical non-dimensionalisation process implemented by Hoek and Brown (1980). Later Brady and Brown (2006) conducted a similar work on other rock types and concluded a table having proposed range of $m$ for different rock types. Same logic is used here for broader range of rock samples with various geological and geographical characteristics to investigate and compare the mathematical functionality of the proposed parameters for 16 failure criteria. Also, different rock types from the same geology are grouped together and in order to overcome the issue regarding the diversity in the strength of the rock samples, they are non-dimensionalised versus their uniaxial compressive strength similar to that accomplished by Hoek and Brown (1980) and Brady and Brown (2006).

**DATA TRANSFORMATION AND MANIPULATION**

Sixteen failure criteria that are a combination of two and three dimensions have been incorporated in the investigation. Each criterion has a different formulation and so the transformation has been applied to allow each criteria to be analysed in the same fashion in which the process was unique to each criterion. The triaxial data sets had to be transformed through the same process as well. The decision to either transform each criterion depended on how the criterion was constructed. In general, two- and three-dimensional stress states criteria were mainly transformed into the triaxial conventional notations consisting of mean stress ($p$) and deviatoric stress ($q$).

Criteria were then non-dimensionalised so the fitting parameters of the criteria could be extracted. This allowed for the calculation of the coefficient of determination.

Equations 1 and 2 outline the procedure for calculating $p$ and $q$ from principal stresses ($\sigma_1$, $\sigma_2$, and $\sigma_3$):

$$ q = \sqrt{(\sigma_1 - \sigma_2)^2 + (\sigma_2 - \sigma_3)^2 + (\sigma_3 - \sigma_1)^2} $$

$$ p = \frac{\sigma_1 + \sigma_2 + \sigma_3}{3} $$

The following steps were applied to Equations 1 and 2 in order to transform the two-dimensional stress states failure criteria into the plane of $p$ and $q$. Generally, in these criteria it is assumed that intermediate principal stress is equal to minor principal stress, subsequently $\sigma_2 = \sigma_3$. Thus, Equations 1 and 2 can be reduced to Equations 3 and 4 according to:

$$ q = \sigma_1 - \sigma_3 $$

$$ p = \frac{1}{3}(\sigma_1 + 2\sigma_3) $$

The $p$ and $q$ expressions can now be expressed in terms of $\sigma_1$ and $\sigma_3$, this process involves re-arranging Equation 3 to Equation 5:

$$ \sigma_1 = q + \sigma_3 $$

Equation 5 is now substituted into Equation 4 to produce Equation 6:

$$ p = \frac{1}{3}(q + \sigma_3 + 2\sigma_3) $$

Equation 6 can now be re-arranged in terms of $\sigma_1$ and $\sigma_3$ as follows:

$$ \sigma_3 = \frac{3p - q}{3} $$

Substituting Equation 7 into 3 leads to the determination of $\sigma_1$ in terms of $p$ and $q$. This relationship can be outlined in Equation 8:

$$ \sigma_1 = \frac{3p - 2q}{3} $$

The following steps were applied to three-dimensional failure criteria that utilise the first stress invariant ($I_1$) and the second invariant of deviator stress ($I_2$) parameters. These criteria included: Shima Oyane (1976), Gurson (1977), Ottosen (1977), Modified Wiebols and Cook (Zhou, 1994) and Christensen (1997). This enabled these criteria to be transformed into the plane of $p$ and $q$, allowing them to be incorporated into Equations 3 and 4. Equations 9 and 10 outline the first stress invariant ($I_1$) and the second invariant of deviator stress ($I_2$) parameters:

$$ I_2 = \frac{1}{6}[(\sigma_1 - \sigma_2)^2 + (\sigma_2 - \sigma_3)^2 + (\sigma_3 - \sigma_1)^2] $$

$$ I_1 = \sigma_1 + \sigma_2 + \sigma_3 $$

In conventional triaxial testing intermediate principal stress is equal to minor principal stress, subsequently Equation 9 can be transformed into Equation 11:

$$ I_2 = \frac{1}{6}(\sigma_1^2 - 2\sigma_1\sigma_3 + \sigma_3^2 + \sigma_2^2 + 2\sigma_2\sigma_3 + \sigma_3^2) $$

Equation 11 can now be reduced to Equation 12 according to:

$$ I_2 = \frac{1}{6}(2\sigma_1^2 - 4\sigma_1\sigma_3 + 2\sigma_3^2) $$

$$ I_2 = \frac{1}{3}(\sigma_1^2 - 2\sigma_1\sigma_3 + \sigma_3^2) $$

$$ I_2 = \frac{1}{3}(\sigma_1 - \sigma_3)^2 $$

Combining Equation 3 and 12, $I_2$ can be transformed into the $p$-$q$ plane according to:

$$ I_2 = \frac{1}{3}q^2 $$

As intermediate principal stress is equal to minor principal stress, Equation 10 can be simplified to Equation 14:

$$ I_1 = \sigma_1 + 2\sigma_3 $$

Combining Equations 4 and 14, $I_1$ can be transformed into the $p'$-$q$ plane according to Equation 15:

$$ I_1 = 3p $$

**FAILURE CRITERIA FORMULATIONS**

This section outlines the formulation of the criteria used in this study and provides a brief description of their parameters.
Two-dimensional criteria
Non-linear failure criteria that consider two-dimensional stress states will be outlined. These criteria assume intermediate and minor principal stresses to be equal (\(\sigma_2 = \sigma_3\)).

Plane Griffith crack theory
The Plane Griffith crack theory is based on the balance between the decrease in potential energy and the increase in surface energy resulting from the presence of a crack (Griffith, 1921). The theory suggests that when a compressive force is applied, micro-fractures within the rock samples will increase and form an elliptical crack that will propagate from the points of maximum tensile stress concentration (Griffith, 1921). Equation 16 and 17 describe this criterion (Franklin, 1977):

\[
(\sigma_1 - \sigma_2)^2 - 8\sigma_1(\sigma_1 + \sigma_2) = 0 \text{ if } \sigma_1 + 3\sigma_2 > 0
\]

(16)

\[
\sigma_3 + \sigma_2 = 0 \text{ if } \sigma_1 + 3\sigma_2 < 0
\]

(17)

where:
\(\sigma_1\) is the uniaxial compressive strength and \(F\) and \(f\) are empirical constants.

Bieniawski criterion
Bieniawski developed this criterion in 1974 from triaxial testing done on a variety of different rock samples. Equation 19 outlines this criterion (Bieniawski, 1974):

\[
\sigma_1 = \sigma_c + \sigma_3 + F'\sigma_3^a
\]

(18)

where:
\(\sigma_c\) is the uniaxial compressive strength and \(F\) and \(f\) are empirical constants.

Franklin criterion
This criterion was developed in 1977 after a comparative study between preceding linear and non-linear criteria. Equation 20 outlines this criterion (Franklin, 1977):

\[
(\sigma_1 - \sigma_3) = A(\sigma_1 + \sigma_3)B
\]

(20)

where:
\(A = \sigma_c^{-B}\) and \(B\) varies between 0.6 to 0.9.

The Franklin criterion would be identical to Griffiths criterion if \(B = \frac{1}{2}\).

Hoek–Brown criterion
The original Hoek–Brown criterion was developed in 1980 from triaxial testing on Andesite from the Bougainville mine in Papua New Guinea (Hoek and Marinos, 2007). The modified Hoek–Brown criterion for intact rock is outlined in Equation 21 (Hoek and Brown, 1980):

\[
\sigma_1 = \sigma_3 + \sigma_c\left(\frac{m_1\sigma_3}{\sigma_c} + 1\right)^\frac{1}{2}
\]

(21)

where:
\(m_1\) is a constant determined by the type of rock.

Ramamurthy criterion
This criterion was developed in 1985 after triaxial testing on multiple sandstone samples. This criterion has also been applied to rock masses by altering input parameters. Equation 22 outlines this criterion (Ramamurthy and Arora, 1993):

\[
\sigma_1 = \sigma_3 + B\sigma_3\left(\frac{\sigma_c}{\sigma_3}\right)^\alpha
\]

(22)

where:
\(\alpha\) and \(B\) are constants obtained by triaxial tests on intact rock.

Johnston criterion
This criterion was developed in 1985 and was applied to three other parameters: \(A, S\) and \(B\). Equation 25 describes the Johnston criterion (Johnston, 1985):

\[
\sigma_1 = \sigma_3 + A\sigma_3\left(\frac{\sigma_c}{\sigma_3}\right)^S
\]

(25)

where:
\(A, S\) and \(B\) are constants determined by the rock type.
Three-dimensional criteria

Non-linear failure criteria that consider three-dimensional stress states will be outlined. These criteria assume major, intermediate and minor principal stresses to be different ($\sigma_1 > \sigma_2 > \sigma_3$).

**Spatially Mobilised Plane**

The Spatially Mobilised Plane criterion was developed by Matsuoka and Nakai (1974). It considers the impact of three-dimensional stress fields. Equation 29 outlines the criterion (Matsuoka and Nakai, 1974):

$$\frac{2}{\sqrt{3}} \sin 3\theta + \left( \frac{3}{k-1} \right) \left( \frac{\sqrt{I_2}}{I_1} \right)^2 + \left( \frac{4}{3} \right) \frac{1}{k-1} = 0$$  \hspace{1cm} (29)

where:
- $k$ is a material constant

**Shima Oyane criterion**

This criterion was developed by Shima and Oyane (1976). It considers three-dimensional stress state, Equation 30 outlines this criterion (Shima and Oyane, 1976):

$$\frac{3I_2}{\sigma_M^2} + a \mu \sigma_2 \left( \frac{I_1}{3\sigma_m} \right)^2 \cdot (1 - n)^5 = 0$$  \hspace{1cm} (30)

where:
- $n$ is porosity
- $\sigma_m$ is the mean stress ($\sigma_m = \frac{I_1}{3}$)

**Gurson criterion**

This criterion is an extension of the von Mises criterion, it has been designed to describe yielding and the evolution of voids in porous materials (Gurson, 1977). Equation 31 outlines the criterion (Gurson, 1977):

$$\frac{3I_2}{\sigma_M^2} + 2n \cosh \left( \frac{I_1}{2\sigma_M} \right) \cdot (1 + n^2) = 0$$  \hspace{1cm} (31)

where:
- $n$ is porosity
- $\sigma_m$ is the mean stress ($\sigma_m = \frac{I_1}{3}$)

**Ottosen criterion**

This criterion considers three-dimensional stress fields. Equation 32 outlines this criterion (Ottosen, 1977):

$$\tau_{oct} = f_i(\sigma_1 + \sigma_2)$$

where:
- $\tau_{oct}$ is octahedral shear stress in failure
- $f_i$ is a monotonically increasing function

Parameters $a$ and $b$ are equal to Equation 27 and 28:

$$a = \frac{2\sqrt{3}}{3} \cos \varphi$$  \hspace{1cm} (27)

$$b = \frac{2\sqrt{3}}{3} \sin \varphi$$  \hspace{1cm} (28)

**Modified Wiebols and Cook criterion**

Zhou (1994) extended the existing Wiebols and Cook criterion which incorporates aspects of the Circumscribed Drucker-Prager criterion. Equation 33 outlines the Modified Wiebols and Cook criterion and Equation 34 presents the value of $I_2$:

$$I_2 = A + B|J_1 + C|J_1|^2$$  \hspace{1cm} (33)

$$I_2 = \frac{1}{3}(\sigma_1 + 2\sigma_2)$$  \hspace{1cm} (34)

where:
- $A$, $B$ and $C$ are determined by rock strength parameters

**Christensen criterion**

Christensen (1997) developed this criterion as a yield function applicable to isotropic materials. Equation 35 outlines this criterion:

$$I_2 = \frac{1}{(1 + \chi)^2} \left[ \left( \frac{\sigma_1}{\sigma_m} \right)^2 + \chi\frac{\sigma_2}{\sigma_m} \right]$$  \hspace{1cm} (35)

This criterion has two yield parameters which include: $\kappa$ and $\chi$. These values can be extracted from the data fitting process.

**COMPARATIVE ANALYSIS**

Following the transformation of all data, the comparative analysis was undertaken. This was completed in the DataFit software package and the coefficient of determination ($R^2$) was used to identify the criteria that best fitted to the data set. All data was classified into their respective groups and imported into the program where a regression analysis was individually applied to each criteria and triaxial data sets. Detailed information on the triaxial data such as uniaxial compressive strength and peak stresses at different confinements can be found in Keaveney (2014) study.

**Results of sedimentary triaxial data analysis**

This analysis consisted of 11 sedimentary triaxial data sets that were extracted from various publications. The analysed data sets included: Gosford sandstone (Ord, Vardoulakis and Kajewski, 1991), Red sandstone (Yang, Jing and Wang, 2011), Vosges sandstone (Besseulle, Desrous and Raynaud, 2000), Berea sandstone (Mogi, 2007), Indiana limestone (Schwart, 1964), Solnhofen limestone (Mogi, 2007), Bunt sandstone (Gowd and Rummel, 1980), Pottsville sandstone (Schwart, 1964), Jinping sandstone (Wang, Zhu and Wu, 2010) and Tyndallstone (Carter, Duncan and Lajtai, 1991). The uniaxial compressive strength varied from 23.75 MPa of Gosford Sandstone (Ord, Vardoulakis and Kajewski, 1991) to 293 MPa of Solnhofen Limestone (Zhu, Xiao and Evans, 2003). There was a total of 113 conventional triaxial results obtained for sedimentary rock samples. Table 1 provides a ranking of the criteria assessed and the coefficient of determination.

From Table 1, the Mogi (1971) criterion provides the best representation of the data set with a coefficient of
determination of 0.914. This criterion was clearly the most suitable one as the second ranked Modified Wiebols and Cook (Zhou, 1994) criterion had a coefficient of determination which was 0.05 lower. The top two criteria were both three-dimensional criteria however the following four criteria were two dimensional. The two-dimensional criteria with two fitting parameters all ranked in similar positions with a variation in their coefficient of determination at a maximum of 0.035. Figure 1 provides a graphical representation of the Mogi (1971) criterion fitted to the sedimentary triaxial data.

**Results of metamorphic triaxial data analysis**

This analysis consisted of seven metamorphic triaxial data sets that were extracted from various publications. The analysed data included: Tennessee marble (Wawersik and Fairhurst, 1969), Midland medium marble (Yang et al., 2008), Midlands coarse marble (Yang et al., 2008), Carrara marble (Haimson, 2006), Nanyang marble (You, 2010), Yamaguchi marble (Mogi, 2007) and Georgia marble (Schwartz, 1964). The uniaxial compressive strength varied from 30.6 MPa of Georgia Marble (Schwartz, 1964) to 137 MPa of Carrara Marble (Fassina, 2000). There was a total of 46 conventional triaxial results obtained for metamorphic rock samples. Table 2 provides a ranking of the criteria assessed and the coefficient of determination.

In the metamorphic analysis the highest four ranking criteria were the Franklin (1977), Johnston (1985), Hoek and Brown (1980) and Bieniawski (1974) criteria, respectively with coefficients of determination greater than 0.92. These four criteria are two-dimensional stress states in which the minor and intermediate principal stresses are assumed equal. However, the Ottosen (1977) and Mogi (1971) criteria were

<table>
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<th>Criterion</th>
<th>$R^2$</th>
<th>Dimension of criterion</th>
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<tr>
<td>1</td>
<td>Franklin (1977)</td>
<td>0.948</td>
<td>2</td>
</tr>
<tr>
<td>2</td>
<td>Johnston (1985)</td>
<td>0.926</td>
<td>2</td>
</tr>
<tr>
<td>3</td>
<td>Hoek and Brown (1980)</td>
<td>0.925</td>
<td>2</td>
</tr>
<tr>
<td>4</td>
<td>Bieniawski (1974)</td>
<td>0.924</td>
<td>2</td>
</tr>
<tr>
<td>5</td>
<td>Ottosen (1977)</td>
<td>0.909</td>
<td>3</td>
</tr>
<tr>
<td>6</td>
<td>Mogi (1971)</td>
<td>0.900</td>
<td>3</td>
</tr>
<tr>
<td>7</td>
<td>Ramamurthy (Ramamurthy and Arora, 1993)</td>
<td>0.881</td>
<td>2</td>
</tr>
<tr>
<td>8</td>
<td>Modified Wiebols and Cook (Zhou, 1994)</td>
<td>0.864</td>
<td>3</td>
</tr>
<tr>
<td>9</td>
<td>Christensen (1997)</td>
<td>0.821</td>
<td>3</td>
</tr>
<tr>
<td>10</td>
<td>Shima Oyane (1976)</td>
<td>0.788</td>
<td>3</td>
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<tr>
<td>11</td>
<td>Hobbs (1970)</td>
<td>0.788</td>
<td>3</td>
</tr>
<tr>
<td>12</td>
<td>Spatially mobilised plane (Matsuoka and Nakai, 1974)</td>
<td>0.787</td>
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<td>13</td>
<td>Sheorey (Sheorey, Biswas and Choubey, 1989)</td>
<td>0.484</td>
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<tr>
<td>14</td>
<td>Hobbs (1970)</td>
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<td>15</td>
<td>Plane Griffith crack theory (Griffith, 1921)</td>
<td>0</td>
<td>2</td>
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<td>16</td>
<td>Yoshida (Yoshida, Morgenstern and Chan, 1990)</td>
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</tbody>
</table>

**FIG 1 – Mogi (1971) criterion fitted to the normalised sedimentary triaxial data.**
ranked fifth and sixth with coefficient of determinations being 0.909 and 0.900. As the difference between the first and sixth ranked criteria is only 0.048, there is no distinct dimensional criteria that is more suitable. Figure 2 provides a graphical representation of the Franklin (1977) criterion fitted to the metamorphic triaxial data.

**Results of igneous triaxial data analysis**

This analysis consisted of four igneous triaxial data sets that were extracted from various publications. The analysed data sets included: Westerly granite (Al-Ajmi and Zimmerman, 2005), Inada granite (Badrul et al., 2014), Manazuru andesite (Mogi, 2007) and Misuho Trachyte (Mogi, 2007). The uniaxial compressive strength varied from 100 MPa of Misuho Trachyte (Yoo, 2011) to 256 MPa of the Westerly Granite (Zhu, n.d.). There was a total of 20 conventional triaxial results obtained for igneous rock samples. Table 3 provides a ranking of the criteria assessed and the coefficient of determination.

In the igneous analysis the highest seven failure criteria all produced coefficients of determination greater than 0.9. The data set that was analysed only utilised four triaxial data sets and the results of this investigation are limited by this shortcoming. Figure 3 provides a graphical representation of the Ramamurthy (Ramamurthy and Arora, 1993) criterion fitted to the igneous triaxial data.

**Results of USA triaxial data analysis**

This analysis incorporated seven triaxial data sets that originated in the United States of America. The data analysed included: Berea sandstone (Chester and Choens, 2009), Dunham dolomite (Mogi, 2007), Indiana limestone (Schwartz, 1964), Pottsville sandstone (Schwartz, 1964), Tennessee marble (Wawersik and Fairhurst, 1969), Georgia marble (Schwartz, 1964), and Westerly granite (Al-Ajmi and Zimmerman, 2005). The uniaxial compressive strength varied from 30.6 MPa of Georgia marble (Schwartz, 1964) to 256 MPa of Westerly Granite (Zhu, n.d.). There was a total of 62 conventional triaxial results obtained for this region. Table 4 provides a ranking of the criteria which were assessed based on the coefficient of determination.

This analysis incorporated a combination of sedimentary, metamorphic and igneous triaxial data sets. The Mogi (1971), Franklin (1977) and Modified Wiebols and Cook (Zhou, 1994) were noticeably the best performing criteria as their coefficients of determination were at least 0.101 greater than the fourth ranked Ottosen criteria (1977). This analysis identified that when sedimentary, igneous and metamorphic are analysed together, these three highest ranked criteria are evidently the most suitable. Figure 4 provides a graphical representation of the Mogi (1971) criterion fitted to the USA triaxial data.

**Results of Japanese triaxial data analysis**

This analysis incorporated four triaxial data sets that originated in Japan. The data analysed included: Yamaguchi marble (Mogi, 2007), Inada granite (Badrul et al., 2014), Manazuru andesite (Mogi, 2007) and Misuho trachyte (Mogi, 2007). The uniaxial compressive strength varied from 81 MPa of Yamaguchi marble (Mogi, 2007) to 211 MPa of Inada granite...
INVESTIGATION INTO THE APPLICABILITY OF THE DIFFERENT FAILURE CRITERIA TO CONVENTIONAL TRIAXIAL DATA

There was a total of 31 conventional triaxial results obtained from this region. Table 5 provides a ranking of the criteria assessed and the extracted coefficient of determination.

The Japanese rock group analysis incorporated both metamorphic and igneous triaxial data sets. The Bieniawski (1974), Johnston (1985) and Modified Wiebols and Cook (Zhou, 1994) were noticeably the best performing criteria as their coefficients of determination were at least 0.074 greater than the fourth ranked Mogi (1971) criteria. In this analysis there was very little difference between the Johnston (1985) and Bieniawski (1974) criteria. This investigation utilised three metamorphic and one igneous triaxial data set. In the previous metamorphic and igneous analysis (Table 2 and 3), the Johnston (1985) and Bieniawski (1974) criteria performed similarly in the metamorphic analysis; however, the Bieniawski (1974) criterion performed significantly better in the igneous analysis. Consequently, the Bieniawski (1974) criterion performed slightly better in the Japanese analysis. Figure 5 provides a graphical representation of the Bieniawski (1974) criterion fitted to the Japanese triaxial data.

Results of European triaxial data analysis
This analysis incorporated four triaxial data sets that originated in Europe. The data analysed included: Solnhofen limestone (Mogi, 2007), Bunt sandstone (Gowd and Rummel, 1980), Vosges sandstone (Besuelle, Desrues and Raynaud, 2000) and Carrara marble (Haimson, 2006). The uniaxial compressive strength varied from 30.1 MPa of Vosges sandstone (Stonecontact, 2003) to 293 of Solnhofen limestone (Zhu, Xiao and Evans, 2003). There was a total of 27 conventional triaxial results obtained from this region. Table 6 provides a
ranking of the criteria assessed and the extracted coefficient of determination.

The European rock group analysis incorporated three sedimentary and one metamorphic triaxial data set. The highest seven failure criteria all produced coefficients of determinations greater than 0.900. The data set that was analysed only utilised four triaxial data sets and the results of this investigation are limited by this shortcoming. There needs to be a larger data set to produce more conclusive results to clearly identify the most suitable criteria for this geographical analysis. Figure 6 provides a graphical representation of the Modified Wiebols and Cook (Zhou, 1994) criterion applied to the European triaxial data.

**DISCUSSION**

By comparing the different failure criteria to the conventional triaxial data it was demonstrated that there is not a singular criterion that provides the greatest agreement with all data sets.

There was, however, a trend in criteria that consistently provided a very good representation ($R^2$ frequently greater than 0.8) of the data sets. These criteria include; the Mogi (1971), Modified Wiebols and Cook (Zhou, 1994), Franklin (1977), Ottosen (1977), Johnston (1985) and Bieniawski (1974) failure models. This is a combination of criteria that consider both two- and three-dimensional stress states. In general, the two-dimensional criteria that had two fitting parameters as well as three-dimensional criteria with three fitting parameters best fitted to the data sets.

There was also a trend in criteria that provided a good representation of the data sets ($R^2$ frequently greater than 0.6). These criteria include: Christensen (1997), Hoek and Brown (1980) and Ramamurthy (Ramamurthy and Arora, 1993) criteria. The Christensen (1997) criterion is three-dimensional types, whereas the Ramamurthy (Ramamurthy and Arora, 1993) and Hoek and Brown (1980) are two-dimensional types. The Hoek and Brown (1980) criterion did provide a relatively good representation of most data sets; however, many other criteria did consistently perform better than it.

Currently, the Hoek and Brown (1980) failure criterion is the industry standard in mining engineering for predicting rock failure. This is mainly due to the way it has been applied in rock mechanics software, its simplicity of application and the manner in which it has been marketed. This study has identified that many alternative criteria provide consistently
better representations of the data sets. The Shima Oyane (1976), Gurson (1977), Sheorey (Sheorey, Biswas and Choubey, 1989) and Spatially mobilised plane (Matsuoka and Nakai, 1974) failure criteria did not produce a consistent result as the coefficient of determination varied significantly between each data set.

The Sheorey (Sheorey, Biswas and Choubey, 1989) criterion tends to suit metamorphic and igneous rock types as it performed well in these analyses. It also performed well in the Japanese analysis which consisted of metamorphic and igneous rocks. The Shima Oyane (1976) and Gurson (1977) criteria performed well in the metamorphic and igneous analysis where porosity was low. However, both criteria performed poorly in the Japanese analysis which consisted of both metamorphic and igneous rocks. This could be associated with the limited number of triaxial data sets used in the Japanese analysis. Further testing needs to be conducted to justify that these criteria are suited to metamorphic and igneous rocks.

The Plane Griffith crack theory (Griffith, 1921), Yoshida (Yoshida, Morgenstern and Chan, 1990) and Hobbs (1970) criteria provided poor representations of all data sets as these criteria returned coefficients of determinations of zero for the majority of the analyses. These criteria were incorporated into this study as they had not been analysed in details previously. The Yoshida criterion is based on the results from eighteen triaxial sets (Yoshida, Morgenstern and Chan, 1990). This highlights that this criterion had not been vigourously applied to a variety of triaxial data during development.

The Hobbs criterion was developed for the use on nine different coals and was later applied to broken rock samples (Hobbs, 1970). This criterion has never been practically applied to intact rock samples. The Plane Griffith crack theory was developed as a hypothetical criterion and has never been practically applied (Griffith, 1921).

**CONCLUSIONS**

This investigation has identified that there is not a singular criterion that provides the best representation of the conventional triaxial data sets. This investigation identified criteria that consistently provided a good fit ($R^2$ frequently greater than 0.8) to the data sets. These criteria included the Mogi (1971), Modified Wiebols and Cook (Zhou, 1994), Franklin (1977), Ottosen (1977), Johnston (1985) and Bieniawski (1974) failure models. The Hoek and Brown (1980) criterion is currently the industry standard in mining engineering for predicting rock failure. This research suggests that alternative criteria can be utilised to predict rock failure with better accuracy. This research also suggests that criteria that do not incorporate porosity produce better representations of the conventional triaxial data sets. There was also no correlation between the inclusion and exclusion of uniaxial compressive strength influencing the accuracy of the representing rock failure.

Finally, the study recognised criteria that produced very poor representations of data sets. The Plane Griffith crack theory (Griffith, 1921), Hobbs (1970) and Yoshida (Yoshida, Morgenstern and Chan, 1990) criteria produced the coefficient of determination of zero for the majority of the analyses. This research suggests that these criteria should not be used for rock failure prediction.

**TABLE 6**

Results for European analysis.

<table>
<thead>
<tr>
<th>Rank</th>
<th>Criterion</th>
<th>$R^2$</th>
<th>Dimension of criterion</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>Modified Wiebols and Cook (Zhou, 1994)</td>
<td>0.971</td>
<td>3</td>
</tr>
<tr>
<td>2</td>
<td>Franklin (1977)</td>
<td>0.97</td>
<td>2</td>
</tr>
<tr>
<td>3</td>
<td>Ottosen (1977)</td>
<td>0.954</td>
<td>3</td>
</tr>
<tr>
<td>4</td>
<td>Johnston (1985)</td>
<td>0.951</td>
<td>2</td>
</tr>
<tr>
<td>5</td>
<td>Hoek and Brown (1980)</td>
<td>0.949</td>
<td>2</td>
</tr>
<tr>
<td>6</td>
<td>Bieniawski (1974)</td>
<td>0.947</td>
<td>2</td>
</tr>
<tr>
<td>7</td>
<td>Mogi (1971)</td>
<td>0.929</td>
<td>3</td>
</tr>
<tr>
<td>8</td>
<td>Ramamurthy (Ramamurthy and Anora, 1993)</td>
<td>0.865</td>
<td>2</td>
</tr>
<tr>
<td>9</td>
<td>Shima Oyane (1976)</td>
<td>0.816</td>
<td>3</td>
</tr>
<tr>
<td>9</td>
<td>Spatially mobilised plane (Matsuoka and Nakai, 1974)</td>
<td>0.816</td>
<td>3</td>
</tr>
<tr>
<td>11</td>
<td>Gurson (1977)</td>
<td>0.816</td>
<td>3</td>
</tr>
<tr>
<td>12</td>
<td>Christensen (1979)</td>
<td>0.738</td>
<td>3</td>
</tr>
<tr>
<td>13</td>
<td>Sheorey (Sheorey, Biswas and Choubey, 1989)</td>
<td>0</td>
<td>2</td>
</tr>
<tr>
<td>14</td>
<td>Hobbs (1970)</td>
<td>0</td>
<td>2</td>
</tr>
<tr>
<td>14</td>
<td>Yoshida (Yoshida, Morgenstern and Chan, 1990)</td>
<td>0</td>
<td>2</td>
</tr>
<tr>
<td>14</td>
<td>Plane Griffith crack theory (Griffith, 1921)</td>
<td>0</td>
<td>2</td>
</tr>
</tbody>
</table>

**FIG 6** – Modified Wiebols and Cook (Zhou, 1994) criterion fitted to the European normalised triaxial data.
ACKNOWLEDGEMENTS

The authors would like to thank Shane Keaveney from the School of Mathematics at UNSW Australia for his assistance during the curve fitting process.

REFERENCES


Integration of Ultramafic Mine Tailings and Acid Mine Drainage for Carbon Sequestration and Mine Waste Management

H Malli\(^1\), W Timms\(^2,3\) and S Bouzalakos\(^3,4\)

ABSTRACT
Mining operations are increasingly expected to minimise their environmental footprint whilst maximising shareholder returns. In this study the possibility of mineral carbonation and carbon sequestration technology are evaluated to assist in reducing the footprint of acid mine drainage, tailings and carbon dioxide emissions produced from mining operations. The technology firstly consists of dissolution to leach magnesium from ultramafic tailings using acidic mine waters. Carbonation is subsequently undertaken by reacting carbon dioxide with the leached magnesium solution to form carbonates. Ultramafic tailings were synthesised from brucite and serpentinite ore. Through variances in solid to liquid ratios and temperatures of dissolution, a maximum five per cent magnesium dissolution efficiency was achieved at 50 g/L and 100°C. It was significant that the modelled mine water was effectively treated by its reaction with the tailings resulting in pH neutralisation (pH 6.5 to 8.5) and a 100 fold decrease in the concentration of heavy metals (iron, copper, zinc). By adopting this technology, a net present value of AS24.5 M over a 14 year life-of-mine was estimated for a hypothetical Australian mine despite significant constraints. The potential economic viability of the technology warrants further research to determine more suitable conditions to maximise efficiency of the dissolution and carbonation process.

INTRODUCTION
Carbon sequestration using ultramafic mine tailings has been a recent focus amongst industry, as a means of promoting sustainability through effective management of carbon emissions and mine waste. The natural weathering of rocks rich in Calcium (Ca) and Magnesium (Mg) silicates results in carbon sequestration. Such silicates react with atmospheric carbon dioxide (CO\(_2\)) to precipitate solid Ca\(_2\)Mg carbonates (Lackner et al, 1995; Seifritz, 1990). Though this process is said to sequester 100 Mt of carbon per year (Seifritz, 1990), the rate of reaction is too slow to render any significant reduction of the greenhouse effect.

However, potential exists for carbon sequestration through the use of mine waste in mineral carbonation technology. Mineral carbonation involves the dissolution of metal ions present in ultramafic tailings/mineral ores, and subsequent carbonation and precipitation of carbonates. Whilst both steps can be performed in the same medium, it is preferred to separate the stages to allow greater flexibility in altering reaction parameters to promote greater carbonation efficiency (Dri, Sanna and Maroto-Valer, 2013). As dissolution has been identified as the rate limiting stage (Haug, Munz and Kliev, 2011; O’Connor et al, 2002) separation of both stages in an indirect means is ideal.

Though Ca/Mg mineral ores are typically used in the process to replicate natural silicate weathering, the use of ultramafic tailings has become an attractive alternative. As these tailings have previously been milled, energy requirements due to comminution of Ca/Mg rich rocks to reduce particle size, are virtually non-existent (Bou Karam et al, 2011).

Dissolution requires the addition of acidic reagents for the release of metal ions, thereby increasing costs and process complexity. Acid mine drainage (AMD) has been recently suggested as an alternative for such reagents. As AMD is readily available at mine sites this presents an attractive substitute. In the context of carbonation, an added benefit of its use in dissolution is the potential treatment of mine water. This negates additional costs associated with conventional means of mine treatment, including the use of hydroxides such as lime and subsequent filtration. However, its effectiveness as a mineral carbonation reagent has not yet been studied in detail (Bouzalakos et al, 2014).

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Thus, a research initiative has been proposed to simulate the use of ultramafic tailings and mine water in the carbonation technology with emphasis on the dissolution component. The broad objective is to assess the suitability of ultramafic mine tailings and mine water in the carbonation technology for both carbon sequestration and mine waste treatment. This involves simulation of the carbonation technology in the laboratory. As the work of Thapa (2013) remains the sole contributor in evaluating the use of mine water and ultramafic tailings with regards to dissolution, further research is necessary.

**EXPERIMENTAL METHODOLOGY**

Simulation of the carbonation technology in the laboratory consisted of:

- feedstock preparation where AMD and ultramafic tailings are synthesised
- dissolution where prepared feedstock are reacted under different conditions to leach Mg from tailings into solution
- carbonation where the leached Mg solution is reacted with CO₂ to form carbonates in which the gas is stored.

Research has indicated dissolution as the rate limiting stage with efforts directed towards enhancing metal extraction from feedstock (Mazzotti et al., 2005). Simulation of this stage under varying conditions (temperature and solid to liquid ratio) allowed for the extent of dissolution, ie amount of Mg leached to be enhanced. Based on these dissolution conditions, the environment that resulted in maximised dissolution was identified and subsequent carbonation undertaken.

**Feedstock preparation**

Feedstock consisting of ultramafic tailings and acid mine drainage were recreated in the laboratory. Mg rich serpentinite rock, likely sourced from the Coolac Serpentinite Belt in New South Wales (NSW) was ground and sieved to a target particle size of less than 100 μm (Sanna et al., 2013; Vogeli et al., 2011). Additionally, to be representative of mine site tailings pure brucite was added at a 95 per cent and five per cent split to the original ground serpentinite. Although brucite can be found between one per cent and 15 per cent by weight in tailings (Harrison, Power and Dipple, 2013), a conservative five per cent composition was chosen.

As the composition of mine water varies amongst mining operations, it is difficult to identify a basic acid mine water model for mine water preparation. Thapa (2013) undertook analysis using synthesised mine water of pH 1.45, although research into Australian operations where AMD has been a major issue highlight variances in pH from 2 to 4 (Chapman and Simpson, 2005; Jacobson and Sparksman, 1988). Based on available literature on the composition of AMD in Australia, mine water was synthesised based on AMD at Captains Flat NSW and Mt Morgan, Queensland. Both operations are volcanogenic massive sulfide deposits, the broader range of classification of ultramafic deposits. AMD at these operations is similar with some discrepancies. This presented a baseline of AMD composition based on available data. Table 1 outlines the target composition of the mine water.

The equivalent mass of salts to obtain the composition outlined in Table 1 was determined, and mine water formed through dissolving these salts in water. To ensure the salts had dissolved, the mine water solution was placed in an ultrasonic vessel for 30 minutes.

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Value</th>
</tr>
</thead>
<tbody>
<tr>
<td>Final pH</td>
<td>2.7–2.9</td>
</tr>
<tr>
<td>Major ions (mg/L)</td>
<td></td>
</tr>
<tr>
<td>Ca</td>
<td>450</td>
</tr>
<tr>
<td>Mg</td>
<td>1250</td>
</tr>
<tr>
<td>Zn</td>
<td>160</td>
</tr>
<tr>
<td>Fe</td>
<td>250</td>
</tr>
<tr>
<td>Cu</td>
<td>35</td>
</tr>
</tbody>
</table>

**Dissolution**

For dissolution, two independent variables were tested – temperature and solid to liquid (S/L) ratio. The reaction variables for dissolution are summarised in Figure 1.

The reaction variables in Figure 1 correspond to those used by Thapa (2013). Controls include a stirring speed of 800 rev/min (Sanna et al., 2013), composition of mine water and tailings, and length of reaction. Since the use of mine waste (tailings and mine water) in the technology has only been simulated by Thapa (2013), reactions variables were kept consistent with the previous study for validation and comparison of results.

Dissolution was modelled using a 500 mL three necked flask to allow for temperature and pH monitoring, as well as sample extraction. The three necked flask was situated in a heating mantle with a condenser attached to minimise evaporation losses during the reaction. For each S/L ratio the amount of tailings per 450 mL of mine water was prepared. The mine water contained in the flask was heated to the required temperature and the pH monitored. At time zero the amount of mine tailings corresponding to the desired S/L ratio (50 g/L or 70 g/L) were added and the reaction left for three hours. Temperature and pH were monitored across the reaction at 5, 15, 60, 60 and 180 minute intervals. This was repeated for all temperatures and S/L ratios, with six reaction scenarios in total. The dissolution experimental set-up is shown in Figure 2.

**Carbonation**

Following identification of the optimal dissolution environment from preceding experimentation, carbonation can be carried out using these ideal dissolution conditions. Carbonation within the study was assessed from a ‘proof of concept level’. As Thapa (2013) had only experimentally...
simulated dissolution, this study aimed to highlight the potential use of the leached Mg solution obtained from the mine waste to sequester CO$_2$. Thus, no optimisation steps were undertaken. Essentially carbonation involved filtration of the optimal dissolution condition that contained the greatest amount of leached Mg. A 10 mL sample of this was extracted to identify the original Mg concentration. The leached Mg solution from dissolution was transferred into a volumetric flask and CO$_2$ bubbled through over a 20 minute period. After 20 minutes, 10 mL of the carbonated solution was extracted for analysis to identify the Mg concentration after carbonation. Inductively coupled plasma optical emission spectroscopy (ICP-OES) analysis of the extracted samples would allow any reduction in Mg to be identified. Such a reduction in free Mg would presumably be due to the precipitation of Mg with CO$_2$ to form magnesium carbonates.

RESULTS AND DISCUSSION

Mine water treatment

The effectiveness of dissolution on mine water neutralisation can be assessed through analysis of pH variation with time. This is summarised in Figure 3.
More than a two-fold increase in pH was apparent within the first 15 minutes across all reaction conditions. This gradually increased to the 30 and 60 minute marks before plateauing. This plateau indicates the systems to be in chemical thermal dynamic equilibrium. The results presented in Figure 3 can be benchmarked against treated effluent standards in the mining industry to assess the effectiveness of treatment. Although this varies amongst countries, essentially all regulatory regimes require treated effluent to have a pH between 6.5 and 8.5 (Younger, Banwart and Hedin, 2002). As shown in Figure 3 this is met for the majority of reaction conditions in the first 30 minutes of dissolution. Thus, effective neutralisation with respect to pH from dissolution allows the mine water to be recycled and reused in mining operations. Potential may also exist to further discharge treated water into the environment. However care must be taken as local water quality guidelines are generally site specific, and may be more stringent than the general guidelines referred to in this study.

Samples of mine water sent for ICP-OES analysis at specified time intervals (5, 15, 30, 60, 120 and 180 minutes) during dissolution, allowed the variation in metal concentration of the mine water to be assessed. As the concentration of heavy metal ions in mine water is important due to their toxicity, their reduction in mine water is an integral objective of mine water treatment methods. The qualitative effect of dissolution on mine water is illustrated in Figure 4. The change in colour of the mine water is due to the reduction of the concentration of heavy metal ions in the mine water.

Samples of the mine water sent for ICP-OES analysis at the outlined intervals during dissolution, allowed the variation in metal concentration to be quantitatively assessed. These include iron (Fe), copper (Cu) and zinc (Zn).

Figure 5 illustrates the variation in Fe concentration during dissolution. Across all reaction conditions there was a substantial decrease in heavy metal concentration as reflected in Figures 5, 6 and 7. The rate of decrease in concentration was greatest for Cu with a more than 200 fold decrease on average, and lowest for Fe with an average 150 fold decrease. Notably there appears to be no general trend between temperature, S/L ratio and the concentration of heavy metal ions aside from that shown in Figure 7. Instead, a general decrease regardless of the initial reaction conditions is evident. In Figure 7 the Zn concentration appears more sensitive to temperature relative to Cu and Fe, with equilibrium reached at a faster rate at higher temperatures.

The demonstrated reduction in heavy metal concentrations is due predominately to precipitation of metal hydroxides, which are reversible reactions and affected by pH (Younger, Banwart and Hedin, 2002). Essentially as the pH of the mine water increases, the solubility of metal ions decrease resulting in the precipitation and reduction of metal concentrations. Benchmarking of the results in Figures 5, 6 and 7 with common treated effluent standards outlined by Younger, Banwart and Hedin (2002) highlight general compliance for Fe, Cu and Zn concentrations. These standards are related to discharge into watercourses that are ecologically sensitive. Such standards require treated effluent standards for Zn to be less than 0.5 mg/L, Cu less than 0.2 mg/L and total Fe less than 1 mg/L. These standards were met for most reactions conditions with respect to Cu and Fe concentration at some point over the reaction. Interestingly, the Zn standard was only achieved in 33 per cent of the reaction environments. Further potential exists to achieve this standard if the dissolution reaction is continued for longer. Similar to Thapa (2013) a significant reduction in heavy metal concentration was apparent although the magnitude of this varies. Notably in the previous study, Fe concentration in the mine water remained below 1 mg/L throughout dissolution.
Coupled with the results from pH neutralisation in Figure 3 and to maximise the reduction in heavy metal concentration, a reaction time of 30 to 60 minutes is efficient for mine water treatment. The technology capitalises on the upside potential to release the treated mine water into watercourses, and for mine water to be successfully recycled in operations following dissolution. Moreover, as effluent is usually mixed and diluted before discharging, a greater reduction in heavy concentration can later be achieved prior to discharge.

Dissolution efficiency

Figure 8 illustrates the variation in Mg concentration throughout dissolution.

Dissolution efficiency refers to the amount of magnesium leached through dissolution, as a percentage of the total amount of magnesium available in the tailings. Equation 1 defines this, noting that the Mg leached into solution is equal to the difference in Mg concentration and the initial concentration of Mg in the mine water at sampled interval times.

\[ \text{Mg dissolution efficiency (\%)} = \left( \frac{\text{Mg leached into solution (mg)}}{\text{Mg available from extraction (mg)}} \right) \times 100 \]  

Application of Equation 1 to the results presented in Figure 8 result in low dissolution efficiencies for experimentation with a maximum of five per cent obtained at 100°C and 50 g/L. The range of dissolution efficiencies varied from two per cent to five per cent across reaction conditions. This contrasts significantly with the work of Thapa (2013) who achieved efficiencies of up to 55 per cent, although different feedstock and reaction time were utilised. Such low efficiencies may be related to the effectiveness of the mine water as an acidic reagent to break the bonds present in the tailings, and leach Mg ions. From literature hydrochloric, sulfuric and acetic acid are typically used to break the strong covalent bonds in Mg-silicates. Thus, the acidic mine water synthesised with a pH of 3 may not be of a comparable strength to be effectively utilised in the process. Under the same reaction conditions, Thapa (2013) achieved significantly higher dissolution efficiencies (>50 per cent) with a varying mine
water composition, and pH of 1.45. Pure brucite also formed the tailings composition in Thapa (2013). Brucite is composed of weak residual bonds (Dutch, 2011). Hence, bonds between magnesium and hydroxide in the mineral are significantly easier to break, in comparison with magnesium and silicate bonds present in serpentine minerals. In this context, for Thapa (2013) mine water may have been sufficient as an acidic substitute. Within this study however, the use of Mg silicate rich minerals requires greater energy to break the bonds due to the complex tetrahedron structure and strong covalent bonds (Kroeger, n.d) of serpentine minerals. The mine water may not effectively provide this energy, which was less acidic relative to Thapa (2013). This potentially inhibited significant amounts of Mg being leached and questions the effectiveness of particular mine water utilised in the technology. Potentially a minimum mine water pH and composition may be required for significant dissolution efficiencies.

Extent of carbonation
Following identification of conditions to achieve maximum dissolution, carbonation was simulated at 100°C and 50g/L. Due to the low maximum dissolution efficiency (five per cent) obtained for these conditions, the subsequent extent of carbonation was uncertain. It was unclear if there would be sufficient Mg to react with CO2 for substantial precipitation and sequestration.

Re-simulation of the maximum dissolution conditions and subsequent carbonation of this Mg rich solution resulted in a 11 per cent reduction of Mg ions in the carbonation solution. It was assumed this reduction was due to the reaction between Mg ions and CO2 during carbonation to form carbonates. As carbonation was undertaken at a ‘proof of concept’ level to illustrate that carbonation can be obtained using the mine waste, optimisation was not undertaken. The reduction in Mg concentration following carbonation although minor, provides evidence on a ‘proof of concept’ level that leached Mg from the reaction between the aforementioned mine waste in dissolution, can potentially be used for carbon sequestration.

COST BENEFIT ANALYSIS
The next component of the project involved a cost benefit analysis. This evaluation was done in comparison with existing mine water treatment costs in an emissions trading scheme context. As the area of research is relatively conceptual with no mines currently operating such technology, it was necessary to assume a hypothetical nickel mine operating in the Great Serpentinite Belt (GSB) in NSW, known as Mine X.

With regards to Mine X to undertake a cost benefit analysis various other assumptions were made. These include:
- Mine X would be producing 3.5 Mt per year of ultramafic tailings with enough AMD on-site to be used in the technology.
- Mine X would have an existing water treatment plant that would transition to a carbonation and waste treatment facility.
- Mine X would form part of a larger nickel operation consisting of a smelter and concentrator. Mine X and its parent company would be accountable for CO2 emissions released from processing Mine X’s output.
- Mine X would be producing 200 000 t of CO2 equivalent emissions per year from operations including downstream processing emissions that are attributable to Mine X’s output.
- Mine X is in close proximity to a power station such as those in the Hunter Region retrofitted with CO2 capture and storage technology.
- A fully established ‘cap and trade’ emissions trading scheme (ETS) is in place, similar to the current European ETS and North American ETS.

Such assumptions were benchmarked from existing literature and appropriate concessions applied.

Cost model parameters
Cost benefit analysis (CBA) consists of development of a cost model based on the assumptions of Mine X. Discounted cash flow analysis can then be completed to estimate the net present value (NPV) that would be added to Mine X if it implemented the technology for carbon sequestration and mine water treatment. Figure 9 provides an overview of the project’s CBA process.

In the case of Mine X, cost benefit analysis is applied on the premise that the operation is able to offset all its carbon emissions, with the mine’s net carbon emissions being zero. Hence, it does not need to purchase any carbon credits on the ETS market. Cost savings can be taken as revenue generated from carbon offsets. However, in light of Mine X’s tailings
production and AMD an additional 510 000 t of CO₂ can be sequestered inclusive of their own 200 000 t of emissions based on the relationship between feedstock requirements and CO₂ sequestered as outlined by Priestnall (2013). The close proximity of Mine X to operations such as Maules Creek as well as being within 200 km of power stations in the Newcastle region, it can further sequester their carbon emissions at a price. This price however is substantially lower than the market price of carbon, thereby allowing Mine X to engage in forward contracts with other emitters to sequester their carbon emissions and reduce their carbon liability. This presents application of Industrial Symbiosis as highlighted by Brent et al (2011). As the nature and legality of such contracts concerning carbon prices has not been established, this is an optimistic and conceptual revenue stream.

An additional benefit of the technology is that it can be used to treat mine water. Mine X incurs cost savings as it does not have to treat its mine water, resulting in annual cost savings amounting to A$7.4 M per year (Veolia, 2013). Notably the cost savings associated with carbon offsets and mine water treatment are truly a negative cost, however, for simplicity and ease of understanding were referred to as a revenue (benefit) in this study.

The aforementioned production model and cost model outlined in Figure 9 was derived from literature, namely Hitch and Dipple (2012), Power et al (2014) and Priestnall (2013). In such literature the use of Mg rich ore such as serpentinite in the carbonation process is analysed, rather than the use of mine water with little emphasis on Mg rich tailings. Thus, appropriate concessions were applied to researched values to account for the reduction in costs associated with using AMD and ground tailings. As the technology is conceptual and not currently implemented the basic nature of costs and prices from literature is stressed to highlight the preliminary nature of the analysis.

Discounted cash flow analysis
Discounted cash flow (DCF) analysis was completed for Mine X over a life-of-mine (LOM) of 14 years based on

Figure 9. Incorporation of the technology into Mine X results in a NPV of $24.5 M and internal rate of return (IRR) of 12 per cent. This is summarised in Figure 10.

Sensitivity analysis
Due to the wide nature of assumptions required for CBA, sensitivity analysis was undertaken to test these assumptions and their impact on NPV. Sensitivity analysis is commonly done with respect to a ±10 per cent change in variables and their effect on NPV. However due to the conceptual nature of the technology and numerous assumptions undertaken this was extended to ±30 per cent. The results of this sensitivity analysis are presented in Figure 11.

As seen in Figure 11, the greatest parameters to influence the NPV of the technology are operating cost and tailings
produced. As these parameters amongst others were benchmarked and appropriate concessions applied, it is integral that they be validated and implicitly derived to obtain true costs and prices. This will allow for a more realistic NPV to be determined. As shown in Figure 11 minor deviations of ten per cent have major implications on the economic feasibility of the project. Essentially however this cost benefit analysis does highlight potential markets for the technology, and provides a foundation for further financial modelling.

CONCLUSIONS

Based on the results of this study, potential exists from both an economic and technical standpoint to integrate mineral carbonation technology into operations for minimising mine waste and for carbon sequestration. Additional testing and enhancement of the experimental method is needed to produce more conclusive and accurate results. The results indicate alternative pathways must be studied in order to develop the technology into a value adding chain in mining operations.

The CBA analysis in this study is a conceptual first pass evaluation. The lack of established markets and operations to implement the technology make it difficult to undertake CBA. However, using the method outlined potential exists to add economic value to mining operations through incorporation of the technology, namely a $24.5M NPV through its use of a 14 year period.

In relation to carbon sequestration via mineral carbonation, this study provides ‘proof of concept’ of utilising Mg rich mine tailings and acidic mine water as feedstock for the carbonation process. This was evident in the extraction of Mg from the synthesised tailings and mine water, followed by the subsequent precipitation of CO₂, assumed as carbonates. However, the low dissolution efficiencies (maximum of five per cent achieved) obtained question the applicability of the particular mine water used in the process. As the composition of mine water is site specific, on an industrial scale the extent of dissolution and carbonation will vary amongst operations. Thus, the applicability of the technology may be site specific amongst operations.

Further testing and analysis is needed to further enhance dissolution, and develop a more reliable assessment of the technical viability of the technology. Whilst the effectiveness of dissolution was limited under the conditions selected for these experiments, mine water was successfully treated. A significant reduction in heavy metal concentration of more than 100 per cent of the original concentrations in the mine water, occurred during dissolution and was generally compliant with generic standards for discharge to fresh waters. More than a two-fold decrease in pH neutralisation was also apparent in the treated mine water. Although low dissolution efficiencies were obtained for carbonation and in turn carbon sequestration, an alternate pathway for mine water treatment is presented. Moreover, potential exists to increase the effectiveness of the mine water as a leaching agent in dissolution through improving the experimental method such as the addition of chemicals and preactivation of tailings, as well as analysing the effect of the composition of mine water on dissolution. Further research is necessary to assess whether the benefits from such enhancements outweigh additional costs.

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FIG 11 – Sensitivity analysis.
MINING EDUCATION AUSTRALIA – RESEARCH PROJECTS REVIEW


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INTRODUCTION

There is an abundant amount of resources that can be found on asteroids in outer space. The majority of asteroids are found in the asteroid belt between Mars and Jupiter. Even though this may be the case, there are at least one thousand near-Earth asteroids (NEAs) that are ready to be mined provided that it is feasible (Sonter, 2013).

The problem with mining these asteroids is that there is currently no suitable design of equipment that is feasible to extract minerals from an asteroid for prospecting. To design potential mining equipment, methods to break rock, reducing excessive transportation costs and the maintenance of the equipment need to be considered.

One proposed solution is the application of shock heating and freezing to fragment regolith found on asteroids for exploration drilling.

SPACE ENVIRONMENT

Over the centuries, the power of technological advancements has enabled the global demand of scarce or non-renewable resources to be satisfied (Neumayer, 2000). As of 2014, there are over 100 000 asteroids that have been discovered in outer space (The Minor Planet Center, 2013). Like most deposits on Earth, asteroids have been classified into three main categories (NASA, 1996):

- **C-group (carbonaceous)** – over 75 per cent of known asteroids are classified as C-type. These are made up of hydrogen, helium and other volatiles.
- **S-group (siliceous)** – approximately 17 per cent of known asteroids are classified as S-type. Their composition is of metallic iron mixed with iron and magnesium silicates.
- **X-group (metallic)** – majority of remaining asteroids. The composition of these asteroids is mostly metallic iron.

This classification system was reinforced by utilising Earth-based remote sensing such as those obtained by spectroscopic analysis, Galileo flybys, and laboratory analysis of meteorites (Bus and Binzel, 2002).

The use of spectroscopy has concluded that S-type asteroids are predominately found in the inner main belt while C-type asteroids are found in the outer main belt. Nevertheless, there are asteroids with different spectral properties present among these (Nesvorny et al, 2005). For C-type asteroids, the absorption band of bound water is observed by spectroscopy that motivate the assumption made by Shestopalov and Shustarev (2008) which states that a C-type asteroid’s surface consists of hydrated clay silicates due to a low temperature aqueous metamorphism.

From observations and numerical and experimental simulations, the surface properties of asteroids are made up of regolith, which is built up of irregular particles that make the near surface density seem lower than the bulk density. It is said that the deeper regolith layers may be relatively porous and filled with water (Levasseur-Regourd, Hadamcik and Lasue, 2006). Price (2004) concluded that smaller sized asteroids have high levels of porosities of at least 20 per cent while larger asteroids have lower levels of porosities. It was also observed that larger asteroids have a regolith layer of...
at least a few centimetres deep while smaller asteroids had a much thinner layer. Price (2004) also noted that this is debatable as there are always exceptions, such as the case observed in ‘433 Eros’, which has an abnormally deep regolith layer of several metres.

The numerous collisions that occur in the main asteroid belt results in a lot of fragmented particles and highly elliptical orbits, which is due to the gravitational pull of Jupiter. NEAs are mainly made up of these asteroids from the main belt but possibly also include nuclei of evolved or extinct comets (Harris and Davies, 1999). It has been suggested by Campins et al (2010) that the Earth’s current supply of water was delivered by asteroids. This presence of water or ice on asteroids has been inferred from several comet-like asteroids such as 24 Themis. The composition of a comet is mainly of ice, organic material and other minerals that reflect the chemistry of the outer regions of the solar system (Bockelee-Morvan, 2011). Some of these comets have lost all their ices or are coated by an insulating dust mantle (Levasseur-Regourd, Hadamcik and Lasue, 2006).

In the context of asteroids, the temperature of asteroids is warmer than outer space because the asteroid absorbs heat from the sun. An example of the temperature of an asteroid is the NEA ‘433 Eros’, which has a temperature of 100°C during the day when the sun is radiating heat and a temperature of negative 150°C at night (Haberman, 1998). It should be noted that asteroids further away from the sun will have a lower temperature than the asteroids closer to the sun but will be warmer than outer space (-270°C).

There are three ways that the surface of any planetary body may be heated up that are thermal: conduction, convection and radiation (Kömle et al, 2011).

ROCK BREAKAGE

Satish, Radziszewski and Ouellet’s (2005) conclusion of the medium to high applicability of thermal and mechanical rock-breakage techniques motivated the proposal for drilling equipment to employ a hybrid system consisting of thermal rods and impact hammers to fracture rock.

Fire-setting

Fire-setting is described as the softening or cracking of the working face of a lode, to facilitate excavation, by exposing it to the action of a wood fire built against it (Weisgerber and Willies, 2000). Diodorus Siculus was a Greek historian who quoted Agatharcides (2nd century BCE) on the use of fire-setting:

_The Earth which is hardest and full of gold they soften by putting fire under it, and then work it out with their hands. The rocks thus softened and made more plant and yielding, several thousands of profligate wretches break in pieces with hammers and pickaxes._ (Hoover and Hoover, 1950)

Fire-setting is still used in small scale operations but not as common as drill and blast. The use of fire-setting still occurs whenever it is economic such as in some granite quarries in India (Craddock, 1996) and during special geological circumstances to weaken tough rocks and then conventional methods such as picking and wedging are applied to extract the rock.

PREVIOUS EXPERIMENTS

There are numerous amounts of experiments that assist and challenge the idea of utilising heating and freezing as a rock-breakage technique in space. Lindroth and Krawza (1971) hint that a temperature less than 600°C will be sufficient for breaking rock to be effective. Charles (2011) experiment involved the heating and freezing of a meteorite sample in atmospheric conditions. The heating was completed with a water bath at a temperature 80°C while the freezing was completed with the use of liquid nitrogen. His results indicated that the meteorite sample eventually fractured after a sufficient amount of cycles of heating and freezing was performed. Fuji and Osako’s (1972) and Sakatani et al (2012) point out that the vacuum environment impedes the transfer of heat as it essentially acts as an insulator. The factors that increase the transfer of heat is using a higher temperature or going deeper (compacting the rock sample). Haynes and Mellor’s (1977) results conclude that the UCS of ice is approximately 2.06 MPa with the use of conventional UCS testing. They also suggested modifying the platens of the universal testing machine to better reflect the true UCS of ice.

EXPERIMENTAL APPARATUS

Figure 1 illustrates the design of the experimental set-up that was utilised to determine the applicability of shock heating and freezing. This experiential apparatus’ function specifically completes the shock heating process. It should be noted that it is used to simulate the vacuum environment that exists in space.

SIMULATED ROCK SAMPLE

It was suggested that asteroids are quite porous (Price, 2004); however, they are insulated by a dust layer (Levasseur-Regourd, Hadamcik and Lasue, 2006). As a result, it was decided to utilise lunar soil simulant with the addition of water to simulate the asteroid sample used in testing.

The lunar simulant used for the experiments was the Australian Lunar Regolith Simulant (ALRS-1). Table 1 gives a comparison of the ALRS-1 with JSC-1. The use of ALRS-1 was suitable as it was quite similar to JSC-1, which is regarded as an ideal lunar simulant (MaKay et al, 1994).

It was decided to have three test methodologies; samples that contained 25 per cent, 50 per cent and 100 per cent water. Figure 2 shows the mould that was used, which consists of a copper pipe with an internal radius of 64 mm as well as the specially designed polyvinyl chloride (PVC) cap with a rubber washer to seal any leaks that may occur. The use of
copper allowed the sample to be cooled rapidly when frozen but also allowed the sample to be removed quite quickly due to copper’s high thermal conductivity (Chung, 2001).

**UNIAXIAL COMpressive STRENGTH TEST**

The operation of the loading frame was utilised to compare the compressive strength of the rock samples. As the rock samples were relatively weak, a 100 kN transducer was used. The use of this transducer allowed much more sensitive readings to be obtained as seen in Figure 3.

**METHODOLOGY**

The experiments were aimed at determining whether the rock sample became weaker after the application of heat and freezing. The independent variables were the vacuum environment, dry ice temperature control, heating temperature and the size of the sample. The dependent variables were the water content of the sample(s) and the amount of cycles that the samples were subjected to the application of heating and freezing.

Determining the effectiveness of heating and freezing rock as an alternative rock-breakage technique to be used in asteroid drilling will be dependent on the degradation in strength due to shock heating and freezing. The procedure that was followed for a sample to undergo shock heating and freezing was:

1. The frozen sample was placed into the vacuum chamber.
2. The sample was heated at 300°C for three minutes as it was found to be suitable based on Charles’ (2011) experiment, which recommended that the freezing time should be four times less than the thawing (heating) time.
3. The sample was removed from the vacuum chamber and immediately placed in dry ice (sublimation temperature of -78.5°C).
4. The sample was frozen for five minutes in the dry ice, which achieved an effective temperature of approximately -50°C. The reason why this prolonged freezing time was used was to compensate for the lower freezing temperature in comparison to Charles’ (2011) experiment.
5. Steps 1–4 were repeated five or ten times depending on the number of cycles of heating and freezing that was required as decided for the three methodologies. (In this experiment the samples are heated and frozen for zero, five and ten cycles).
6. The sample was removed from the mould by taking off the cap first and then sliding out the sample. This was found to be easy as the mould was copper.
7. A handsaw was used to ensure the faces were flat so that results were accurate for the loading frame.

**TABLE 1**

Comparison of composition of ALS-1 and JSC-1 (Bernold, 2013).

<table>
<thead>
<tr>
<th>Chemical oxide</th>
<th>ALS-1</th>
<th>JSC-1</th>
<th>Difference</th>
</tr>
</thead>
<tbody>
<tr>
<td>SiO₂</td>
<td>42.36</td>
<td>47.71</td>
<td>-5.35</td>
</tr>
<tr>
<td>TiO₂</td>
<td>2.73</td>
<td>1.59</td>
<td>1.14</td>
</tr>
<tr>
<td>Al₂O₃</td>
<td>13.48</td>
<td>15.02</td>
<td>-1.54</td>
</tr>
<tr>
<td>Fe₂O₃</td>
<td>12.55</td>
<td>3.44</td>
<td>9.11</td>
</tr>
<tr>
<td>MnO</td>
<td>0.18</td>
<td>0.18</td>
<td>0.00</td>
</tr>
<tr>
<td>MgO</td>
<td>10.23</td>
<td>9.01</td>
<td>1.22</td>
</tr>
<tr>
<td>CaO</td>
<td>8.61</td>
<td>10.42</td>
<td>-1.81</td>
</tr>
<tr>
<td>Na₂O</td>
<td>3.29</td>
<td>2.70</td>
<td>0.59</td>
</tr>
<tr>
<td>Cr₂O₃</td>
<td>0.00</td>
<td>0.04</td>
<td>-0.04</td>
</tr>
<tr>
<td>FeO</td>
<td>0.00</td>
<td>7.35</td>
<td>-7.35</td>
</tr>
<tr>
<td>K₂O</td>
<td>1.49</td>
<td>0.82</td>
<td>0.67</td>
</tr>
<tr>
<td>P₂O₅</td>
<td>0.53</td>
<td>0.66</td>
<td>-0.13</td>
</tr>
<tr>
<td>SO₃</td>
<td>0.02</td>
<td>0.00</td>
<td>0.02</td>
</tr>
<tr>
<td>Loss on ignition</td>
<td>4.50</td>
<td>0.71</td>
<td>3.79</td>
</tr>
<tr>
<td>Total</td>
<td>99.97</td>
<td>99.65</td>
<td>0.32</td>
</tr>
</tbody>
</table>

**FIG 2** – Sample mould (copper pipe and PVC cap with internal rubber washer).

**FIG 3** – Universal testing machine load frame with a 100kN transducer.
8. The cut sample was placed into the dry ice container while the settings of the loading frame were inputted.
9. The sample was placed into a plastic bag prior to loading it into the testing machine as it was found that the regolith sample melted during the test.

RESULTS

Figure 4 shows the test results for 25% water sample. This sample was observed to be the most plastic out of all the samples. This because of the observed change in the force was relatively steady in comparison to the other samples. The application of ten cycles of heating and freezing resulted in a UCS of 2.31 MPa from an original UCS of 2.71 MPa for this sample.

Figure 5 shows the test results of the 50% water sample. This sample was observed to be the strongest out of the three samples tested. The resultant UCS was found to be 8.78 MPa after 10 cycles of heating and freezing, which is down from 10.25 MPa from the control case.

Figure 6 shows the test results of the 100% water sample. It can be seen that the ice (100% per cent water) sample is quite brittle as there is little data available in comparison to the other rock samples (results stop being relevant at a displacement of 6 mm). As a result, it was the weakest sample out of the three regolith sample types tested with a maximum UCS for ten cycles of 0.61 MPa in comparison to 1.85 MPa obtained from an undisturbed sample.

Sublimation

It was observed that through the heating and freezing process, the sample lost weight due to the sublimation of the water. Sublimation occurs as the boiling point of water is much lower in a vacuum environment and as a result the water vapour is expelled from the chamber. Figure 7 shows that during the initial stages of heating, sublimation is most effective, which would explain why there was the most weight lost during these periods. The 25% water sample initially had the highest weight loss as it was observed that dust particles were projected from the sample during sublimation. It should be noted that the 100% water would eventually boil off as seen in Figure 7 due to the consistent weight loss.

SIGNIFICANCE OF UNIAXIAL COMpressive STRENGTH RESULTS

Transport cost is estimated to be over $10 000 per kilogram sent from the Earth to outer space (Sonter, 2013). Mechanical rock-breakage equipment such as roadheaders are generally limited for us in intact rocks that have an unconfined compressive strength of less than 100 MPa (Rostami, 2011).

It was found that utilising shock heating and freezing repeatedly lowers the UCS of various samples that were tested by promoting expansion and sublimation, causing the rock to internally fracture. The use of heating and freezing rock may also be ideal as it may lower the need for frequent bit replacement associated with harder rock types when utilising mechanical rock-breakage methods. There is potential for further investigation in optimising the rate (time between cycles) of heating and cooling, which might impact the UCS of the samples.

CONCLUSIONS

There will be a time when resources will be depleted from the Earth as it is a finite body. It is inevitable that if we are to progress as humans we must eventually obtain minerals from outer space. The major hurdle for this is the excessive costs needed to combat the challenging space environment. Novel or unconventional methods must be developed to enable humans to mine in outer space. The idea of heating and freezing rock was suggested, which was based on the ancient rock-breakage technique known as fire-setting. In a possible off-Earth mining operation, the surface of an asteroid will be
heated and naturally frozen due to the space environment. This process could be repeated numerous times due to the limited physical contact associated with this technique.

An investigation on the suitability of shock heating and freezing was undertaken, which involved designing a vacuum environment with a sample that resembled an asteroid. To assess this, a universal testing machine was used to measure the degradation in strength.

It was seen that rock strength decreases with more heating and freezing cycle. The 25 per cent water sample’s UCS decreased from 2.71 to 2.31 MPa after the application of ten cycles of heating and freezing. Similarly the 50 per cent water sample’s UCS decreased from 1.84 MPa to 0.61 MPa. It was also seen that the rock composition is vital in determining the rock strength as a sample containing 50 per cent water had a UCS approximately four times greater (10.2 MPa) than a sample containing 25 per cent (2.71 MPa). It was observed that sublimation occurred during the heating process, which also projected dust away.

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INTRODUCTION

In the global competitive mineral commodities market there are high pressures on mining operations to maximise reserves, optimise production and increase profits by reducing costs. The conventional stability graph by Mathews et al. (1981) is a qualitative design tool used to determine the potential stability of stopes. When used correctly it can provide mine planners with optimal stope dimensions. Optimal stope sizes result in less development costs and in turn, lower mining costs and dilution.

The paper presents a generalised quantitative dilution-based stability graph independent of orebody width for open stope design. This is achieved by applying Logistic regression and Bayesian likelihood discriminant methods to stope performance data collected from various operating underground metalliferous mines in Australia. A generalised quantitative dilution-based stability graph has more attached value to miners and mine planners as they appreciate dilution numbers more and their implication to the profitability of the operation than merely knowing whether a stope is stable, unstable or cave in the case of the conventional stability graph by Mathews et al. (1981). As the proposed stability graph is orebody width independent it can easily be applied effectively to both wide and narrow-vein orebodies, whereas the conventional stability graph developed by Mathews et al. (1981) and the equivalent linear overbreak slough (ELOS) stability graph (Clark and Pakalnis, 1997) are only applicable to wide and narrow-vein orebodies respectively.

BACKGROUND

The stability graph method for open stope design was developed by Mathews and co-workers at Golder Associates (Mathews et al. 1981) for predicting stable spans in open stope mining at depths below 1000 m.

The stability graph is a plot of the stability number N against a hydraulic radius \( HR \) and shape factor \( S \) (Figure 1). In Figure 1, if the stability state of a stope plots in the stable zone it means that surface has a high probability of being stable, unstable or cave. The unstable zone is sometimes referred to as the supportable zone. A stope surface plotting in the cave zone implies there is a high probability about 30 per cent of the stope surface will slough but not in the sense of caving as in block caving. The three stope stability states in Figure 1 are separated by what are referred to as transition zones.

ABSTRACT

With decreasing profit margins, optimising the size of stopes to minimise dilution is a step towards achieving a productive and profitable mining operation. The stability graph was developed to provide guidance in the determination of stope sizes to control dilution in bulk mining. Unfortunately, this graph is qualitative and stopes can only be described as stable, unstable or cave. The alternative to the qualitative stability graph is the equivalent linear overbreak slough (ELOS) stability graph. The original stability graph is only applicable to wide orebodies while the ELOS stability graph only applies to narrow-vein orebodies and does not provide explicit quantitative dilution values. The objective of this research is to develop a generalised quantitative dilution-based stability graph independent of orebody width. To achieve this objective a total of 226 stope case studies on open stope performances were gathered from six current underground metalliferous mining operations located across Australia. The data was statistically analysed using Logistic regression and the Bayesian likelihood discrimination method to produce quantitative dilution-based stability graphs independent of orebody widths. The graphs provide the mining engineer the flexibility to design open stope sizes based on what dilution amounts are acceptable to a given operation. While the graphs were developed from a statistically significant database, additional data could improve the confidence and reliability in their use. It should also be noted that while the database is from Australian mines, the graphs could be used elsewhere with similar geological and mining conditions to those used in the study.

Development of a Generalised Dilution-based Stability Graph for Open Stope Design

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The stability number and hydraulic radius are defined in Equations 1 and 3 respectively.

\[
N = Q' \cdot A \cdot B \cdot C \tag{1}
\]

\[
Q' = \frac{RQD \cdot J_r \cdot J_n}{J_a \cdot SRF} \tag{2}
\]

where:
- \(Q'\) is the modified tunnelling quality index defined in Equation 2
- \(Q'\) is obtained from the Tunnelling Quality Index, \(Q\) (Barton, Lien and Lunde, 1974) defined in Equation 3 by setting \(J_a\) and \(SRF\) to 1.

\[
Q = \frac{RQD \cdot J_r \cdot J_n}{J_a \cdot SRF} \tag{3}
\]

In Equation 1:
- \(A\) is defined as the stress factor
- \(B\) is the joint orientation factor
- \(C\) the gravity factor

In Equations 2 and 3:
- \(RQD\) is the rock quality designation
- \(J_n\) is the joint set number
- \(J_r\) is the joint roughness number
- \(J_a\) the joint alteration number
- \(J_w\) the joint water reduction factor
- \(SRF\) is the stress reduction factor

The hydraulic radius, \(HR\), is defined by Equation 4 as:

\[
HR = \frac{Area}{Perimeter} \tag{4}
\]

where:
- \(Area\) refers to the stability of the stope surface investigated (eg the hanging wall of the stope)
- \(Perimeter\) refers to the distance around the stope surface investigated (eg the stope hanging wall whose area is the numerator)

Note that in the stability graph, stability of each surface of the stope is determined independently.

As time has progressed there have been several significant developments which have improved the reliability of the stability graph. These developments are based on an increasing database as well as continued experience in the use of the stope design method.

The original database used to create the stability graph composed of 26 case histories of stopes from three mines. This database was expanded by Potvin (1988) to 175 case histories from 34 mines resulting in the recalibration of the parameters \(A\), \(B\) and \(C\) in Equation 1. When the recalibrated factors \(A\), \(B\) and \(C\) are used to determine the stability number, \(N\), it is referred to as \(N\)-prime, \(N'\) (the modified stability number). Potvin also redefined the original stability zones into either stable or caved with an inferred supportable limit in between the unstable and cave zones. Potvin’s transition zones were ‘eyeballed’. Nickson (1992) redefined the zones statistically using discriminant analysis. The zones are shown in Figure 1.

Potvin’s redefinition of the stability graph transition zones was criticized by Stewart and Forsyth (1995) who argued that the stability graph method is non-rigourous and that Potvin’s definition of the zones implied the stability graph is a rigourous design method when it is not.

Mawdesley, Trueman and Whiten (2001), Suorineni (1998) and Diederichs and Kaiser (1996) used statistics to interpret the stability states of stopes in the stability graph in order to dispel the notion that a stope surface plotting in a zone means that stope will definitely behave in that manner.

Mawdesley, Trueman and Whiten (2001) increased the stability graph database to an impressive 483 case histories. These authors used logistic regression analysis to define new stability graph zones different from those shown in Figure 1 that are commonly used. They referred to them as minor, major and cave zones. The cave zone in Mawdesley, Trueman and Whiten (2001) Extended stability graph is different by definition from those defined in the commonly used stability graph by Nickson (1992) and imply caving (progressive unravelling and vertical propagation) as in block caving.

The stability graph method is empirical and as the database used to develop it grows, so does the reliability and accuracy of the method. However, one has to be cautious in using the stability graph databases as they are not based on the same factors. It is important to note that the extended Mathews stability graph by Mawdesley, Trueman and Whiten (2001) is based on the uncalibrated stability graph factors \(A\), \(B\) and \(C\) that were based on only 26 case histories as basis for proof of the stability graph concept but did not establish confidence and reliability in its application. These authors argue that there is no significant difference between the original stability graph factors and the calibrated factors.

**Limitations of the stability graph method**

Limitations of the stability graph method were identified soon after its introduction. These limitations are comprehensively presented and discussed in Suorineni (2010). Some limitations in the design of the stability graph method include not considering the impact that blasting quality has on the surrounding rock masses. The stability graph does not also consider backfill abutments and assumes all stope surfaces are in situ rock. Most often in open stope mining, backfill is employed and depending on the stoping sequence some stope faces may be backfill, whose stability is important in managing dilution. At the moment backfill stability in open stopes is handled outside the stability graph using the Mitchell theory (Mitchell, Olsen and Smith, 1982). The stability of backfill in open stopes is becoming increasingly significant as more mining operations are looking at extending mine life by undertaking pillar extraction, where the surrounding or nearby walls may be backfill material.

The extended stability graph database case histories include stope stability states performance based on visual observations.
rather than quantitative cavity monitoring survey (CMS) data. CMS is used to monitor stability of cavities, and is frequently used in assessing open stope performance for dilution determination. Also, the inclusion of case studies of caving from underground block caving, longwall coal mining and cut-and-fill is inappropriate (Suorineni, 2010) as this results in mixing entry and non-entry mining methods with a consequence on safety. Some amount of failure is acceptable in non-entry mining methods but cannot be tolerated in entry methods such as cu-and-fill.

The original stability graph was designed for large bulk ‘non-entry’ stopes. Recently, the stability graph method has been inappropriately applied to the design of open stopes in narrow vein orebodies resulting in misleading stability outcomes, further hindering the reputation of the stability graph method (Suorineni, 2010).

DILUTION

While there are numerous definitions for dilution, this project primarily focuses on unplanned overbreak, which occurs as a result of instability within a stope. The Equivalent Linear Overbreak Slough, ELOS (Clark and Pakalnis, 1997), is an indirect quantitative measure of dilution. ELOS is defined in Equation 5. The ELOS stability graph database is developed from narrow vein mines.

While the ELOS concept can be applied to narrow and wide orebodies when it is converted into per cent dilution in stopes for the two types of orebodies the difference in values and potential impact on profitability is huge. Equation 6 provides a means for converting ELOS values into per cent dilution and forms the basis for this paper.

\[ ELOS = \frac{\text{Volume of slough}}{\text{Stope wall surface area}} \]  

Dilution (%) = \( \frac{ELOS}{\text{Orebody width}} \) \times 100  

METODOLOGY

In order to develop a generalised quantitative dilution-based stability graph independent of orebody width for open stope design, a new stope stability graph database is required.

Data collection and validation

Data collection was undertaken at a number of mines in the New South Wales Central West. The data was collected firsthand via stope notes, geotechnical reports and cavity monitoring surveys. At each site, a baseline data set was developed to allow for benchmarking and validation of all other data provided. Additional data was sourced from a number of underground metalliferous operations in Western Australia, Queensland and Victoria.

Data validation was undertaken to identify any outliers within the database, and discussing results with site geotechnical engineers. More importantly, all geotechnical stability data used within the project were determined using the same prescribed calibrated stability number parameters.

As this project focused on the development of a dilution-based stability graph, all dilution values provided were assessed for accuracy. This was achieved by the following techniques:

- When availability of site data allowed, other dilution equations were used to cross reference dilution values determined from ELOS. Comparisons were made between calculated dilutions and site provided stope dilutions for validation.
- Dilution was determined via a comparison between the initial designed stope shape and the final resultant stope shape measured by CMS. This method is the most objective way of determining stope dilution and overbreak. By comparing the initial design size/volume and the final design size/volume, and using Geovia Surpac™ to determine the difference in volumes and size a dilution value could be determined and validated. Figures 2 and 3 illustrate this method.

![FIG 2 – Stope design and cavity monitoring survey outline.](image1)

![FIG 3 – Cross-sectional view open stope dilution validation with cavity monitoring survey outline.](image2)
Due to the sensitive nature of the stope stability data the names of the mines have been kept confidential by identifying them by letters only.

**Statistical approaches**

In order to increase confidence and reduce subjectivity and bias, statistical analysis was used to define the boundaries between dilution categories. Two statistical approaches were used: Logistic regression and Bayesian likelihood statistic.

Logistic regression is used to analyse and statistically delineate dilution zone boundaries within the stability graph. Logistic regression analysis has advantages over other traditional regression techniques such as ordinary linear regression, which does not effectively consider the discrete nature of the dependent variable being examined (Laio, 1994). Logistic regression applies a non-linear transform to transform a linear combination of independent variables to a binary output of zero or one. For each of the resultant outcomes of case variables, a logit function was used to estimate the probability of the event occurring.

The logit model (Equation 7) was adopted from Mawdesley (2002). Data input into the logit regression model composed of two independent variables: the modified stability number, $N'$, and the hydraulic radius, $HR$. Then, using a binary logit function the probability of a stope surface fitting into the two dependent variables is assessed. This predicted logit probability value can be compared with the original values, allowing for analysis of misclassified cases. In binary logistic regression the logit value produced represents the natural logarithm of the odds, in which the odds indicate the relative probability of the case being classified into one of the two categories (Mawdesley, 2002).

The Logit model is given by Equation 7, which incorporates both $N'$ and $HR$ for use within the stope stability graph. The values $\alpha$, $\beta_1$ and $\beta_2$ are estimated using the maximum likelihood method which is derived from binomial distribution (Bergerud, 1996).

$$ z = \alpha + \beta_1 \log HR + \beta_2 \log N' $$  

(7)

Microsoft Excel add-on software XLSTAT was used to determine the Logit values and predicted Logit probability values.

Another statistical method for determining the dilution category boundaries in the stability graph is the Bayesian likelihood discriminant analysis using equiprobability contours and the likelihood ratio approach. This approach was used by Suorineni (1998). The likelihood ratio approach in discriminant analysis is derived from Bayes’ Discrimination rule which states that ‘assign the object to the group with the highest conditional probability’. When applied to the stability graph it considers all the posteriori probabilities of a specific case point falling into one of two discrete categories based on the highest probability of the dependent variable and how other case points have been assigned. The likelihood ratio ($\Lambda$) and equiprobability methods assume that two data groups are normally distributed (Suorineni, 1998). The likelihood ratio is given by Equation 8, where the example of the two dilution categories <5 per cent and 5–10 per cent is used.

$$ \Lambda = \frac{f_{<5\%}(X)}{f_{5\%}(X)} $$  

(8)

The equiprobability contour method also known as the approximate method is used to determine the degree of overlap between data categories from which the transition boundaries between the data groups can be inferred.

The equiprobability-contour method also known as the approximate method is used to determine the degree of overlap between data categories from which the transition boundaries between the data groups can be inferred.

Discriminant analysis is characterised by its ability to successfully determine the degree of overlap in data categories. An increasing degree of overlap can result in diminishing success and usefulness of discrimination. Consequently, a separability index, $q$, was developed by Suorineni, Tannant and Kaiser (2001) in order to account for the amount of overlap between categories. This separability index, $q$, is defined as the ratio of the distance between the centroids of two dilution categories such as dilution zone categories $X_{<5\%}$ and $X_{<5\%}$ and the pool variance of both of the two data categories $S$ as in Equation 9.

$$ q = \frac{X_{<5\%} - X_{<5\%}}{S} $$  

(9)

By applying the equiprobability-contour method to the data, contours of equiprobability can be determined surrounding the centroid of a category, where the overlap between the equiprobability contours of different categories, define the boundaries between the categories.

The likelihood ratio, $\Lambda$, defined in Equation 8 represents the likelihood function of two data categories or normal density functions of a bivariate system in a pair of data categories which form a multivariate data set (Suorineni, 1998). For dilution-based stability graph analysis, the pair of data categories is the two dilution categories for case stope surfaces. Each category is assessed with its following category to determine the degree of overlap and to develop the equiprobability contours and resulting boundaries. The likelihood ratio $\Lambda$ defined in Equation 8 can be represented by Equations 10 and simplified in Equation 11. Equation 11 defines the transition boundaries between different categories. Coefficients $\beta_1$ and $\beta_2$ can be determined from simultaneous equations of two data groups (Suorineni, 1998).

$$ \beta_1 \log 10 HR + \beta_2 \log N' + k = 0 $$  

(10)

$$ HR = 10^{\beta_0 + \beta_1 \log 10 N'} $$  

(11)

**RESULTS AND DISCUSSION**

**Results**

After the data validation, all data was composited within a Microsoft Excel database where it was preprocessed to identify any outliers. In total 226 case histories were used from six operating Australian underground metalliferous open stope mines.

After the data validation, all data was composited within a Microsoft Excel database where it was preprocessed to identify any outliers. In total 226 case histories were used from six operating Australian underground metalliferous open stope mines.

In order to subject the 226 case study stope surfaces to the logistic regression procedure Microsoft Excel add-on software XLSTAT was used to perform the logistic regression. Three parameters where entered into XLSTAT as variables: dilution category, (the dependent variable) the modified stability number and hydraulic radii. The binary
logit model utilised limited data from the two categories at a time. In order to develop the transition boundary lines between dilution categories the upper and lower dilution categories were compared together in the binary logit model. As such <5 per cent category was compared with 5–10 per cent category and 5–10 per cent category compared with 10–15 per cent category and so on. This process was repeated to create four distinctive logit data sets.

Logistic regression is used to determine the location of the transition boundaries which can be used to demarcate different dilution categories. In order to determine the position of boundaries cumulative logit values were determined for each dilution category pair (<5 per cent and 5–10 per cent, etc). To achieve the cumulative distributions each dilution category is plotted along with the inverse of each respective category on a cumulative distribution graph as shown in Figure 4. The crossover point between the upper bound dilution category such as <5 per cent cumulative distribution function intersects the inverse lower point cumulative distribution function representing the logit probability value which defines the separation line (Mawdesley, 2002). The intersecting Logit value determines the vertical axis intercept for the dilution category boundary while, the inclination of the transition boundary is represented by regression coefficients $\alpha$, $\beta_1$, and $\beta_2$.

Using Equation 12 the predicted logit probability value $p$ from Figure 4 can be transformed into a predicted log odd’s value $z$ and regression coefficients $\alpha$, $\beta_1$, and $\beta_2$ in Equation 6 can be written as Equation 13 which represents the boundary line between dilution categories.

\[
Z = \ln\left(\frac{p}{1-p}\right)
\]  

\[
N' = e^{-\frac{(z - \alpha - \beta_1 R)}{\beta_2}}
\]  

Repeating the process for each paired dilution categories and applying Equation 13, the generalised dilution-based stability graph independent of orebody width for open stope design was developed with the logistic regression (Figure 5).

Subjecting the 226 case study stope surfaces to the Bayesian discriminant analysis is similar to the logistic regression approach. The statistical software SYSTAT developed by Sigmaplot was utilised to analyse the data using discriminant analysis and to represent dilution categories with equiprobability contours. Again, three parameters were entered into SYSTAT as variables: dilution category (the dependent variable), the modified stability number and hydraulic radii. Similar to binary logistic regression two dilution categories were analysed at a time. The resultant graph of equiprobability contours formed the basis for the boundary lines between dilution category pairs.

Using Equation 11, the boundary line between two dilution category pairs can be determined. The boundary line corresponds with the intersection between corresponding equiprobability contours as shown in Figure 6. Figure 6 represents the <5 per cent dilution and 5–10 per cent dilution category pairs. Each equiprobability contour determined by SYSTAT surrounds the centroid of the dilution category with outwards increasing probability. Probability ranges in Figure 7 are ten per cent, 20 per cent, 40 per cent, 60 per cent and 80 per cent. The point A in Figure 6 is the point where the stope surface has 40 per cent chance of having a dilution...

![Figure 4](Cumulative distribution function <5 per cent and 5–10 per cent.)

![Figure 5](Generalised dilution-based stability graph independent of orebody width for open stope design developed with logistic regression.)
between five and ten per cent and a 60 per cent chance of having a dilution <5 per cent.

The resultant equiprobability contours between dilution categories is shown in Figure 7 that are used to define the boundaries between dilution categories (Figure 8). Figure 8 shows the stability graph developed with the Bayesian likelihood discrimination and the respective dilution values.

The ability to statistically develop boundary lines based upon the likelihood method and equiprobability contours results in reduced subjectivity. Bayesian likelihood discrimination can be extended to estimate predictive errors in the stability graphs.

**Discussion**

Logistic regression analysis is capable of defining isoprobability contours representing boundaries between sets of data but it does not correct for misclassification errors. Furthermore, when the data categories are not equal to each other due to unequal prior probabilities, the discriminant is bias towards the larger data group which is the >5 per cent and 5–10 per cent dilution categories in this study. The logistic regression model used to develop boundary lines is limited by its fixed gradients, whereas the Bayesian likelihood discrimination model used can account for varying gradient of boundary lines, there by accounting for data inequalities in the data groups.

**CONCLUSIONS AND RECOMMENDATIONS**

The project involved a data collection campaign between May and July 2014. In total, 226 case histories were collected from six current operating underground metalliferous open stope mine sites located across Australia.
The collected data was preprocessed to identify and eliminate outliers. The resultant data was used to develop a quantitative dilution-based stability graph independent of orebody width using two methods of statistical analysis, Logistic regression and the Bayesian Likelihood discrimination method.

A quantitative dilution-based stability graph is more meaningful to a miner than a qualitative or average overbreak depth because miners clearly understand the implications of dilution numbers for their mines. The graph also gives mine planners and designers the flexibility to design stope sizes based on their acceptable dilution levels. The quantitative dilution-based stability graph is also orebody width-independent. An orebody width independent stability graph has the benefit of not being misapplied. The conventional and ELOS stability graphs which are orebody-width dependent are currently being misused as it is not obvious to users that these graphs are orebody-width (narrow versus wide) dependent. The use of statistical methods to define transition boundaries between dilution categories eliminates subjectivity and bias.

It is recommended that additional data be collected and the dilution boundaries fine-tuned to improve the predictive reliability of the graphs. It is also recommended to continue the research using the Bayesian likelihood discriminant method because of its capacity to account for data category in-balance.

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Detection of Changes in the Wetland Conditions adjacent to a Longwall Mining Area in the Southern Coalfields, New South Wales Using Radar Satellite Data

A Poon¹, S Raval²,³, B P Banerjee³ and A Shamsoddini³

ABSTRACT

Wetlands are significantly important due to their capacity to store floodwater, improve water quality, and recharge groundwater aquifer. In the western and southern coalfields of the Sydney Basin, underground longwall coal mining poses one of the potential threats to wetlands. Continuous mapping and monitoring of wetlands has become critical for any mine operating in the vicinity of these sensitive ecosystems. Although field sampling may provide the detailed data for monitoring wetland conditions, it can be expensive and time-consuming, as well not able to be carried out in inaccessible areas. This study investigates the effectiveness of the synthetic aperture radar (SAR) data acquired from the Envisat satellite for detecting changes between 2008 and 2010 in the Thirlmere lakes wetland located adjacent to longwall mining area in the southern coalfields of the Sydney Basin. The change detection result from the radar data was compared against optical high resolution data for accuracy assessment. It was found that the changes in water bodies and associated vegetation types, such as forest trees, were detected at higher accuracy compared to areas classified as ‘thin vegetation’ and ‘grass’. It is recommended that the potential of L band and X band SAR imagery should be investigated as well as different polarisations to fully determine the potential of radar satellite data in detecting the changes in these wetland conditions.

INTRODUCTION

Wetlands, are a distinct ecosystem, commonly occurring at the intersection of land and water (Commonwealth of Australia, 2014). Wetlands are crucial ecosystems through providing habitat as well as trapping a significant amount of organic carbon. Accurate mapping of wetlands enables the quantification of one of the main carbon pools (Page, Rieley and Banks, 2011) and its possible linkage to climate changes.

In the western and southern coalfields of the Sydney Basin, underground longwall methods of coal mining poses one of the potential threats to wetlands, including upland peat swamps (Jenkins and Frazier, 2010; Commonwealth of Australia 2014), prompting the New South Wales (NSW) Scientific Committee to announce that the upland swamps are sensitive areas with Key Threatening Process Listing under the Threatened Species Conservation Act 1995 (Hughes, 2005). Surface subsidence, an inevitable consequence of the extraction of the longwall panels, has potential to change the terrain laying above the extraction panels (Booth, 2006; Guo, Adhikary and Craig, 2009). These changes, including slope changes and ground fracture, can lead to the change of surface and subsurface flow and consequently affect the groundwater and soil moisture (Booth, 2006; Jenkins and Frazier, 2010). According to Murray et al (2003), swamps or wetland as ecosystems depending on groundwater can react to soil and groundwater changes occurring due to mining activity. Vegetation cover and its net primary productivity (NPP) change, detected through long-term data collection and monitoring (Eamus and Froend, 2006), could represent the visual reactions of the swamps affected by hydrological changes (Jenkins and Frazier, 2010). In other words, species composition and the structure of vegetation are functions of hydrological conditions in wetland areas (Mumba and Thompson, 2005). For example, the change in woody species and tall grasses in wetlands are influenced by the duration of inundation and the change in the hydrological regime (Nadirima, 2007). Accordingly, sustainable management of wetlands is important and requires continuous mapping and monitoring of this sensitive ecosystem.

Conventionally, monitoring of wetland conditions focused on determination of the vegetation extent and identification of surface and subsurface flow and consequently affect the groundwater and soil moisture (Booth, 2006; Jenkins and Frazier, 2010). According to Murray et al (2003), swamps or wetland as ecosystems depending on groundwater can react to soil and groundwater changes occurring due to mining activity. Vegetation cover and its net primary productivity (NPP) change, detected through long-term data collection and monitoring (Eamus and Froend, 2006), could represent the visual reactions of the swamps affected by hydrological changes (Jenkins and Frazier, 2010). In other words, species composition and the structure of vegetation are functions of hydrological conditions in wetland areas (Mumba and Thompson, 2005). For example, the change in woody species and tall grasses in wetlands are influenced by the duration of inundation and the change in the hydrological regime (Nadirima, 2007). Accordingly, sustainable management of wetlands is important and requires continuous mapping and monitoring of this sensitive ecosystem.

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of the types of species present. Although field surveying and in situ inspections provides the detailed set of data, they suffer from several limitations. The field survey-based methods are not applicable for inaccessible areas and are often not able to provide continued monitoring of the wetlands due to being time-consuming and expensive (Harvey and Hill, 2001; Rundquist, Narumalani and Narayanan, 2001). Remote sensing techniques could provide continuous monitoring of wetlands over time, including those being difficult to access, at a reduced cost (Rundquist, Narumalani, and Narayanan, 2001). Moreover, the information related to temporal land cover changes can be acquired using satellite images (Ozesmi and Bauer, 2002). Different types of remotely sensed data including optical (Harvey and Hill, 2001; Jenkins and Frazier, 2010), radar (Baghdadi et al, 2001) and lidar data (Jenkins and Frazier, 2010) have been used for the wetland classification and monitoring during last decades.

Studies have been conducted in the United States (Kasischke and Bourgeau-Chavez, 1997; Kasischke et al, 2003; Whitcomb et al, 2009; Wdowinski et al, 2008), Canada (Baghdadi et al, 2001) and other parts of the world (Souza-Filho et al, 2011) utilising radar satellite data for wetland monitoring. However, limited attempts have been made in Australia to create wetland inventories using radar satellite after its potential being highlighted by Finlayson et al (1999). This study has utilised three different types of remote sensing data (ie optical satellite, radar satellite and optical airborne) and is the first attempt to investigate the effectiveness of the synthetic aperture radar (SAR) data for detecting changes in the wetland conditions adjacent to longwall mining areas in the southern coalfields of the Sydney Basin.

STUDY AREA

The Thirlmere Lakes National Park, located approximately 100 km south-west of Sydney in the southern coalfields of NSW in Australia was considered for this study due to its proximity to the Tahmoor longwall mining operations. The Thirlmere Lakes National Park was established to protect the five freshwater lakes due to their geomorphological and biological significance. The native vegetation of the lake area, ridge tops and slopes comprise of rich eucalypt woodland and a diverse range of trees and plants. In addition to extensive woodlands, the colluvial/alluvial flats and the lake margins all have a diverse and extensive range of aquatic flora (NSW Department of Environment and Heritage, 2012; NSW National Parks and Wildlife Service, 1997).

According to the NSW Department of Environment and Heritage (2012), the lakes are embedded within the Hawkesbury Sandstone and have fluctuated between substantially dry and completely full conditions over the past few decades. Despite the number of ground investigations performed in this area, there is not enough evidence to establish the relationship between the conditions of the lakes and progress of the Tahmoor longwall mining operation (NSW Department of Planning and Infrastructure, 2007). Whilst there is some evidence to suggest that mining operations have contributed to changes in groundwater over time, there is little to no evidence to separate groundwater changes due to mining from other factors such as climate and private bore usage. The limited information and knowledge regarding the Thirlmere lakes suggests that additional research and utilisation of more effective monitoring methods are needed in order to continuously monitor the changes in the condition of the lakes.

REMOTE SENSING DATA

Three types of remote sensing data; radar satellite data, high resolution optical satellite data and high resolution optical airborne data have been utilised for this study.

Radar satellite data

The European Space Agency (ESA)’s Envisat satellite data acquired in C-band, commonly referred as ASAR imagery, for the Thirlmere Lakes National Park was searched using the ESA’s ‘Eoli-sa’ catalogue search tool. The only available ASAR imagery of the study area was in the form of Envisat’s ‘Global Monitoring Mode’ and ‘Image Mode’. The ‘Image Mode’ products were adequate for application oriented analyses and for deriving the backscattering coefficient of the study area (European Space Agency, 2014). Since use of backscattering coefficient was required for this study, the ‘Image Mode’ products were selected.

Two Envisat C-band ASAR Image Mode Precision images were acquired for the study area free of charge from the ESA. The details of each SAR data set are outlined in Table 1.

### Table 1

<table>
<thead>
<tr>
<th>Date of data</th>
<th>ASAR swath</th>
<th>Incidence angle (°)</th>
<th>Polarisation</th>
<th>Spatial resolution (m)</th>
</tr>
</thead>
<tbody>
<tr>
<td>9 March 2008</td>
<td>IB</td>
<td>25.92–31.4</td>
<td>VV</td>
<td>30</td>
</tr>
<tr>
<td>14 March 2010</td>
<td>IB</td>
<td>25.92–31.4</td>
<td>VV</td>
<td>30</td>
</tr>
</tbody>
</table>

High-resolution airborne data

High resolution airborne optical data acquired over the study area in March 2008 by the NSW Division of Land and Property Information was utilised to validate the radar satellite derived results.

The airborne image captured through Leica ADS40 airborne digital sensor had 50 cm spatial resolution. The image comprised of a red band (608–662 nm wavelength), a green band (533–587 nm wavelength), a blue band (608–662 nm wavelength), and a near-infrared band (833–887 nm wavelength). The image was captured through multiple flights between 30 March 2008 and 2 April 2008 (NSW Land and Property Information, 2008).

High-resolution optical satellite data

Since there was no high resolution airborne data available apart from the March 2008 ADS40 image for the study area, high resolution satellite image acquired by the WorldView-2 (WV2) satellite on 13 April 2010 over the study area was purchased for this study.

The WV2 data comprised of a single-band panchromatic image with a spatial resolution of 50 cm and a four-band (red, green, blue, near-infrared) multispectral image with a spatial resolution of 2 m.

METHODOLOGY

Analysis of the radar satellite data

The 2008 and 2010 Envisat ASAR images were preprocessed, calibrated and analysed using open-source SAR analysis software ‘Next ESA SAR Toolbox’ (NEST) version 5.1. Radiometric calibration was performed on both the images to obtain correct radar backscatter. Corrections to the incidence angle and the incorporation of an absolute calibration
constant were applied. The calibrated data was filtered using NEST’s single-product Lee speckle filter in order to minimise influence of speckle in the final results. The Range Doppler ortho-rectification method was applied to each data set in order to produce a terrain corrected ortho-rectified image. The backscatter coefficient ($\sigma^0$) of each image pixel was then computed.

Subsequently, using the ArcGIS software package, the processed ASAR images were geometrically corrected and adjusted from the GCS WGS 1984 coordinate system to the WGS 1984 UTM coordinate system. This ensured that the ASAR images were spatially consistent, with the ADS40 and WV2 images.

Markov random field (MRF) supervised classification was undertaken using an MRF supervised image segmentation program (Berthod et al, 1996). Five classes were used; namely ‘trees’, ‘thin vegetation’, ‘grass’, ‘bare land’ and ‘water’. Four MRF based classification algorithms were tested namely – Metropolis, modified Metropolis dynamics (MMD), Gibbs Sampler, and iterated conditional modes (ICM). All the four algorithms were used to classify the ASAR images, resulting in a total of eight classified ASAR images, four images for the 2008 ASAR data and four images for the 2010 ASAR data.

Classification pseudo-accuracy assessment was undertaken for each of the eight ASAR images using the ArcGIS software package. Pseudo-accuracy assessment is a technique to assess the classification accuracy using same or different high resolution images as a reference in the absence of ground information. Reference points were taken in accordance to the five classes and used to generate a confusion matrix. The overall classification accuracy and kappa coefficient for each of the eight ASAR images was derived using the confusion matrix. The ASAR MRF classification with the highest classification accuracy was used to produce the change detection map.

A final change detection map was produced using ArcGIS in which the 2008 ASAR MRF classification results were used as the reference and the 2010 ASAR MRF classification results were subtracted using a class to class subtraction approach. A class-based change detection approach was preferred over pixel based approach to identify whether one of the five classes had undergone any change.

### Analysis of the high-resolution airborne data

The acquired 2008 ADS40 image of the study area was already preprocessed. However, in order to prepare the data for change detection analysis with the WV2 data, additional processing was required.

Subsequently, the ADS40 data was reprojected to the WGS 1984 UTM coordinate system as well as atmospherically corrected using the Exelis ENVI software package. The final ADS40 image was then classified into five separate classes. The five classes used were the same as the ones used for the ASAR imagery (ie ‘trees’, ‘thin vegetation’, ‘grass’, ‘bare land’ and ‘water’).

A maximum likelihood supervised classification was subsequently performed on the image using the Exelis ENVI software package, resulting in a final classified image of the study area. A classification pseudo-accuracy assessment was undertaken for the ADS40 image using the ArcGIS software package. Reference points were taken in accordance to the five classes and used to generate a confusion matrix. The overall classification accuracy and kappa coefficient for the ADS40 was derived using the confusion matrix. The ADS40 image was subsequently used to produce a final optical high-resolution change detection map.

### Analysis of the high-resolution optical satellite data

The panchromatic and multispectral WV2 data were processed and fused using the Exelis ENVI software package to produce an image of the study area with spatial resolution of 50 cm and reprojected to the WGS 1984 UTM coordinate system. The resultant image of the study area was subsequently atmospherically corrected using the FLASH tool in Exelis ENVI software package.

The processing of the WV2 data was done to match the ADS40 specifications and to ensure consistency between the 2008 and 2010 data sets in order to minimise any errors that may arise during the change detection phase.

The classification and accuracy assessment methods performed on the ADS40 image was again used for the WV2 image. The WV2 image was subsequently used with the ADS40 image to produce the final optical high-resolution change detection map.

### RESULTS

#### Change detection through the radar data

A total of 344 and 354 ground truth points were used to perform pseudo-accuracy assessment for each of the four MRF classification results on 2008 and 2010 ASAR data respectively. The resultant overall classification accuracies and kappa coefficients for each ASAR MRF classification is outlined in Table 2.

<table>
<thead>
<tr>
<th>Markov random field classification</th>
<th>Overall accuracy (%)</th>
<th>Kappa coefficient</th>
<th>Overall accuracy (%)</th>
<th>Kappa coefficient</th>
</tr>
</thead>
<tbody>
<tr>
<td>Metropolis</td>
<td>55.5</td>
<td>0.42</td>
<td>53.0</td>
<td>0.41</td>
</tr>
<tr>
<td>Gibbs sampler</td>
<td>54.4</td>
<td>0.4</td>
<td>51.9</td>
<td>0.4</td>
</tr>
<tr>
<td>Iterated conditional modes</td>
<td>70.9</td>
<td>0.62</td>
<td>59.8</td>
<td>0.5</td>
</tr>
<tr>
<td>Modified Metropolis dynamics</td>
<td>52.9</td>
<td>0.38</td>
<td>53.3</td>
<td>0.42</td>
</tr>
</tbody>
</table>

Analysis of the overall accuracies and visual representation of each ASAR MRF classification result showed that the ICM MRF classification was the optimal choice for classifying 2008 and 2010 ASAR images. It was evident that the ICM MRF classification is able to distinguish each of the five classes better than the Metropolis, MMD and Gibbs Sampler MRF classifications.

The producer’s accuracy refers to the conditional probability that a certain land-cover of an area on the ground is correctly mapped, while the user’s accuracy refers to the conditional probability that a pixel labelled as a certain land-cover class in the map is actually belongs to that class (Stehman and Czaplewski, 1998). Overall accuracy refers to percentage of classified map correctly allocated (Foody, 2002). The producer’s and user’s accuracy for the ICM MRF classification was derived as part of the accuracy assessment and is outlined in Table 3.

The producer’s and user’s accuracy for the ICM MRF classification of the 2008 and 2010 ASAR imagery suggests that the areas belonging to the ‘trees’ and ‘water’ classes were
classified at higher accuracy compared to ‘thin vegetation’, ‘grass’, and ‘bare land’ classes. The fact that dense vegetation usually has more pure reflectance compared to sparse vegetation, provides the explanation to the achieved accuracy results.

Based on the derived accuracies, the ICM MRF classification of the 2008 and 2010 ASAR imagery was deemed to be accurate enough to use for the final ASAR change detection map. A change detection map was generated in ArcGIS for the possible 21 changed classes as shown in Figure 1. It was found that a substantial part of the study area underwent no change from 2008 to 2010.

### Change detection through the high-resolution optical data

Using the maximum likelihood supervised classification results of the ADS40 and WV2 images, pseudo-accuracy assessment was undertaken using a total of 388 and 334 ground truth points respectively. The overall classification accuracy and the kappa coefficient for the ADS40 imagery were determined to be 78.9 per cent and 0.73 per cent respectively. Similarly, overall classification accuracy and kappa coefficient for the WV2 imagery were determined to be 87.1 per cent and 0.84 per cent respectively. The producer’s and user’s accuracy for the ADS40 and WV2 classification were also derived as shown in Table 4.

### CONCLUSIONS

It was determined that forest tree vegetation changes were detectable across the Thirlmere Lakes National Park wetland between 2008 and 2010 located adjacent to longwall mining area in the southern coalfield using Envisat C-VV ASAR imagery. No major change was detected for most of the study area, except the areas classified in the ‘trees’ and ‘water’ classes recorded some notable changes in both radar as well as high resolution optical data. With the producer’s accuracy of the ASAR imagery being comparable to the producer’s accuracy of the optical high-resolution imagery, it was determined that ASAR imagery is able to monitor and detect changes in water bodies and large plant species to a sufficient level of accuracy.

The derived kappa coefficient, producer’s accuracy, user’s accuracy, and overall accuracy for the Envisat C-VV imagery was found to be sufficiently acceptable overall and comparable to the optical high-resolution imagery, especially for areas in the ‘trees’ and ‘water’ classes across the study area.

However, the classification accuracy was lower for the ‘thin vegetation’, ‘grass’, and ‘bare land’ classes. The considerable difference in producer’s accuracy for the ASAR imagery when compared to the optical high-resolution imagery confirms that discrimination for non-homogeneous pixel such as thin vegetation is a challenge with the ASAR data. This result can be pinpointed to the lower (30 m) spatial resolution of the C-VV ASAR imagery which makes classification of smaller vegetation types exceptionally difficult.
DETECTION OF CHANGES IN THE WETLAND CONDITIONS ADJACENT TO A LONGWALL MINING AREA

**FIG 1** – The change detection map generated using radar data.
FIG 2 – The change detection map generated using optical high resolution data.
It can be concluded that the usage of Envisat C-VV ASAR imagery is sufficient in detecting change in water bodies and species of sizable vegetation, such as trees, but it is relatively ineffective for detailed delineation and discrimination of smaller vegetation types such as grass, thin vegetation types, and bare land.

RECOMMENDATIONS

Based on the results of this project, it is recommended that a number of factors and considerations be taken into account for advancing the study for using radar data for monitoring wetlands.

Although this study resulted in some promising outcomes in the usage of ASAR for detection of vegetation change, the project scope was limited by the availability of only Envisat C-VV ASAR imagery over the study area. It is recommended that the potential of higher resolution radar data with L band as well as X band should be investigated. Furthermore, the use of full polarisation SAR imagery rather than just single polarised SAR imagery could improve the classification accuracy of the results. A field survey conducted concurrently to the satellite acquisition could provide robust validation compared to the pseudo-accuracy assessment done in this study.

ACKNOWLEDGEMENTS

The authors would like to thank Dr Scott Hawken, Lecturer School of Built Environment UNSW for providing the ADS 40 data. Funding support to purchase the WV2 satellite data was provided by the School of Mining Engineering, UNSW under School Research Grant 2014 (A novel approach to monitor longwall coal mining, provided by the School of Mining Engineering, UNSW under School of Built Environment UNSW for providing the ADS 40 satellite data).

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The Effect of Tool Geometry on Cutting Performance

E Sarwary¹ and P Hagan²

ABSTRACT
Many machines used in excavation of weak and medium strength rock such as sandstone utilise conical style picks. An understanding of the factors that cause high wear rates is crucial in the selection and design of an excavator and appropriate type of cutter tools and also the layout of optimum cutting geometry. This paper explores how the initial onset of tool wear that is associated with an increase in pick tool angle affects the cutting performance in two different rock types and with changes in depth of cut. Rock cutting tests were performed in Gosford sandstone and Gambier limestone. The rock samples were cut at depths ranging between 5 mm and 20 mm using a standard conical pick having tool tip angles of 70°, 90°, 100° and 110°. The cutting and normal forces, specific energy, and yield were correlated against tip angle and depth of cut. The results reveal the effect of wear was more pronounced with normal force that is the force required to sump a cutting machine into the face compared to cutting force that is the force associated with machine torque and power. Over the range of tool angles, there was a near three-fold increase in normal force compared to a two-fold increase in cutting force. In all instances the rate of increase in force and specific energy with tool angle increased with depth of cut. Changes in specific energy confirmed cutting efficiency increases with tool angle and hence wear.

INTRODUCTION
Many mechanical rock cutting machines in rock excavation in mining and tunnelling use tungsten-carbide conical picks such as shown in Figure 1, mounted on a cutting head to fracture the rock in situ prior to its removal and further processing (Lloyd, 1985). As theory relating to the mechanical excavation of rock has emerged and evolved over time, so too has the utilisation of these machines, often replacing traditional drill and blast methods, resulting in an increase in safety performance and a reduction in operating cost. As a consequence, laboratory-scale rock cutting facilities such as the portable linear rock cutting machine (PLCM) in the Machine Cuttability Research (MCR) facility within the School of Mining Engineering at UNSW Australia are able to provide data to aid in machine selection, design, and performance prediction for a given rock formation (Balci and Bilgin, 2007; Jacobs and Hagan, 2009; Langham-Williams and Hagan, 2014).

Normal force as shown in Figure 2 is used to estimate the effective mass and thrust required of an excavator. This is a critical parameter as it provides an insight into the range of necessary forces provided by the excavator in order for the cutter to effectively penetrate the rock and maintain the cutting depth. Likewise, the cutting force is related to the torque requirements of a machine and can be used to estimate the energy requirements of a machine. Cutting force is used

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to calculate the specific energy requirements, defined as the amount of energy required in excavating a unit volume of rock. Specific energy is a direct measure of cutting efficiency. Lower specific energy correlates to more material being produced by a given machine; therefore, lower specific energy indicates an increase in cutting efficiency (Roxborough, 2009).

Despite picks being typically constructed from tungsten carbide due to its hardness, thermal resistance, high compressive strength and high impact resistance, they are still susceptible to wear (Hudson et al., 1993). There are several mechanics of wear, such as frictional wear, abrasive wear, microfracturing, thermal fatigue, impact damage, and chemical erosion, all of which contribute to tool wear. The consequence of wear on the cutting is familiar in the mining and tunnelling industry, since the performance of the machine deteriorates significantly as the tools become blunt. The mining output will fall, repairable dust production will rise, and the risk of sparking increases (Roxborough, 2009).

**METHODOLOGY**

A project was undertaken to assess the effect of changes in cutting performance with changes in the tool tip angle, \( \phi \), of a pointed pick that is commonly used by machines in the excavation of soft to medium strength rock. Cutting tests were undertaken in a combination of a pick of differing tool angles and at differing depths of cut in two rock types. Changes in cutting performance were assessed in terms of changes in:

- cutting force, \( F_c \)
- normal force, \( F_n \)
- specific energy, \( SE \)
- yield, \( Q \).

The research involved tests conducted using the newly installed the newly commissioned PLCM as shown in Figure 3. The linear rock cutting tests were performed using blocks of Gosford sandstone and Gambier limestone at a constant cutting speed of 0.06 m/s and attack angle of 55°. Tests were undertaken at depths ranging from 5 mm to 20 mm with conical picks at four different pick angles. A data acquisition system was used to record the cutter forces measured by a triaxial dynamometer during linear rock cutting tests.

**Sample preparation**

Blocks of test samples having dimensions of 260 × 180 × 100 mm were set in plaster within a small steel box frame to provide the necessary confinement during testing, as depicted in Figure 4. The preparation of the plaster involved mixing with water at a ratio of 5:3.25 (that is 5 kg of powder to 3.25 kg of water). The samples were cured for at least 24 hours prior to any testing to ensure the plaster had hardened sufficiently.

**Preparation of test picks**

New conical picks with short-tailed 25 mm shank were machined to provide four different tip angles of 70°, 90° 100° and 110° representing a pick at various states of wear. An illustration of the pick used in the tests is shown in Figure 5.

The pick holder system was designed with a fixed insert as shown in Figure 6. The design allowed the pick to be mounted at an attack angle of 55°. According to Mostafavi et al. (2011) this is within the range of angles when mounting picks on continuous miners, road-headers and shearers.

**FIG 3** – Portable rock cutting machine in the Machine Cuttability Research Facility at UNSW.

**FIG 4** – Method of securing the rock samples for cutting tests.

The cutting \( F_C \) and normal \( F_N \) forces were measured using an integrated triaxial dynamometer. The length of the cut was measured using a linear variable displacement transducer (LVDT). The LabVIEW software package was used for real time monitoring and recording of the forces and displacement.

The force values in conjunction with mass of collected debris were used to calculate the specific energy and yield based on the following formulas:

\[
Q = \frac{m \rho}{l} \tag{1}
\]

where:
- \( Q \): yield \((\text{m}^3/\text{km})\)
- \( \rho \): density of the rock sample \((\text{kg}/\text{m}^3)\)
- \( m \): mass of the debris collected \((\text{kg})\)
- \( l \): length of cut \((\text{km})\)

\[
SE = \frac{F_C}{Q} \tag{2}
\]

where:
- \( SE \): specific energy \((\text{MJ}/\text{m}^3)\)
- \( F_C \): cutting force \((\text{kN})\)

**Strength and density of test samples**

Uniaxial compressive strength (UCS) tests were conducted on specimens of Gambier limestone, in accordance with the ISRM suggested method for uniaxial compressive strength determination (Brown, 1981).

The testing procedure involved six limestone rock specimens with a diameter and length of 52 mm and 104 mm respectively using an MTS universal test machine. The tests were conducted at a constant displacement rate of 0.003 mm/sec. The strength of the Gosford sandstone was earlier determined by Masoumi (2013).

Nine core samples were weighed and the diameter and length of each sample recorded. Sufian and Russell (cited in Masoumi, 2013) conducted an X-ray CT scan on Gosford sandstone. By using a resolution of 5 \( \mu \text{m} \) they calculated the porosity to be approximately 18.5 per cent with a density of 2.5 \( \text{t}/\text{m}^3 \). Table 1 summaries the strength and density of the two rock types indicating a ten-fold difference in strength and near doubling in rock density between the sandstone and limestone samples.

**RESULTS**

A series of cutting tests was conducted using sandstone and limestone samples with a typical result as shown in Figure 7.

The results of the cutting and normal forces, specific energy, and yield were correlated against picks at different tool angles representing various states of wear. Figures 8 and 9 show the effects of pick wear on cutting and normal force for the two rock types. The trends in each of the graphs indicate cutting and normal force increase with tool tip angle with a strong correlation more evident in the sandstone with \( R^2 \) typically of 0.98. The magnitude of forces is much greater for Gosford sandstone compared to Gambier limestone, nearly three times greater for cutting force and six times greater for normal force. This is in line with the sandstone’s much greater strength and density. It is also evident that the magnitude of cutting force for both types of rock is greater than the magnitude of normal force. Cutting force is also on average approximately 1.6 times greater than the normal force for sandstone and 2.8 times greater for limestone.

Table 2 shows the variation in cutting force and normal force with depth of cut in the two rock samples. In the case of sandstone, the gradient increases with depth of cut.
indicating the impact of wear increases with depth of cut. The gradients are of similar magnitude level for cutting force and normal force.

The situation is less consistent for the softer limestone whereby there is little significant change in gradient with depth and the values for cutting and normal force are again comparable. Earlier work has found that increasing wear usually has a much more deleterious effect on normal force than on cutting force (Roxborough, 2009). The consequence of this effect is that machines, such as continuous miners and road-headers, are likely to become thrust limited rather than torque limited with increasing wear.

The graphs shown in Figures 8 and 9 indicate a linear relation between forces of cutting and tool tip angle albeit over limited range of tip angles such that shown in Equations 3 and 4:

\[
\begin{align*}
F_c &= a \phi - b \\
F_n &= a \phi - b
\end{align*}
\]

where:
\[
\begin{align*}
F_c &= \text{cutting force (kN)} \\
F_n &= \text{normal force (kN)} \\
\phi &= \text{tool tip angle}
\end{align*}
\]

In the case of a 15 mm depth of cut in sandstone, the variation in cutting and normal forces with tool angle can be expressed as:

\[
\begin{align*}
F_c &= 0.92 \phi - 21.9 \\
F_n &= 0.95 \phi - 47.1
\end{align*}
\]

The values of the force gradients stated in Table 2 after being normalised with respect to depth of cut, \(d\), are shown in Table 3.

The results in Table 3 again reflect a higher level of consistency in test results for the stronger more consistent sandstone. Considering the average values for normalised gradient, a generalised form of the variation in forces with tool angle and depth of cut in Gosford sandstone can be expressed as:

\[
\begin{align*}
F_c &= (0.62 \phi - 1.86) d \\
F_n &= (0.68 \phi - 3.81) d
\end{align*}
\]

where:
\[
\begin{align*}
d &= \text{depth of cut (mm)}
\end{align*}
\]
Figure 10 shows specific energy increase with tip angle. Hence as would be expected cutting efficiency decreases with increasing tip angle and hence with tool wear. Also specific energy decreases with increasing depth of cut and hence this is in agreement with the general principle that cutting efficiency varies depth of cut.

The increase in normal force required to successfully achieve pick penetration will result in the machine becoming thrust limited, consequently leading to eventual stalling.

Although the general trend indicates that wear has a negative impact on cutting efficiency, there are some outlier results. Closer analysis of the result reveal that cutting sandstone with pick tip angle of 110° at 20 mm depth of cut results in a slightly lower specific energy compared to cutting with pick tip angle of 90° and 100° which represent a slightly less worn out pick. A possible explanation is that not only the groove cut with a 110° pick tip angle is wider but it has been able to achieve the same penetration depth as a sharp pick under the same constant thrust force provided by the PLCM, resulting in greater yield which consequently would have yielded a lower efficiency. These variations may also be due to differences in microfractures, grain size distribution, and varying joint structure within the different rock samples, suggesting that the rocks tested are not perfectly homogenous. There are also similar outliers observed when cutting sandstone with a pick tip angle of 90° and cutting limestone at a pick tip angle of 110° at 5 mm depth of cut which suggests that it is more efficient to cut with a more worn out pick.

From Figure 10, it is evident that the trend indicates that it is more cost effective because it requires less energy to excavate Gambier limestone than Gosford sandstone. However, whether it is more efficient to mine the two different types of rock would require further exploration. It is further observed that wear has a dramatic effect on cutting efficiency at shallow depths of cut and higher specific energy, but at the deeper depths of cut, such as 15 mm and 20 mm, a more worn out pick (ϕ = 110°) has little effect on cutting efficiency. This indicates that a slightly more worn pick will perform just as well at a higher depth of cut.

Table 4 shows the reduction in specific energy with an increase in depth of cut from 5 mm to 10 mm in limestone and sandstone. The data in the table indicates a greater reduction in specific energy and hence increase in efficiency for a mechanical excavator cutting deeper in the sandstone than in the limestone, given all other conditions remaining constant. This trend is related to the brittleness of the rock, which is a function of compressive and tensile strength of the rock (Goktan and Yilmaz, 2005). Generally, if the tensile strength of the rock is similar, a higher compressive strength value means that the rock would be more brittle (Goktan and Yilmaz, 2005). Since the strength of Gosford sandstone is 50.3 MPa, it is more brittle compared to the Gambier limestone of 5.0 MPa. In this case, when a pick penetrates the rock at the same depth of cut, the ease with which fractures propagate with sandstone is higher compared to limestone, thus resulting in more rock fragments. This is consistent with experimental results for this research project.

### TABLE 3
Values of force gradients normalised with respect to depth of cut.

<table>
<thead>
<tr>
<th>Rock type</th>
<th>Depth (mm)</th>
<th>Normalised cutting force gradient, C’ (kN/deg)</th>
<th>Normalised normal force gradient, N’ (kN/deg)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Sandstone</td>
<td>5</td>
<td>0.059</td>
<td>0.074</td>
</tr>
<tr>
<td></td>
<td>10</td>
<td>0.069</td>
<td>0.067</td>
</tr>
<tr>
<td></td>
<td>15</td>
<td>0.061</td>
<td>0.063</td>
</tr>
<tr>
<td></td>
<td>20</td>
<td>0.059</td>
<td>0.067</td>
</tr>
<tr>
<td></td>
<td>5</td>
<td>0.0041</td>
<td>0.010</td>
</tr>
<tr>
<td></td>
<td>10</td>
<td>0.0032</td>
<td>0.010</td>
</tr>
<tr>
<td></td>
<td>15</td>
<td>0.0187</td>
<td>0.006</td>
</tr>
<tr>
<td></td>
<td>20</td>
<td>0.0143</td>
<td>0.004</td>
</tr>
<tr>
<td>Limestone</td>
<td>5</td>
<td>0.0041</td>
<td>0.010</td>
</tr>
<tr>
<td></td>
<td>10</td>
<td>0.0032</td>
<td>0.010</td>
</tr>
<tr>
<td></td>
<td>15</td>
<td>0.0187</td>
<td>0.006</td>
</tr>
<tr>
<td></td>
<td>20</td>
<td>0.0143</td>
<td>0.004</td>
</tr>
</tbody>
</table>

### TABLE 4
Variation in specific energy with tool angle.

<table>
<thead>
<tr>
<th>Rock type</th>
<th>Tool angle (°)</th>
<th>Reduction in specific energy (%)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Limestone</td>
<td>70</td>
<td>9</td>
</tr>
<tr>
<td></td>
<td>90</td>
<td>33</td>
</tr>
<tr>
<td></td>
<td>100</td>
<td>49</td>
</tr>
<tr>
<td></td>
<td>110</td>
<td>32</td>
</tr>
<tr>
<td>Sandstone</td>
<td>70</td>
<td>58</td>
</tr>
<tr>
<td></td>
<td>90</td>
<td>23</td>
</tr>
<tr>
<td></td>
<td>100</td>
<td>61</td>
</tr>
<tr>
<td></td>
<td>110</td>
<td>58</td>
</tr>
</tbody>
</table>
More fragments indicate a higher yield, given that other parameters are constant, which leads to a lower specific energy.

CONCLUSIONS

In the initial stages of wear of a rock cutting tool, there is effectively an increase in the tool tip angle reflecting blunting of the tool. As the tool tip angle increases with wear, both cutting force and normal force increase. Over the range of tool angles investigated from 70° to 110°, there was a near three-fold increase in normal force, whereas the rate of increase in cutting force was less than two-fold. Hence tool wear has a greater degradation effect on the sumping capability of a rock cutting machine than the increase in torque or power requirements.

The rate of increase in forces was not constant but varied with depth of cut and rock type. While cutting deeper is generally found to be more efficient, the impact of wear was greatest at the deeper cuts. Similarly, the effect of wear was enhanced when cutting in stronger rock.

There was also found to be an increase in specific energy and hence a fall in cutting efficiency indicating a reduction in machine production rate.

These results explain what is often observed in practise with changes in operating mode of a cutting machine. Usually a cutting machine such as a continuous miner or a roadheader with all new picks is initially torque limited meaning the machine is more likely to stall at the face while still capable of cutting deeper. But as the picks on the machine wear over time there is a reduction in maximum cutting depth per revolution leading to a reduction in production rate and the machine becomes thrust limited.

ACKNOWLEDGEMENTS

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REFERENCES


INTRODUCTION

The hydrogeological character of faulting depends on a number of factors and conditions, which are unique to local geology and conditions. The hydrogeological character of faulting has not been evaluated on a regional scale, across the Southern Coalfields, which is part of the Sydney Basin, Australia (Figure 1). This is an area of interest and significance as it is the major source of high quality coking coal, and is located within part of Sydney’s water catchment and water reservoirs. The Sydney water catchment and associated infrastructure provides drinking water for over 4.3 million people, a fifth of Australia’s population (Sydney Catchment Authority, 2014b).

Coal mining commenced in the region in 1857 (New South Wales Department of Planning, 2008). There are currently six underground metallurgical coalmines operating in the Southern Coalfields: Metropolitan, Appin, Westcliff, Russell Vale, Dendrobium and Tahmoor. Current operations utilise longwall methods or are highly mechanised pillar extraction mines. These mines produce a total of approximately 11 million tonnes per annum (Mt/a) of high quality, coking coal (New South Wales Resources and Energy, 2014). The Department of Primary Industries (DPI) estimated in 2008 that the total recoverable coal reserves for the Southern Coalfields to be 670 million tonnes (Mt). At the current mining rate of 11 Mt/a these total recoverable coal reserves could extend for a further 60 years of mining activity (New South Wales Department of Planning, 2008).

Coal seams and overburden rock are often intersected by geological structures such as faults, dykes and sills, along with fractures and joints, and can have a significant impact...
on groundwater flow. Bense et al (2013) reviewed various structural geological methods and hydrogeological methods to analyse fault zone hydraulics. Odling, Harris and Knipe (2004) used similar methods of fault zone analysis to determine the permeability (or hydraulic conductivity) of different zones of damage surrounding faults. Both studies agreed that the principal variables in the hydraulic conductivity of fault structures include both physical characteristics (eg damage zones, length, displacement, orientation), and geochemical characteristics including infill and weathering that determine clay content. The petroleum industry has published general relationships between clay content of the fault and permeability or fault seal behaviour (Yielding, Freeman and Needham, 1997).

The objective of this research was to evaluate the hydrogeological character of major geological structures within the Southern Coalfields by reviewing available evidence, developing conceptual models, and outlining future strategies for evaluating the significance of geological structures for groundwater flow. In parallel with this paper, preliminary testing of rock permeability in the absence of structures was also undertaken. These results will be reported with tests of fault-infill materials in the next phase of research.

**METHODOLOGY**

International literature on the hydrogeology of geological structures was reviewed, along with collation and evaluation of information from the Southern Coalfields. Collation and review of information supplied by the New South Wales (NSW) Dams Safety Committee (DSC), with the approval of mining companies was used to augment relevant reports that are publically available from Tahmoor Colliery, and Illawarra Coal. For the first time, information from approximately 100 reports was brought together, including internal mining
company reports, and consultancy reports commissioned by mining companies or by the DSC.

**DRivers For LEading Practices**

Leading mining practices are being driven by increased expectations and scrutiny of environmental outcomes and enhanced risk management in terms of safety and project finances. The proximity of reservoirs, creeks and wetlands to mining operations in the Southern Coalfields has led to many site based studies, regulatory reviews and industry led research on prediction of aquifer inflow into longwall mines (eg Gale, 2008). The most recent review recommended a regional approach to monitoring and evaluation of the potential cumulative effects of coal and coal seam gas developments in the catchment (New South Wales Chief Scientist and Engineer, 2014).

The potential for the dam structures, associated reservoirs and the surrounding catchments to be affected by mining have resulted in more stringent regulatory and environmental requirements for mine approvals and operations. The Sydney water supply draws in part from the Woronora, Cataract, Cordeaux and Avon reservoirs that overlie the Southern Coalfields. The Thirlmere Lakes also contribute overflow to the Nepean part of the Sydney’s water supply. These requirements are imposed by the Sydney Catchment Authority and the NSW Dams Safety Committee (DSC), which was formed as part of the New South Wales (NSW) Government’s Dams Safety Act 1978 (NSW DSC, 2011).

**Influential Processes and Mechanisms in the Southern Coalfields**

Whether a fault acts as a conduit or barrier to groundwater flow in the Sydney Basin and Southern Coalfields is dependent on various geological and geotechnical factors as described in this section.

**Geological setting and history**

The Sydney Basin is comprised of at least 4000 metres (m) of Permian-Triassic age (201–298 million years (Ma)) sedimentary rocks (Moffitt, 1998). The Sydney Basin was deposited by deltaic, estuarine and alluvial processes unconformably over Palaeozoic basement rocks and contains several coal deposits, including the Illawarra Coal Measures. The latter are up to 500 m thick, a sequence of interbedded coal, sandstone, siltstone, claystone and shale, with minor igneous intrusion (Moffitt, 1998).

The Jurassic-Cretaceous (65–201 Ma) periods resulted in significant plate tectonic movements, reactivation of faults, and deep weathering. Tectonic activity reactivated geological structures and forced igneous intrusions through weaknesses in older strata resulting in dykes, sills and cindered zones (O’Neill and Danis, 2013). These intrusion features are common in the Bulli and the underlying Wongawilli coal seams of the Illawarra Coal Measures.

Recent research on gouge, or fault infill within joint swarms and surrounding brecciated rock that are characteristic of fault zones has been done by Och, Offer and Zwingmann (2014). Clay minerals such as illite and smectite form as a result of fluid infiltration and geochemical weathering of igneous rock. Major geological structures that outcrop are thus deeply weathered by water seepage. K-Ar dating of illite and illite-smectite in fractions extracted from fault gouges indicate the clays are 120 to 196 Ma age, depending on thermal overprinting. These dates confirm igneous intrusions were associated with the early stages of rifting of eastern Gondwana during the early Cretaceous period.

Faulting within the region occurs on both a major regional scale and at a mine site scale. Major regional faults and fault structures are found through surface based exploration, as displayed in Figure 1. These major structures have clearly formed with processes that occurred during the deposition of the basin, as the major structures have similar characteristics and orientations to the basin (Moffitt, 1998).

By contrast, deep faults encountered by mines are between 350 and 850 m of depth (Merrick, 2009). Faults encountered at the mine-scale are normal faults and trend north-northwest or south-southeast (Moffitt, 1998). This is parallel to the fold axes in the eastern section of the Southern Coalfields where mines are operating. The throw on this type of faulting is up to 95 m and increases with depth. This indicates these faults formed with the sedimentation processes (Moffitt, 1998). Mine-scale faults were typically formed in Triassic strata, buried by younger strata, and thus typically lack a surface expression (GHD Geotechnics, 2007). These faults have experienced less weathering processes than that of major faults with connections to the surface, however they have been reactivated through seismic events. Reactivation of faults generally results in a reduction of the fault’s ability to transfer fluids or act as a conduit (Drummond, 2013).

**Stress, depth and orientation of geological structures**

Applied stress can close up faults or joint sets. Bense et al (2013) established a correlation between the hydraulic conductivity (or hydraulic aperture) a rock mass and applied stress. Vertical stress is predominantly a function of the density and thickness of overburden rock. For example, a new empirical relationship that included data from the Southern Coalfield indicates a very significant decrease in hydraulic aperture between the ground surface and a depth of approximately 150 to 200 m (Zoorabadi, 2014). Beyond this ‘threshold’ depth, the hydraulic aperture (or hydraulic conductivity) continues to decrease with increasing depth, though at a slower rate.

The orientation of stresses relative to geological structures also has implications for groundwater flow. Horizontal stresses within the Sydney Basin are two to three times greater than vertical stresses (New South Wales Department of Planning, 2008). The boundaries of the Sydney Basin and the boundary plate forces define the direction and magnitude of stresses of the Sydney Basin (Hillis, Enever and Reynolds, 1999). The Sydney Basin is bounded by the Lachlan and New England Fold Belts, and the edge of the continental shelf. However, the direction of the stresses varies from site to site according to geological structures, igneous intrusions and natural features of the rock mass (Thomas, 2008).

The horizontal stress direction is generally north-northeast or south-southwest in the Southern Coalfields, due to these Basin boundaries (O’Neill and Danis, 2013; Ward and Kelly, 2013). This is a similar orientation to the orientation of major structures of the Southern Coalfields. However, the predominant orientation of mine scale faults within the Southern Coalfields is north-northwest or south-southeast (Moffitt, 1998). The orientation of the mine-scale faults, opposite to the regional stress direction ensures tight closure of apertures, particularly with increasing depth of cover.
HISTORICAL MINE INFLOW EVENTS

There are four historical cases of inflows into mines within the region due to geological features. These events occurred in the 1960s to 1980s.

Mount Kembla Colliery

Mount Kembla Colliery experienced an inundation in 1964 in Gilmore’s shunt, whilst extracting the Bulli seam. Limited details were recorded, however it is known that the site geologist associated the inflows with a 3 m fault and extraction below the Goondarrin Creek (Doyle, 2007). There is no documentation of the flow rates or the depth of the workings. It is highly likely that the depth of the workings was less than 100 m as the Kemira Colliery extracted the Wongawilli seam at depths of between 100 and 120 m, below the Mt Kembla workings.

Blue Panel, Wongawilli Colliery

Wongawilli Colliery was extracting under the Avon reservoir in 1982, using board and pillar mining techniques (Anderson, Stapledon and Mattes, 1989). The surface topography was irregular and thus the depth of cover varied between 80 and 100 m, from the reservoir to the coal seam. Seam extraction was adjacent to an igneous sill and cindered zone of coal (McNally and Evans, 2007). The inflows rapidly built up to over 2.4 mega litres per day (ML/day) in the panels Blue 3–4 and Blue 2 (New South Wales Chief Scientist and Engineer, 2014). This caused significant damage, the DSC to withdraw their approval for mining in these panels, and the abandonment of the panels.

These inflows decreased in the Blue 2 panel in contrast to inflows to Blue 3–4 that remained constant and subsequently increased with rainfall. This correlation with rainfall was attributed to a connection to the Avon reservoir (Anderson, Stapledon and Mattes, 1989). Furthermore, the algal content of the Avon reservoir water and water from throughout the Wongawilli mine indicated that the inflow to panel Blue 3–4 was from the reservoir. Inflow to the mine could have occurred in part through fractures created by strain concentrations near the seam extraction. A recent review also indicated that inflows could have occurred due to igneous sills and cindered coal that were hydraulically connected to the Avon reservoir and a confined aquifer (New South Wales Chief Scientist and Engineer, 2014).

Kemira Colliery

Unexpected inflows occurred at Kemira Colliery in 1989, during the extraction of the Wongawilli seam at approximately 100 to 120 m depth of cover (Whittall, 1990). These inflows occurred over one month and peaked at 0.65 ML/day (Whittall, 1990). The extra pumping that was required resulted in four months loss of longwall production. The inflows were a result of the saturation of the Mt Kembla workings in the Bulli seam. The Kemira Colliery workings exist within the Wongawilli seam 30 m below the Bulli seam and Mt Kembla board and pillar workings (Whittall, 1990).

Tahmoor Colliery

The sinking of the inclined drift for the initial development access at Tahmoor Colliery in the 1970s encountered fault structures (Fawcett and Rose, 1978). Minor inflows of up to 0.2 ML/day were expected as the drift and shafts were driven through the water bearing Hawkesbury Sandstone and the water table (Fawcett and Rose, 1978). In one of the shafts, at 37 m of depth steel rods separated from the drilling rig and water spouted 20 m up the shaft. The estimated inflow was 1.8 ML/day. Subsequent investigations determined the inflow source was a void within the sandstone that allowed for the formation of a stream path which was under considerable pressure (Fawcett and Rose, 1978). Further inflows occurred during the sinking of the decline and shafts one and two (Fawcett and Rose, 1978; Pells, 2011). Inflows were attributed to the fault zone within the Hawkesbury Sandstone. No similar inflows have been encountered in Tahmoor Colliery since that time (Pells, 2011).

Trends and potential risk of future inflows

It is important to note that these mines were operating in less than 100 m depth of cover and where surface to seam tensile cracks were created. The absence of a significant mine inflow event in the Southern Coalfields during the past 25 years could be attributed to a number of factors including; increasing depth of mines, increasingly higher stress regimes, in addition to improved mine design and subsidence management. The risk of inflows to mines in the Southern Coalfield appears to have reduced over time, although a small risk remains due to inherent geological uncertainty in the subsurface. This is in contrast to underground coal operations with high risk geological conditions, such as limestone aquifers overlying coalmines in China (Zhang et al, 2014).

FAULTING AND FLOW ASSESSMENTS

Nepean Fault

Recent operations at the Tahmoor Colliery intersected the Nepean fault zone, a major fault structure within the Southern Coalfields. The Nepean fault is part of one of the predominant regional structures, known as the ‘Lapstone Structural Complex’ a set of subparallel, high-angle, discontinuous, en echelon, reverse faults (Moffitt, 1998). The structures have formed through compression and wrenching. The Nepean fault zone lies along the northeast side of the mine, which is the southern end of the structure, as represented by the green line in Figure 2. The fault intersected can be detected over 200 m on the ground surface and at approximately 480 m of depth, in a northeast direction with a displacement of over 20 m (Maddock, 2014).

The Nepean fault is the only hydraulically active geological structure encountered during mining at this site to date (Merrick, 2012). Hypothetically, there is a safety risk if drilling or mining intersects a large source of water with a significant pressure head. The water pressure head at the base of an open, fluid filled aperture extending from the surface to 500 m of depth would be 5 MPa (Equation 1):

\[
P = \rho g h + P_{atmosphere}
\]

\[
P = (1000 \times 9.81 \times 500) + 101.3
\]

\[
P = 5006.3 \text{ kPa or } 5 \text{ MPa}
\]

where:

- \( P \) is pressure (kPa or MPa)
- \( \rho \) is density of water (kg/m³)
- \( g \) is acceleration due to gravity (m/s²)
- \( P_{atmosphere} \) is atmospheric pressure (kPa)

For example, if drilling was to encounter such a pressurised water source there could be significant flows of water and potentially breakouts. It is improbable that this hypothetical
GEOLOGICAL STRUCTURES AND FAULT-INFILL IN THE SOUTHERN COALFIELDS AND IMPLICATIONS FOR GROUNDWATER FLOW

Railway cutting fault

In 2013 the surface railway line above Tahmoor was reconstructed to divert the railway line from the Redbank Tunnel. This was due to the substantial subsidence impacts predicted that mining below the tunnel could induce (Maddocks, 2014). During the relocation of the surface railway line a fault was encountered at a cutting. The presence of this fault resulted in a significant slope failure at the cutting, as depicted in Figure 3. The Principal Subsidence Engineer from the NSW Department of Trade and Investment ordered a geological investigation due to the future potential impact the structure could have on the Main Southern Railway line (GHD Geotechnics, 2013a.).

The geological investigation included structural mapping that concluded the structure was a reverse fault, filled with weathered clay, tight and existed over a distance in excess of 20 m, as shown in Figure 4.

Subsurface drilling and mining had already been completed below the location of the fault intersection on the surface and there was no expression of this shallow fault in the mine at depths of approximately 400 to 500 m. However, a strike-slip fault was intersected in the mine below the old railway line, as shown in Figure 5. The deep fault was filled by a 5 mm band of fine grained mylonite, which did not extend to the other side of longwall 29 and was not detected at the ground surface. The location of the fault overlaid on the longwall panels is displayed in Figure 6. This is an example of a fault that is discontinuous over a significant thickness of overburden strata. The shallow fault and the deep fault were not connected. Both faults were infilled with a low permeability, clay material and thus acted as barriers to flow.

Corrimal Fault

The Corrimal fault is a normal fault that extends from the seam outcrop at the escarpment inland over a distance of approximately 3200 m, trending northwest (Mills, 2014). It is down thrown to the north-east and has a displacement of 1.3 to 3 m in the vicinity of the mine workings (GeoTerra, 2012).

The fault has been identified as a ‘hinge fault’, which means the fault has extended from the initial point along the strike (GeoTerra, 2012). The fault has been intersected in the development of longwall block six, in the both the maingate and tailgate development, as displayed in Figure 7. Although no inflows have occurred, the fault has resulted in the deterioration of the gate road for longwall six (Sydney Catchment Authority, 2014a).

The operators of Russell Vale Colliery present a case that the Corrimal fault is not continuing and the structure does not occur beyond longwall 7, as shown in Figure 7. The discontinuity of a fault structure is not unusual in the Southern Coalfields. However, the presence of the Corrimal fault on the surface and then its disappearance closer to Lake Cataract could be due to alluvial deposits that are concealing the structure (Sydney Catchment Authority, 2014a). The operators anticipate that mining within the vicinity of Lake Cataract will not have significant impacts on the Corrimal fault, subsidence behaviour or the hydraulic conductivity of the overburden strata (GeoTerra, 2012).

Due to the potential for significant impacts, approval of this mine has been referred to the NSW State Government’s Planning Assessment Commission (PAC). The PAC process required...
the submission of geological investigations and groundwater modelling. Groundwater investigations completed attempted to quantify inflows into the mine and develop a prediction of the fault’s hydrogeological character to determine the potential magnitude and extent of inflows through the structure (GeoTerra, 2012). It is possible that permeability of the structure could increase with stress induced fracturing, however there is no history of increased flow associated with previous mining activities on both sides of the fault from Cordeaux and Bulli Colliery in the Bulli seam.

The Independent Expert Scientific Committee on Coal Seam Gas and Large Coal Mining Development (IESC) did not explicitly discuss the fault but was concerned about the impacts of subsidence (IESC, 2014). ISEC concluded that the subsidence and groundwater modelling completed was not comprehensive and did not give sufficient consideration to the impacts on surface water (IESC, 2014). Given these concerns, the advice of ISEC was not in favour of allowing this mine to operate closer to the dam. It was subsequently

FIG 4 – Structural mapping from the geological investigation (GHD Geotechnics, 2013a).

FIG 5 – Old railway line before overlaid on Tahmoor mine map (Maddocks, 2014).
FIG 6 – Fault location in relation to Tahmoor workings (GHD Geotechnics, 2013a).

FIG 7 – Russell Vale workings, Lake Cataract and the Corrimal Fault circled in green (GeoTerra, 2012).
decided to permit the first 365 m of longwall six to be mined (Planning Assessment Commission, 2014).

Geological structures in Dendrobium Area

A review of inflow data for geological structures within Illawarra Coalmines including Wongawilli, Kemira and Dendrobium was completed by Doyle (2007) for submission to the DSC. Geological mine maps were scanned to identify 1660 points that corresponded with a fault, dyke, sill intersection or numerous joint related features that were recorded during first workings prior to pillar extraction or longwall mining. The data set specifically covered areas adjacent to stored waters where the DSC required information, which in these mines did not include major structures (fault structures with greater than 2 m of displacement nor of dykes greater than 2 m in thickness).

Over 95 per cent (1580/1660) of geological structures reviewed were not associated with any inflow. There were 65 points of inflow associated with a geological structure and another 15 possible conduits for water flow (total of 80). Recorded inflows at these points were less than 0.001 ML/day, except for two instances in the Dendrobium mine of up to 0.01 ML/day flow. The lack of inflow meant that it was not possible to correlate flows with depth of cover or distance from a reservoir.

Kemira Dyke and Fault Systems

There have been a large number of dyke-fault systems encountered at Dendrobium Colliery, within the 500 m notification zone of the Cordeaux reservoir. Mills (2005) investigated the Kemira dyke and a small adjacent fault system at Dendrobium. The fault had a displacement of less than 0.5 m and was not present at the surface, however the Kemira dyke outcrops on the surface, and is thus noted in surface mapping. The dyke outcrop creates ‘potential for there to be a substantial connection along the Kemira dyke/fault system to the overlying reservoir’ (Mills, 2005). The overlying reservoir is approximately 300 m above Dendrobium Colliery. The mine is also surrounded by old workings including Nebo, Kemira, Elouera, and directly overhead, the Mt Kembla workings. No flows are entering from the Mount Kembla workings (Mills, 2005).

Mills hypothesises that old mine workings could act as a buffer to potential flows from the reservoir; however, there is no barrier between the Mount Kembla working and the dyke system. There are currently no flows and it is evident that the dyke and fault are acting as barriers to flow. A dyke recently investigated by Mills (2013) was anticipated to experience significant horizontal stress changes due to mining. The investigation found that the increased horizontal stress has slightly reduced the hydraulic conductivity of the dyke structure.

CONCEPTUAL MODELS OF FAULTS

Mines in the Southern Coalfields intersect two styles of faulting. Firstly, major fault structures can extend for kilometres laterally but are rarely continuous vertically from the ground surface to coal seam. Mining through such structures may cause significant movement and surface deformation. The second style is much smaller, mine scale faults, which are generally infilled and rarely have associated groundwater flow. Intersection with and mining through these structures is possible with minimal impacts.

Based on the faults investigated a series of three conceptual models have been developed to classify the types of faults that occur in the Southern Coalfields. The first of these models is a discontinuous fault, as shown in Figure 8. This fault is present at the surface and at depth. Geological processes, tectonic movement and induced stresses can result in the fault being discontinuous. This style may be a conduit closer to the surface and a barrier at depth, due to the significant horizontal stresses. An example of a fault of this style is the Nepean Fault.

The second model is a fault with expression at the surface and not at significant depths, as displayed in Figure 9. This type of fault is generally affected by weathering, with either clay rich materials or a poorly structured silica-based mix of materials. This style of fault is a probable conduit for water in shallow ground, if it is saturated. However, there were no reports of such shallow structures being intersected by a mining operation, even with limited depth of cover. An example of this type of fault is the fault intersected at the Tahmoor rail cutting.

The third conceptual model describes a mine-scale fault as displayed in Figure 10. Thousands of these structures have been intersected with a lack of significant inflow, attributed to a high stress environment and thus the relatively low permeability of any geological structures. The faults examined in Doyle’s (2007) review of faulting in the Dendrobium area are examples of this style of fault.

This research project has determined that the faults of the Southern Coalfields are frequently acting as a barrier to groundwater flow. The interrelated nature of the groundwater system of the Southern Coalfields and Sydney Water Catchment makes it difficult to identify and quantify sources of water into longwall panels.
are illustrated in Figure 11 and summarised as follows:

- coal seam - lateral flow of water within the seam
- previous mine workings – water stored within old mine voids
- subsidence and fracture zone – interaction with aquifers either directly or indirectly
- surface water – reservoirs, lakes and creeks.

However, it is possible to identify the major sources of water into longwall panels. These sources and their pathways are illustrated in Figure 11 and summarised as follows:

- coal seam - lateral flow of water within the seam
- previous mine workings – water stored within old mine voids
- subsidence and fracture zone – interaction with aquifers either directly or indirectly
- surface water – reservoirs, lakes and creeks.

CONCLUSION AND RECOMMENDATIONS

There is a lack of evidence for direct vertical flow paths via geological structures from the ground surface, through overburden strata to coal seams, in the Southern Coalfields and no significant mine inflow events have occurred in the past 25 years.

There are several factors that reduce the potential for inflows to mines via geological structures in the Southern Coalfields:

- geological structures that are generally discontinuous, with a lack of expression from the ground surface to coal seams at depth
- infill of fault zones with low permeability clayey materials derived from weathering of igneous intrusions into the fault structures
- increasing depth of cover of recent mining operations, resulting in increased vertical stress, and reduced permeability of strata
- high horizontal stresses that are generally orientated perpendicular to mine-scale faults, reducing the aperture of joints and fractures
- improved mine design and subsidence management practices.

In the absence of significant flows via geological structures, water can seep into underground mine voids from old mine workings, from within the worked coal seam, and from subsidence induced fractures. However, groundwater flow systems in the region are complex due to inherent geological variability and the cumulative impacts of mining over the last 150 years. Quantifying groundwater hydraulics (flow directions, rates and storage) near mine voids on a site-specific basis is therefore challenging.

Targeted geotechnical and hydrogeological monitoring at surface and mine workings in high-risk areas is recommended to identify inactive structures and structures that may be reactivated by mining. For example, high-resolution seismic surveys can detect fault displacements as small as 2 to 2.5 m (Zhou and Hatherly, 2014). The significance of mining-induced seepage near the surface and within shallow strata can be evaluated with a complimentary suite of techniques including: geochemical-isotope tracers (David, Timms and Baker, 2014), aquifer interference tests and hydraulic tomography (Illman, 2014) and refinement of empirically based predictions of mine induced fracture zones (DeBono et al., 2014). Advanced numerical modelling may also be appropriate in some cases, if suitable data is available, to better understand coupled geomechanical and hydrogeological processes. The next phase of this research will evaluate the permeability of intact rock matrix and fault-infill materials under various stress and geochemical conditions.

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Development of Dozer Push Optimisation Software for Commodore Coal Mine

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ABSTRACT

If significant improvements in mining performance are to be achieved, new software and technology must be developed and implemented. This in turn will help to maximise productivity and minimise production costs. This paper details the development and evaluation of a recently created dozer push program, Dump Designer, that was used to maximise dozer push horizons at Commodore Coal Mine in the Surat Basin, Queensland. The necessary data required for Dump Designer was collected from the geological model of Commodore. From this, Dump Designer was trialled and the dozer push horizons were maximised where possible. Operating costs were then applied to show the benefits of maximising waste allocated to dozers. After processing 556 mining blocks, it was concluded that the coal dip significantly affects the amount of waste the dozers are able to push. It was also found that there are some implementation constraints with regards to mine planning that may prevent the dozers from being able to work to maximum dozer push horizons.

INTRODUCTION

There are only a few dozer push programs on the market that are relevant to the mining industry, in particular coal mining operations. Most mining programs require basic training and large quantities of site specific data to be imported into the program before use. The technical skill and licensing requirements makes the cost of using these programs expensive. There is a lack of user- and cost-friendly software on the market.

At Commodore Coal Mine, a majority of the previously mined blocks have had dozers remove the waste below the prestrip horizon to the top of the Millmerran Upper Seam. Currently at Commodore there have been no studies carried out on optimising dozer push horizons for each mining block. Therefore, by developing a program that focuses on a mass balance between the available capacity of the dozer waste dump and the overburden material pushed with dozers, the horizons for all remaining mining blocks can be determined. This program can then be incorporated into the short- and medium-term planning at Commodore to ensure the maximum amount of waste is removed with dozers when changes in the pit occur. The results of the dozer push software trial at Commodore could be used to help further develop the software.

BULK DOZER PUSH

Background

Bulk dozer push is an alternative method to conventional truck and shovel. Dozers have the ability to move large amounts of waste material over a short distance very cheaply whereas over long distance truck and shovel fleets are more economical. With the addition of bigger machines, dozers have become a typical component in mining operations, removing overburden down to the top seam of coal (Hayes, 1997). For this method to be effective and economically viable, certain favourable geological conditions are required, including:

- flat to gentle dipping coal seam deposit
- suitable ground pressure
- shallow coal seams
- suitable overburden material.

Advantages

In comparison to a truck and shovel system, bulk dozer push has many advantages and disadvantages that sets both systems apart. The advantages include:

- lower capital cost
- lower operating cost
- less disturbances to neighbours
- ability to operate during poor weather conditions.

The capital cost associated with dozers is relatively low with Caterpillar D11 Carry Dozers costing approximately A$2.2 million (Bertinshaw et al, 2013). Dozers are able to support themselves and do not require any ancillary equipment such as graders to operate. The operating cost of a dozer is dependent on operator experience, correct gear selection, grade and push distance. Despite this, dozers still have a lower operating cost in comparison to a standard truck.
and shovel system. There is a lower economic risk associated with a dozer system due to the lower capital and operating expense.

Dozers largely operate down below the crest of the highwall, whereas trucks run along the top of the mining pit and frequently dump high (Seib, 2013). Thus, dozers tend to have less impact on neighbours regarding dust, noise and light disturbances. Dozers can operate whilst it is raining; however, if the rain becomes excessive, the dozers will be required to cease operation to avoid becoming bogged.

Disadvantages
The main disadvantages with using a bulk dozer push system compared to a truck and shovel system are that:

- one is unable to selectively mine the coal deposit
- specialised training is required
- the operator is exposed to continuous vibrations
- there is a requirement to cut the highwall.

Once dozers begin on a strip they are required to finish that strip before moving on. If the dozers cut and change from strip to strip no coal will be uncovered and this will lead to delays in production. Therefore, dozers have a higher geological risk associated with them, especially if the coal mine has highly variable coal quality and quantity constraints. Using dozers as the primary equipment to remove the waste is relatively new so training will be required for operators, supervisors and technical people (Seib, 2013). This could be challenging and will require a change in mining culture. The cutting of the highwall is a difficult task for the operator to undertake. This requires the operator to scale the top of the blast and clear it away from the highwall, illustrated in Figure 1.

Dozer push operation
A cast blast is employed at Commodore with the intention of reducing the amount of overburden the dozers have to push. Figure 2 shows a typical cast blast profile seen at Commodore.

Coal Mine is a privately owned operation and are operated by the Millmerran Operating Company (Rickuss, 2015). At present, Downer EDI Mining is contracted to operate the mine for the next five years. Commodore currently consists of three pits: A, B and C. A combination of conventional truck and shovel as well as dozer push methods are used to remove the overburden and uncover the coal.

Downer EDI Mining is responsible for the operation and management of the mine and associated activities, including: providing mine planning and design expertise, drill and blast services, overburden stripping, coal mining and rehabilitation (Downer Group, 2014). The strip mining method is used at Commodore with active mining occurring in 100 m long by 50 m wide strips.

FIG 1 – A dozer cutting the highwall at Commodore.

As shown in Figure 1, there is marginal room for error; therefore, the difficulty level is high. The cutting of the highwall is time consuming and is best to be performed on day shift when visibility is adequate.

COMMODORE COAL MINE

Site overview
Commodore Coal Mine is an open cut thermal coal mine that is located in the Surat Basin near Millmerran, Queensland. Millmerran is approximately 80 km south-west of Toowoomba (InterGen, 2014). The coal mined from Commodore directly supplies the Millmerran Power Station, which is adjacent to the mine. The Millmerran Power Station and Commodore

FIG 2 – A typical blast profile at Commodore.

The dozers will begin pushing the overburden down and sliding it over into the waste dump. Once the dozers have reached the desired pivot point height they will begin pushing up at approximately 10–15° for 60–100 m to build the dump. The pivot point is the point that all material must be pushed past to maintain the coal edge (Pettigrew, 2012). The general rule of thumb is that the pivot point is located 45 degrees from the bottom of the coal seam to the dozers cut line (Pettigrew, 2012). By doing this it enables a clean coal edge to be established.

Slot dozing and scalloping techniques are employed to increase the productivity of the dozers as is evident in Figure 3.

Slot dozing is a technique were the dozers will work in a slot confinement. The slots help to contain the material within the blade area instead of spilling over the sides (Hayes, 1997). All slots are perpendicular to the highwall to ensure the length of the push does not exceed the intended length – the dozer will not deviate off course. Scalloping allows the dozer operator to pick up a full load of material and stack the front material at the back of the dump and the back material at the front (Pettigrew, 2012). Therefore halfway through the push a dozer can take a cut and push that material to the front of the dump. This technique essentially halves the push distance. The final product of the push can be seen in Figure 4.
The wedge left against the highwall is left for truck and shovel to remove. The dozers are physically unable to get behind the wedge and push it forward into the waste dump. This wedge is considered to be prime dirt as no mechanical work has been carried out on it.

**Geology**

At Commodore there are two main seams, the Millmerran Upper (MU) and the Millmerran Lower (ML) Seam. There are many thin rider seams present; however, they are generally of poor quality and are often not recovered. The main reason for not recovering the rider seams is due to the coal quality constraints put in place by Millmerran Power Station (Pettigrew, 2012). In Figure 5, the different seams that are found at Commodore can be seen.

The MU and ML have a combined seam thickness of approximately 5 m; however, this does vary slightly throughout the three pits. In Pit A and B the coal seams are relatively flat; however, in Pit C the coal seams dip at approximately ten per cent. The overburden consists of sandstone, mudstone and clay material. It is classed as soft dirt; however, blasting still takes places for various other productivity reasons. The coal is relatively shallow in Pit A and B but it deepens in Pit C, where it is covered with approximately 40 m of overburden.

**Commodore dozer push horizons**

There were two methods that were being employed to find the maximum dozer push horizons at Commodore. The first method, was a Microsoft Excel Spreadsheet that required the user to input certain variables regarding the highwall, dozer push length and angle. A pivot point height could be entered in and the dozer waste volume and dump volume were calculated. This method was relatively accurate when there was no coal dip; however, it could not factor into account the change in the coal dip and the change this would have on the advancing waste dumps. No diagrams were generated and the user was unable to see how the next waste dump is also dependent on the previous waste dump. Therefore, this method was not appropriate for Commodore.

The second method employed was using Maptek’s Vulcan 8.1 to recreate the highwall height. Slices of the highwall are then made which the dozer dumps are based on. This is a trial-and-error process that is extremely time consuming as each waste dump needs to be drawn manually. Figure 6 shows the range diagrams created in Vulcan and the slices that made until an appropriate dozer push horizon has been found.

These two methods set a foundation for developing a dozer push software specifically to analyse and maximise horizons. The positives from both of these methods have been employed and were crucial in the early stages of development.

**DUMP DESIGNER**

**Overview**

Dump Designer was designed with the purpose of assisting the technical service team at Commodore Coal Mine with determining dozer push horizons. The purpose of creating this program was to maximise the amount of overburden moved with dozers. It combines trigonometry and polygon area calculations to calculate the dozer push and dump areas. A volume can then be found by multiplying the area by the block length. It uses two methods to create optimum
dozer waste dumps, the first being dumps based on a pivot point height and the second is dumps based on the level the dozers are going to push to. In this paper, only the dozer push method will be discussed. Dump Designer allows the user to change various aspects of the dumps and highwall to create dumps specific to the environment. The graphical display alters as the user changes the required inputs and this graphical display shows how the previous dumps affects the next one.

Variables of Dump Designer
In order for Dump Designer to work, several variables are required to be entered:
- block gap
- low wall angle
- dozer push length
- dozer push angle
- angle of repose
- coal dip angle
- ash pad length
- ash pad depth
- coal seam thickness
- interburden thickness
- swell factor
- highwall height
- highwall angle
- dozer push depth
- pivot point height.

Figure 7 outlines the purpose that each variable plays in order to create the highwall and waste dump.

The dozer push volume is the amount of overburden the dozer will cut from the blasted volume and push into the waste dump. In both the pivot point and dozer push method, the changing of the pivot point height and dozer push depth will increase or decrease the total dozer push volume. The available waste dump volume is the total volume of the waste dump minus the ash volume. If there is no ash space in the dump, then it will just be the total volume of the waste dump.

Dozer push method
The dozer push method was created after the pivot point method; thus, it is a more practical approach and it has been further developed. The dozer push method requires the user to enter in a depth the dozer will push to. This depth is based on the top of the highwall and is projected vertically down until the desired depth has been reached. From this, an imaginary line is drawn across the highwall until it intercepts with the low wall of the waste dump. This imaginary line can be seen in Figure 7 (the yellow line). Using the variables entered in by the user the dump is able to be created.

The wedge of material between the highwall and waste dump is considered rehandle material that the dozer is unable to move. The dozers are unable to move this, as the dozers cannot get behind the material and push it. Thus, this material is left behind for truck and shovel to take so it is excluded from the dozer push volume. The range diagram output from Dump Designer is shown in Figure 8.

METHODOLOGY
The recently updated geological model of Commodore Coal Mine for the remaining life-of-mine was imported into Maptek’s Vulcan. From the geological model the following data was collected:
- overburden thicknesses
- coal seam thicknesses (Millmerran Upper and Lower combined)
- Millmerran Lower Seam floor dip.

This data was collected for Pit A Strip 29 to 56 for Blocks 24 to 42 and for Pit B Strip 31 to 56 for Blocks 7 to 22. Pit C was not analysed and this will not be discussed in this paper. Correspondence was made with the site surveyor of Commodore to determine relevant swell factors for Pit...
A and B. It is important to note that the thicknesses of the overburden and coal seams were found using grids and the average thicknesses for each block was taken.

Dump Designer is able to analyse five consecutive strips from a particular mining block; however, the first strip within the block is only used to draw the previous waste dump. Therefore, only four strips can have actual data collected from the range diagrams. Microsoft Excel was used and the data collected was placed into a spreadsheet that was tailored to Dump Designers requirements.

End of month scans of the current dozer waste dumps for all Pit A and B at Commodore was used to gather the necessary data to build the existing waste dump. In order to do this, the height of the low wall was found as well as the coal dip for that particular block. From this, the pivot point height could be calculated for all necessary blocks. For all waste dumps, Dump Designer generates the pivot point height so the dozer push depth for the first waste dump was a trial an error process until the desired pivot point height was found. It was crucial to get this waste dump accurate, as it would result in all other waste dumps being inaccurate. As the strips progress the inaccuracy of the first waste dump would be less, however, for the first couple it would be significant. This is because the next waste dump capacity is highly dependent on the previous waste dump. Thus, if the dump level of the previous waste dump is too high, the next waste dump capacity will be affected.

The Australasian Institute of Mining and Metallurgy’s Cost Estimation Handbook, published in 2013, was used to calculate the required operating costs for excavators, trucks and dozers. The operating costs in the Cost Estimation Handbook needed to be converted to the equipment that is found at Commodore Coal Mine and this was carried out by ratios dependent on the size of the equipment. This value is inclusive of capital cost, fuel, lubrication, parts, maintenance cost, overhaul, tyres or tracks, ground engineering tools and operating cost exclusive of labour. Labour costs were not included in the operating cost due to all equipment requiring an operator. The calculated operating costs are shown in Table 1.

<table>
<thead>
<tr>
<th>Equipment</th>
<th>Operating cost ($/h)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Hydraulic excavator (15 m³)</td>
<td>1986.8</td>
</tr>
<tr>
<td>Rear end dump truck (136 t)</td>
<td>893.1</td>
</tr>
<tr>
<td>Dozer (D11)</td>
<td>954.4</td>
</tr>
</tbody>
</table>

The budgeted production rates of Commodore Coal Mine for the required equipment were used to convert truck and shovel and dozer volumes into hours. From this the operating costs could be applied to calculate the total savings.

RESULTS OF DOZER PUSH ANALYSIS

Performance variation over distance

Across the data collected, it was found on average; by pushing 100 m the dozer waste dump capacity increases by seven per cent. Although this increase in capacity means that more waste can be moved with dozers, which may be beneficial in terms of production costs, there is a negative impact on the productivity of the dozers. Pushing the waste an extra 25 m decreases the productivity of the dozers by approximately ten per cent. This information is highlighted in Figure 9.

As shown in Table 2 the dozer waste has increased significantly and this can be attributed to analysing the dozer push horizons block by block. Through the use of range diagrams it became evident how the previous waste dump affects the next one and so on. The dozers were able to take a section of the prestrip waste due to the lack of overburden coverage and the coal seam dipping upwards.

<table>
<thead>
<tr>
<th>Pit A</th>
</tr>
</thead>
<tbody>
<tr>
<td>Dump Designer dozer volume (m³)</td>
</tr>
<tr>
<td>Original dozer volume (m³)</td>
</tr>
<tr>
<td>Difference (m³)</td>
</tr>
</tbody>
</table>

For Pit B, out of the 231 blocks that had data collected for them only 132 could be analysed using Dump Designer. From the blocks that were analysed, it was found that a majority of them could have dozers take more than the original 15 m of waste.

The change in the coal dip in comparison to Pit A has had a significant impact on the dozer waste dump capacity. Unlike Pit A, only 11 of the blocks analysed in Pit B could have the dozers take a pass of the prestrip waste. Similarly to Pit A, once all the data had been processed in Dump Designer, the total dozer volumes could be compared against the original dozer volumes. These volumes can be seen in Table 3.
was due to the variations in operating costs over different dozers grades have not been used in the costing data. This distances, change in elevation over the haul route and differing it was kept constant. It is important to note that the haulage of analysing the dozer push horizons. While the overall hydraulic backhoes were calculated as another means The operating costs for dozers, rear-end dump trucks and Cost analysis

The coal dip plays a significant part in the waste dumps and dictates the amount of waste the dozers are able to move. In a majority of the cases in Pit A, the Millmerran Lower Seam floor is on an upward dip. The greater the coal is dipping upwards, the greater the dump capacity will be. The coal dip can also impact on the productivity of the dozer. In this case the coal dipping upwards lets the dozers push with a negative grade for longer. This in turn will increase the productivity of the dozers as less time is spent pushing with a positive grade.

An analysis of the effect the coal dip plays on the dozer push level with respect to differing overburden thicknesses was explored. Using Dump Designer, the dump variables remained the same with the only changes being the coal seam dip, overburden thickness and dozer push level. A graph of the results found can be seen in Figure 10.

From Figure 10 the amount of waste the dozers are able to move increases significantly from a negative ten per cent dip in comparison to a positive five per cent dip. It must be noted, that in the positive five per cent dip and the zero per cent dip, starting points are the same. In both cases the overburden thickness of 20 m can be removed without the dump reaching full capacity.

Cost analysis

The operating costs for dozers, rear-end dump trucks and hydraulic backhoes were calculated as another means of analysing the dozer push horizons. While the overall production cost may fluctuate, for the purpose of this study it was kept constant. It is important to note that the haulage distances, change in elevation over the haul route and differing dozers grades have not been used in the costing data. This was due to the variations in operating costs over different blocks and the purpose of this study is not comparing dozer operating costs to truck and shovel operating costs.

The operating costs of dozers against truck and shovel were applied to the differences in volumes generated by Dump Designer and the original dozer push volumes. The cost of using truck and shovel in comparison to dozers can be seen in Table 4.

Effects of coal dip

The coal dip plays a significant part in the waste dumps and dictates the amount of waste the dozers are able to move. In a majority of the cases in Pit A, the Millmerran Lower Seam floor is on an upward dip. The greater the coal is dipping upwards, the greater the dump capacity will be. The coal dip can also impact on the productivity of the dozer. In this case the coal dipping upwards lets the dozers push with a negative grade for longer. This in turn will increase the productivity of the dozers as less time is spent pushing with a positive grade.

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### TABLE 3
Comparison on total dozer volumes for Pit B.

<table>
<thead>
<tr>
<th></th>
<th>Pit B</th>
</tr>
</thead>
<tbody>
<tr>
<td>Dump Designer dozer volume (m³)</td>
<td>22 083 614</td>
</tr>
<tr>
<td>Original dozer volume (m³)</td>
<td>19 561 470</td>
</tr>
<tr>
<td>Difference (m³)</td>
<td>2 522 144</td>
</tr>
</tbody>
</table>

As shown in Table 3, the dozer waste has increased significantly; however, not to the extent seen in Pit A. This is because of the change in the coal dip that occurs in Pit B. These volumes shown in Table 3 are only the volumes from the blocks that could be analysed using Dump Designer.

### TABLE 4
Total savings for maximising dozer push horizons.

<table>
<thead>
<tr>
<th></th>
<th>Pit A</th>
<th>Pit B</th>
</tr>
</thead>
<tbody>
<tr>
<td>Truck/shovel cost ($)</td>
<td>13 101 923</td>
<td>6 118 670</td>
</tr>
<tr>
<td>Dozer cost ($)</td>
<td>11 714 840</td>
<td>5 470 895</td>
</tr>
<tr>
<td>Savings ($)</td>
<td>1 387 083</td>
<td>647 775</td>
</tr>
</tbody>
</table>

Table 4 highlights the savings that can be made when maximising the amount of waste moved with dozers. These savings are not a representation of the savings Downer EDI Mining will incur if the dozer push horizons found from Dump Designer are used. This has been completed with the intention to show the benefits that may be achieved by maximising waste moved with dozers. To calculate the possible savings made, an in-depth mine plan would be required with dump locations for the trucks. It would also require a differing operating cost for dozers over various sections of the push.

### Implementation of updated dozer push horizons

It has been shown that maximising the dozer push horizons at Commodore will most likely lead to a reduction in production costs; however, there are some constraints in the implementation of increasing the waste allocated to dozers, which includes:

- mine planning changes
- additional dozers may be required
- cultural change.

Prestrip waste was allocated to the dozers if there was still available volume in the waste dump. Although, in some cases the available volume exceeded the prestrip section allocation, this was to prevent benches not being able to be drilled. The constant changing of drill bench heights results in more work needed to be carried out during the prepping of the benches. The worst case scenario is the bench is unable to be drilled. Despite the attempt to prevent the constant changing of the drill bench heights, this has not been checked, and when added into the annual mine plan changes may be required.

The dozers are unable to selectively mine the deposit, thus when they begin on a strip they are unable to move on until it is complete. If the dozers have been working in areas of concern in terms of the coal quality, issues may arise when sending that coal to the power station. Therefore, more blending may be required and this will increase the daily production cost. Dozers have a higher geological risk associated with them and thus further studies need to be conducted to ensure allocating more waste to dozers will not affect the coal mining process.

The increase in waste allocated to dozers is likely to lead to a reduction in production costs; however, capital expenditure may be required to purchase more dozers to assist with the waste removal and meet production targets. A cultural change will be required, as currently, Commodore is predominantly a truck and shovel operation with dozers...
used to assist in waste removal. Thus, for dozers to be seen as an integral part of the waste removal fleet, a cultural change will be required.

ANALYSIS OF DUMP DESIGNER

Advantages
There are two main advantages of using Dump Designer to maximise dozer push horizons at Commodore:

1. the graphical output of range diagrams allows the user to understand the effects of coal dip on the waste dumps
2. it is easy to use, quick in generating range diagrams and its volumes are accurate.

The range diagrams generated from Dump Designer are the main feature that sets it apart from any other known dozer program currently on the market. There are several benefits of having a graphical output that changes by the user changing a variable input. The key one being; the user is able to see the effects the previous waste dump can have on the next waste dump. If the dozer push horizon has not been maximised in the previous waste dump, the next waste dump will have a decrease in capacity. This will continue to flow on for the following waste dumps. This however, can be seen by instantly looking at the range diagram output and adjusting accordingly to correct this.

Disadvantages
Whilst running scenarios for the all the mining blocks in the next ten years of mine life, it was found that not all blocks were able to be analysed using Dump Designer. There were two main issues that were found:

1. Dump Designer is constrained to the geometry of the mining block
2. only the top section of the highwall was analysed as dozer material.

For a mining block to be analysed accurately it needs to be a rectangle or square, no polygons can be analysed. If the mining blocks are not rectangles or squares the volumes generated will be inaccurate and thus the dozer push horizons will be incorrect. This issue was found when analysing mining blocks situated along the pit boundaries. Although, this did not affect many blocks it still impacted on the analysis of dozer push horizons. This geometry constraint was not considered when developing the program and is something that can be improved on in the future.

There were other issues that were encountered; but, this had to do with the way in which the code has been written. The author’s moderate experience with PHP suggests that numerous improvements could be made. This has directly impacted on Dump Designer’s ability to load the range diagrams. It was also found that only one user at a time can access Dump Designer.

CONCLUSIONS
The paper investigated the dozer push horizons of Commodore Coal Mine with the newly developed, Dump Designer. In total, 556 mining blocks were examined using Dump Designer at Commodore for the next ten years of mine life. The key finding from this paper was the effect the coal dip has on the dump capacity. It was found that with an overburden thickness of 30 m and a coal dip of five per cent the dozers are able to remove 22.4 m of the overburden, in comparison when the coal dip is -10 per cent the dozers can only take 13.5 m. This understanding of the effects the coal dip plays on bulk dozer push is crucial to Commodore as the dip varies between each strip.

By maximising the dozer push horizons for Pit A and Pit B an extra 5.4 and 2.5 million bank cubic metres of waste can be moved with dozers over the next ten years, respectively. 140 out of the 325 blocks analysed in Pit A could have dozers take all of the prestrip or a pass of the prestrip that is usually allocated to truck and shovel. The main reason for this was the coal predominantly dips upwards and the coal is relatively shallow. Not all blocks in Pit B could be analysed and this was due to the overburden thickness significantly increasing as the pit advances. It was identified that truck and shovel operations would be more economical in taking the top section of waste material. A line between when truck and shovel operations stops and dozers takes over needs to be defined in the deeper parts of the pit.

Regarding the overall unit cost, the dozers were the cheapest equipment to run and this was evident as approximately $2 million in savings can be seen by maximising the dozer push horizons at Commodore for the next ten years.

There are some implementation constraints that may prevent the dozers being able to work to maximum horizons and further studies will be required. From this experience, improvements to Dump Designer can now be made especially in areas where issues were encountered.

ACKNOWLEDGEMENTS
The author would like to thank her late father, Simon Uren for his guidance, support and help whilst developing Dump Designer.

REFERENCES


Selb, W, 2013. Personal communication, October.

Economic Selection of Surface Mine Haulage Options

D van Hest and P F Knights

ABSTRACT

Bulk material haulage contributes significantly to the operating cost of mining operations. This study introduces a new graph for determining economic haulage options for Australian surface mines. It is designed to be used during the conceptual and prefeasibility stages of projects to assess the most cost-effective haulage options for mines having a range of capacities of between 100 t and 1 Mt per day, and over a range of haulage distances of between 50 m and 100 km. Over very short haulage distances (<200 m), dozer push operations offer the most economical alternative. As t-km limits are increased, the next most economic option is medium capacity front-end loaders, followed by small, medium and then large off-highway truck/shovel systems. Conveyor haulage is the cheapest option when large throughputs are required (>500 000 t/d), or as haulage distance becomes more significant (>20 km).

INTRODUCTION

This study was inspired from an economic haulage curve developed for teaching purposes at Queens University, Canada in 1981 (see Figure 1). For a given range of production and haulage distance, this graph indicates which method of bulk material movement is lowest cost.

The graph is of relevance to mining engineers as an aid to selecting mine haulage options during the conceptual and prefeasibility stages of the surface mine design process. However, since the data on which the graph is based is over 30 years old, the information contained is no longer accurate.

This paper introduces an updated version of the graph and includes a range of haulage options that were not previously considered.

METHODOLOGY

This study aims to construct an up-to-date cost model (A$/t) for conventional material haulage options in Australian surface mining operations. These costs are varied based on the horizontal distance material is to be moved and the throughput, in tonnes per day (t/d) required from a mining operation. This will ultimately serve as a useful reference for selecting material haulage systems having lowest cost under a set of simple assumptions.

The scenarios analysed were limited by varying two factors; production rate and haulage distance. A series of typical haulage profiles were created to simulate those found in operations around Australia. The model should not be applied to scenarios that radically depart from standard profiles.


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Each haulage method has representative equipment assigned to represent their class of haulage. Different equipment capacities were analysed to determine the effect of scale on haulage economics. All classes of equipment are presented in the final model.

The analysis was conducted for a range of discrete haulage methods including dozer push, front end loader trams, dozer assisted scrapers, articulated dump trucks, rigid body off-highway, rear dump trucks as well as overland conveyors. The latter is fed via an in-pit crushe supplied by a small shovel/truck fleet, and includes a spreader discharge system.

The final analysis was conducted for haulage scenarios between 50 m to 100 km haul distance and daily productions of between 100 t and 1 Mt and presented on a logarithmic scale.

**OWNING COSTS**

The capital investment into any piece of machinery is a large expenditure. Depreciation of an item is accounted for based on one of several depreciation models (Ferguson, 2014). Depreciation costs are used to calculate the daily instalment on the capital cost of the item. This method requires the expected economic life of the machine, as well as the hours used per day. Equation 1 illustrates how to calculate the daily depreciation of an item.

\[
\text{Depreciation} \left( \frac{\$}{\text{day}} \right) = \frac{\text{Capital Cost} \cdot 24 \cdot \text{Availability}}{\text{Economic Lifetime (hrs)}}
\]  

(1)

Capital costs have been derived from documented typical costs presented in the Australian coal cost guide (R2 Mining, 2011) with the application of linear modelling. Aghajani et al. (2010) provide a non-linear cost model for excavators.

Finance costs address the opportunity cost of capital that is tied up in machinery. According to Bertinshaw et al. (2013) the finance cost can be calculated according to Equation 2, where WACC is the weighted average cost of capital and is calculated according to Equation 3 (Brailsford, Heaney and Bilson, 2011). E/V is the amount of equity financing, Rd the cost of debt and D/V the ratio of debt financing, Rd the cost of debt and TC is the corporate tax rate.

\[
\text{Finance} \left( \frac{\$}{\text{day}} \right) = \frac{(\text{Capital + Salvage}) \cdot \text{WACC}}{2 \cdot 365.26}
\]  

(2)

\[
\text{WACC} = \frac{E}{V} \Re + \frac{D}{V} \Rd (1 - t_c)
\]  

(3)

Insurance is another cost that is taken on, whether through an outside insurer or by self-insuring (McCarthy, 2013). Bertinshaw et al. (2013) states that for mobile surface plant the insurance cost is approximately 2.5 per cent of the capital cost, whilst, for fixed surface plant, this is reduced to two per cent of capital cost.

**OPERATING COSTS**

Diesel consumption is taken from manufacturers’ specifications; however, when that is not available Equation 4, derived by McCarthy (2013), provides a useful empirical equation to determine the fuel cost per machine per day. This equation assumes diesel costs of $1.50 per litre. The fuel job factor ranges between 0.1 and 0.2, depending on many variables such as engine revolutions, duty cycle and elevation.

\[
\text{Fuel Cost} \left( \frac{\$}{\text{hr}} \right) = \text{Power (kW)} \cdot \text{FJF} \cdot \$1.50
\]  

(4)

Lubricant consumption varies by machine type and function. A machine with more hydraulics working under high stress conditions will consume more hydraulic fluid than a machine with less hydraulics used occasionally. McCarthy (2013) recommends that, if possible, field data be used to accurately predict future lubricant consumption. When these figures are unavailable, a rule of thumb can be applied stating that lubricant cost ranges from 15 per cent to 40 per cent of the fuel costs of a piece of machinery.

Maintenance labour depends significantly on operators, maintenance crew, conditions of the mine and age of equipment. McCarthy (2013) gives a series of maintenance ratios (MR) for different large open pit pieces of equipment. These are listed in Table 1 and can be used with Equation 5 to determine total hourly maintenance costs per item of equipment.

\[
\text{Maintenance Cost} \left( \frac{\$}{\text{hr}} \right) = \text{Unit Cost} \left( \frac{\$}{\text{hr}} \right) \times \text{MR}
\]  

(5)

Various models exist for calculating the true value of maintenance supplies including McCarthy’s (2013) model, whereby equipment is considered to be made up of spare parts, each having a different lifetime. The model developed in this study estimates repair parts costs according to Equation 6. This model assumes that the average lifespan of spare parts on mining machinery is 10 000 hours.

\[
\text{Repair Cost} \left( \frac{\$}{\text{hr}} \right) = \frac{\text{Capex} \cdot \text{RPCF} \cdot \text{RPLF} \cdot \text{RPJF}}{10 000}
\]  

(6)

The repair parts cost factor (RPCF) varies between 0.15 and 0.25, and allows for adjustments in premiums of repair items to be made. The repair parts life factor (RPLF) allows for correction based on expected lifetime decrease or increase for all parts. It ranges from 0.8 to 1.2. For example, a conveyor idler will have a longer expected life than the hydraulic hoses of a dozer in high stress conditions. Most items will be treated as mid field and given RPLF of 1.0. The repair parts job factor (RPJF) allows for correction against job conditions, and is site specific.

Operating labour covers the cost of an operator on each shift panel and makes allowances for days off and absenteeism. It will be assumed that all mines use a four panel roster with the annual wage reflecting this. Using Equation 7 the total operating cost per hour can be estimated. Note that the effective utilisation of a machines is given by its physical availability multiplied by its utilisation. Typical operator wages are listed in Table 2 (note that these reflect peak Australian market values). On-costs are incurred, such as superannuation, insurance and tax. Bertinshaw et al. (2013) quotes that, for an Australian cost structure, 1.31 people must be paid per year to maintain one person per equipment item. This also accounts for all the leave types that each person is able to receive.
TABLE 2

Typical yearly operator wages including on-costs (Bertinshaw et al., 2013).

<table>
<thead>
<tr>
<th>Haulage option</th>
<th>Operator wage ($)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Shovel/excavator</td>
<td>215 000</td>
</tr>
<tr>
<td>Truck</td>
<td>195 000</td>
</tr>
<tr>
<td>Front end loader</td>
<td>215 000</td>
</tr>
<tr>
<td>Track dozer</td>
<td>202 000</td>
</tr>
</tbody>
</table>

Operating consumables can be broken down into two broad categories: ground engaging tools (GETs) and tyres. Ground engaging tools include bucket teeth and wear plate and cutting edges. The consumption of ground engaging tools varies with digging conditions with coarse and hard rocks leading to higher wear rates and incurring greater costs. Equation 8 can be used to calculate GET costs as an hourly rate, however the costs are usually a small fraction of total operating costs. Wear parts cost factor (WPCF) is a constant between 5 and 20 millionths, varying on ground and digging conditions, and wear parts job factor (WPJF) is a constant varying between 0.3 and 3.0 based on frequency of changing and difficulty to replace (McCarthy, 2013).

Tyres can incur significant costs and require significant management of haul roads, pit floors and waste dumps. Equation 9 (McCarthy, 2013) provides an estimate for the cost of tyres. This equation assumes an average life of 4000 hours. The tyre job factor (TJF) is a value ranging between 0.3 and 4.0, and varies based on the ease of replacing tyres.

\[
\text{Wear Parts Cost} = \text{Capex} \times \text{WPCF} \times \text{WPJF} \times \text{TJF} \tag{8}
\]

Job factors are chosen to adjust for different scenarios of cost estimation. They are applied to account for the effects of difficult and easy job conditions on operating costs. Similarly, they can also be used to adjust the operating cost based on operator experience. These factors are useful when the operating conditions for each piece of equipment is known. For the purposes of this study, job factors will be nominated to a midfield number to represent ‘average’ operating conditions.

COST PER TONNE ANALYSIS

For each piece of equipment, owning and operating costs were calculated and summed to produce a total daily cost per tonne per equipment.

The average cycle time for a machine over a haulage range of 50 m to 100 km was calculated and used to determine the tonnes each piece of equipment could move per day. For haul trucks, 40 per cent of cycle time was assumed on grade. Fleet size was then determined to meet the required production rate.

The total daily cost per equipment was then multiplied by the resulting fleet size to determine the total daily fleet cost for moving a required daily production over the specified distance. This value was then divided by horizontal haulage distance (km) and the daily throughput (tonnes) to give a cost per t-km for each haulage method and scenario. Each of these values was then compared, allowing the lowest cost haul method for a given production and distance to be determined and placed in a graph, ultimately completing the process of generating the new economic haulage diagram.

Appropriate pairing of equipment is critical and goes beyond ensuring a whole number of passes to fill haulage equipment. A match factor calculation was used to verify calculated fleet sizes. This was done to ensure that the excavators (or loaders) were achieving full effective utilisation.

In these scenarios, it is assumed that the excavator or loader limits the required production. This is then matched with the number of trucks required to deliver that daily production as the overall cheapest method of delivering throughput. Whilst this is adequate for the purposes of this study, at the Feasibility stage of a study stochastic aspects of the truck/shovel systems should be considered.

CONVEYORS

The capital and operating costs of conveyor systems vary according to both distance and throughput. This is moderated by the existence of belt capacity limits and as well as limits on the length of individual belt sections.

The maximum conveyor throughput considered in this study was 10 000 t/h, or 240 000 t/d while the maximum haulage distance was 2.2 km. It is been assumed that maximum length is independent of throughput and vice versa.

The information required for cost estimation was sourced from the Conveyor Equipment Manufacturers Association (CEMA, 2007). Conveyors are estimated to have a 20 year economic lifetime, the equivalent of 158 000 hours. A conveyor system has been assumed to have an availability of 90 per cent; however, this varies with the number and length of conveyors used.

The Australian coal cost guide (R2 Mining, 2011) quotes conveyor costs as a base cost for a turnkey installation with an adjustment for length, as a function of different throughputs. Using this information, a model was developed as seen in Equation 10, where C is capacity and L is length. The limits to this equation are that of the conveyor length limit (2200 m) and capacity limit (10 000 t/d).

\[
\text{CapCost} = 0.0048LC + 21.66C + 359L + 2664 000 \tag{10}
\]

Equation 10 was derived by using cost information provided by the Australia coal cost guide (R2 Mining, 2011). A linear regression was run to estimate intermediate belt costs, resulting in a model with a fixed cost component and an adjustable cost according to belt capacity. The coefficients of correlation were 93 per cent and 89 per cent respectively.

The limits to this formula cause some issues when estimating capital costs for conveyor set-ups. When the length of one conveyor is exceeded, a second needs to be installed in a series arrangement, and when the maximum desired capacity is greater than the available conveyors, two or more can be run in parallel.

When multiple parallel conveyors are required the capital cost for the arrangement is calculated by evenly distributing capacity over all the required conveyors. For example, if 25 000 t/d is desired then three conveyors are installed in parallel, each taking 8300 t/d capacity. Equation 11 describes how to calculate the total capital costs for a complete conveyor system installation.

Owing costs are calculated in exactly the same way as for discreet systems using the equations found in the owning costs section and the inputs found in Table 3.
TABLE 3
Accounting cost parameters for conveyor haulage.

<table>
<thead>
<tr>
<th>Parameter</th>
<th>Value</th>
<th>Unit</th>
</tr>
</thead>
<tbody>
<tr>
<td>Weighted average cost of capital</td>
<td>10</td>
<td>%</td>
</tr>
<tr>
<td>Salvage value</td>
<td>5</td>
<td>%</td>
</tr>
<tr>
<td>Insurance index</td>
<td>2</td>
<td>%</td>
</tr>
<tr>
<td>Economic lifetime</td>
<td>157 680</td>
<td>hours</td>
</tr>
<tr>
<td>Effective utilisation</td>
<td>90</td>
<td>%</td>
</tr>
</tbody>
</table>

\[
\overline{C} = \frac{C_{\text{required}}}{\text{roundup}\left(\frac{C_{\text{required}}}{C_{\text{max}}}ight)}
\]

\[
n_{\text{full}} = \text{rounddown}\left(\frac{L_{\text{required}}}{L_{\text{max}}}ight)
\]

\[
L_{\text{partial}} = L_{\text{required}} - n_{\text{full}} \cdot L_{\text{max}}
\]

\[
\text{CapCost}_{\text{total}} = \text{roundup}\left(\frac{C_{\text{required}}}{C_{\text{max}}}ight)
\]

\[
n_{\text{full}} = \text{rounddown}\left(\frac{\text{CapCost}_{\text{full}} + \text{CapCost}_{\text{partial}}}{\text{CapCost}_{\text{max}}}ight)
\]

The operating costs of conveyors are quoted by CEMA (2007) as five per cent of the conveyor capital cost each year. This can then be adjusted to a daily cost using the physical availability of 90 per cent and 365 days in a year. Equation 12 shows the daily owning cost relationship as a function of conveyor capital cost.

\[
\text{Operating Cost} = \frac{\text{CapCost}_{\text{total}} \times \text{Opex Ratio}}{\text{Days} \times \text{Availability}}
\]

OR

\[
\text{Operating Cost} = 0.05 \cdot \frac{\text{CapCost}_{\text{total}}}{365 \cdot 0.9}
\]

A conveyor, however, also requires a feed and discharge system. Conveyors can handle material sizes typically only up to 30 per cent of the belt width with the majority of material significantly smaller than this. This means that crushing must be undertaken in order to transport the material (Humphrey and Wagner, 2011). In addition, a discharge system is required to unload the conveyor to a stockpile or dump. Additional costs for continuous haulage systems are estimated using fixed values for:

- crushing cost of $0.25 per tonne
- spreading cost of $0.15 per tonne
- loading cost determined by that for a small shovel/truck fleet.

The crushing and spreading costs were sourced from a conveyor equipment manufacturer. The loading cost however can be taken from previous truck and shovel calculations and relies on the distance over which material will be hauled.

It has been assumed that the average truck and shovel haul fleet will operate over a 2 km distance from face to crusher.

On the basis of these assumptions, the most economic haulage options were calculated and are summarised in Table 4. $12 000 corresponds to a very low production rate of 1 t per day and is obviously not a practical application for a conveyor.

TABLE 4
Loading cost summary for continuous haulage methods.

<table>
<thead>
<tr>
<th>Haul method</th>
<th>Upper capacity (t/d)</th>
<th>Lower cost limit ($/t)</th>
<th>Upper cost limit ($/t)</th>
</tr>
</thead>
<tbody>
<tr>
<td>Small articulated dump truck</td>
<td>5000</td>
<td>$3.48</td>
<td>$12 000</td>
</tr>
<tr>
<td>Small off-highway truck</td>
<td>80 000</td>
<td>$1.72</td>
<td>$3.27</td>
</tr>
<tr>
<td>Medium off-highway truck</td>
<td>1 000 000</td>
<td>$1.40</td>
<td>$1.62</td>
</tr>
</tbody>
</table>

FLEET LIMITATIONS

The minimum number of load and haul units required to satisfy a daily production scenario was dependant on two separate constraints; a production and a tyre constraint. The production constraint is as expected, the minimum number of machines required to satisfy the capacity however this value alone ignores the practical limitations that tyres apply to each haulage method. Tyres have a tonne kilometre per hour (TKPH) limit that should not be breached in order to stop tyres overheating and affecting the safety of the operator and nearby mine workers. As such a TKPH limit was also applied.

The minimum number of trucks required to break even with the TKPH limit of a tyre is expressed in Equation 13 where distance is expressed in kilometres. This formula uses the TKPH rating as a limit to the maximum speed a machine can travel at when fully laden. The assumption behind this formula is that each of the tyres carry an even distribution of the machines weight when fully laden. The maximum of each of the TKPH limited scenarios and the production limited scenarios is the minimum number of trucks required to meet safe production, hence the number used for analysis.

\[
n_{\text{trucks}} = \frac{\text{Production per hour} \times \text{GVM Laden} \times \text{Dis tan ce Per Cycle}}{\text{Payload} \times \text{TKPH Rating} \times \text{Time Per Cycle}}
\]

It is unrealistic for a mine to operate with very large equipment fleets and as such, realistic limits were applied to the number of machines in each fleet. These limits are summarised in Table 5.

RESULTS

Daily cost data was combined with the production data in two steps; firstly the total required number of machines was determined, then the cost per tonne of owning these equipment was calculated.

Calculating the number of pieces of equipment required was done by taking the maximum of each of the TKPH limited equipment numbers and the production limited equipment numbers as per Equation 14.

\[
n = \text{Maximum}\left(\frac{\text{Prod}_{\text{required}}}{\text{Prod}_{\text{per machine}}}, \text{TKPH Limited}\right)
\]

OR

\[
n = \text{Maximum}\left(\text{roundup}\left(\frac{\text{Prod}_{\text{required}}}{\text{Prod}_{\text{per machine}}}, 0\right), \text{TKPH Limited}\right)
\]
The number of machines required to meet production given by Equation 14 is then multiplied by the daily owning cost and divided by daily production. In the case where paired systems are used, this process is repeated for the load and then the haul machines with the results summed to give a paired owning cost for each operational scenario.

In order to determine the minimum total cost for each operational scenario, a function was used in Microsoft Excel to retrieve the lowest cost per tonne for each combination of production and haul distance. This value was then matched to the corresponding equipment or equipment pair and reported on a separate spreadsheet giving the final economic haulage curve by both machine and cost. The final economic haulage curve is shown in Figure 2.

The graph places required daily production of an operation on the vertical axis and the horizontal haulage distance on the horizontal axis. Given a required production rate over an expected horizontal haulage distance, the graph can be used as a first port of call to assess the most feasible material transport option.

Conveyors form a large portion of the most economic haulage range for high capacity, long range production. They are significantly cheaper than the truck haulage method, being up to just ten per cent the total cost of a discrete truck and shovel operation. However, there are several features associated with conveyors that may negate their use, even if they are the cheapest option. These are: the large upfront capital costs associated with acquiring and installing conveyors; relocation difficulties; and suitability to mines having large ore reserves, high capital availability and a clear mining plan. Given that these requirements cannot always be met, the economic region for conveyors in Figure 2 overlaps some of the regions where off highway truck haulage options are the cheapest option.

From Figure 2 it can be seen that, over very short haulage distances (<200 m), dozer push operations offer the most economical alternative. As t-km limits are increased, the next most economic option is medium capacity front-end loaders, followed by small, medium and then large off-highway truck/shovel systems. Conveyor haulage is the cheapest option when large throughputs are required (>500 000 t/d), or as haulage distance becomes more significant (>20 km).

**CONCLUSION**

This work has resulted in a new graph for determining economic haulage options for Australian surface mines. It is designed to be used during the conceptual and prefeasibility stages of projects to assess the most cost-effective haulage options for mines having a range of capacities of between 100 t and 1 Mt per day, and over a range of haulage distances of between 50 m and 100 km. Haulage options include commonly used discrete mining systems as well as continuous conveyor systems. It is recommended that a more detailed cost analysis of haulage options be conducted at the feasibility stage of a mining project in order to validate the selected option.

**REFERENCES**


![FIG 2 – Updated economic haulage curve for surface mining operations (SM = small, MD = medium, LG = large, HW = highway).](image-url)


**Queens University**, 1981. Materials handling, course notes, Department of Mining Engineering, Canada.

INTRODUCTION

This paper details a research project to assess the suitability of recycled composite materials for use in drawpoint support systems in place of timber lagging. The project had the major aim of determining the engineering properties of a given recycled composite material, particularly those that pertain to drawpoint support. In addition to this the project aimed to use these found engineering properties and those known for timber to make an assessment as to whether the recycled composite material would be a suitable substitute for timber in drawpoint support systems.

An extensive literature review was conducted that examined:
- the support requirements at drawpoints
- current drawpoint support systems
- previous studies on the use of recycled materials; as mining supports
- the engineering properties of recycled; composites used as structural supports
- the engineering properties of timber used in the Australian mining industry.

Resulting from reviewing past research it was concluded that the most likely mode of failure when using a material as lagging in a drawpoint environment would be through tension in the outer fibres caused by bending stresses (Suorineni, 2014). In conjunction to this the materials compressive strength, failure modes, and susceptibility to damage due to the environmental conditions of drawpoints were considered important aspects to be investigated.

With the major aim being to make an assessment of the materials suitability in comparison to timber, the ASTM standard D6109 ‘Standard test method for flexural properties of unreinforced and reinforced plastic lumber and related products’ was deemed the most appropriate method to determine the materials flexural strength. This allowed for an unbiased assessment of the material against the properties of other materials. Similarly ASTM D6108 ‘Standard test method for compressive properties of plastic lumber and shape’ was largely followed to determine the compressive strength of the material.

These methods were deviated from where appropriate to perform investigations on the effect that varied loading rates

Assessing the Suitability of Recycled Material as a Timber Substitute in Drawpoint Design

K Williams¹, D Chalmers² and F T Suorineni³

ABSTRACT

The use of timber as a support in the mining industry is becoming less sustainable as hardwood timber prices increase and supply dwindles. As the industry moves to become more sustainable it must consider how it can innovate to achieve this. Recycling technology now allows for composites made from recycled materials to be used in structural applications cost effectively. This research project aimed to investigate the use of a recycled material composite as a timber substitute in drawpoint support systems. It was hoped that it would provide a way for the industry to become more sustainable and innovative, making better use of its underutilised waste streams.

In line with this, a series of tests were conducted to examine the mechanical properties of a recycled composite material made of polymer blend and woodchip filler. This aimed to predict how the material might behave when used in drawpoints through examining the flexural and compressive properties of the material. A methodology comprising a series of four point bending tests, compression tests and indirect tensile strength tests was developed from a series of standards and research into drawpoint support requirements.

The outcomes were positive and support further research into the use of recycled composite materials in drawpoint support and other applications in secondary mining support. The methodology allowed the material to be evaluated against timber, to which it was found to be comparable in the magnitude to which it could provide support. It also allowed observations of the failure mode; these differed from those of timber and further supported the use of this material in a drawpoint support application.

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had on the materials engineering properties and the effects of sequential loading with the aim of determining the materials likely behaviour when used as lagging in a drawpoint.

**DRAWPOINT SUPPORT**

Drawpoints vary in the level of ground support required based on geotechnical conditions and their intended operating life. Drawpoints in block cave mines generally are required to remain open for the life of stope, which in cases may be as long as the life of the mine and hence require extensive support to ensure the longevity of the excavation (Duffield, 1999). The surrounding rock requires considerable assistance to remain in place for the operational life of the opening hence there is an economic incentive to install the correct reinforcement during the development of the openings to avoid costly remedial work (Vandewalle, 1998). For permanent drawpoints the construction cycles typically require a steel set and lining system. Currently a number of lining systems are employed in the industry including steel fibre reinforced shotcrete. A typical block cave drawpoint support system is illustrated in Figure 1.

Ma, Stankus and Faulkner (2011) emphasise steel set design procedures for the use in roof falls ignore the adequacy of the wood lagging, and state that this is likely the weak link of an impact resistant steel set system citing the wide variation in engineering properties for wood products. The anisotropic nature of wood leading to an unreliability for steel sets that are lagged with timber cribbing is cited as the main reason for this.

Once a support has been installed and has full contact with the rock it will start to deform elastically in response to tunnel closure and the in situ stress of the surrounding rock mass. Figure 2 details the response of support systems for tunnel wall displacement as described by Vandewalle (1998). The maximum elastic displacement that the support can handle is \( U_m \) and the maximum support pressure \( P_{sm} \); these two variables are defined by the yield of the support system. Equilibrium is reached if the support reaction curve intersects the rock mass displacement curve before either has progressed too far

A number of other considerations must also be made when considering ground support for drawpoints. These include the underground environmental conditions such as moisture and wet muck, the flammability of products being used in the support. In addition to this the complex stress states of drawpoints must be considered. High levels of dynamic loading can be expected through drawpoints as can point loading and unloading, and blast induced stresses.

**RECYCLED POLYMER COMPOSITES**

Much research has been done in recent years towards developing recycled materials to make structural load bearing elements. A series of studies conducted at the University of New South Wales (UNSW) during the period 2002 to 2004 have compared and assessed the suitability of plastic lumber as an element in secondary support for means as a replacement for timber. Axion International Holdings (2013), an American manufacturer of recycled structural products has recently entered the marketplace with a mining support and shows that there is a growing market for the use of recycled materials in mining supports.

**Description of specimens tested**

The composites tested in this research paper were sourced from Integrated Recycling, a manufacturer located in Victoria, Australia. Their products generally contain a 65 per cent mixture of low-density polyethylene (LDPE) and high-density polyethylene (HDPE), 20 per cent Polystyrene and approximately 15 per cent woodchip filler derived from red gum sawdust (Curran, 2014). The materials tested ranged in their content of the various polymer types and filler content, however this was undisclosed at the time of writing.

The samples were manufactured using two methods, injection moulding and extrusion. In the case of injection moulding the curing process in the mould forms a gaseous mixture, there is no means for the gas to escape during the process leaving products manufactured with voids throughout the centre. The samples manufactured using this process and tested in this project exhibited continuous voiding the entire length of the specimen. It was noted that the voids, however, were not interconnected. The density of the samples was affected as a direct result of voiding and it was noted that an increase in the diameter or width and depth of specimens decreased in density as a result of voiding. The
extrusion manufacturing method forces the mixture through an opening in the desired shape, in this method any gas can freely escape through the ends of the product. None of the samples produced using this method exhibited any voiding.

A number of rod shaped specimens from four composite types containing varying undisclosed ratios of LDPE to HDPE were tested to ascertain the compressive strength of each composite type. In addition to this plank shaped specimens were tested to evaluate the material’s flexural strength. Figure 3 illustrates a cross-section of road and plank sample types.

TESTING METHODOLOGY

Compressive strength
The compressive strength tests were undertaken at UNSW using the MTS model 815 Rock Mechanics Test System. The system contains two feedback transducers, a differential pressure transducer used to measure the force and liner variable differential transducer that provides control and measurement of the actuator displacement (MTS Systems Corporation, 2014).

The MTS loading station set up for this experiment can be seen in Figure 4. The method used was as follows:

• The loading rate for the MTS was calculated as a function of strain to the equivalent of 0.03 mm/mm/min.
• The applicable loading rate was set on the MTS loading station and the specimen placed on the lower platen of the loading station, in the case that the sample was too short to allow for adequate closure a series of steel cylinders were used to decrease the closure distance. All samples were cut to adhere to a height to width ratio of two.

Four point bending
The MTS 815 loading station was also utilised for the four point bending tests with the addition of a loading frame and load cell. The set-up is illustrated in Figure 5. The experiment was carried out using the following method:

• The MTS loading station was adjusted by manual control to close the distance between the platens applying contact to both ends of the specimen.
• The loading station was then set to load at the required speed and the load displacement data recorded every half-second.
• specimens were marked at 32 mm from each end to allow for accurate alignment in the bending frame allowing for a ten per cent overhang on a support span of 256 mm
• the appropriate sized loading noses and supports for the specimen were attached to the bending frame and the frame centred on the platen of the MTS loading station
• specimens were placed in the loading frame with the markings aligned with the centre of the loading noses and supports
• a load was applied at a constant rate equivalent to a strain rate of 0.03 mm/mm/min
• the load displacement data was recorded automatically at half-second intervals
• the test was continued until a break had occurred in the specimen or the bending frame reached its closure limitations
• stress and strain curves were plotted using the calculations detailed in Equation 1 and Equation 2 following adjustments in force to account for the weight of the upper section of the bending frame, loading noses, and any cylindrical weights
• as deflection had occurred on the beams from the weight force of the frame the linear elastic portion of the graph was extrapolated backwards to the strain axis and then the data offset to more accurately represent where zero stain occurred.

The outer fibre stress and strain was calculated using Equation 1 and Equation 2:

\[ r = \frac{4.70 D d}{L^2} \]  
\[ S = \frac{P L}{b d^2} \]

where:
- \( D \) = midspan deflection, in (mm)
- \( r \) = strain, in/in (mm/mm)
- \( d \) = depth of the beam, in (mm)
- \( L \) = support span, in (mm)
- \( S \) = stress in outer fibre throughout load span (MPa)
- \( P \) = total load on beam at any given point on the load deflection curve (N)
- \( L \) = support span (mm)
- \( b \) = width of beam (mm)
- \( d \) = depth of beam (mm)

**RESULTS OF COMPRESSIVE STRENGTH TESTS**

A range of compressive strength values were obtained. Composite four was deemed unsuitable for use in drawpoint support applications as it exhibited failure at a low compressive strength and in a brittle manner, both characteristics not being conducive to use in this application. The remaining composites exhibited load deformation characteristics that were deemed favourable with composite one showing the most promising results deforming in a linear manner to a strain value of 40 per cent where the test was ceased. The compressive strength of the specimens tested to the loading rate prescribed by ASTM D6108 (ASTM International, 2013a) are shown in Table 1. The average values for all specimens of each composite are tabulated in Table 2. The effects of loading rates were examined and were shown to affect the results for compressive strength. Some observations were made on the effects of sequential loading and material the results of which were promising but require further investigation. Toe compensations were performed on the stress-strain curves to account for any slack in the machine or uneven seating of the specimen as per the tangent line in Figure 6.

**Load deformation and failure modes**

A range of properties were exhibited from the different composites as they underwent deformation. Composites one, two and three demonstrated an ability to hold their strength during a prolonged period of deformation up to 40 per cent strain. Composite one, two and three exhibited barrelling as can be noted in Figure 7. It can be noted that as a result of the barrelling the outer fibres of the specimen are beginning to fail in tension. This was common amongst the variety of composites tested.

Composite one was observed to be the most stable whilst undergoing deformation with plastic deformation remaining linear throughout the testing period; this is clearly demonstrated in Figure 8. There were three main failure modes exhibited by the different composites, these were:

- barrelling
- lateral buckling
- compression.

Figures 8 to 11 show the stress-strain curve for each composite. The stability of each composite can be noted from these curves in addition to the composites compressive strength and load deformation characteristics.

Each composite with the exception of composite four shows good material stability with the ability to hold load over a wide range of deformation. Small depressions can be noted in composites two and three; these depressions were observed to occur when the material was failing along seams in the outer surface. These seams were common in both composites produced by injection moulding and extrusion; however, those formed by the latter had many surface creases whereas those produced by injection moulding only had one seam. Typically the outer fibres failed in tension along these seams as a result of the specimen undergoing barrelling.

**TABLE 1**  
Compressive strength results at 0.03 mm/mm/min strain rate.

<table>
<thead>
<tr>
<th>Composite</th>
<th>Ultimate compressive strength (MPa)</th>
<th>Strain at ultimate strength</th>
<th>Stress at 3% strain</th>
<th>Number of samples tested</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>25.1</td>
<td>0.188</td>
<td>20.4</td>
<td>1</td>
</tr>
<tr>
<td>3</td>
<td>20.0</td>
<td>0.174</td>
<td>14.8</td>
<td>1</td>
</tr>
<tr>
<td>4</td>
<td>17.8</td>
<td>0.036</td>
<td>17.5</td>
<td>4</td>
</tr>
</tbody>
</table>

a. Composite 2 was not tested.

**TABLE 2**  
Average compressive strength results.

<table>
<thead>
<tr>
<th>Composite</th>
<th>Ultimate compressive strength (MPa)</th>
<th>Standard deviation (MPa)</th>
<th>Stress at 3% strain (MPa)</th>
<th>Standard deviation (MPa)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>25.2</td>
<td>0.1</td>
<td>20.0</td>
<td>0.3</td>
</tr>
<tr>
<td>2</td>
<td>19.3</td>
<td>2.3</td>
<td>14.9</td>
<td>2.0</td>
</tr>
<tr>
<td>3</td>
<td>19.4</td>
<td>2.7</td>
<td>13.1</td>
<td>1.9</td>
</tr>
<tr>
<td>4</td>
<td>17.8</td>
<td>0.6</td>
<td>17.5</td>
<td>0.4</td>
</tr>
</tbody>
</table>
Composite four differed highly from the other composites in its deformation characteristics, reaching its elastic limit at between three and four per cent strain before failing in compression between five and six per cent strain for all samples tested. This is inherent of the composite having a high stiffness, exhibiting very little ability to hold strength whilst yielding. The stress-strain curve for composite four is seen in Figure 11. Composite four exhibited a brittle failure through compression owing to the materials stiffness. A clear and defined failure can be seen in the specimen shown in Figure 12.

Sequential loading
Figure 13 details the stress strain curve for the testing of Specimen 3a. Specimen 3a was initially tested at a strain rate of 0.0059 mm/mm/min, the test was stopped at 8.2 per cent strain during the stage of liner plastic deformation. This specimen exhibited full recovery to the specimen’s original dimensions over a 12-hour period. The specimen was then re-tested at a rate of 0.012 mm/mm/min, this test examined the materials behaviour under sequential loading. The recycled composites exhibited varied levels of post testing recovery throughout the investigation.

The initial test appeared to have no adverse effect on the materials performance during the re-test inferring that the material will maintain its integrity after loading events, given that a certain strain has not been exceeded. This is a desirable characteristic considering the variety of loading events that occur at drawpoints that are both varied in magnitude and duration. No conclusive analysis on the materials performance during sequential loading can be made due to only one re-test having been conducted, nor can it be inferred at what level of strain the composite ceases to exhibit recovery characteristics.

Effects of loading rates
Composite two was tested at a variety of strain rates. The strain rate is calculated as millimetres of displacement per millimetre of the specimen’s original height per minute. A clear correlation between loading rate variations and stress at three per cent strain is illustrated from composites two in Figure 14. It can be concluded that the loading rate applied to testing will affect the results obtained for compressive strength and stress at a given strain. Similar results were attained for composite three.

These results confirm the need to use standardised testing when defining the materials engineering properties for the purpose of unbiased comparison. This test infers that the composites exhibit a higher modulus of elasticity under higher loading rates as is evident when considering the stress strain curves for composite two.
FIG 8 – Stress-strain curve of composite one showing a largely linear plastic deformation.

FIG 9 – Stress-strain curve of composite two showing a linear elastic region and non-linear plastic region.

FIG 10 – Stress-strain curve of composite three showing a linear elastic region and both linear and non-linear plastic regions.
RESULTS OF FLEXURAL STRENGTH TESTS

Flexural strength

The material performed well under four point bending tests achieving an average flexural strength of 26.32 MPa with a standard deviation of 3.55 MPa. The voiding in some samples appeared to have an effect on stiffness. Failure mode was observed to be a clean tensile break propagating from the outer fibres of the material. The results for series one through three are tabulated in Table 3 and conform largely to the

<table>
<thead>
<tr>
<th>Specimen series</th>
<th>Average flexural strength (MPa)</th>
<th>Standard deviation (MPa)</th>
<th>Tests completed</th>
<th>Support span to depth ratio (mm:mm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>22.51</td>
<td>2.22</td>
<td>3</td>
<td>14.28:1</td>
</tr>
<tr>
<td>2</td>
<td>27.29</td>
<td>2.21</td>
<td>2</td>
<td>14.88:1</td>
</tr>
<tr>
<td>3</td>
<td>29.15</td>
<td>0.41</td>
<td>2</td>
<td>20.82:1</td>
</tr>
<tr>
<td>Average</td>
<td>26.32</td>
<td>3.55</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

Table 3

Flexural strength results for series one, two and three.

Failure mode

Rupture occurred during the testing as the result of tearing caused through tension failure in the outer fibres. In most cases the tear propagated vertically through the specimen with very little to no lateral propagation. The common mode of failure observed is illustrated in Figure 15. The absence of a grain structure is advantageous with failure not propagating laterally as does occur with timber. In the case of failure due to bending stresses in a drawpoint this behaviour infers somewhat that the remainder of the member will still provide

FIG 11 – Stress-strain curve of composite four with failure occurring between four and six per cent strain.

FIG 12 – Composite for post compression testing showing a clear compression failure.

FIG 13 – Stress-strain curve of sample 3a initial test and post material recovery test.
support and remain unaffected by a failure caused by a nearby point loading of the material.

This mode of failure is more favourable than that of timber. Timber beam failure is highly variable, the most common modes are displayed in Figure 16.

**Load transfer characteristics**

The primary purpose of this material in drawpoint design is to transfer the load occurring from dynamic loading events evenly to the steel set. Due to the profile of tunnel perimeters it is expected that bending of any lagging material will occur due to some level of point loading as convergence of the roof occurs.

Figure 17 can be alluded to as a model for this situation. The spike in stress seen in is a result of the bottom of the beam

---

**TABLE 4**

Flexural strength results for all series.

<table>
<thead>
<tr>
<th>Specimen series</th>
<th>Average flexural strength (MPa)</th>
<th>Standard deviation (MPa)</th>
<th>Number of specimens tested</th>
<th>Support span to depth ratio (mm:mm)</th>
<th>Standard deviation (mm)</th>
</tr>
</thead>
<tbody>
<tr>
<td>1</td>
<td>22.51</td>
<td>2.22</td>
<td>3</td>
<td>14.28:1</td>
<td>0.08</td>
</tr>
<tr>
<td>2</td>
<td>27.29</td>
<td>2.21</td>
<td>2</td>
<td>14.88:1</td>
<td>0.10</td>
</tr>
<tr>
<td>3</td>
<td>28.98</td>
<td>0.41</td>
<td>3</td>
<td>20.82:1</td>
<td>0.41</td>
</tr>
<tr>
<td>4</td>
<td>16.83</td>
<td>2.32</td>
<td>3</td>
<td>6.93:1</td>
<td>0.04</td>
</tr>
<tr>
<td>5</td>
<td>22.52</td>
<td>1.57</td>
<td>3</td>
<td>5.93:1</td>
<td>0.03</td>
</tr>
<tr>
<td>6</td>
<td>24.66</td>
<td>1.70</td>
<td>3</td>
<td>10.70:1</td>
<td>0.05</td>
</tr>
<tr>
<td>Average</td>
<td>23.80</td>
<td>4.32</td>
<td>17</td>
<td></td>
<td></td>
</tr>
</tbody>
</table>

---

**FIG 14** – Stress at three per cent strain for composite two at varied loading rates.

**FIG 15** – Failure mode of plank sample side on (top); outer fibres of beam showing tension cracking (bottom).

**FIG 16** – Common failure modes of timber beams (Record, 2004).
conforming to the surface of the bending frame, in addition with the two upper extremities of the beam conforming to the upper assemblage of the frame. The stresses caused by the constant rate of closure in the MTS are being transferred to the steel of the frame in an effective manner providing an example of how this composite may behave in a drawpoint. It is however important to make note that this example involves a uniform loading rate across the length of the specimen, an unlikely case in a drawpoint environment.

COMPARISON WITH TIMBER

The characteristic values of timber discussed apply to rectangular beam sections with depths up to 300 mm as defined by the Australian Standard AS1720.1 (Standards Australia, 2010). It can be noted that the recycled composite is comparable with timber in a range between an F8 and F11 stress graded timber that ranges from 22 to 31 MPa with an average flexural strength of 26.32 MPa.

Evaluating the compressive strength of the composites it can be noted that the best performing composite achieved results exceeding that of F11 stress graded timber (22 MPa). Composites two and three exceeded that of an F8 stress graded timber (18 MPa) and the poorest performing composite, composite four, exceeded that of an F7 graded timber (18 MPa).

Of timbers used in the mining industry Blue Gum was found by Galvin and Offner (2000) to be the most common species used in Australia. Blue Gum is stress graded as F8, F11, F14 and F17 for unseasoned and F11, F14, F17 and F22 for seasoned by the Department of Agriculture, Fisheries and Forestry (2013). It has been cited that timber stress graded as F8, F11, and F14 are used as mine timbers (Auswest Timbers, 2012) and as such it can be concluded that the compressive and flexural strength of the recycled composites tested is similar to that of timber used in the Australian mining industry.

In addition the material’s mechanical properties remained unaffected after being submerged in water for a 24-hour period, in contrast to the strength of timber, which is known to decrease as a function of the timbers saturation (Record, 2004). The material was found to have a water absorption of less than one per cent, as no differences in strength were shown the results form the wet specimens form part of the averages shown in Table 4.

CONCLUSIONS

It can be concluded from the investigation that the engineering properties of the recycled polymer composites tested suggest that recycled composites would be a suitable substitute for timber in drawpoint design.

The flexural and compressive strength of the recycled composite material is similar to that of timbers being used in the Australian mining industry, supporting the conclusion that it is a suitable substitute. In addition to the magnitude of the result achieved and the way in which material tested performed under compressive force, sustaining its ability to hold load under high levels of strain again supports its suitability in this application. The apparent ability of the material to recover from a certain level of strain and exhibit the ability to provide support to original capacity under sequential loading are desirable characteristics in drawpoint support where dynamic loading is expected as are periods of point loading and unloading.

It can be concluded that not all recycled composites will have appropriate properties for this application with some showing brittle failure characteristics under small amounts of strain. This suggests that there also exists the ability to refine the composition of recycled composites in order to produce a product that is specifically suited to this application.

In addition, the structure of the recycled composite material is advantageous to the way it fails under bending stresses. The consistency in quality and performance for each specimen type suggested that these materials have predictable mechanical performance and little variation can be expected. This supports the use of these materials over timber, as it is known a major disadvantage of timber is its variance in quality and performance owning to its organic nature, orthotropic properties and susceptibility to degradation.

ACKNOWLEDGEMENTS

The authors would like to thank PDB Support Solutions Pty Ltd for assistance in obtaining the necessary samples to undertake the project. The authors gratefully acknowledge the assistance of Kanchana Gamage during the laboratory testing.
REFERENCES

Curran, C, 2014. Personal communication (Manager), Integrated Recycling, 30 August.

INTRODUCTION

As the search for more new mineral deposits continues both in Australia and other countries, resource and energy industry companies are having to utilise and develop strategies to be competitive and be approved for new projects or expansions in areas with agricultural or biodiversity significance. One way companies are addressing their impact on biodiversity is through the use of biodiversity offsets (Gardner et al., 2013). An example of this is Xstrata Coal’s Biodiversity and Land Management Factsheet which states how the company aims to ‘avoid net losses or degradation of natural habitats, biodiversity and landscape functions’ (Xstrata Coal, 2011). Other reasons for mining companies to invest in these innovations include conforming to state or national regulations as well as maintaining their ‘social licence to operate’ and minimising reputational risks (Virah-Sawmy, Ebeling and Taplin, 2014, Gardner et al., 2013).

Biobanking is a scheme that was introduced in New South Wales in 2008 to address these issues by incorporating agricultural landholder involvement in the biodiversity offsetting process (Office of Environment and Heritage, 2014b). However, this scheme is yet to prove a success with a number of criticisms directed towards contract terms, and regulation and compliance of offset areas (Office of Environment and Heritage, 2014a). Recent adjustments have been made by the New South Wales (NSW) Government to address these (Office of Environment and Heritage, 2014a).

The aim of this research was to investigate how mining biodiversity offsets can be better incorporated with the agricultural industry by assessing potential negotiations from members who feel that these offsets are not achieving the task they were designed to do. This is backed up by academics such as Gardner et al. (2013) and Eglinton (2011) who claim that there are still a number of significant challenges that need to be addressed and that it is time for another ‘step-change’ in environmental stewardship particularly, biodiversity offsets.

Biobanking is a scheme that was introduced in New South Wales in 2008 to address these issues by incorporating agricultural landholder involvement in the biodiversity offsetting process (Office of Environment and Heritage, 2014b). However, this scheme is yet to prove a success with a number of criticisms directed towards contract terms, and regulation and compliance of offset areas (Office of Environment and Heritage, 2014a). Recent adjustments have been made by the New South Wales (NSW) Government to address these (Office of Environment and Heritage, 2014a).

The aim of this research was to investigate how mining biodiversity offsets can be better incorporated with the agricultural industry by assessing potential negotiations from...
the landholder’s perspective. To achieve this, two objectives were defined as follows:

1. Investigate landholders preferred biodiversity offsetting structure by generating qualitative data from interviews with landholders in the research focus area.
2. Compile and analyse data to develop a biodiversity offsetting structure that would satisfy landholders and mining company requirements.

BACKGROUND

Biodiversity offsets

Biodiversity offsetting in mining is performed for a number of reasons. It can be used as a tool for mining companies to demonstrate good environmental stewardship or social and environmental responsibility (International Council on Mining and Metals (ICMM) and International Union for Conservation of Nature (IUCN), 2013). This is especially important in the extractive industries sector where a demonstration of good biodiversity performance and risk management can lead to gaining access to land and resources or a ‘licence to operate’ (Virah-Sawmy, Ebeling and Taplin, 2014; ICMM and IUCN, 2013). With the growing environmental awareness of consumers, it is also important to demonstrate good environmental stewardship to maintain or improve company reputation with the public. In some cases this can even help with government approvals because of a positive public interest in a project (Gardner et al, 2013). Examples of other business drivers for biodiversity offsetting practices are government regulation and operations financing (ICMM and IUCN, 2013). Today it is not uncommon for government policies to refer to biodiversity offsets as a required tool for any development project that impacts biodiversity and that project approval cannot be granted without a comprehensive biodiversity offset plan. Financial institutions such as the International Finance Corporation (IFC) and Asian Development Bank (ADB) are essential to mining developers to provide capital investment for the commencement of new project. However, many of these financial institutions are now including offset requirements that prospective customers must comply with (ICMM and IUCN, 2013).

Challenges of biodiversity offsetting

Gardner et al (2013) is a key paper in the field that discusses the vast challenges of effective biodiversity offsetting. One key concern that is highlighted in this paper is the lack of standard methods and definitions required for biodiversity accounting. This is echoed by Virah-Sawmy, Ebeling and Taplin (2014), who argue that implementing an effective strategy for biodiversity is difficult for mining companies and often varies between site and organisation. Gardner et al (2013) discussed the deficiency of evidence showing actual effectiveness of biodiversity offsetting programs and argued that these policies have a risk of serving a symbolic purpose whereby environmental concerns are neutralised and limited environmental gain is achieved. The industry ‘hype’ around the words ‘biodiversity’ and ‘conservation’ as well as the concern that the preliminary mitigation measures of avoidance and minimisation of biodiversity impacts are forgotten or overlooked has also been criticised publically (ABC National Radio, 2014).

Focus area – Mudgee Upper Hunter Valley, New South Wales

This project focused on mining operations in the Upper Hunter Valley area of NSW, concentrating particularly on the direct and indirect impacts from the mining operations on agricultural land as well as environmentally significant areas.

Apart from mining, another major industry in the Hunter Valley region is agriculture. Agriculture often requires significant land use in specific environments and can therefore be a land-use competitor with mining (Planning Institute Australia, 2011). Agriculture in the region can be divided into subcategories, with equine/thoroughbred breeding and viticulture (wine production) being the major components, but there is still a significant presence of dairy farming (James-Elliott, 2012). These agricultural industries thrive in the area which is home to some of Australia’s most fertile land and produces some high quality products that are competitive nationally and globally (Planning Institute Australia, 2011).

Current trends in offsetting

Current practices in biodiversity offsetting are quite diverse with not only each company, having different frameworks for offsetting, but each individual mine site being unique as well. This is demonstrated by irregularities between documents such as Xstrata’s Ulan Coal Mines Biodiversity Management Plan (2011) and Mt Owen Mine Biodiversity Offset Strategy (2005), Anglo American’s Draton Management System Standard Offset Strategy (2012) and Rio Tinto’s Mount Pleasant Biodiversity Management Plan (2013). However, as previously mentioned, Biobanking is one scheme that is regulated by the NSW Government and is therefore relatively streamlined across all mining projects in NSW.

Biobanking

The NSW Biobanking Scheme Overview (Department of Environment and Climate Change NSW, 2007) is a government document that details an environmental initiative for addressing biodiversity loss to mining and other companies who may be interested or obliged to introduce biodiversity offsetting strategies. It introduces the concept of Biobanking before explaining how exactly it works and which stakeholders will be involved. It utilises biodiversity credits that developing companies want to buy to offset the biodiversity impact of their development. Landholders can sell credits for cash if they meet the requirements; in theory, the offset site needs to have a biodiversity value that correlates with the affected site, and needs to be managed in terms of the conditions of contract. The Biobanking scheme also details how the number and price of credits are calculated for any particular project and concludes with the successful and effective outcomes that should result from the scheme. However, it does not provide evidence of potential effectiveness in terms of biodiversity offsets which is one of the significant flaws that Gardner et al (2013) highlighted. As a result it is impossible to tell if the scheme is at all practical or enticing for landholders (NSW Farmers’ Association, 2011). The Biobanking review – summary of submissions (Office of Environment and Heritage, 2014) is a document published by the Office of Environment and Heritage that summarises 50 written submissions in response to the effectiveness of the Biobanking Scheme introduced in 2008. It included responses from all groups of stakeholders including individuals, private companies, government bodies and environmental groups. Its findings provide a fundamental discussion on the different views and reactions of each of the stakeholder groups involved and include ways in which the scheme can be improved to increase success. At June 2012, the scheme had been in place for four years however, there were only nine landowners who held approved Biobanking agreements and four developers held approved Biobanking statements (Office of Environment and Heritage, 2014).
This strongly suggests that the scheme is unattractive and ineffective and therefore further research and refinement is required. Other documents such as the New South Wales Aboriginal Land Council’s Biobanking and Development – A Guide for Local Aboriginal Land Councils (2011) and the Law Society of New South Wales’ Review of: ‘Biobanking Review: Discussion Paper’ (2012), do provide additional specific insights into otherwise overlooked challenges such as the impact of Biobanking on Local Aboriginal Land Councils.

METHODOLOGY

Research approach
To gather an accurate understanding of landholders’ awareness, perceptions and preferences concerning biodiversity offsetting of mining projects, a semi-structured interview schedule was developed to facilitate an in-depth, but open discussion with agricultural landholder participants. Despite having some specific questions, the interview schedule was designed to stimulate discussion for production of qualitative data to provide details in preference to a quantitative survey.

The interview schedule was divided into two main sections; the first section gathered details of the landholder, their enterprise and their management from both a production and a conservation perspective. This was followed by a ‘choice experiment’ which highlighted six main areas of discussion for a new biodiversity framework. These areas are: conservation requirements, biodiversity credit valuation and payment, trading arrangement, contract length, flexibility of conditions and monitoring and performance. These areas were selected based on the literature of research into similar schemes including Biobanking by the NSW Government as mentioned previously.

The interviewees were all from the Mudgee region and surrounding areas on the western border of the Upper Hunter Valley as shown in Figure 1. They were contacted by both telephone and email initially to organise a time and a place where interviews would be conducted. Initial contact was made with landholders via those with details found online and a snow-balling process identified other landholders in the area. A total of 15 landholders were interviewed, comprising of a wide variety of enterprises. Figure 2 shows the locations and varying agricultural commodities of the interviewees.

RESULTS

General questions
After completing all interviews in the Mudgee area is was clear that most landholders in the area were aware of biodiversity offsetting practices by mining companies. However, there were mixed opinions about the effectiveness of these offsets as well as some major criticisms of how they are currently being managed.

The main criticism was that the biodiversity offset areas were being bought outright by the mining company and then not properly maintained. This biodiversity offset mismanagement would in some cases impact on the landholder by incurring additional time and financial costs associated with noxious weeds and pests spreading across the property boundary onto the interviewee’s production land.

When asked the question, ‘do you think environmental damage through mining can be ‘offset’ via farming practice change?’ a range of conflicting views were provided by the landholders. One landholder stated:

“It’s better than doing nothing. I don’t really agree with the amount of mining going on but I also don’t think we can stop it because the world has these needs. I think this could actually be a better thing than planting a bunch of trees around a mine because if no one looks after those trees then it gets taken over by exotic species.” [Landholder 13]

The trends of responses are proportionally illustrated in the pie chart in Figure 3.

Despite all landholders having a view on biodiversity offsets, very few had actually heard of the NSW Government Biobanking Scheme when asked the question; ‘are you aware of Biobanking?’ and only one participant out of 15 could describe any details of the scheme accurately as shown in Figure 4.

These results identify the need to rethink biodiversity offsetting in agricultural areas. While six participants agreed that environmental offsets were viable, the five participants who didn’t agree put forward the proposition that the damaged land would never ‘go back’ to its previous state.

Others explained that just fencing off and managing areas that have never been utilised for production would not offset eradication of biodiversity in another landscape.

With only one participant being aware of the Biobanking scheme, these results illustrate the lack of marketing and uptake of the scheme. These interviews confirmed the ineffectiveness and unattractiveness of this scheme.

**Choice experiment**

To determine the factors which make biodiversity offset schemes unattractive a choice experiment was enacted. This experiment identified six key aspects of contracts and explored the options which were most attractive to landholders.

**Conservation requirements**

The extent to which farming practices are present on the biodiversity offset area varied as shown in Table 1. Some suggestions included minimal cattle grazing on the offset land:

“...it could be a combination of both setting some land aside and allowing managed grazing on it at the same time.” [Landholder 8]

All 14 agricultural landholders agreed that there had to be some form of management of the land. Others suggested that all production land should be counted in the offset because improved farming techniques such as rotational grazing and regular soil testing can result in improved biodiversity, for example, increased ground cover.

**Biodiversity credit valuation and payments**

As shown in Table 2, nine landholders explained that conditions of a contract would have to be based on the results
gained from an implemented scheme to ensure biodiversity was actually benefitting and to keep farmers accountable:

Some form of calculating the actual amount of improvement. You shouldn’t be rewarded for just having it.

[Landholder 5]

Trading agreement

As shown in Table 3, multiple trade options had five responses and involved having different options available for farmers to choose from. For example, some farmers would like to use a broker for the convenience, protection and advice while others would rather not and make a higher profit. Five respondents suggested the mining company should contact the landholder with the offer since the offset credits are ultimately in their interest.

<table>
<thead>
<tr>
<th>Option</th>
<th>Number of responses</th>
</tr>
</thead>
<tbody>
<tr>
<td>Multiple trading options</td>
<td>5</td>
</tr>
<tr>
<td>Mining company application to farmer</td>
<td>5</td>
</tr>
<tr>
<td>Landholder application to miner</td>
<td>2</td>
</tr>
<tr>
<td>Common landholder group</td>
<td>1</td>
</tr>
<tr>
<td>University managed scheme</td>
<td>1</td>
</tr>
<tr>
<td>Self-listed auction</td>
<td>1</td>
</tr>
</tbody>
</table>

Table 3: Landholders’ preferred trading agreement.

Contract length

This question asked landholders to comment on how the length of a contract should be determined. Roughly half believed that it should be related to the life of the mine, with most of these suggesting that it would be until the on-site mine rehabilitation, after mine closure, was deemed satisfactory. They suggest that the contract should only be relevant until the offset was no longer required. Other interviewees suggested a time frame in years which could be renegotiated at the end of the period. A summary of these responses is provided in Table 4.

<table>
<thead>
<tr>
<th>Option</th>
<th>Number of responses</th>
</tr>
</thead>
<tbody>
<tr>
<td>Related to mine life</td>
<td>7</td>
</tr>
<tr>
<td>Year value provided</td>
<td>5</td>
</tr>
<tr>
<td>In years, but no value provided</td>
<td>2</td>
</tr>
<tr>
<td>Depends on situation</td>
<td>1</td>
</tr>
</tbody>
</table>

Table 4: Landholders’ preferred contract length.

Flexibility of conditions

The majority of landholders interviewed agreed that the scheme would have to incorporate a large degree of flexibility to both encourage landholder participation and satisfy the different requirements of both mining operations and farmers as shown in Table 5.

<table>
<thead>
<tr>
<th>Option</th>
<th>Number of responses</th>
</tr>
</thead>
<tbody>
<tr>
<td>High flexibility</td>
<td>7</td>
</tr>
<tr>
<td>Goals set with flexibility in how they are achieved</td>
<td>4</td>
</tr>
<tr>
<td>Goals and targets set</td>
<td>2</td>
</tr>
<tr>
<td>Legal contract</td>
<td>1</td>
</tr>
</tbody>
</table>

Table 5: Landholders’ preferred flexibility option.

One interviewee suggested that a mine employee should perform the offset inspections but also acknowledged that this process should be audited.

DISCUSSION

After completing all interviews in the Mudgee area it was clear that most landholders were aware of biodiversity offsetting practices by mining companies. However, there were mixed opinions about the effectiveness of these offsets as well as some major criticisms of how they are currently being managed.

The main criticism was that the biodiversity offset areas were being bought outright by the mining company and then not properly maintained. The perceptions were that biodiversity offset mismanagement would in some cases impact on farming landholders by incurring additional time and financial costs associated with noxious weeds and pests spreading across the offset property boundary onto interviewees’ production land. The results of the choice experiment and the relevance of the research to other locations are discussed below.

Choice experiment

Conservation requirements

The first question: ‘what would be your preferred conservation requirement for a biodiversity offsetting scheme?’ for the choice experiment involved a discussion of conservation requirements for a biodiversity offsetting framework.

As presented in the results, both managing offset land with grazing and improving farming practices was the most preferred option. It is interesting to note that even landholders who indicated that they have an area of land in mind that they would be prepared to offer as a biodiversity offset still preferred to have some kind of managed grazing on that land, rather than ‘locking it out’ completely.

There were a number of reasons presented to support this statement. One of the strongest opinions presented was from Landholder 14 who had been the neighbour of a mining biodiversity offset area for over five years. The landholder explained how previous agriculturally productive land had been purchased by a mining company and left unmanaged which has resulted in an uprising of noxious weeds and pests such as wild dogs. Although the intent of the biodiversity offset was sound, to restore the land to its native state, this landholder stated that this result has not been achieved.
This was echoed by Landholder 5, who also highlighted that financial costs could be a severe barrier:

- Setting the land aside and managing pests and weeds could be a big problem here because I can see it getting frighteningly expensive. [Landholder 5]

With offsets for mining operations consisting of substantial areas and often rugged terrain, it would not be unreasonable for the financial costs of maintaining exotic weeds and pests to be a significant portion of a mining company’s offsetting budget. Landholders also expressed concerns that shortcuts are taken by mining companies in this area which result in poorly managed offsets due to poorly trained or under-equipped staff. Most landholders interviewed agreed that this is a factor that gives a clear advantage to a scheme where the farmer is paid to look after the land because the farmer will in most cases, already have the experience and equipment required to address such land management issues.

Despite farmers having an advantage over mining staff in farmland management, there were also other criticisms offered by some of the interviewees with regard to farmers maintaining biodiversity conservation areas. These included the lack of time or manual labour many farmers battle with day-to-day. They would much rather spend their time on improving their production land or on tasks that improved the profitability of their business. If farmers were to spend time in looking after biodiversity offset areas, these issues would have to be addressed. Another criticism was that many farmers refrain from investing in updating their education and training and that this may be needed in relation to offsets and conservation management:

- Education is one important thing. One thing I don’t understand is that farmers don’t go and get more educated. Other careers require ongoing training and refreshers. I think the farmers should be educated in sustainable farming practices and have a farm plan. [Landholder 14]

**Credit valuation and condition of payment**

When asked how the biodiversity credits should be valued the most common response given by the landholders interviewed was that the value of the credit should be based on the results, or measured biodiversity improvement on the land. Landholder 2 suggested a possible method for how this measurement could be made:

- I think that the value of the land should be valued first up and then measured later on and the difference, improvement observed, should be then rewarded. [Landholder 2]

This multistage procedure of measuring biodiversity traits before and after a scheme has been implemented, was also preferred by Landholder 5 who disliked the idea of farmers being paid a set amount just to have a given amount of land under contract:

- Some form of calculating the actual amount of improvement. You shouldn’t be rewarded for just having it. [Landholder 5]

The main criticism of a scheme where the landholder receives payments just for having a section of land under contract was that it relieves the farmer of any accountability or incentive to innovate and continually improve the biodiversity of that land. Some interviewees were critical of other farmers highlighting that some would solely be motivated by financial rewards without thinking further about the conservation of biodiversity.

As previously mentioned, there is a general perception that mining companies are wealthy and that they are willing to pay large sums of money to satisfy their offsetting requirements. This is an aspect that some landholders may be willing to exploit.

Another aspect, payment valuation, that was highlighted by multiple landholders was that farmers who are already involved and excelling at land and biodiversity conservation should not be disadvantaged by a possible scheme.

Landholder 3 raised a very important point highlighting a potential loophole or counterproductive strategy that farmers could possibly utilise if the credits were valued on the results measured from an offset. It relies on the principle any given piece of land ultimately has a maximum biodiversity limit and that biodiversity improvement can be much more easily achieved and observed if the land was originally in a very poor state. This contrasts with the harder to achieve and more difficult to observe situation presented with already high biodiversity quality land. To address this issue, it is suggested payments should be scaled according to quality of biodiversity achieved and not just the amount of biodiversity improvement gained.

**Trading arrangement**

Trading arrangement involved a discussion with the landholder about how biodiversity credits should be traded between the landholder and the miner. This was especially interesting because there were a number of different views and responses received from the interview participants.

One of the most common responses was that there should be multiple different trading options that both landholders and mining companies could choose from depending on their specific preference or situation. One of these involved the choice for a landholder to use a broker or not when trading credits. This broker could be in the form or an officiating government department, such as the current Biobanking Scheme or an independent organisation.

Another aspect of having different options was a suggested compulsory divide between a scheme that private farmers would qualify for, and a scheme for farming corporations:

- The other thing is that some of these corporation farms will outbid everyone else just to gain this scheme to offset their costs. So maybe there needs to be a different arrangement for corporations to the one for private enterprises. [Landholder 10]

Another common theme amongst the landholders interviewed was about whether a mine should approach a landholder with a proposal or opportunity. Some of the major reasons for this issue being raised included that it would be in the mine’s main interest to achieve the offsets and that the farmer would not go looking for offset opportunities.

Farmers’ knowledge, or lack thereof, of these opportunities was an area that was discussed with multiple landholders. Many believed that there was major disadvantage to many farmers who are not kept up to date as well as others. This may be particularly the case where credits are traded in an open auction. Despite this, one landholder listed an auction design would be their preferred option.

There were a few other unique options presented by the landholders. One of which involved a common landholder group or body who would oversee all the trading between the miner and landholders. The advantage of this option would be that farmers would not be easily discriminated against and all would have the same access to information. Another option presented was a structure that was managed by a university to eliminate any prejudices that a farmer may have with a government body or mining corporation.
deter many landholders. Without the reassurance of the agreement being made, would the application was submitted. Some believed that hiring costs associated with assessing the land in question before the process by the landholder to the mine would be an acceptable variable depending on the situation and could be better estimated by an ecologist. Flexibility of conditions

The majority of landholders interviewed believed that there had to be some degree of flexibility in the conditions that made up a biodiversity offsetting agreement. There were a number of specific reasons given to support each landholder’s preference.

The most common theme amongst landholders was that there needed to be flexibility in how the land under the agreement was utilised and managed. It was unanimous amongst the interviewees that one set framework, or set of rules would not be adequate to efficiently achieve positive conservation outcomes over a range of different land types. The majority of farmers also believed that managed grazing on offset areas was an essential part of a successful scheme:

[It should] also be flexible to allow grazing on the land. I know here, that to completely remove grazing from one part of the land would be detrimental. It shows time and time again that regeneration goes up with some type of disturbance. Burning and grazing are the two natural occurrences that would have been happening anyway in this area. [Landholder 2]

Even landholders who firmly believed in rigid goals and targets felt that there needed to be some degree of flexibility in how those goals should be achieved. Some interviewees highlighted that the mining company would need some certainty that the offsets they agreed to are actually being achieved and the best way to ensure this was to incorporate a detailed prescriptive method with the landholders:

Goals should be set from the beginning with rules so the developer knows it’s being achieved. Try and keep it simple and there is less to go wrong. [Landholder 6]

Monitoring and assessing performance

As highlighted in the results section, all landholders acknowledged that independent assessments and audits should be an essential part of a biodiversity offsetting scheme. When asked if a report by the landholder would be a valid option, half of the interviewees believed it would be, but this was met with a number of concerns:

With an annual report done by the farmer, I have issues with the assurance and competency of the farmer to do that. Although most farmers are highly educated with a wide skills set, some really do struggle with basic literacy and numeracy. They might love to participate, but if you tell them to do a report then they might not be interested. [Landholder 3]

One landholder also presented an idea that involved consultation between the farmer and a mine employee:

I think the mines should employ someone just to travel around and spend the day. The main reason is for the paperwork. Some people struggle to take care of that. And there is only so long you can sit at a computer for. This person could spend half the day on the farm and then half a day writing up the report saying what the farming is doing. I guess that there should also be audits on the process to ensure that what is being done is actually being done. Pretty standard really, a lot of industries require audits and checks. This mining person should be the farmer liaison officer. I know that in some cases this is someone local from the community. The only problems here arise when there is some kind of conflict and it can divide a community. Just need to be aware of that. [Landholder 14]

Relevance of research to other locations

As the results and findings of this research were gathered in the Upper Hunter Region of NSW around the town of Mudgee, the results included in this paper only reflect this area and it cannot be assumed that other agricultural regions would give the same results. One factor that may affect this is that the type of agricultural land that surrounds Mudgee in most cases will differ significantly from agricultural land elsewhere in NSW and Australia. With the strong mining history and significance in the Mudgee region, most farmer interviewees had either worked in the mines or had close family members who had done so. This could present another possible difference between this research location and other possible biodiversity offsetting areas.

Despite these differences, the findings with regard to farmers’ perceptions about a biodiversity offsetting scheme provided by this research and the recommendations below, should provide a useful starting point for the initiation of a biodiversity offsetting scheme research in other parts of NSW and even Australia.
CONCLUSIONS AND RECOMMENDATIONS

This research outlines the current state of biodiversity offsetting practices by mining operations in the Upper Hunter Valley of NSW and subsequently outlines why there is a current need for a step change in the way biodiversity offsets are treated and achieved.

The findings from this research have shown that there is very real possibility to achieve this step change by incorporating biodiversity offsets in agricultural landscapes. After in-depth discussions with 15 landholders in the Mudgee region, it is clear that most landholders are interested in the idea of being involved in a biodiversity conservation scheme to offset the impacts that a mining operation imposes on the immediate environment.

Due to a range of different options which suit different landholders being collected as part of this research, it is suggested that each offsetting contract be negotiated and tailored to the needs of the individual landholders to ensure uptake.

Also, based on the research, a useful starting point for a biodiversity offsetting scheme for the Upper Hunter, NSW, would have the following characteristics:

- Conservation requirements would involve managing offset land through agricultural methods (grazing) and improving biodiversity traits on production land through improved farming techniques.
- Biodiversity offset credits would be valued using rewards system. This involves a combination of being fixed at a minimum level and being variable, depending on performance.
- Trading between the mining company and the landholder would have to involve multiple options.
- Contract length would be equal to the life of the mine plus a period of time determined by sufficient rehabilitation of the mine.
- The contract would have set goals and targets, but would also be flexible to allow for unforeseen conditions (such as weather).
- Monitoring of the biodiversity offset site would be through a combination of individual reporting and independent inspections.

ACKNOWLEDGEMENTS

The authors would like to sincerely thank the 15 landholders and their families in Mudgee who offered their valuable time and hospitality during the data collection stage of this project. Their personal reflections on the subject and detailed ideas of how biodiversity offsetting can be improved have formed the backbone of this study and are greatly appreciated.

Recognition must also go the Watershed Landcare at Mudgee for their assistance in contacting landholders in the area and also for their keen interest and support.

REFERENCES


# APPENDIX 1 – PROGRAM 2014 MEA STUDENT CONFERENCE

**Date:** 2014 MEA Student Conference, 24 October  
**Venue:** The Vines Room, National Wine Centre, Adelaide

<table>
<thead>
<tr>
<th>Time</th>
<th>Activity</th>
</tr>
</thead>
<tbody>
<tr>
<td>08:30 – 08:45</td>
<td>Welcome &amp; Opening of the Symposium (E. Chancia &amp; P. Dowd)</td>
</tr>
<tr>
<td>08:45 – 10:30</td>
<td>Session 1</td>
</tr>
<tr>
<td>08:45 – 09:00</td>
<td>Russell Brooks, John Forsyth, Ryan Fuller, Zachary Mcleay (UA): Simulation and animation of an Australian underground mine</td>
</tr>
<tr>
<td>09:00 – 09:15</td>
<td>Holly Kiely (WASM): A critical utilisation and availability analysis of La Mancha’s underground long-hole production drill rig</td>
</tr>
<tr>
<td>09:15 – 09:30</td>
<td>Cameron White (UQ): Mount Thorley-Warkworth grader fleet analysis</td>
</tr>
<tr>
<td>09:30 – 09:45</td>
<td>Jason Jung (UNSW): Asteroid Accessibility Rating System for Asteroid Mining</td>
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<tr>
<td>09:45 – 10:00</td>
<td>Harrison Berry, Craig Bridgman, Bartholomew Gardner, J. Schiller (UA): Simulation and animation of an Australian surface mine</td>
</tr>
<tr>
<td>10:00 – 10:15</td>
<td>Daniel Htwe (WASM): An implementation of mixed integer linear Programming for open pit optimization</td>
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<tr>
<td>10:15 – 10:30</td>
<td>Jack Butler (UQ): Squeezing ground prediction at Mount Isa Mines Northern 3500 Orebody via the rockwall condition factor</td>
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<td>10:30 – 10:50</td>
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<td>Session 2</td>
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<td>Claire Tonkin (UNSW): Hydrogeological character of faults of the Southern Coalfields</td>
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<td>11:20 – 11:35</td>
<td>Kelsay Roberts (WASM): The investigation of the optimal stemming length for 229 mm diameters charge holes on a standard production blast at Rio Tinto Brockman 4</td>
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<td>11:35 – 11:50</td>
<td>Zoe Uren (UQ): Dozer push optimisation software for Commodore Coal Mine</td>
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<tr>
<td>11:50 – 12:05</td>
<td>Kelly Williams (UNSW): Assessing the suitability of recycled material as a timber substitute in drawpoint design</td>
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<td>12:05 – 12:20</td>
<td>Amy Royle, Zhen Yang, Yue Zhao (UA): Progressive damage mechanism of rocks subjected to cyclic loading</td>
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<td>Jivan Silva, Thomas Chaplin (UQ): Optimisation of rib and roof support</td>
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<td>14:00 – 14:15</td>
<td>Feng Zhang (UNSW): Analysis of the spectral characteristics of the froth phase</td>
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<td>14:15 – 14:30</td>
<td>Xiang Ji, Jialun Qi, Chuanwei Wang, Zhongyu Xu (UA): Rock fatigue damage evaluation under cyclic loading using acoustic emission</td>
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<td>14:30 – 14:45</td>
<td>Audie Trutwein, Miguel Bachiller (WASM): Pre-scoping feasibility study of re-mining the Brownfield Teutonic Bore Project</td>
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<td>14:45 – 15:00</td>
<td>Dylan Wedel (UQ): The effectiveness of rapid stone dust compliance testing in Central Queensland coals</td>
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<tr>
<td>15:00 – 15:15</td>
<td>Heather Malli (UNSW): Integration of ultramafic tailings and acidic mine drainage for carbon sequestration and waste management of mining operations</td>
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<tr>
<td>15:15 – 15:30</td>
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## APPENDIX 2 – PRESENTERS AT MEA STUDENT RESEARCH CONFERENCES, 2009–2014

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<th>Presentation title</th>
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<td>2014</td>
<td>UoA</td>
<td>Craig Bridgman</td>
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<tr>
<td></td>
<td></td>
<td>Ashton Ingerson</td>
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<td>Zachary Mcleay</td>
<td>Simulation and animation of an Australian underground mine</td>
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<td>Jialun Qi</td>
<td>Rock fatigue damage evaluation under cyclic loading using acoustic emission</td>
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<td>Progressive damage mechanism of rocks subjected to cyclic loading</td>
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<td>UNSW</td>
<td>Jason Jung</td>
<td>Asteroid accessibility rating system for asteroid mining</td>
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<td>Maoyi Hu</td>
<td>Augmented Reality for improving ore selection following blasting</td>
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<td>Holly Kiely</td>
<td>A critical utilisation and availability analysis of La Mancha’s underground long-hole production drill rig</td>
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<td>Kelsey Roberts</td>
<td>The investigation of the optimal stemming length for 229 mm diameters charge holes on a standard production blast at Rio Tinto Brockman 4</td>
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<td>2013</td>
<td>UoA</td>
<td>Paul Bryson</td>
<td>Deep drilling performance estimation using rock mass characterisation</td>
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<td></td>
<td></td>
<td>Jingyu Chen</td>
<td>Simulation and animation of a surface gold mine</td>
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<td>Jesse Clark</td>
<td>Probabilistic stability analysis of rock excavations</td>
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<td>Mathew Harding</td>
<td>Deep open-pit mining: rock haulage optimisation</td>
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<td>Cassandra Lazo Olivares</td>
<td>The study of the strength and deformability of rocks under random and systematic cyclic loading</td>
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<tr>
<td></td>
<td>UNSW</td>
<td>Prudence Fischer</td>
<td>Identify and optimize parameters influencing blast performance within basal till material with ground water inflow</td>
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<tr>
<td></td>
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<td>Jamie Jongebloed</td>
<td>Coal project valuation – a hedonic pricing approach</td>
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<td>Eric Law</td>
<td>The effects of particle stabilisers on bubble generation, coalescence an breakup</td>
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<tr>
<td></td>
<td></td>
<td>Stefan Skorut (equal third prize)</td>
<td>Optimising cut-off grade strategy for seabed massive sulphide operations</td>
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<td></td>
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<td>Jake Small (second prize)</td>
<td>Time benefit analysis of through-seam blasting</td>
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<td>UQ</td>
<td>Nicholas Butel (fourth prize)</td>
<td>Estimation of in situ rock strength using geophysics</td>
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<tr>
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<td>Yanhao Hui</td>
<td>Workforce hazard mapping and its optimization support design at North Goonyella</td>
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<td>Robert Lucas</td>
<td>Optimisation of waste-dump lift heights for pre-strip operations at Meandu Mine, Queensland</td>
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<td>Charles Smith</td>
<td>A critique of SMU sizing at Century Mine to optimize productivity with dilution</td>
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<td>Gareth Steel</td>
<td>Analysis of heat reduction opportunities for the George Fisher north extension project decline</td>
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<td>WASM (Curtin)</td>
<td>Daniel Boxwell</td>
<td>Using artificial neural network (ANN) to predict overbreak at Plutonic Gold Mine</td>
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<tr>
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<td>Mark Kong</td>
<td>Aboriginal land rights – a Western Australia case-study</td>
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<td>He Ren (equal third prize)</td>
<td>Using clustering method for block aggregation in open pit mining</td>
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<td></td>
<td>Kirstan Lee (first prize)</td>
<td>Comparison of methods for opening initial voids in longhole open-stopping blasting</td>
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<tr>
<td></td>
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<td>Yu Yang</td>
<td>Surface fauna as a new mining exploration method- fact or fiction?</td>
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<td>2012</td>
<td>UoA</td>
<td>Dermott Sundquist and Davood Jafari</td>
<td>Multi-objective optimisation of mining-metallurgical systems</td>
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<tr>
<td></td>
<td>UNSW</td>
<td>Paul Carmichael</td>
<td>An investigation into semi-intact rock mass representation for physical modelling of block caving mechanics zones</td>
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<td></td>
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<td>McLeod McKenzie</td>
<td>Prediction and modelling of blast vibration and its effects at Glenella Colliery</td>
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<td>UQ</td>
<td>Tim Graham (third prize)</td>
<td>Strategic project risk management for an emerging miner</td>
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<td>Vanessa Collins</td>
<td>Maximising production rates at Brockman 4 by minimising truck delays at the crusher</td>
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<td>Casey Costello (first prize)</td>
<td>Grizzly modifications at Ridgeway Deeps block cave gold mine</td>
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<td>Brenton Goves (second prize)</td>
<td>Continuous surface miner operations at Fortescue</td>
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<td>WASM (Curtin)</td>
<td>Seung Hyeon Lee</td>
<td>Comparing neural networks with JKSimblast prediction model in purpose of optimal ground vibration induced by blasting</td>
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<td>Vadim Strukov</td>
<td>A comparative study of truck cycle time prediction methods</td>
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<td>Weilin Wang</td>
<td>ANSYS for stress analysis of underground structures</td>
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<td>James Boffo</td>
<td>Acoustic emission monitoring of impregnated diamond drilling for deep exploration</td>
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<td>Owen Riddy (first prize)</td>
<td>Developing a truck allocation model for Bengalla Mine</td>
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<td>Brendan Murphy</td>
<td>Development of an underground coal scheduling and simulation program</td>
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<td>WASM (Curtin)</td>
<td>Colin Thomson</td>
<td>Stoping analysis of Golden Grove under high stress conditions</td>
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<td>Adam Schwartzkopff and Daniel Hardea</td>
<td>Comparisons between three-dimensional yield criteria for fractured and intact rock</td>
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<td>James Tibbett</td>
<td>Failure criteria of three major rock types affecting Ridgeway Deeps</td>
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<td>An investigation into final landform criteria required for a safe, stable, sustainable and non-polluting landform in the Bowen Basin</td>
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<td>Analysis of the principles for sound waste dump design and placement</td>
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<td>Michael Baque</td>
<td>Optimisation of coal scheduling at Dawson Mine</td>
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<td>Kieran Rich (first prize)</td>
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